FIRST QUANTUM MINERALS LTD.

Cobre Panamá Project

Colón Province, Republic of Panamá
NI 43-101 Technical Report

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ITEM 1 SUMMARY

This Technical Report on the Cobre Panamá Project (the property) has been prepared by Qualified Persons David Gray, Michael Lawlor and Robert Stone of First Quantum Minerals Pty Ltd (FQM, the issuer or the Company). This is the second Technical Report prepared by FQM as an issuer, in relation to the subject property, and follows an initial report filed in July 2015 (FQM, July 2015).

The purpose of this Technical Report is to document updated Mineral Resource and Mineral Reserve estimates for the property, and to provide a commentary on the current project development status for the Cobre Panamá Project. The effective date for the Mineral Resource and Mineral Reserve estimates is 31st December 2018.

1.1 Project location and ownership

The Cobre Panamá Project is located in the Donoso and Omar Torrijos Herrera Districts[1] of Colón Province, Republic of Panamá, approximately 120 km west of Panamá City. The centre of the Project area occurs at latitude 8°50' North and longitude 80° 38' West. The Project is accessible from Panamá City via the Pan-American Highway and secondary paved and gravel roads. The property forms the major holding within a total of four concessions of approximately 12,995 hectares, and is the first large scale mining project to be constructed in Panamá.

When the Company acquired Inmet Mining Corporation (Inmet) in April 2013, it also acquired an 80% interest in Minera Panamá S.A. (MPSA), which held the Cobra Panamá Project concessions. At that time, Korean Panamá Mining Company (KPMC), a 50:50 joint venture between Korea Resources Corporation (Kores) and LS-Nikko Copper Inc. (LS-Nikko), held the remaining 20% share of MPSA. In August 2017, the Company increased its effective ownership of MPSA to 90% by acquiring LS-Nikko's holding of KPMC.

1.2 Project background

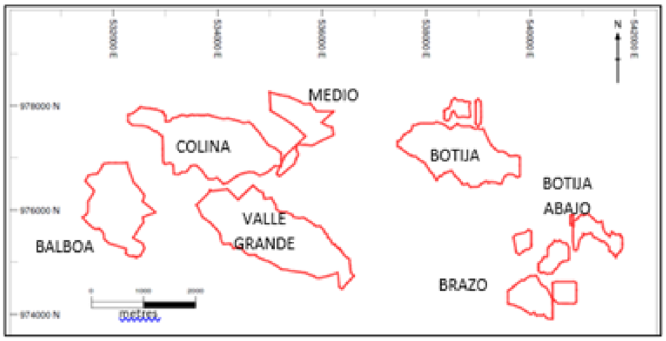
The Cobra Panamá Project will involve large scale and conventional open pit mining at up to approximately 83 Mbcm of ore and waste mined per annum. The multiple pits will be mined in an optimised sequence, with ore crushed in-pit and conveyed overland to a nearby processing plant.

The processing plant design is based upon a conventional sulphide ore flotation circuit to an initial nameplate throughput capacity of 85 Mtpa, expandable to 100 Mtpa capacity, and with differential flotation to produce separate copper and molybdenum concentrate products. Plant tailings will be directed into an area immediately to the north of the mine and plant site, and also into the depleted Botija Pit at the cessation of mining. Figure 1-1 shows the location of each of the Cobre Panamá deposits relative to each other.

Over a distance of 25 km, the copper concentrate product will be piped as a slurry to a port on the northern side of the country (on the Caribbean Sea), from where it will be loaded onto vessels for shipping to world markets. The molybdenum concentrate will be delivered to port by road and shipped in containers.

[1] Previously the Cobre Panamá Project was located completely within the Donoso District of Colón Province. Under Law No. 11 of 20th February 2018 of the National Assembly of the Republic of Panamá, the special district of Omar Torrijos Herreras was subdivided from Donoso District, and the Project now lies partly within each of the amended Districts.

Figure 1-1 Location of the Botija, Colina, Valle Grande, Balboa, Medio, Botija Abajo, and Brazo deposits



Project power will be generated by a coal-fired power station at the port and transmitted to the mine site along a new access corridor, which also incorporates the concentrate pipeline. Surplus power can be supplied into the national grid, as available.

Aspects of the Project have changed since those described in the Technical Report of July 2015 (FQM, July 2015):

- process commissioning and first concentrate production was then planned for Q4 2017
- mine development is now proposed as a sequence of terraces, rather than phased cutbacks, better suited to ultra class (UC) equipment productivity and pit water inflow management
- the Mineral Resource model and estimate, and the Mineral Reserve estimate, have been updated
- the processing plant ramp-up profile and ultimate capacity have been changed and increased, respectively

As at the end of 2018, the mine pre-strip had been completed to an extent sufficient to expose the initial Botija orebody for process plant start-up. Two in-pit primary crushers had been installed, along with their associated overland conveyors to the crushed ore stockpile at the plant site. The final stages of construction and assembly on three semi autogenous (SAG) mills and two ball mills was also completed by year end.

In February 2018, the Company announced the ramp-up expectations for the Project. Phased commissioning will commence in 2019, continue to ramp-up during 2019 towards 85 Mtpa capacity in 2021, and then to 100 Mtpa capacity by 2023. As a consequence of the increased capacity, the expected Project life is now 36 years from 2019 to 2054.

The date for declaration of commercial production is expected to be announced in late 2019.

1.3 Project approvals

The Project Environmental and Social Impact Assessment (ESIA) was approved by the *Autoridad Nacional del Ambiente* (National Authority of the Environment, ANAM; now referred to as *MiAmbiente*) in December 2011. The approved Project was for a mining operation comprising open pits at Botija, Colina, Medio and Valle Grande.

Since then, the Project definition and development scope has changed to include aspects that will need to be addressed in a new ESIA, as follows:

- mining of additional open pits at Balboa and Botija Abajo/Brazo (also referred to as the BABR Pit)
- construction of additional waste rock storage facilities and realignment of approved facilities
- increased site clearing for development of additional open pits, waste rock storage facilities and tailings expansion
- impacts on subsistence farming, on possession rights and private properties, implying physical and economic resettlement.

The expected timeframe for submitting the new ESIA for approval is mid-2024. The proposed expansion of the Project will be subject to a high level of environmental and social impact assessment by MPSA, in anticipation that the proposal will be similarly classified (Category III) by the Panamánian Government. Environmental management and controls will be applied to the development of the expansion areas that are equivalent to the controls for the existing Project.

Provided that the same process and assessment is conducted as has been for the existing Project, and with similar commitments made, there are therefore no known reasons to indicate that there will be environmental and permitting issues that could materially impact on expanded mining and tailings disposal activities.

1.4 Project development status

The Project has two main development/operating areas: the mine and plant site, and the port and power station at Punta Rincón on the Caribbean coast.

After commencing in 2015, the target for completion of the mining pre-strip at the Botija Pit was essentially achieved by the end of 2018. The starter pit boxcut had been developed, as had the excavation and formation of four in-pit primary crusher pockets. In-pit water management sumps had been excavated and equipped, and the pit was progressing towards the formation of mining terraces to suit the deployment of UC primary mining equipment.

The first UC rope shovel was assembled and began operating in March 2018, followed by the first of the UC haul trucks in November 2018. At the end of 2018, two rope shovels and 16 UC trucks were operating in the Botija Pit. An additional two rope shovels, three UC loaders and 14 UC trucks will be progressively commissioned during 2019. In addition, an ongoing pioneering and intermediate scale of mining equipment, comprising diesel hydraulic excavators plus 100 t capacity trucks and 40 t capacity 6 x 6 trucks, continue to operate in the Botija Pit on pioneering works.

Grade control drilling was underway throughout 2018 and this incorporated a geometallurgical sampling and testing programme to validate processing recovery projections for the initial production years.

As at the end of 2018, inching and lining of one of the three SAG mills and two of the ball mills was completed. Progress towards completion of other mechanical and structural works was well advanced, as was progress on storage tanks, piping works, electrical installations and instrumentation.

The coastal Punta Rincón facilities include a deep water berth for concentrate and coal shipments, a conventional ship loading wharf and a 300 MW coal fired power plant. In 2018, the first of the two power station generators (150 MW) was synchronised to the national grid. The second generator is expected to be connected to the grid in early 2019. A 120 km long dual circuit, 230 kV transmission line connecting the Project facilities (including the power plant, mine and process plant) to the national grid was completed in 2018 and was supplying backfeed electricity for early commissioning activities.

A 25 km long access road has been constructed between the mine/plant site and Punta Rincón. The port has been operational since 2015, and has enabled more than 100 vessel dockings for Project equipment and consumables deliveries.

1.5 Geology and mineralisation

The Cobre Panamá Project consists of several copper (Cu) – gold (Au) – molybdenum (Mo) – silver (Ag) porphyry mineralised systems, which were first discovered in Panamá during a regional geological survey by a United Nations Development Programme team in 1968. Exploration by numerous companies since, led to discovery of four large deposits (Botija, Colina, Valle Grande and Balboa) as well as a number of smaller deposits (Botija Abajo, Brazo and Medio). A total of 1,813 diamond drillholes totalling 348,775 m have been drilled from discovery to February 2018, with many of the deposits drilled to a spacing of 50 m by 50 m, to 200 m by 200 m or greater. Reverse circulation (RC) grade control drilling was started at the Botija deposit late in 2017. A total of 2,911 by 45 m deep RC holes have been drilled at a spacing of 15 m by 15 m.

The deposits occur at the southern margin of a large granodiorite batholith of mid-Oligocene age (36.4 Ma). Mineralisation is hosted in granodiorite, feldspar-quartz-hornblende porphyry and some andesite volcanics. Host lithologies and mineralisation are cross-cut by later dykes of either andesitic or felsic composition. Hydrothermal alteration is primarily silica-chlorite, a form of propylitic alteration. Local potassic alteration is noted to be found at Botija. Higher grade mineralisation is associated with intense quartz stockworks.

The dominant copper bearing sulphide is chalcopyrite, with only minor bornite. Sulphide mineralisation is disseminated or as micro-veinlets and quartz-sulphide stockworks. Traces of molybdenite are commonly found in quartz veinlets. There is no evidence of supergene copper enrichment at Botija, Colina, Valle Grande or Balboa. However, at Brazo, supergene mineralisation, consisting of chalcocite-coated pyrite and rare native copper, occurs to a depth of at least 150 m. Some local supergene gold enrichment has also been identified at Colina.

1.6 Metallurgical summary

The predominantly copper/molybdenum sulphide ore is amenable to conventional differential flotation processing, with lesser gold and silver recovered into the copper and gravity concentrate.

Various metallurgical test work programmes have been undertaken on the Cobre Panamá Project since 1968, commensurate with the various levels of preliminary feasibility and prefeasibility studies that were completed up until 1998.

In 1997, an extensive programme of metallurgical testing was designed to confirm earlier studies on the metallurgical response of the Botija and Colina ores. Work included grinding, flotation, dewatering and mineralogical testing. Further testing was completed, including locked-cycle flotation testwork and modal analysis to assist in defining grind requirements for both rougher and cleaner flotation. Copper-molybdenum separation by means of differential flotation was also tested.

Confirmatory batch laboratory flotation testwork was conducted during 2014 by ALS Metallurgy in Perth, Western Australia (ALS Metallurgy, 2014).

Based on all of this testwork, variable processing recovery relationships were determined for copper and gold, whilst fixed recovery values were determined for molybdenum and silver. The design recoveries vary for each deposit, as summarised in Table 1-1.

Table 1-1 Cobre Panamá process recovery relationships and values

Deposit	Recovery			
	Cu (%)	Mo (%)	Au (%)	Ag (%)
Botija	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Colina	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Medio	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Valle Grande	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775))-4)	52.0%	MIN(80,MAX(0,(15.993*LOG(auppm))+92.138))	47.3%
Balboa	MIN(96,((2.4142*LOG(cutpct))+92.655))	55.0%	MAX(0,MIN(80,(7.6009*LOG(auppm))+85.198))	40.0%
Botija Abajo	6.6135*Ln(Cu%) + 92.953	55.0%	50.0%	30.0%
Brazo	6.6135*Ln(Cu%) + 92.953	55.0%	50.0%	30.0%

During 2018, a confirmatory geometallurgical testwork programme was commenced using grade control samples to validate the above processing recovery relationships, especially in regard to the ore from initial mining horizons. At this time, however, an update to the processing recovery relationships is not warranted.

1.7 Mineral Resource summary

Block model resource estimates for Botija, Valle Grande, Colina, Medio, Balboa, Botija Abajo and Brazo were completed in January 2014 by consultants from Optiro Pty Ltd (Optiro) and by FQM geologists. A review of sample preparation methodology, sample analyses and security was completed by Optiro in support of these Mineral Resource estimates. Since these estimates, FQM has updated the Botija Mineral Resource by including the added RC grade control drilling results. All of this work was completed under the supervision of David Gray (QP) of FQM.

The Mineral Resource estimates have used drillhole sample assay results together with geology models that relate to the spatial distribution of copper, molybdenum, gold, and silver mineralisation. Block grade estimation parameters have been defined based on the geology, drillhole spacing, and geostatistical analysis of the data. Block grade estimation is by ordinary kriging into an optimal panel size, as considered appropriate for the distribution of sample data and the deposit type. Post-processing by local uniform conditioning of the copper and gold panel estimates has provided estimates based on a selective mining unit block size of 10 mE by 25 mN on 15 m benches or 15 mE by 15 mN by 15 m benches; this is considered appropriate to the expected ultra class scale of mining. Potentially deleterious elements [arsenic (ppm), bismuth (ppm), iron (%), sulphur (%), lead (ppm) and zinc (ppm)] were also estimated by ordinary kriging.

The Mineral Resource estimates have been classified according to the drilling density, geological confidence, and confidence in the panel grade estimate. They have been reported in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014), which in turn complies with the guidelines of the Australasian JORC Code (JORC, 2012). The resulting Mineral Resources have been stated for a 0.15% copper cut-off grade as per Table 1—2. The Mineral Resources have been reported inclusive of the Mineral Reserve. Botija Mineral Resources have been depleted of mining volume as at 31st December 2018.

Table 1-2 Cobre Panamá Mineral Resource statement, at 31st December 2018, using a 0.15% copper cut-off grade

Deposit	Category	Tonnes (Mt)	TCu (%)	Mo (%)	Au (g/t)	Ag (g/t)
Botija	**Measured**	310	0.47	0.008	0.11	1.44
Botija	**Indicated**	660	0.36	0.007	0.07	1.13
Colina	**Indicated**	1,032	0.39	0.007	0.06	1.58
Medio	**Indicated**	63	0.28	0.004	0.03	0.96
Valle Grande	**Indicated**	602	0.36	0.006	0.04	1.37
Balboa	**Indicated**	647	0.35	0.002	0.08	1.37
Botija Abajo	**Indicated**	114	0.31	0.004	0.06	0.93
Brazo	**Indicated**	228	0.36	0.004	0.05	0.81
Total Meas. plus Ind.		**3,657**	**0.37**	**0.006**	**0.07**	**1.34**
Botija	**Inferred**	198	0.23	0.004	0.05	0.87
Colina	**Inferred**	125	0.26	0.006	0.05	1.20
Medio	**Inferred**	189	0.25	0.005	0.03	1.25
Valle Grande	**Inferred**	363	0.29	0.005	0.03	1.14
Balboa	**Inferred**	79	0.23	0.003	0.04	0.96
Botija Abajo	**Inferred**	67	0.27	0.005	0.06	1.25
Brazo	**Inferred**	76	0.21	0.003	0.01	0.73
Total Inferred		**1,097**	**0.26**	**0.005**	**0.04**	**1.09**

Table 1-3 Cobre Panamá Stockpile Mineral Resource statement, at 31st December 2018

Deposit	Category	Tonnes (Mt)	TCu (%)	Mo (%)	Au (g/t)	Ag (g/t)
Botija	**Indicated**	**3.3**	**0.22**	**0.004**	**0.04**	**1.18**

1.8 Mineral Reserves summary

The detailed mine planning for the Project, including conventional optimisation processes, detailed designs, surface layout planning and life of mine (LOM) production scheduling, was completed by FQM staff under the supervision of Michael Lawlor (QP) of FQM.

At the outset, conventional Whittle Four-X software was used to determine optimal pit shells for each of the various deposits. The optimisation process considered all mined ore to be processed in a conventional sulphide flotation plant, with differential processes to produce separate copper and molybdenum concentrates. The copper concentrate would contain gold and silver. The optimisations were completed on a maximum net return (NR) basis, and with recoveries to metal in concentrate based on different variable and fixed relationships for each deposit. The optimisation process considered pit slope design criteria provided by a geotechnical consultant, in addition to mining and process operating costs derived in detail by MPSA. The pit optimisations have not been updated for this Technical Report.

Geological losses were built into the regularised mine planning models to account for the presence of unmineralised dykes. These losses could be considered as “planned dilution”. In the Whittle optimisation inputs, “unplanned dilution” and mining recovery factors were included to emulate practical mining losses.

Following the optimisation, a series of pit designs were developed for the 2015 Technical Report (FQM, July 2015) using the ultimate and selected intermediate pit shells. The design for the Botija ultimate pit was updated in 2018. The ultimate pit designs were prepared in detail to match the pit shells as close as possible. Specific design criteria were followed when incorporating ramps, berms, benches and in-pit crusher pockets.

The pit development strategy has changed since the 2015 Technical Report; lesser pit shells are no longer used to define phased pit designs. Intermediate pit development planning now follows a terracing strategy which is considered to be more suited for the deployment of UC mining equipment and for the high rainfall environment.

With the adoption of the terracing strategy, detailed life of mine production scheduling was then completed to demonstrate an achievable mine plan. This allows the reporting of a Mineral Reserve as stated in Table 1-4, and in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014), which in turn complies with the guidelines of the Australasian JORC Code (JORC, 2012). In addition to the insitu Mineral Reserve, there is an additional ore stockpile accumulated during the mining pre-strip. The additional Mineral Reserve for the ore stockpile is listed in Table 1-5.

Table 1-4 Cobre Panamá Project Mineral Reserve statement, at 31st December 2018

MINERAL RESERVE AT 31st DECEMBER 2018 (Cu = $3.00/lb, Mo = $13.50/lb, Au = $1,200/toz, Ag = $16.00/toz)										
		Insitu Mining Inventory								
Pit	Class	Mtonnes	TCu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	TCu metal (kt)	Mo metal (kt)	Au metal (koz)	Ag metal (koz)
BOTIJA	Proved	323.2	0.45	75.98	0.11	1.43	1,467.0	24.6	1,106.3	14,886.7
	Probable	641.4	0.35	68.28	0.08	1.13	2,220.2	43.8	1,512.4	23,357.7
	Total P+P	**964.6**	**0.38**	**70.86**	**0.09**	**1.23**	**3,687.2**	**68.4**	**2,669.6**	**38,188.5**
COLINA &	Proved									
MEDIO	Probable	981.3	0.39	66.98	0.06	1.61	3,870.5	65.7	1,986.8	50,646.8
	Total P+P	**981.3**	**0.39**	**66.98**	**0.06**	**1.61**	**3,870.5**	**65.7**	**1,986.8**	**50,646.8**
VALLE GRANDE	Proved									
	Probable	541.1	0.37	67.43	0.05	1.42	2,016.0	36.5	805.2	24,637.3
	Total P+P	**541.1**	**0.37**	**67.43**	**0.05**	**1.42**	**2,016.0**	**36.5**	**805.2**	**24,637.3**
BALBOA	Proved									
	Probable	437.1	0.35	16.10	0.08	1.36	1,509.0	7.0	1,126.9	19,168.2
	Total P+P	**437.1**	**0.35**	**16.10**	**0.08**	**1.36**	**1,509.0**	**7.0**	**1,126.9**	**19,168.2**
BABR	Proved									
	Probable	219.7	0.40	41.31	0.07	0.87	882.1	9.1	527.5	6,163.9
	Total P+P	**219.7**	**0.40**	**41.31**	**0.07**	**0.87**	**882.1**	**9.1**	**527.5**	**6,163.9**
TOTAL	Proved	323.2	0.45	75.98	0.11	1.43	1,467.0	24.6	1,106.3	14,886.7
	Probable	2,820.5	0.37	57.48	0.07	1.37	10,497.7	162.1	6,009.6	123,918.1
	Total P+P	**3,143.7**	**0.38**	**59.38**	**0.07**	**1.37**	**11,964.7**	**186.7**	**7,116.0**	**138,804.8**

As at the end of December 2018, the total Mineral Reserve inclusive of the stockpile inventory is estimated as 3,147.1 million tonnes at 0.38% TCu, 59.36 ppm Mo, 0.07 g/t Au and 1.37 g/t Ag. The estimate for the insitu pit inventory is entirely within the Measured and Indicated Mineral Resource estimate.

The reported Mineral Reserve is based on an economic cut-off grade which accounts for the longer-term copper metal price projection of $3.00/lb ($6,615/t).

Table 1-5 Cobre Panamá Project Stockpile Mineral Reserve statement, at December 2018

MINERAL RESERVE AT 31st DECEMBER 2018 (Cu = $3.00/lb, Mo = $13.50/lb, Au = $1,200/toz, Ag = $16.00/toz)										
		Stockpile Inventory								
Pit	Class	Mtonnes	TCu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	TCu metal (kt)	Mo metal (kt)	Au metal (koz)	Ag metal (koz)
BOTIJA	Proved									
	Probable	3.3	0.22	43.12	0.04	1.18	7.4	0.1	4.5	126.9
	Total P+P	**3.3**	**0.22**	**43.12**	**0.04**	**1.18**	**7.4**	**0.1**	**4.5**	**126.9**

1.9 Production schedule

Features of the LOM mining and production schedule associated with the detailed pit designs are as follows:

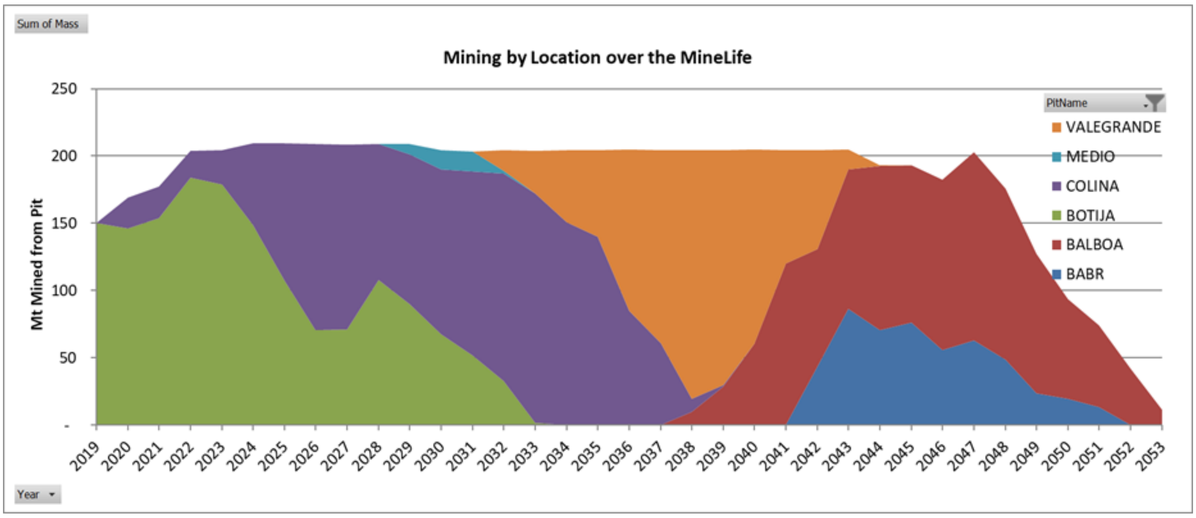
- Mining (ie, the pre-strip period) commenced in July 2015 and processing commences in Q1 2019. From 2015, the Project life is 39 years to 2054.
- The total material to be mined from all pits from the end of 2018, amounts to 6,306.1 Mt (2,436.7 Mbcm), of which 3,143.7 Mt is ore (including high grade ore, medium grade ore, low grade ore and saprock ore) and 3,157.8 Mt is waste (including saprolite, saprock waste and mineralised waste).
- The starting stockpile balance as at the end of 2018 is 3.3 Mt at a grade of 0.22% Cu.
- The direct feed ore is 2,813.1 Mt at a grade of 0.39% Cu and 334.0 Mt at a grade of 0.21% Cu is active and long term stockpile reclaim.
- The direct feed ore from the open pits is 89% of the total plant feed tonnage.
- The total high grade and medium grade ore mined to active stockpiles and reclaimed throughout the Project life is 45.7 Mt at a grade of 0.31% Cu.
- In addition, 239.9 Mt of low grade ore at a grade of 0.17% Cu is mined to stockpile and reclaimed in the final ten years of the Project.
- 48.4 Mt of saprock ore is mined to stockpile from the outset of mining, but not reclaimed (with a degraded recovery assignment) until the final two years of the Project.
- The crusher feed ramps up from 47 Mtpa in 2019 to 85 Mtpa in 2021, and ultimately to 100 Mtpa in 2023 at which rate it remains until 2041.
- The rate drops to 75 Mtpa between 2042 and 2053.
- The average annual copper metal production in the first five years is 310.0 ktpa. For the next five years, when processing at 100 Mtpa, the average annual copper metal production rises to 377.0 ktpa. Thereafter, the annual average is 275.1 ktpa.
- The annual average by-product production is approximately 2,717 tonnes of molybdenum, 107 thousand ounces of gold and 1,701 thousand ounces of silver.
- The overall life of mine strip ratio (tonnes) is 1 : 1.

Table 1-6 summarises the life of mine production schedule, whilst Figure 1-2 shows the life of mine mining sequence. The mined ore and crusher feed grades and insitu metal in this table are diluted.

Table 1-6 Cobre Panamá Project life of mine production schedule

Year	Mined Ore	Grade	Metal	Crusher Feed	Grade	Metal	Cu Metal Recovery		HG/MG Stockpiles			LG Stockpiles			Saprock Ore Stockpiles			Waste		Ore+Waste	
	(Mt)	(% Cu)	(kt Cu)	(Mt)	(% Cu)	(kt Cu)	(%)	(kt Cu)	On (Mt)	Off (Mt)	Bal. (Mt)	On (Mt)	Off (Mt)	Bal. (Mt)	On (Mt)	Off (Mt)	Bal. (Mt)	(Mt)	Strip Ratio	(Mbcm)	(Mt)
<2019	3.3	0.22	7.4						3.3												
2019	76.0	0.31	235.9	47.2	0.38	177.6	81.5	144.8	17.8	9.8	11.4	22.0	5.2	16.9	3.8		3.8	62.7	0.83	57.3	138.7
2020	94.2	0.36	340.8	81.6	0.40	323.1	91.0	293.9	2.6	10.0	4.0	19.8		36.7	0.3		4.1	74.7	0.79	67.9	169.0
2021	101.0	0.36	363.3	85.0	0.40	342.8	91.2	312.6	0.0	3.7	0.3	19.6		56.3			4.1	76.1	0.75	68.2	177.1
2022	109.8	0.40	438.3	90.2	0.45	406.8	91.9	373.7	1.4	1.2	0.5	19.4		75.7			4.1	94.2	0.86	77.8	204.0
2023	124.8	0.41	506.6	100.0	0.46	461.4	92.1	424.9	5.5	3.9	2.2	22.0		97.7	1.0		5.2	79.3	0.64	78.4	204.1
2024	109.1	0.42	455.2	100.0	0.44	436.7	91.9	401.5	0.2		2.3	6.0		103.7	3.0		8.1	100.4	0.92	81.4	209.5
2025	110.9	0.40	446.4	100.0	0.42	420.3	91.6	385.1		1.1	1.2	5.6		109.3	6.4		14.6	98.3	0.89	82.9	209.3
2026	123.8	0.38	466.3	100.0	0.41	412.5	91.2	376.0	1.4		2.6	16.1		125.4	6.3		20.9	85.3	0.69	79.9	209.1
2027	123.4	0.34	417.4	100.0	0.37	370.0	90.5	334.9	0.0	1.2	1.5	18.4		143.8	6.2		27.1	85.1	0.69	82.2	208.4
2028	108.2	0.41	445.1	100.0	0.42	423.5	91.5	387.5	2.3		3.8	3.5		147.2	2.4		29.5	100.9	0.93	80.0	209.1
2029	119.8	0.40	480.9	100.0	0.45	449.3	91.8	412.3	0.2		4.0	18.0		165.2	1.7		31.2	89.3	0.75	80.6	209.1
2030	116.3	0.38	438.6	100.0	0.42	417.3	91.3	380.8		3.3	0.6	18.3		183.5	1.3		32.4	88.0	0.76	79.4	204.3
2031	111.1	0.40	443.8	99.6	0.43	427.1	91.5	390.7		0.6	0.0	11.6		195.1	0.5		32.9	92.2	0.83	78.0	203.2
2032	102.8	0.43	446.0	100.0	0.44	439.5	91.6	402.6	1.8		1.8	0.8		195.9	0.3		33.2	101.4	0.99	78.3	204.2
2033	103.7	0.37	385.5	100.0	0.38	376.9	90.7	341.9	2.2		3.9	0.9		196.9	0.6		33.8	100.1	0.97	77.0	203.8
2034	102.8	0.37	383.8	99.7	0.38	381.6	90.8	346.3		3.9	0.0	6.9		203.7	0.1		33.9	101.4	0.99	76.9	204.2
2035	104.8	0.34	360.9	100.0	0.35	347.2	90.2	313.2	0.2		0.2	0.3		204.0	4.3		38.3	99.4	0.95	77.0	204.2
2036	110.6	0.36	395.1	100.0	0.37	372.2	90.6	337.1	2.0		2.2	7.0		211.0	1.5		39.8	94.2	0.85	77.2	204.7
2037	114.5	0.36	414.7	100.0	0.39	385.1	90.8	349.6	1.0		3.2	10.2		221.2	3.4		43.1	89.6	0.78	77.2	204.2
2038	101.6	0.38	381.6	100.0	0.38	377.1	90.8	342.4		2.0	1.1	0.8		222.0	2.8		45.9	102.6	1.01	78.7	204.2
2039	106.5	0.39	410.9	100.0	0.39	391.0	91.1	356.0	1.5		2.6	4.2		226.2	0.8		46.8	97.7	0.92	79.0	204.2
2040	105.6	0.33	347.7	100.0	0.34	340.4	90.2	307.1		1.2	1.5	6.3		232.5	0.5		47.3	99.1	0.94	80.6	204.7
2041	103.6	0.35	362.0	100.0	0.35	354.3	90.4	320.4	2.1		3.6	1.5		234.0	0.0		47.3	100.5	0.97	80.4	204.2
2042	81.9	0.33	268.4	85.0	0.33	279.6	89.7	250.9		3.4	0.2			234.0	0.3		47.5	122.2	1.49	80.7	204.1
2043	75.5	0.33	249.7	75.0	0.33	247.9	88.7	220.0	0.1		0.3	0.0		234.0	0.4		47.9	129.2	1.71	78.3	204.7
2044	63.1	0.34	214.7	75.0	0.31	233.6	88.1	205.8		0.0	0.3		11.9	222.1	0.0		47.9	130.2	2.06	77.4	193.3
2045	62.0	0.35	214.0	75.0	0.31	235.5	88.5	208.4		0.3	0.0		13.1	209.0	0.4		48.3	131.2	2.12	74.2	193.1
2046	50.7	0.36	180.5	75.0	0.29	219.2	89.2	195.5	0.0		0.0	0.7	25.0	184.7	0.0		48.3	131.7	2.60	72.7	182.5
2047	66.4	0.35	234.5	75.0	0.33	248.0	89.1	220.9		0.0			8.7	176.0	0.1		48.4	136.3	2.05	78.4	202.7
2048	66.3	0.39	256.7	75.0	0.36	270.6	89.4	241.9					8.7	167.3			48.4	109.4	1.65	67.1	175.7
2049	61.0	0.36	219.9	75.0	0.32	242.2	89.1	215.7					14.0	153.3			48.4	65.9	1.08	47.8	127.0
2050	56.7	0.40	229.4	75.0	0.34	258.6	89.5	231.3					18.3	135.0			48.4	37.0	0.65	35.4	93.7
2051	37.4	0.39	147.0	75.0	0.28	207.0	88.6	183.4				0.0	37.6	97.5			48.4	36.8	0.98	28.0	74.2
2052	27.2	0.31	84.7	75.0	0.21	161.0	88.1	141.8					47.8	49.7			48.4	14.6	0.54	15.8	41.8
2053	10.7	0.55	58.8	75.0	0.25	185.9	84.0	156.1					49.7			14.7	33.8	0.8	0.07	4.4	11.4
2054	0.0			33.8	0.33	110.3	71.8	79.2								33.8					
Diluted	**3,147.1**	**0.37**	**11,732.7**	**3,147.1**	**0.37**	**11,732.7**	**90.2**	**10,586.4**	**45.7**	**45.7**		**239.9**	**239.9**		**48.4**	**48.4**		**3,157.8**	**1.00**	**2,436.7**	**6,301.6**

Figure 1-2 Graph showing mining sequence



1.10 Environmental and social summary

The Project site is located in the Donoso and Omar Torrijos Herrara Districts of Panamá, an area of recognised high biodiversity which is subject to heavy tropical rainfall. The location is also relatively isolated, undeveloped and sparsely populated (some 3,300 people in nearby villages and ranches). Subsistence farming is the primary occupation for the local people and the area has been faced with increasing deforestation and changes in land use. Due to the high annual rainfall, increased water erosion and sedimentation have resulted.

Extensive environmental studies and social impact assessments were completed between 2007 and 2012 for the 2010 ESIA process. Over 600 impacts and management commitments are addressed in the ESIA, during the construction, operations and closure phases.

An environmental management system (EMS) has been developed to include the environmental and social management plans and commitments.

During 2017, *MiAmbiente* (the Panamánian environmental authority) announced the establishment of the Donoso Multiple-Use Protected Area and a Management Plan was being developed by the Government of Panamá.

In 2018, the Company continued to implement it's environmental and management plans to meet ESIA commitments. No material environmental incident was reported at Cobre Panamá up to 2018 and no notices of violation or penalties were imposed by any applicable regulatory authority.

1.11 Capital and operating cost estimates

Table 1-7 lists the development capital costs included in the Project cashflow model. $6,300 M is the updated estimate for completion of the Project to 85 Mtpa capacity. This includes an amount of $6,070 M which has been spent. The new estimate is a reduction from the $6,425 M that was reported in the 2015 Technical Report (FQM, July 2015), and arises through a combination of construction cost savings, construction synergies and productivity improvements.

Table 1-7 shows an additional capital cost in 2019 amounting to $240 M, required to enable commencement of the expansion to 100 Mtpa capacity, along with initial development and engineering work allowing mining to proceed to the Colina Pit[2].

Table 1-7 Development capital cost provisions

Capital Item		< 2019	2019	2020	2021	2022	2023	2024	2025	2026	TOTAL
Development capital											
Project capital for 85 Mtpa plant	USD$M	$6,070.0	$230.0								**$6,300.0**
100 Mtpa plant upgrades	USD$M		$90.0	$87.0	$110.0	$40.0					**$327.0**
Colina pioneering, access, engineering, IPC and OVC	USD$M		$45.0	$6.0		$45.0	$16.0				**$112.0**
River diversion works for Colina	USD$M							$33.0	$33.0	$34.0	**$100.0**
Site and other capital	USD$M		$105.0	$7.0	$12.0	$4.0	$2.0				**$130.0**
Total development capital	**USD$M**	**$6,070.0**	**$470.0**	**$100.0**	**$122.0**	**$89.0**	**$18.0**	**$33.0**	**$33.0**	**$34.0**	**$6,969.0**
Total development capital to end 2019, and from 2020	**USD$M**	**$6,540.0**		**$429.0**							**$6,969.0**

$105 M of the $240 M in 2019 relates to site and other capital expenses, inclusive of allowances for:

- completion of the Botija Pit pre-strip
- a brought forward amount for completion of the Botija Pit diversion channel and an associated water control embankment
- pioneering mining equipment for the Colina pre-strip
- Colina infill and geotechnical drilling
- early deployment of trolley assist haulage and required infrastructure
- mine dewatering expenses
- additional spares

Table 1-7 lists the continuing plant expansion and related engineering cost estimates from 2020. Listed site and other capital items from 2020 include allowances for miscellaneous infrastructure at the port, power plant and accommodation facilities.

Sustaining capital cost allowances have been included in the Project cashflow model, for ongoing mining, processing and G&A expenditure. The average annual sustaining capital figure is approximately $69 M. A cashflow model closure cost provision for $124.2 M has been split over the final six years of the Project.

A detailed derivation of operating and metal costs was originally completed by MPSA prior to the 2015 Technical Report (FQM, July 2015). This derivation has been updated to yield:

- overall average ore mining cost = $1.94/t
- overall average waste mining cost = $1.76/t
- overall average processing operating cost (inclusive of port fixed cost) = $3.84/t processed
- overall mine and plant fixed cost (equivalent G&A cost in variable terms) = $0.62/t processed
- stockpile reclaim cost = $1.19/t reclaimed
- copper metal cost:
  = metal price x royalty + Cu metal cost
  = $3.07 x 2% + $0.338 = $0.40/lb Cu

[2] Initial Colina expenditure has been brought forward due to the sooner requirement of plant feed from Colina arising from the capacity expansion (ie, not envisaged in the 2015 Technical Report).

- molybdenum metal cost:
  = metal price x royalty + Mo metal cost
  = $8.83 x 2% + $1.334 = $1.46/lb Mo
- gold metal cost:
  = metal price x royalty + Au refining cost x Au payable
  = $1,310 x 2% + $5.10 x 90% = $28.08/oz Au ($409.46/lb Au)
- silver metal cost:
  = metal price x royalty + Ag refining cost x Ag payable
  = $18.87 x 2% + $0.44 x 90% = $0.73/oz Ag ($10.63/lb Ag)

The updated mining costs listed above take account of ore haulage to designed in-pit crusher locations, and to waste dumping locations. The unit cost estimates include a factored allowance reflecting the adoption of trolley-assisted haulage.

1.12 Economic analysis

An economic analysis in the form of a basic cashflow model to support the Mineral Reserve estimate is summarised in Table 1-8 (the cashflow is listed annually in Item 22). Table 1-9 provides a summary of the average annual physicals and unit costs. The model shows the indicative cashflow and does not replace a more comprehensive financial model that exists for the Project. The annual revenues are calculated from late 2018 consensus metal price projections, for which the long term values (beyond 2023) are as follows:

- copper = $3.07/lb ($6,768/t)
- molybdenum = $8.83/lb ($19,467/t)
- gold = $1,310/oz
- silver = $18.87/oz

The modelled overall average processing recoveries (after downrating in the initial production years) are:

- copper = 90.2%
- molybdenum = 53.4%
- gold = 55.9%
- silver = 45.0%

The modelled payable metal factors are:

- copper = 96.15%
- molybdenum = 86.20%
- gold = 90.00%
- silver = 90.00%

Mining, process and G&A operating costs for 2019, in addition to metal costs for that year, have been expensed.

Table 1-8 Mineral Reserve cashflow model summary

MINERAL RESERVES - CASHFLOW SUMMARY	UNIT	TOTAL	< 2019 < Year 1	2019 to 2023 Year 1 to 5	2024 to 2028 Year 6 to 10	2029 to 2033 Year 11 to 15	2034 to 2028 Year 16 to 20	2039 to 2043 Year 21 to 25	2044 to 2048 Year26 to 30	2049 to end Year 31 to end
MINING										
Total ore	Mt	**3,143.7**		505.7	575.4	553.7	534.2	473.1	308.5	193.0
Total waste (incl. MW not reclaimed)	Mt	**3,157.8**		387.1	470.0	471.0	487.1	548.7	638.7	155.1
Total Mining	Mt	**6,301.6**		892.8	1045.4	1024.7	1021.3	1021.9	947.3	348.2
Strip ratio	W:O	**1.0**		1.3	1.2	1.2	1.1	0.9	0.5	1.2
Stockpile reclaim	Mt	**334.0**		33.7	2.3	4.0	6.0	4.5	67.8	215.8
PIT TO MILL DIRECT										
TOTAL	t	**2,813.1**		370.3	497.7	495.6	493.7	455.5	307.2	193.0
Cu	%	**0.39**		0.43	0.41	0.42	0.37	0.35	0.36	0.38
Mo	ppm	**60.19**		75.05	68.77	76.62	70.09	48.03	26.51	24.29
Au	ppm	**0.07**		0.09	0.09	0.07	0.05	0.06	0.07	0.08
Ag	ppm	**1.38**		1.37	1.36	1.51	1.53	1.34	1.11	1.28
PIT TO STOCKPILE										
TOTAL	Mt	**330.6**		135.5	77.7	58.1	40.5	17.7	1.3	0.0
Cu	%	**0.21**		0.21	0.23	0.17	0.22	0.25	0.22	0.15
Mo	ppm	**41.31**		38.33	47.74	33.16	59.64	21.10	35.12	43.90
Au	ppm	**0.05**		0.05	0.07	0.04	0.04	0.06	0.03	0.03
Ag	ppm	**1.02**		0.93	1.06	0.94	1.29	1.11	1.06	0.59
STOCKPILE TO MILL										
TOTAL	Mt	**334.0**		33.7	2.3	4.0	6.0	4.5	67.8	215.8
Cu	%	**0.21**		0.32	0.39	0.29	0.26	0.39	0.16	0.20
Mo	ppm	**41.38**		62.63	63.33	62.73	48.20	35.22	34.04	39.68
Au	ppm	**0.05**		0.07	0.05	0.03	0.03	0.05	0.04	0.05
Ag	ppm	**1.07**		1.36	1.51	1.51	1.46	1.32	0.94	1.03
TOTAL FEED TO PLANT (after mining dil'n & recovery)										
TOTAL	Mt	**3,147.1**		404.0	500.0	499.6	499.7	460.0	375.0	408.8
Cu	%	**0.37**		0.42	0.41	0.42	0.37	0.35	0.32	0.28
Mo	ppm	**58.19**		74.02	68.74	76.51	69.83	47.91	27.87	32.41
Au	ppm	**0.07**		0.09	0.09	0.07	0.05	0.06	0.07	0.07
Ag	ppm	**1.35**		1.36	1.36	1.51	1.53	1.34	1.08	1.15
AVERAGE RECOVERIES										
Cu	%	**90.2%**		90.5%	91.4%	91.4%	90.6%	90.2%	88.9%	86.5%
Mo	%	**53.4%**		50.2%	55.0%	55.0%	53.7%	52.4%	58.1%	49.1%
Au	%	**55.9%**		57.4%	58.6%	53.8%	48.3%	52.9%	60.5%	58.0%
Ag	%	**45.0%**		44.4%	47.3%	47.3%	47.3%	44.6%	39.9%	39.4%
METAL RECOVERED										
Cu	kt	**10,586.4**		1,549.9	1,885.0	1,928.3	1,688.7	1,454.4	1,072.6	1,007.5
Mo	kt	**97.8**		15.0	18.9	21.0	18.7	11.5	6.1	6.5
Au	koz	**3,876.2**		654.1	825.6	599.4	371.5	440.6	485.0	499.9
Ag	koz	**61,226.2**		7,870.2	10,334.6	11,458.2	11,593.7	8,838.5	5,193.1	5,937.9
METAL PAYABLE										
Cu	kt	**10,193.0**		1,494.5	1,817.0	1,859.2	1,625.6	1,399.0	1,030.7	967.0
Mo	kt	**84.3**		12.9	16.3	18.1	16.2	10.0	5.2	5.6
Au	koz	**3,488.5**		588.7	743.0	539.5	334.4	396.5	436.5	449.9
Ag	koz	**55,103.6**		7,083.2	9,301.2	10,312.3	10,434.3	7,954.6	4,673.8	5,344.1
GROSS REVENUE (based on varying concentrate grade)										
Cu	$M	$69,693.1		$10,787.5	$12,304.3	$12,590.5	$11,008.6	$9,474.2	$6,979.6	$6,548.4
Mo	$M	$1,680.9		$291.0	$317.3	$352.9	$314.7	$193.8	$102.0	$109.3
Au	$M	$4,565.7		$767.8	$973.2	$706.6	$437.9	$519.4	$571.7	$589.2
Ag	$M	$1,030.2		$124.2	$175.5	$194.6	$196.9	$150.1	$88.2	$100.8
Subtotal - Gross Revenue	**$M**	**$76,969.9**		**$11,970.5**	**$13,770.3**	**$13,844.5**	**$11,958.1**	**$10,337.4**	**$7,741.4**	**$7,347.8**
CAPITAL COSTS										
Development capital	$M	$6,300.0	$6,070.0	$230.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Expansion capital	$M	$669.0		$569.0	$100.0	$0.0	$0.0	$0.0	$0.0	$0.0
Sutotal - Development Capital Costs	**$M**	**$6,969.0**	**$6,070.0**	$799.0	$100.0	$0.0	$0.0	$0.0	$0.0	$0.0
SUSTAINING COSTS										
Sustaining (Mining, Processing, G&A) capital	$M	$2,493.9		$461.9	$420.9	$372.2	$371.3	$363.3	$326.3	$178.0
Closure and reclamation	$M	$124.2		$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$124.2
Subtotal - Sustaining Capital Costs	**$M**	**$2,618.1**		**$461.9**	**$420.9**	**$372.2**	**$371.3**	**$363.3**	**$326.3**	**$302.2**
OPERATING COSTS										
Mining ore	$M	$6,136.8		$886.3	$1,149.0	$1,124.2	$1,007.5	$844.0	$643.5	$482.3
Mining waste	$M	$5,539.6		$661.4	$858.8	$830.5	$830.3	$906.1	$1,141.2	$311.4
Processing	$M	$12,088.2		$1,792.4	$1,902.7	$1,861.2	$1,861.6	$1,717.8	$1,409.9	$1,542.6
G&A	$M	$1,957.9		$551.7	$267.4	$253.6	$253.7	$233.5	$190.4	$207.5
Stockpile reclaim	$M	$397.3		$40.1	$2.7	$4.7	$7.1	$5.4	$80.6	$256.7
Subtotal - Operating Costs	**$M**	**$26,119.9**		**$3,931.9**	**$4,180.6**	**$4,074.3**	**$3,960.3**	**$3,706.8**	**$3,465.6**	**$2,800.5**
METAL COSTS (TCRCs AND ROYALTIES)										
Cu	$M	$9,107.6		$1,312.2	$1,590.0	$1,618.2	$1,455.7	$1,268.8	$951.4	$911.2
Mo	$M	$315.5		$49.1	$60.8	$67.6	$60.3	$37.1	$19.5	$20.9
Au	$M	$108.8		$18.3	$23.2	$16.8	$10.4	$12.4	$13.6	$14.0
Ag	$M	$44.5		$5.5	$7.5	$8.4	$8.5	$6.4	$3.8	$4.3
Subtotal - Metal Costs	**$M**	**$9,576.4**		**$1,385.2**	**$1,681.6**	**$1,711.0**	**$1,534.9**	**$1,324.8**	**$988.4**	**$950.6**
CASHFLOW	**$M**	**$31,686.5**	**-$6,070.0**	**$5,392.5**	**$7,387.2**	**$7,687.1**	**$6,091.6**	**$4,942.6**	**$2,961.2**	**$3,294.4**

Table 1-9 Mineral Reserves cashflow model average annual physicals and costs

MINERAL RESERVES - CASHFLOW SUMMARY	UNIT	TOTAL	< 2019 < Year 1	2019 to 2023 Year 1 to 5	2024 to 2028 Year 6 to 10	2029 to 2033 Year 11 to 15	2034 to 2028 Year 16 to 20	2039 to 2043 Year 21 to 25	2044 to 2048 Year26 to 30	2049 to end Year 31 to end
ANNUAL AVERAGES - PHYSICALS										
Mining tonnage, ore plus waste	Mtpa	**180.0**		178.6	209.1	204.9	204.3	204.4	189.5	69.6
Mill feed	Mtpa	**87.4**		80.8	100.0	99.9	99.9	92.0	75.0	91.7
Cu recovered	kt	**294.1**		310.0	377.0	385.7	337.7	290.9	214.5	167.9
Mo recovered	kt	**2.717**		3.0	3.8	4.2	3.7	2.3	1.2	1.1
Au recovered	koz	**107.7**		130.8	165.1	119.9	74.3	88.1	97.0	83.3
Ag recovered	koz	**1,700.7**		1,574.0	2,066.9	2,291.6	2,318.7	1,767.7	1,038.6	989.6
ANNUAL AVERAGE UNIT COSTS										
Mining ore	$/t mined	**$1.94**		$2.29	$2.44	$2.39	$2.07	$1.54	$1.01	$3.11
Mining waste	$/t mined	**$1.76**		$1.31	$1.49	$1.50	$1.55	$1.92	$3.70	$1.61
Processing (excluding sustaining)	$/t processed	**$3.84**		$4.44	$3.81	$3.73	$3.73	$3.73	$3.76	$3.77
G&A (excluding sustaining)	$/t processed	**$0.62**		$1.37	$0.53	$0.51	$0.51	$0.51	$0.51	$0.51
Stockpile reclaim	$/t rehandled	**$1.19**		$1.19	$1.19	$1.19	$1.19	$1.19	$1.19	$1.19
Metal costs (including royalties)	$/t processed	$3.04		$3.43	$3.36	$3.42	$3.07	$2.88	$2.64	$2.33

The Project is cashflow positive from 2020 and payback on the $6,540 M capital spend occurs in 2024 (ie, payback on the capital inclusive of $6,070 M spent, $230 M of remaining capital for the 85 Mtpa expansion, and $240 M for initial 100 Mtpa expansion costs and associated development and engineering costs).

The total undiscounted cashflow for the Project, from the outset, is $31,686.5 M. As at 31st December 2018, the NPV of the projected gross operating income (ie, excluding the precious metal stream) and capital expenditure of the Project is $14,307 M, calculated assuming mid-period cash flows.

The NPV is stated on the basis that the $6,070 M of historic capital spend for the development and construction of the Project has been spent. In looking forward from the end of 2018, the adoption of an 8.5% discount rate is considered appropriate and aligned with rates recommended by the Company's independent valuation experts. With the exclusion of the historic capital spend from the discounted cashflow, the presentation of an IRR value is considered to be not applicable.

1.13 Conclusions and recommendations

It is the opinion of David Gray (QP), that the Mineral Resource classifications applied to the mineralisation at Cobre Panamá fairly reflect the levels of geological and grade confidence. While there are uncertainties with the geological and structural framework of the Cobre Panamá mineralisation, the risk to the overall estimated tonnage and grade is considered to be low.

In the opinion of Michael Lawlor (QP), the Mineral Reserve estimate reflects an achievable longer term mining plan and production sequence, and one which has taken account of a newly devised terrace mining strategy.

There is considered to be minimal risk attributable to the mining method and primary equipment selected for the Project. The method and equipment items are conventional and suitable for a large scale, bulk mining operation. The terrace mining strategy is being implemented in recognition of the need to provide spacious working areas suited to the deployment of ultra class mining equipment. Furthermore, the layout of terraces within the design pits enables an improved means of managing the potential for production disruptions due to prolonged rainfall events.

Uncertainty in operating and metal costs (ie, treatment costs, transport costs, and royalty charges), to the extent identified, poses minimal risk to the selection of optimal pit shells as the basis for all following pit design and production scheduling work supporting the Mineral Reserve estimate. Not withstanding this, there is a need to continuously review the pit optimisations so that they reflect the ongoing engineering and planning that is currently focussed on the siting of in-pit crushers and associated infrastructure. Updates on operating and metal costs should be included in future optimisations for completeness.

The 2015 Technical Report (FQM, July 2015) mentioned a need for operating cost review as the Project proceeds into the operations phase. This has been done to some extent leading up to this Technical Report, through the inclusion of short to medium term site budgeted costs in the current cashflow model. It is recommended that this cost review procedure be continued as an item of continuous improvement.

Mine geotechnical risks are considered to be manageable, largely through the adoption of the terraced bench layout, and the ability to map and analyse available exposures on interim terrace slopes before committing to final design slopes. Wide geotechnical berms are included in the overall slope designs as a risk mitigation measure. The Botija Pit has benefited from a reappraisal of pit slope design parameters now that pre-strip batters and slopes are exposed for mapping and structural review. Geotechnical drilling will commence in 2019 at Colina (the next pit mined in sequence) to provide new information updating that collected in earlier drilling programmes.

The drilling of horizontal drain holes in the Botija Pit has addressed the drained pit slope assumption made in the mine geotechnical studies to date. Piezometers are to be installed in newly drilled vertical bores to assess groundwater drawdown effectiveness. The continued drilling of horizontal drains and the monitoring of piezometer information is endorsed. It is recommended that this information be appraised before committing to expensive drilling and equipping of vertical dewatering bores.

The risk of surface water inflows to the pits has been mitigated by the design of surface diversions, the proposed stepped pit terraces and a staged/booster pumping system from pit bottom. Work is well advanced on the surface diversion channel for the Botija Pit, and planning and engineering work is underway for a similar channel that will be required for the Colina Pit.

There is essentially no new information on the processing of Cobre Panamá ores which would warrant a change to the metal recovery projections that were reported in the 2015 Technical Report (FQM, July 2015).

ITEM 2 INTRODUCTION

2.1 Purpose of this report

This Technical Report on the Cobre Panamá Project (the property) has been prepared by Qualified Persons (QPs) David Gray, Michael Lawlor and Robert Stone of First Quantum Minerals Pty Ltd (FQM, the issuer).

The purpose of this Technical Report is to document updated Mineral Resource and Mineral Reserve estimates for the property, and to provide a commentary on the current development status for the Cobre Panamá Project.

2.2 Terms of reference

This Technical Report covers all seven deposits of the Cobre Panamá Project and has been written to comply with the reporting requirements of the Canadian National Instrument 43-101 guidelines: 'Standards of Disclosure for Mineral Properties' of April 2011 (the Instrument) and with the 'Australasian Code for Reporting of Mineral Resources and Ore Reserves' of December 2012 (the 2012 JORC Code) as produced by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC).

The effective date for the Mineral Resource and Mineral Reserve estimates is 31st December 2018.

2.3 Qualified Persons and authors

The Mineral Resource estimates were prepared under the direction and supervision of David Gray (QP). Mr Gray of FQM meets the requirements of a Qualified Person according to his Certificate of Qualified Person attached in Item 28. FQM personnel completed an updated estimate for Botija in 2018, and for the Botija Abajo and Brazo deposits in 2014. Optiro Pty Ltd prepared the Mineral Resource estimates for the Colina, Valle Grande, Balboa and Medio deposits, and authored those items of this Technical Report relating to geology and Mineral Resource estimation.

The Mineral Reserve estimates were prepared under the direction of Michael Lawlor (QP), with the assistance of FQM staff. Mr Lawlor of FQM meets the requirements of a Qualified Person according to his Certificate of Qualified Person attached in Item 28. Mr Lawlor takes responsibility for those items not addressed specifically by the other QPs.

Metallurgical testing, mineral processing and process recovery aspects of this Technical Report were addressed by Robert Stone (QP). Mr Stone of FQM meets the requirements of a Qualified Person according to his Certificate of Qualified Person attached in Item 28.

Table 2-1 identifies which items of the Technical Report have been the responsibility of each QP.

2.4 Principal sources of information

Information used in compiling this Technical Report was derived from previous technical reports on the property, and from the reports and documents listed in the References item (Item 27).

Table 2-1 QP details

Name	Position	NI 43-101 Responsibility
David Gray *BSc (Geology), MAusIMM, FAIG*	Group Mine and Resource Geologist, FQM (Australia) Pty Ltd	Author and Qualified Person Items 3 – 12, 14
Michael Lawlor *BEng Hons (Mining), MEngSc, FAusIMM*	Consultant Mining Engineer, FQM (Australia) Pty Ltd	Author and Qualified Person Items 1, 2, 15 and 16, 18 to 26
Rob Stone *BSc (Hons), CEng, ACSM*	Technical Manager, FQM (Australia) Pty Ltd	Author and Qualified Person Items 13 and 17
Ian Glacken *BSc (Hons)(Geology), MSc (Geology), MSc (Geostatistics), FAusIMM(CP), CEng, MIMMM, DIC*	Principal & Director, Optiro Pty Ltd	Contributing author Items 3-12. 14

2.5 Site visits

The Qualified Persons (QPs) have visited the site, as follows:

- David Gray last visited the Project in August 2018. Mr Gray inspected drill core and drilling sites, reviewed geological, data collection and sample preparation procedures, and carried out independent data verification.
- Michael Lawlor last visited the Project in September 2018. Mr Lawlor visited all accessible areas of the site.
- Robert Stone was on site regularly through 2017 and continuously from Q3 2018, overviewing technical aspects associated with the construction of the process plant, port and TMF, and also involving himself with environmental control infrastructure requirements.

2.6 Conventions and definitions

Reference in this Technical Report to dollars or $, relates to United States dollars. Copper metal production is reported in (metric) tonnes and (imperial) pounds, where the conversion factor is 1 tonne (t) = 2,204.62 pounds (lb). Gold and silver production is reported in (troy) ounces.

The conventional chemical abbreviation for copper of Cu is used throughout this report, whilst the abbreviation for molybdenum is Mo, for gold is Au, and for silver is Ag. ASCu is used to denote Acid Soluble Copper and TCu is used to denote Total Copper.

Where not explained in the text of this report, specific terms and definitions are as listed in Table 2-2.

Table 2-2 Terms and definitions

Term	Definition	Term	Definition
bcm	bank cubic metres	µm, mm, cm, m, km	microns, millimetres, centimetres, metres, kilometres
csv	coma separated value	NPV	net present value
g, kg	grams, kilograms	oz	ounces
g/t, kg/t	grams per tonne, kilograms per tonne	P_{80}	80% passing
ha	hectares	pH	potential of hydrogen
IRR	internal rate of return	Q1, Q2, Q3, Q4	quarter 1 to 4
kWh/t	kilowatt hours per tonne	t, kt, Mt	tonnes, thousands of tonnes, millions of tonnes
lb	pounds	tpa	tonnes per annum
LOM	life of mine	MW, LG, MG, HG	mineralised waste, low grade, medium grade, high grade
Ma	mega annum (million years)	V, kV	volts, kilovolts
masl	metres above sea level	W, MW	watts, megawatts
mE, mN	coordinates: metres East, metres North	WGS	Western Geodetic System

ITEM 3 RELIANCE ON OTHER EXPERTS

The authors of this Technical Report do not disclaim any responsibility for the content contained herein.

ITEM 4 PROPERTY DESCRIPTION, LOCATION AND TENURE

4.1 Project ownership

In acquiring Inmet Mining Corporation (Inmet) in April 2013, the Company also acquired an 80% interest in Minera Panamá S.A. (MPSA), which held the Cobre Panamá Project concessions. At that time, the remaining 20% share of MPSA was held by Korean Panamá Mining Company (KPMC), a 50:50 joint venture between Korea Resources Corporation (KoRes) and LS-Nikko Copper Inc (LS-Nikko). In August 2017, the Company increased its effective ownership of MPSA to 90% by acquiring LS-Nikko's holding of KPMC.

4.2 Project description components

The Cobre Panamá Project has the following primary components, aspects and the development status of which are described in this Technical Report:

1. Seven open pit mines (with the first of these, at Botija, currently completing pre-strip) plus associated waste dumps (also referred to as waste rock storage facilities or WRSFs)[3].
2. Primary crushing carried out in pit, with crushed ore conveyed overland to stockpiles at the plant site (IPCC technology).
3. A process plant comprising secondary and pebble crushing, semi-autogenous grinding (SAG) and ball milling, and conventional flotation to produce separate copper and molybdenum concentrate products.
4. An initial tailings management facility (TMF) located north of the open pits and plant site, with the concept of subsequent tailings backfill into the depleted Botija Pit, and future expansion of the initial TMF towards the north.
5. A port facility for the receival of construction materials and operating consumables (fuel, reagents, coal, maintenance parts and spares), and the shipment of concentrates.
6. A coal-fired power generation plant at the port, and transmission line connecting the Project to the national grid.
7. A slurry pipeline for the delivery of copper concentrate to the port, and an adjoining access road and power transmission line.

Other Project attributes include:

- owner procurement, construction and management (ie, self-perform)
- owner mining
- own power generation and transmission
- own port facility

4.3 Project location

The Cobre Panamá Project is located in the Colón Province, approximately 120 km west of Panamá City (Figure 4-1). Colón is in the north central part of the Republic of Panamá, bounded by the Caribbean Sea to the north and the Coclé Province to the south. The Project area is characterised by rugged topography with heavy rainforest cover.

[3] ESIA approval is currently in place for four of the open pits.

The Project contains two main development sites; a mine and plant site located within the tenement concession boundaries, and a port site at Punta Rincón, situated on the Caribbean coast, 25 km north of the plant site, and approximately 100 km south-west of Colón City. The location of the processing plant site is N8°50’ and W80°38’ and the location of the port site at Punta Rincón is N9°02’ and W80°41’.

Figure 4-1 Cobre Panamá Project location map (source: Rose *et al*, 2012)

4.4 Mineral tenure

In February 1996, the Republic of Panamá and Minera Panamá SA (MPSA), a Panamánian subsidiary of the Company, entered into a mining concessions contract in respect of the Cobre Panamá Project.

In February 1997, Contract Law No. 9 (Law No. 9) was passed by the Panamánian National Assembly. Law No. 9 granted the status of national law to the mining concessions contract, establishing a statutory legal and fiscal regime for the development of the Cobre Panamá Project. The legal regime established by Law No.9 is supplemented by the Mineral Resources Code of Panamá. Law No. 9 had an initial twenty year term ending in 2017 with provision for two consecutive twenty year extensions.

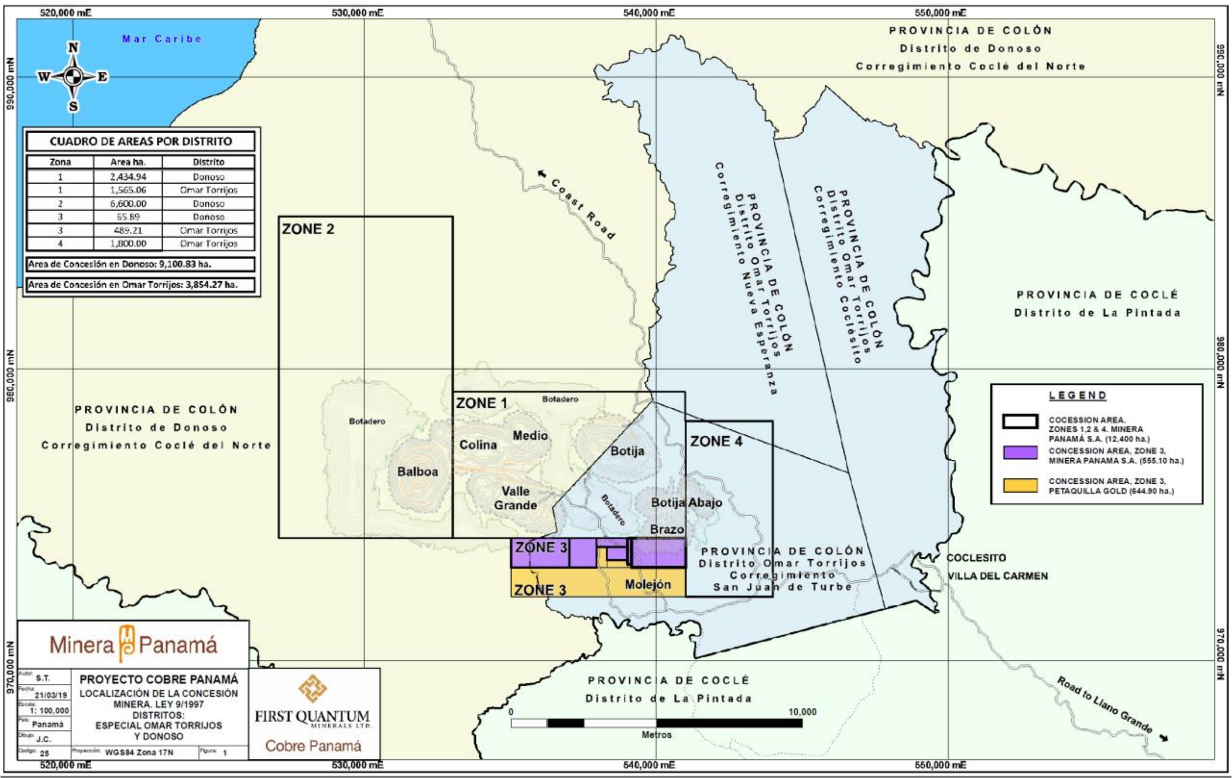
In December 2016, the Government of Panamá signed and issued Resolution No. 128 by which it extended the Law No. 9 mineral concessions for a second twenty year term commencing on 1st March 2017 (through to 28th February 2037). MPSA remains eligible for consideration of a third twenty year term commencing on 1st March 2037.

Under the mining concessions contract and Law No. 9, MPSA has the rights to explore for, extract, exploit, beneficiate, process, refine, transport, sell and market the copper, gold and other minerals on the Cobre Panamá concessions. MPSA is entitled to rights of way on state-owned lands and easements, and to use surface lands on concessions adjacent to the Cobre Panamá concessions. Furthermore, MPSA has the right to build upon, and to use and maintain such lands, in addition to easements for building, maintaining and using facilities and installations that MPSA deems convenient for the development of the Cobre Panamá concessions.

4.5 Project concessions

The property consists of four concessions totalling 12,955.1 hectares (Figure 4-2). The geographic co-ordinates for each zone are provided in Table 4-1.

Figure 4-2 Property location map – Mina de Cobre Panamá concessions



- Zona No. 1: Has a total area of 4,000 hectares and lies within the jurisdictions of Northern Coclé and San José del General, Donoso District and within the jurisdiction of San Juan de Turbe, Omar Torrijos District in the Province of Colón.
- Zona No. 2: Has a total surface area of 6,600 hectares and lies within the jurisdiction of Northern Coclé, Donoso District, Province of Colón.
- Zona No. 3: Has a total surface area of 1,200 hectares and lies within the Jurisdiction of San José del General, Donoso District and within the jurisdiction of San Juan de Turbe, Omar Torrijos District in the Province of Colón. It is contiguous to the south of Zona No. 1.
- Zona No. 4: Has a total surface area of 1,800 hectares and lies within the jurisdictions of Nueva Esperanza and San Juan de Turbe, Omar Torrijos District in the Province of Colón. Zona No. 4 is contiguous to the east of Zona No. 1 and contiguous to the east of Zona No. 3.
- The Molejón sub-concession area formed part of the Zona No. 3 concession and related to the property and mineral rights to develop the Molejón gold deposit on a stand-alone basis by Petaquilla Minerals Ltd (PML). The Molejón sub-concession covers 644.9 hectares and is not included in the 1,200 hectares of Zona No. 3[4].

[4] PML formerly had the right to explore and mine gold deposits in the larger Zona No. 3 concession area provided that they did not interfere with MPSA's ability to operate within the area. MPSA retained the right to develop any copper deposits on the Molejón sub-concession. Following a transaction dated May 2014, there is now complete separation of the current operations of PML's Molejón Gold mine and the Cobre Panamá Project.

Table 4-1 MPSA mineral concessions under Law No. 9, 1997 [Geographic Coordinates – NAD27 UTM Zone 17 (Canal Zone)]

Zone	Longitude	Latitude	Direction	Distance (m)	Area (ha)
Zona No. 1	80°41'59.02"	8°51'25.11"	East	8,000	4,000
	80°37'38.15"	8°51'25.11"	South	5,000	
	80°37'38.15"	8°48'42.07"	West	8,000	
	80°41'59.02"	8°48'42.07"	North	5,000	
Zona No. 2	80°45'14.67"	8°54'40.76"	East	6,000	6,600
	80°41'59.02"	8°54'40.76"	South	11,000	
	80°41'59.02"	8°48'42.07"	West	6,000	
	80°45'14.67"	8°48'42.07"	North	11,000	
Zona No. 3	80°40'53.80"	8°48'42.07"	East	6,000	1,200
	80°37'38.15"	8°48'42.07"	South	2,000	
	80°37'38.15"	8°47'36.85"	West	6,000	
	80°40'53.80"	8°47'36.85"	North	2,000	
Zona No. 4	80°37'38.15"	8°50'52.55"	East	3,000	1,800
	80°36'00.48"	8°50'52.55"	South	6,000	
	80°36'00.48"	8°47'36.85"	West	3,000	
	80°37'38.15"	8°47'36.85"	North	6,000	
Molejón sub-concession (now excluded from Zona No. 3	80°40'53.82"	8°48'09.90"	East	2,930	644.9
	80°39'17.97"	8°48'09.70"	North	700	
	80°39'17.92"	8°48'32.68"	East	370	
	80°39'61.87"	8°48'32.67"	South	450	
	80°39'06.20"	8°48'17.87"	East	700	
	80°38'43.29"	8°48'17.84"	North	150	

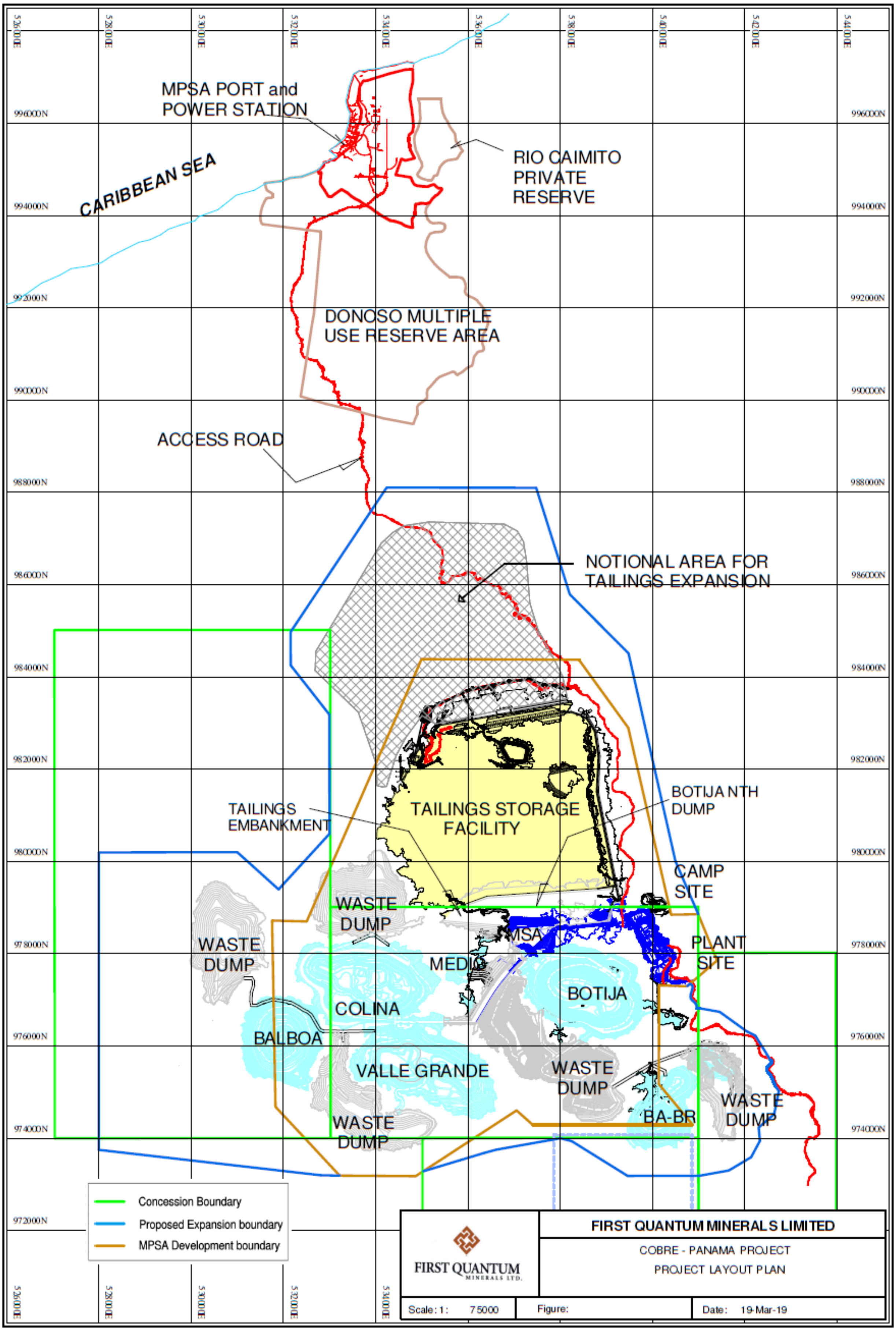
Under Law No. 9, 1997, MPSA has exercised its rights to acquire or lease state lands located within the proposed tailings storage area (Figure 4-3). MPSA has the land rights on the entire land polygon of 7,477.5 ha (shown as the orange line in Figure 4-3).

Most of the land required for the proposed port facilities at Punta Rincón, including land required for construction and permanent use of the site, was acquired by MPSA during 1998 and 2000.

Mine expansion, including mining of the Balboa, Botija Abajo and Brazo deposits and adjacent waste rock dumping, will require access to additional properties covering up to 6,800 ha. MPSA intends to initiate the acquisition of these properties in accordance with the procedures established by Law No. 9 and other applicable Panamánian laws.

Where lands are occupied, which is the case for the mine expansion area, MPSA intends to adhere to the International Finance Corporation's Performance Standard 5 (IFC PS 5) in connection with relocation and resettlement of affected persons and communities. Where Project infrastructure is located within protected areas, such as the Donoso Multiple-Use Protected Area, MPSA will conform to the requirements of IFC PS 6.

Figure 4-3 Location of open pits and facilities for the Cobre Panamá Project



4.6 Royalties and taxation

As governed by Law No. 9, MPSA is required to pay to the Government of Panamá, a 2% royalty on "Negotiable Gross Production". This is defined as "the gross amount received from the buyer due to the sale (of concentrates) after deduction of all smelting costs, penalties and other deductions, and after deducting all transportation costs and insurances incurred in their transfer from the mine to the smelter".

A land rental tax of 3 Panamánian Balboas (US$3.00) per hectare per year for the total concession area also applies.

Under Law No. 9, corporate income tax payable at a rate of 25% on taxable earnings is exempted for the period during which the Company has outstanding debt relating to the construction and development of the Project.

In August 2012, MPSA entered into a precious metals stream agreement with a subsidiary of Franco-Nevada Corporation for the delivery of by-product precious metals based on copper concentrate production from the Cobre Panamá Project.

4.7 Environmental liabilities

The QPs of this Technical Report are unaware of any environmental liabilities currently existing on the property.

4.8 Permitting

An Environmental and Social Impact Assessment (ESIA or *Estudio de Impacto Ambiental Category III*) was submitted to the Panamánian environmental authority, *Autoridad Nacional del Ambiente* (ANAM or the National Environmental Authority; now referred to as *MiAmbiente*) and was approved on December 28th, 2011. This ESIA covers 7,586 hectares of the total concession, plus the port site and the four mining areas at Botija, Colina, Medio and Valle Grande.

MPSA intends to expand this to cover the other mining areas at Balboa and BABR by submitting a new ESIA for approval in mid-2024.

The proposed expansion of the Project will be subject to a high level of environmental and social impact assessment by MPSA, in anticipation that the proposal will be similarly classified (Category III) by the Panamánian Government. Environmental management and controls will be applied to the development of the expansion areas that are equivalent to the controls for the existing Project. Provided that the same process and assessment is conducted as has been for the existing Project, and with similar commitments made, there are therefore no known reasons to indicate that there will be environmental and permitting issues that could materially impact on mining and tailings disposal activities expanding to the limits shown by the blue perimeter in Figure 4.3.

4.9 Factors and risks which may affect access or title

In September 2018 the Company became aware of a ruling of the Supreme Court of Panamá in relation to the constitutionality of Law No. 9. The Company understands that the ruling of the Supreme Court with respect to the constitutionality of Law No. 9 relates to the enactment of this law and does not affect the legality of the MPSA mining concession contract itself, which remains in effect, and allows continuation of the development and operation of the Cobre Panamá Project by MPSA.

The Supreme Court ruling is not yet in effect and is subject to various procedural processes. Based on support from the Government of Panamá, the Chamber of Commerce and Industries of Panamá, the Panamánian Mining Chamber, other Panamánian businesses and industry chambers, and its own legal advice, the Company is confident of resolving the Law No. 9 clarification in the near to medium term.

The QPs of this Technical Report are unaware of any other significant factors and risks that may affect access, title, or the rights or ability to perform work on the property.

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

5.1 Accessibility

Access to the property is via the southern Pan-American Highway from Panamá City to Penonomé, heading north to La Pintada along surfaced all-weather roads and then along sealed roads from Coclecito to the mine site. An extensive network of roads has been constructed around the Project site including a 25 km long access road between the mine/plant site and the port at Punta Rincón.

Helicopter pads have been retained for occasional use. There is an existing airplane runway at Coclecito, however, frequent thick cloud cover impedes aircraft visibility and therefore limits its availability.

5.2 Climate and physiography

Climatic conditions in Panamá are equatorial with uniformly high temperatures (25°C to 30°C) and relative humidity with little seasonal variation. Climatic regions are determined on the basis of rainfall which can vary from less than 1,300 mm to 4,700 mm annually. In general, rainfall is much higher in coastal areas, particularly on the Caribbean side of the continental divide. Heavy tropical rains are prevalent throughout the year, with storms generally of short durations, ranging from 1.5 to 2 hours.

The concession zones are characterised by rugged topography with dense rainforest cover. The topography is characterised by a relatively low elevation (less than 300 m) although ranging from 70 masl to 300 masl. Narrow ridges that parallel major geological structural trends are the dominant landforms which are cross cut by numerous surface water drainage channels.

Elevations at the port site range from sea level to 60 masl with the terrain characterised as much gentler.

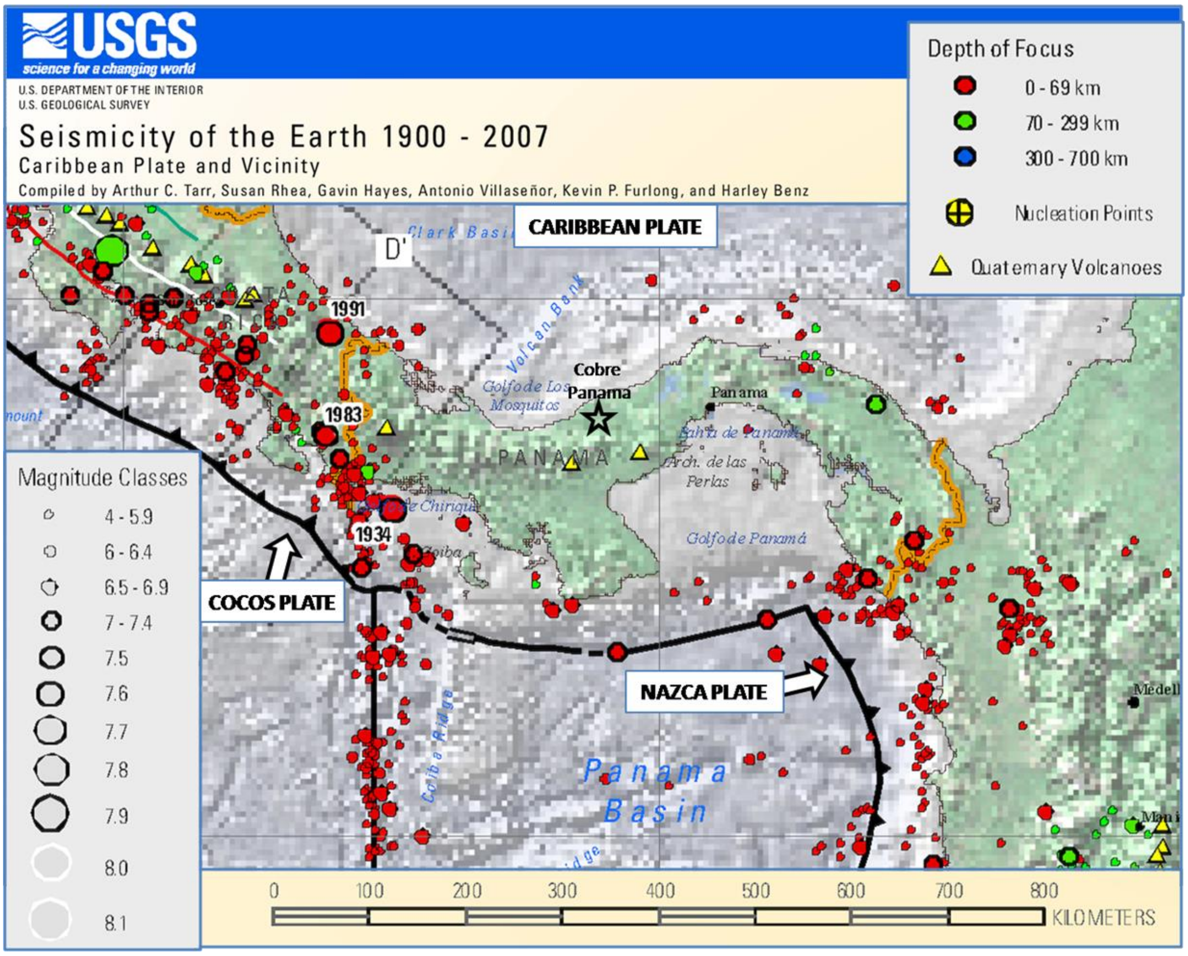
Operations at the Project will be conducted year round and are not expected to be adversely affected by the climatic conditions, despite the high rainfall.

5.3 Seismic conditions

Figure 5-1 indicates that relative to other areas of the isthmus, the Project site is located in an area relatively unaffected by seismicity.

Work by the *Universidad de Panamá, Instituto de Geociencias* (IGC, 2013), confirms the tectonic setting shown in Figure 5-1, but states that there is the potential for seismic activity of similar magnitude to an earthquake that occurred in April 1991 at a location approximately 50 km from the Project site. IGC completed a probabilistic seismic hazard study for the Project and provided a range of peak ground acceleration parameters, at varying return periods, for the design of the tailings facility and structures at the plant and port sites.

Figure 5-1 Seismicity map (Tarr et al, US Geological Survey)



5.4 Availability of power, water, personnel and areas for Project infrastructure

5.4.1 Power supply

Project power will be generated by a coal-fired, thermoelectric power station located at the port site, and transmitted to the mine and plant sites along a new access corridor. Excess electricity and backfeed supply is possible from the national grid.

Electric power into the mine is provided by a 34 kV transmission line ring main from the Botija substation.

5.4.2 Water supply

The water requirements for the Project are expected to be amply met by rainfall alone due to the high annual precipitation across the area. Collection and containment facilities will be required, much of which will be within the tailings basin and surrounding the mine and plant areas.

All run-off from the active areas will be treated to remove solids and recycled to the process plant where applicable.

5.4.3 Availability of personnel

The nearest sizeable communities to the Project are Coclecito and Villa del Carmen (~1,440 people), located 8 km southeast of the proposed plant site. Smaller nearby communities are located at San

Benito (~200 people), Nuevo Sinai (~350 people), Chicheme (~300 people) and Rio Caimito (~240 people). Subsistence farming is the primary occupation of the local population, with no industrial development in this part of Panamá. The city of Penonomé, the capital of the Colcé Province, is located 49 km southeast of Coclecito.

Efforts have been made to hire a workforce from the local communities. However, as this region of Panamá is relatively sparsely populated, additional personnel have been recruited from other areas of the country. Skilled expatriate personnel have been required in the early stages of the Project, for initial project management and to help establish the substantial training programmes required to educate and train a national workforce.

As at December 2018 the operations workforce comprised 2,133 direct employees out of the projected total of 3,421 full staff complement. A substantial training and development programme for Panamánian employees in the operations team is underway.

5.4.4 Infrastructure

As there is no other industrial development in this region of Panamá, all current Project infrastructure has been built by MPSA. This includes several camps on the mine and plant site, as well as several smaller construction camps in other parts of the concession, including at the TMF and at Punta Rincón. The main site camps are accessible by road and can collectively accommodate over 10,000 people. Figure 5-2 depicts the current Project layout and infrastructure locations.

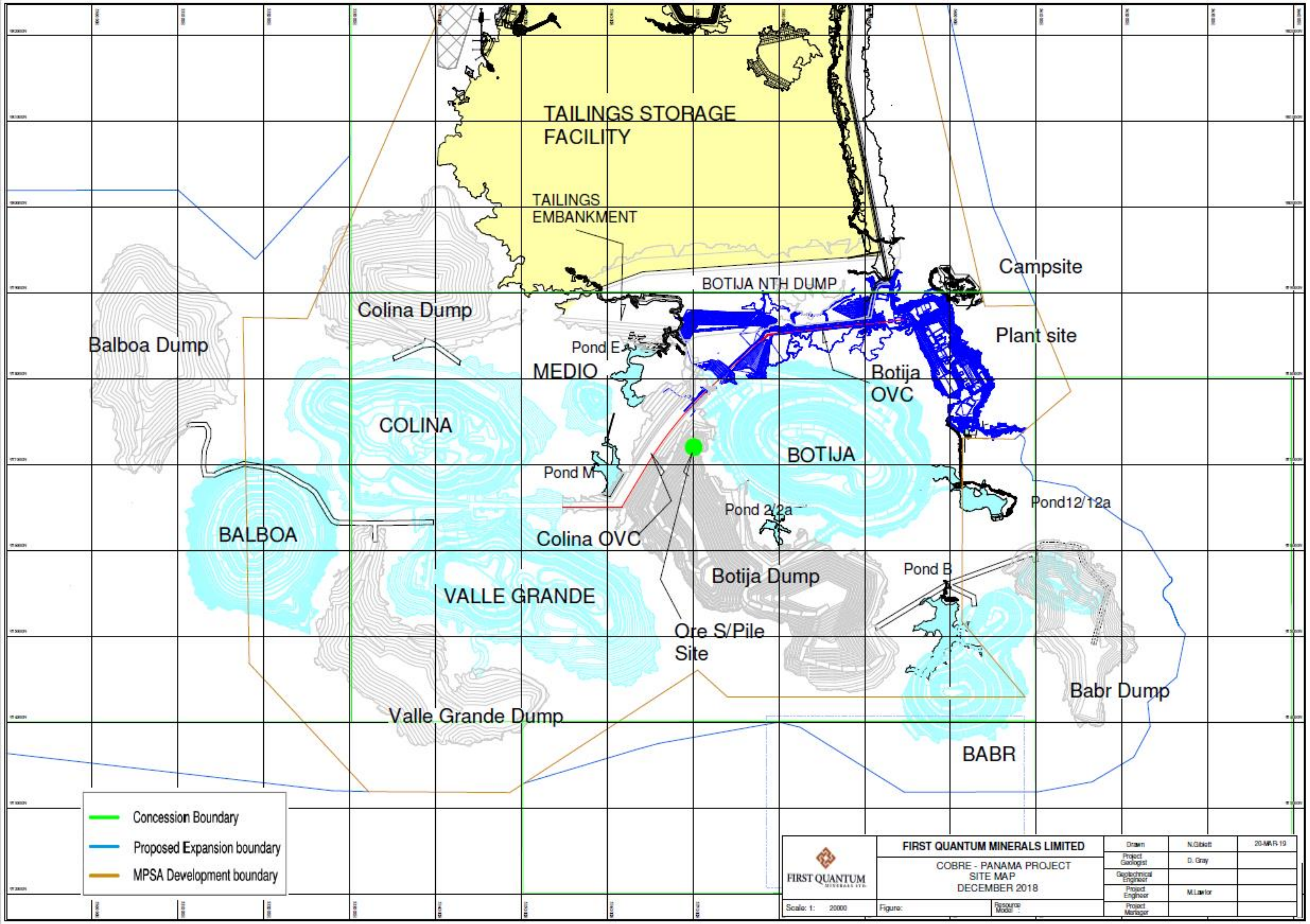
MPSA has constructed an extensive all-weather road network to improve access across the property. This includes upgrades to existing roads, including the main access road from Llano Grande, as well as building of an Eastern Access road to link in to the national road network.

Development of the Project required the construction of a port facility and an adjacent 300 MW power plant at Punta Rincón, approximately 25 km north of the mine and plant area. An all-weather road has also been constructed between Punta Rincón and the plant site. Other required developments have included a 120 km long double circuit, 230 kV power transmission line connecting to the national grid, pipeline infrastructure for both water and metallurgical concentrates, worker accommodation and other support facilities (maintenance and storage areas, etc).

5.5 Sufficiency of surface rights

Land required for the provision of mine facilities, including waste rock storage and tailings facilities, stockpiles, mill and port sites, has been acquired or leased by MPSA as per its rights under Law No. 9, 1997. Mine expansion will require the acquisition of an additional 6,800 ha beyond the current ESIA polygon.

Figure 5-2 Cobre Panamá site map



ITEM 6 HISTORY

6.1 Ownership and exploration

Copper-gold-molybdenum anomalism was first discovered in the Rio Petaquilla region of central Panamá by a United Nations Development Programme (UNDP) team in 1968. The UNDP team were conducting regional geological and geochemical surveys of the area. Further exploration by various companies has since continued, leading to the discovery of four large porphyry deposits (Botija, Colina, Valle Grande and Balboa) and a host of smaller mineralised porphyry systems (including Brazo, Botija Abajo and Medio). One zone of epithermal gold mineralisation (Molejón Gold deposit) has also been discovered. In 2009 the Project, formerly known as the Petaquilla Project, was renamed Mina de Cobre Panamá, or Cobre Panamá.

Companies that have conducted exploration over the Project area include: United Nations Development Programme (1968-1969), Panamá Mineral Resources Development Company (PMRD), a Japanese consortium (1970-1980), Inmet – Adrian Resources – Teck (Formerly Teck Cominco) (1990-1997), Petaquilla Copper (PTC)(2006-2008) and Minera Panamá S.A. (MPSA) (2007-2013). A total of 1,813 diamond drillholes (348,775 m) have been completed up to February 2018.

A brief summary of ownership and exploration across the Project area is summarised in Table 6-1.

Table 6-1 Exploration and ownership history of Cobre Panamá (source: Inmet)

Year	Party	Description
1968	UNDP	Regional geological and geochemical survey of central Panamá by the United Nations Development Programme (UNDP); widespread silicification and copper mineralisation discovered in the area of Colina and Botija deposits; silt samples and 200 line-km of soil samples revealed several copper and molybdenum anomalies, including Valle Grande, Botija Abajo, Brazo, and Medio; vertical field magnetics identified areas of magnetite alteration and magnetite destruction.
1969	UNDP	27 short (Winkie drill) holes and 10 long holes drilled in Botija, Colina, Vega (ie, part of Valle Grande), and Medio areas.
1969	PMRD	Panamánian government tendered Cobre Concession exploration rights to international bidding; concession awarded to Panamá Mineral Resources Development Company (PMRD), a Japanese consortium.
1970-1976	PMRD	Geological mapping at Botija and Colina; 48 short (Winkie drill) and 51 long (diamond) holes drilled at Botija, Colina, Medio, and Vega (part of Valle Grande), totalling approximately 14,000 m. Botija and Colina deposits drilled on approximately 200 m centres.
1977	PMRD	Preliminary reserves calculated and pre-feasibility report completed.
1978-1979	PMRD	Feasibility work updated; unsuccessful negotiations with the Panamánian government over terms of production.
1980	PMRD	Property abandoned by PMRD.
1990-1992	Minnova	Property acquired by Minnova (later Inmet), 80%, and Georecursos International S.A., 20%. Exploration activity included regional lithogeochemical sampling.
1992-1993	Adrian	Adrian Resources Ltd. (Adrian) granted an option to earn 40% of Minnova's interest through cash payments, work commitment, and production of a feasibility study. Adrian subsequently acquired Georecurso's interest, bringing its total interest to 52%.
1992-1995	Adrian	Adrian carried out grid-based soil sampling and magnetic measurements, geologic mapping of selected areas, and drilling of approximately 396 diamond drill holes in Colina, Botija, and exploration targets. Investigation of Valle Grande deposit and discovery of epithermal Au mineralisation at Molejón, as well as identification or investigation of several other targets (Botija Abajo, Brazo, Faldalito (north west of Colina), Cuatro Crestas (part of Balboa), Lata, Orca (gold deposits, both NW of Colina)). Initiation of baseline environmental studies. Scoping study and pre-feasibility study produced.

Year	Party	Description
1994	Teck	Teck was granted the right to acquire half of Adrian's share (26%) of the deposit by funding a feasibility study and arranging Adrian's portion of the financing needed to bring the deposit into production.
1996	Teck	Infill and deposit condemnation drilling and mapping for feasibility study carried out. Teck drilled 91 infill and 33 condemnation holes totalling 26,837 m. Feasibility study completed.
1997	Teck	Infill and drilling for metallurgical samples to update feasibility study. Teck drilled 43 holes totalling 8,099 m. Feasibility study updated.
2005	MPSA	Molejón Gold Agreement – shareholders transfer rights to any gold deposits on concession to Petaquilla Minerals Limited for a 5% NSR.
2007-2008	MPSA	Activity resumes on copper deposits with the JV drilling condemnation, metallurgical, infill, and pit geotech holes. Lidar topo survey of concession.
2008	Inmet	Inmet acquires Petaquilla Copper Ltd (PTC) including the 26% interest in MPSA, taking Inmet to 74% interest in MPSA.
2008	Inmet	Inmet acquires Teck's 26% interest in MPSA, taking Inmet to 100% interest in MPSA.
2009	MPSA	2007-2009: 288 holes (73,481 m) of infill, metallurgical, and condemnation drilling at Botija, Colina, Valle Grande, and Brazo deposits. Comminution testing. Condemnation of proposed tailings area and seismic, resistivity, and geotech drilling at port site and infrastructure locations.
2009	KORES/LS Nikko (KPMC)	KORES and LS Nikko Copper agree to option a 20% interest in MPSA.
2011	MPSA	Discovery of Balboa deposit announced on March 11th.
2011	MPSA	ESIA (Environmental Social Impact Assessment) approved by government of Panamá on December 28th – allows MPSA to proceed with the development of the Project.
2012	KPMC	KPMC elects to exercise their option and acquires a 20% interest in the Cobre Panamá Project on January 10th.
2012	Inmet/MPSA	Inmet Board of Directors makes production decision on May 18th to proceed with the development and construction of the Cobre Panamá Project.
2012	Inmet/ Franco-Nevada	August 20th, Franco Nevada acquires an interest in the precious metal stream in exchange for up to $1 Billion for Inmet to finance their portion of the development costs.
2013	FQM	FQM purchases Inmet, acquiring an 80% stake in MPSA
2017	FQM	FQM acquires an effective 90% stake in MPSA

Several pre-feasibility and feasibility studies have been completed on the Cobre Panamá Project. These include:

- A preliminary feasibility report prepared in 1977 by Panamá Mineral Resources Development Co. Ltd. This was updated in 1979.
- Adrian Resources Ltd. (a pre-cursor of Petaquilla Minerals Ltd and Petaquilla Copper Ltd) commissioned Kilborn Engineering Pacific Ltd to complete a prefeasibility study in 1994. The study was updated in 1995.
- In November 1996, Teck Corporation commissioned H.A. Simons, now AMEC, to produce a feasibility study which was subsequently updated in January 1998.
- AMEC completed a Front End Engineering Design (FEED) Study Report in 2010.
- A Basic Engineering Completion Report was completed in April 2012.

The 1998 Teck feasibility study was submitted to the Panamánian Ministry of Industry and Commerce in May 1998 and was accepted as the official Feasibility Study to satisfy concession law requirements outlined in Law No. 9 for the delivery of a feasibility study. At that time, the Project was owned by Teck, Petaquilla Copper Ltd and Inmet Mining Corporation under the MPSA holdings.

In January 2017, the Company produced an internal Feasibility Study for the purposes of a due diligence review undertaken by potential Project financiers.

6.2 Previous Mineral Resource estimates

Mineral Resource estimates were generated by the previous Project owner for the porphyry copper-type deposits on the Cobre Panamá property including the Botija, Colina, Medio, Valle Grande, Balboa, Brazo and Botija-Abajo mineralised zones. The last of these estimates were completed as part of the original FEED study in December 2009, with some deposits updated in 2010 and 2012 as exploration and delineation programmes evolved across the Project.

Table 6-2 lists the first Mineral Resource estimate produced by the Company for the 2015 Technical Report (FQM, July 2015). The estimate was stated at a 0.15% Cu copper cut-off grade and was inclusive of the Mineral Reserve.

Table 6-2 Cobre Panamá Mineral Resource statement, at June 2015, using a 0.15% copper cut-off grade

Deposit	Category	Tonnes (millions)	Copper (%)	Molybdenum (%)	Gold g/t	Silver g/t	Contained Cu (ktonnes)
Botija	Measured	336	0.46	0.008	0.10	1.35	1,540
Botija	Indicated	672	0.35	0.007	0.06	1.08	2,349
Colina	Indicated	1,032	0.39	0.007	0.06	1.58	3,983
Medio	Indicated	63	0.28	0.004	0.03	0.96	179
Valle Grande	Indicated	602	0.36	0.006	0.04	1.37	2,169
Balboa	Indicated	647	0.35	0.002	0.08	1.37	2,259
Botija Abajo	Indicated	114	0.31	0.004	0.06	0.93	351
Brazo	Indicated	228	0.36	0.004	0.05	0.81	816
Total Measured and Indicated		**3,695**	**0.37**	**0.006**	**0.07**	**1.32**	**13,646**
Botija	Inferred	152	0.23	0.004	0.03	0.78	354
Colina	Inferred	125	0.26	0.006	0.05	1.20	329
Medio	Inferred	189	0.25	0.005	0.03	1.25	482
Valle Grande	Inferred	363	0.29	0.005	0.03	1.14	1,048
Balboa	Inferred	79	0.23	0.003	0.04	0.96	180
Botija Abajo	Inferred	67	0.27	0.005	0.06	1.25	182
Brazo	Inferred	76	0.21	0.003	0.01	0.73	162
Total Inferred		**1,051**	**0.26**	**0.005**	**0.04**	**1.08**	**2,737**

6.3 Previous Mineral Reserve estimates

A formal Mineral Reserve estimate was publically filed in an NI 43-101 Technical Report dated May 2010 (Rose *at al,* 2010), at an effective date of March 2010. The estimate was based on metal prices of \$2.00/lb Cu, \$12.00/lb Mo, \$750/oz Au and \$12.50/oz Ag. The design and production schedule upon which the Mineral Reserves was determined, accounted for the Botija, Colina and Valle Grande mineralised zones.

Table 6-3 lists the Mineral Reserve estimate produced by the Company for the 2015 Technical Report (FQM, July 2015). The estimate was stated as being based on a copper metal price of $3.00/lb, a molybdenum metal price of $13.50/lb, a gold price of $1,200/oz and a silver price of $16.00/oz.

Table 6-3 Cobre Panamá Project Mineral Reserve statement, at June 2015

Pit	Class	Mtonnes	TCu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	TCu metal (kt)	Mo metal (kt)	Au metal (koz)	Ag metal (koz)
BOTIJA	Proved	345.6	0.45	74.88	0.10	1.33	1,550.2	25.9	1,122.0	14,790.5
	Probable	603.5	0.35	70.79	0.07	1.10	2,124.5	42.7	1,289.8	21,377.4
	Total P+P	**949.1**	**0.39**	**72.28**	**0.08**	**1.19**	**3,674.7**	**68.6**	**2,411.8**	**36,167.9**
COLINA & MEDIO	Proved									
	Probable	1,009.9	0.39	66.27	0.06	1.59	3,898.8	66.9	2,034.9	51,607.8
	Total P+P	**1,009.9**	**0.39**	**66.27**	**0.06**	**1.59**	**3,898.8**	**66.9**	**2,034.9**	**51,607.8**
VALLE GRANDE	Proved									
	Probable	566.0	0.36	67.02	0.05	1.39	2,035.9	37.9	837.8	25,278.9
	Total P+P	**566.0**	**0.36**	**67.02**	**0.05**	**1.39**	**2,035.9**	**37.9**	**837.8**	**25,278.9**
BALBOA	Proved									
	Probable	437.1	0.35	16.10	0.08	1.36	1,509.0	7.0	1,126.9	19,168.2
	Total P+P	**437.1**	**0.35**	**16.10**	**0.08**	**1.36**	**1,509.0**	**7.0**	**1,126.9**	**19,168.2**
BABR	Proved									
	Probable	220.5	0.40	41.25	0.07	0.87	882.5	9.1	529.4	6,179.2
	Total P+P	**220.5**	**0.40**	**41.25**	**0.07**	**0.87**	**882.5**	**9.1**	**529.4**	**6,179.2**
TOTAL	Proved	345.6	0.45	74.88	0.10	1.33	1,550.2	25.9	1,122.0	14,790.5
	Probable	2,836.9	0.37	57.71	0.06	1.36	10,450.7	163.7	5,818.9	123,611.5
	Total P+P	**3,182.5**	**0.38**	**59.57**	**0.07**	**1.35**	**12,000.9**	**189.6**	**6,940.8**	**138,402.0**

6.4 Production from the property

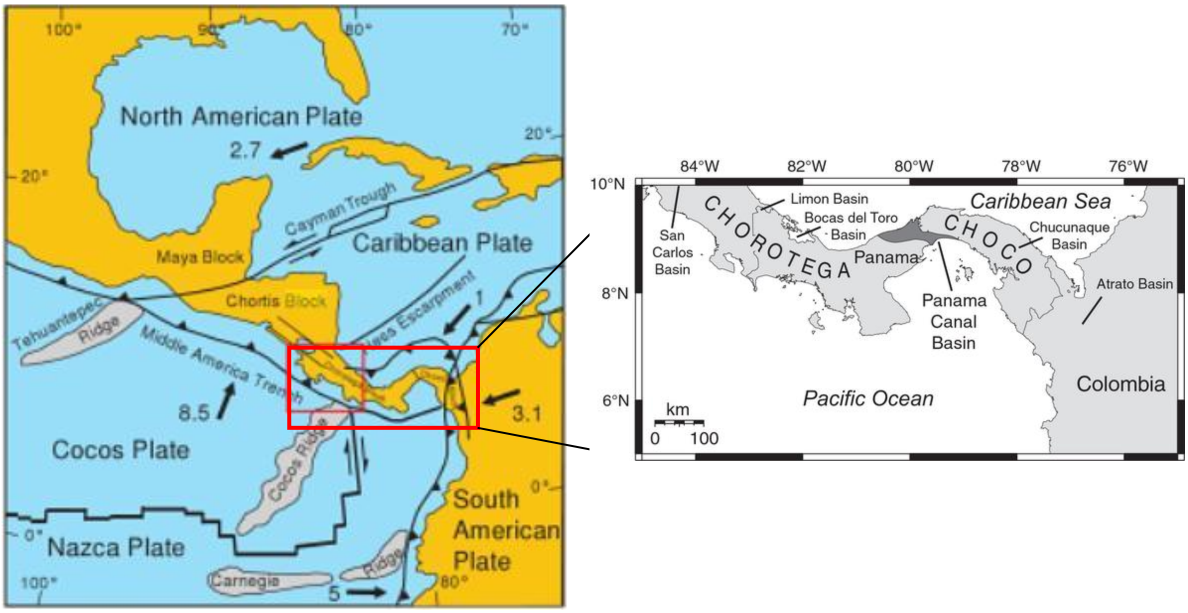
To date there has been no production from the property.

ITEM 7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional geological setting

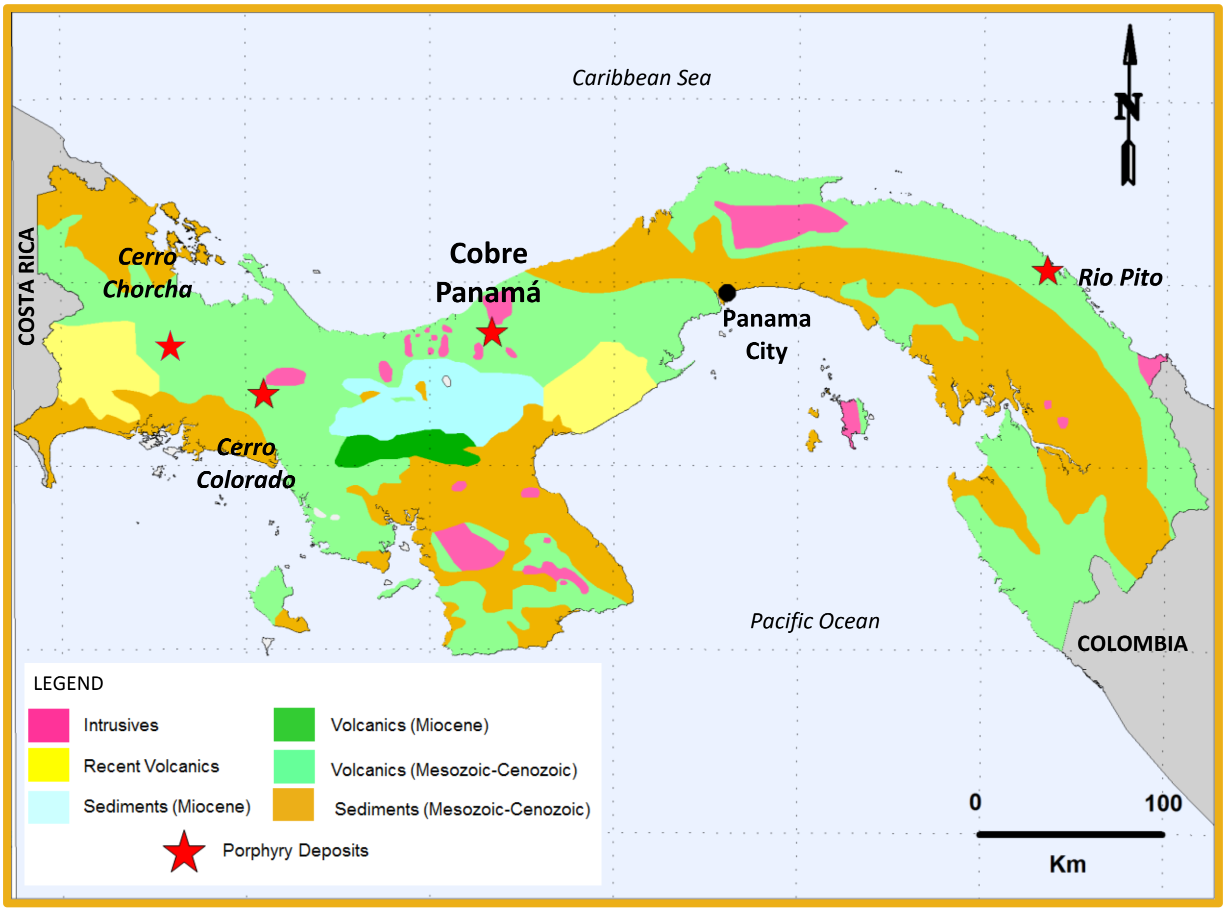
The Panamá Isthmus is a narrow strip of land that connects the North and South Americas at the junction of several tectonic plates, including the Caribbean, Nazca, Cocos and continental North and South American Plates. South vergent subduction and related arc volcanism, high-angle strike-slip and block faulting, as well as north vergent thrust faulting are currently shaping the tectonically active country (Mann, 1995). The Central American land bridge (which includes Costa Rica and Panamá) is a complex assemblage of distinct crustal blocks and includes the Chorotega (southern Coast Rica and western Panamá) and Choco (eastern Panamá) Blocks (Figure 7-1). The Chorotega and Choco Blocks comprise island-arc segments underlain by Mesozoic oceanic crust (Escalante and Astorga, 1994) and are separated by the left-lateral Canal Shear Zone.

Figure 7-1 Plate tectonic map of the Caribbean region (from Kirkby *et al,* 2008)



Within the Chorotega block, the island arc sequence consists of several distinct pulses of volcanism, including Palaeocene-Eocene, mid-Oligocene, late Oligocene to early Miocene, and Pliocene-Pleistocene ages. It is assumed that breaks between the volcanism were related to times of plate reorganisation (de Boer *et al,* 1995). Intrusive rocks of Palaeocene-Eocene age lie along a tholeiitic trend when plotted on an AFM plot; however, no porphyry mineralisation is recognised during this period (Kesler *et al*, 1977) The suites of younger rocks are dominantly calc-alkaline in composition and contain porphyry mineralisation ranging from Oligocene (including the Cobre Panamá deposits ~32Ma) to Pliocene (Cerro Colorado deposit ~5Ma). The regional geology map of Panamá is presented in Figure 7-2.

Figure 7-2 Regional geology of Panamá (source: MPSA, 2014)



Interpretation of satellite imagery over the Project area within the Chorotega Block suggests that major structural trends, expressed as topographic lineations, are orientated northeast and northwest (Figure 7-3). Northwest-trending lineations are parallel to and related to the Canal Shear zone and other large left-lateral shear zones in the region.

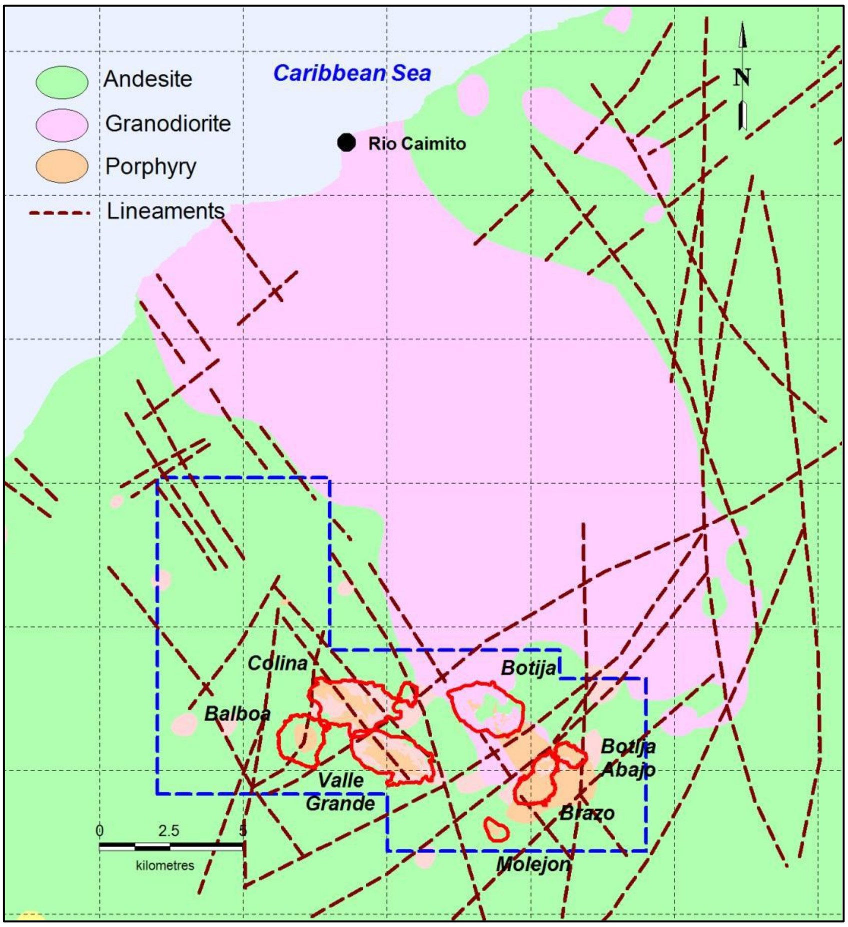
In the area of the Cobre Panamá deposits, the oldest rocks are submarine andesites, basalt flows and tuffs, intercalated with clastic sedimentary rocks and reef limestones, of probably Eocene to early Oligocene age. This suggests that the volcanic arc became emergent during the mid-Oligocene period, with terrestrial flows and volcaniclastic rocks and lesser intercalated submarine tuffs. Miocene and younger rocks comprise the bulk of volcanic rocks in western Panamá and consist of both terrestrial and marine volcanic and volcanic-derived rocks of progressively more felsic composition (Rose *et al*, 2010).

7.2 Local geological setting

7.2.1 Lithology

Regional geological mapping in the Cobre concession and surrounding areas was first conducted in 1966 to 1969 by the United Nations Development Programme (UNDP). The region is underlain by altered andesitic to basaltic flows and tuffs and clastic sedimentary rocks of presumed early to mid-Tertiary age, intruded by the mid-Oligocene Petaquilla batholith which is granodiorite in composition. Numerous satellite plutons of equigranular to porphyritic granodiorite, tonalite, quartz diorite and diorite are found on the batholith margins, especially to the south. A detailed geological map for the concession based on limited field mapping and geology taken from drill holes is presented in Figure 7-3.

Figure 7-3 District-scale geology and structural map of the Cobre Project showing lithology and structural lineaments (source: Rose *et al*, 2012)

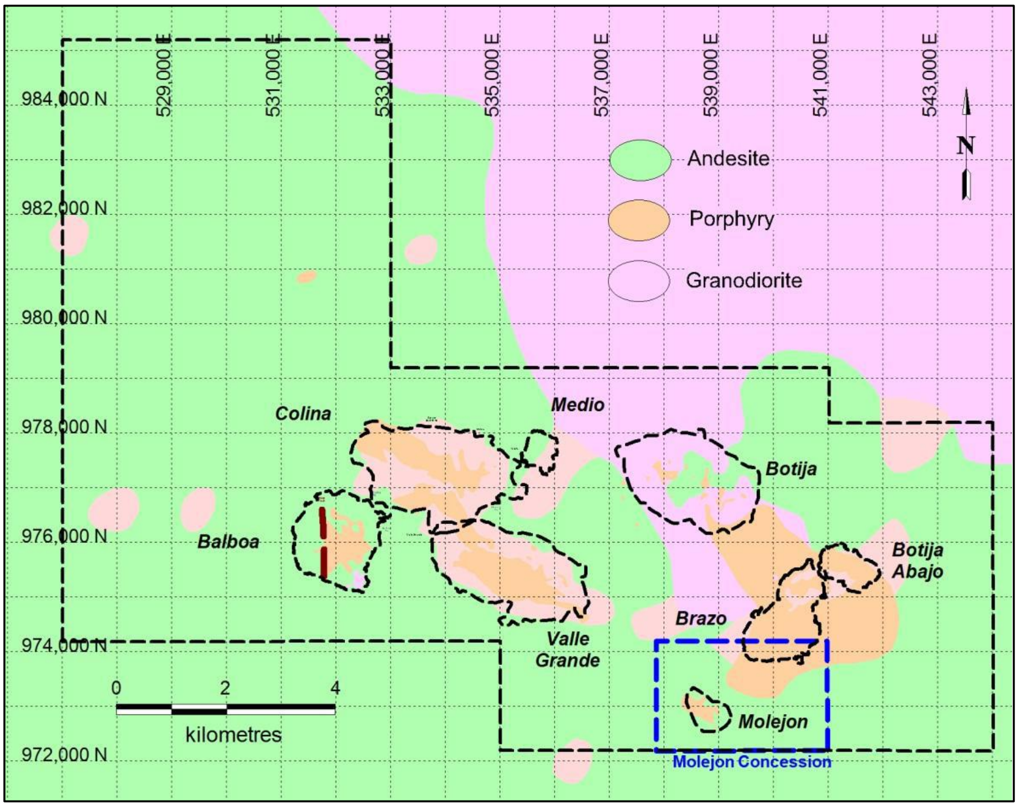


Seven main lithological units have been recognised at the Cobre Panamá Project. From oldest to youngest these units are:

- a sequence of undifferentiated andesite volcanics and volcaniclastics
- granodiorite
- a suite of a feldspar-quartz-hornblende porphyries of granodiorite composition
- a suite of feldspar-quartz porphyry dykes
- a suite of fine-grained mafic dykes
- a suite of andesite porphyry dykes
- saprolite

Cross-cutting relationships suggest that the andesite volcanics and volcaniclastics were intruded by the granodiorite, which forms a batholith and hosts most of the copper mineralisation. This was followed by the intrusion of dykes and apophyses of the feldspar-quartz-hornblende porphyry suite. These dykes are the most extensive of the intrusives and contain slightly higher grade copper mineralisation than the granodiorite. There are few cross-cutting contacts observed between the feldspar-quartz-hornblende porphyry and the granodiorite and texturally can be difficult to distinguish. This suggests that they were intruded over a short period of geological time and most likely form a continuum of the same melt and progressive differentiation of a regional batholith.

Figure 7-4 Camp-scale geology of the Cobre Panamá Project with deposit pit outlines shown for reference (source: Rose *et al*, 2012)



A second series of feldspar-quartz porphyry dykes have also intruded the area, but are volumetrically minor and have not been identified at all the deposits. A set of volumetrically small, mafic dykes cross cut the aforementioned lithologies; these are likely to be a final differentiation phase of the feeder batholiths. Late-stage andesite dykes clearly cross-cut and post-date all other lithologies, exhibiting well-developed chill margins which indicate that they intruded after the main porphyry event had cooled. Surface weathering has allowed the development of a saprolite profile that is typical of tropical environments.

Emplacement of the large porphyry bodies at Cobre Panamá was likely through magmatic stoping. Evidence supporting this geological model includes:

- the abundance of roof pendants
- the flat-lying geometry of the lower porphyry contact at the majority of deposits
- the proximity of a large batholith underlying the magma chamber
- the presence of fracturing at the lower porphyry contact without significant displacement, caused by the deflation of the underlying batholith allowing subsidence of the lower contact

It is interpreted that the feldspar-quartz-hornblende porphyry intruded the sequence in response to cantilever subsidence of the lower contact of the intruding granodiorite (Cruden, 1998). At the major deposits the feldspar-quartz-hornblende porphyry forms sill-like bodies, thought to have been fed from high-angle feeder dykes; situated in the north of the deposit at Botija and Colina and to the northwest at Valle Grande. The porphyry at Balboa intruded passively toward the south from a source located northwest of the deposit and is also thought to be influenced by a high angle structure to the west of the deposit.

A detailed description of each of these lithologies including the respective logging codes is presented in Table 7-1.

7.2.2 Structure

Three generations of structures have been identified from mapping at the Cobre Panamá Project (Model Earth, 2013). The earliest features trend east-west to northeast-southwest and are moderate to north dipping features characterised by fracturing and a pervasive fracture cleavage with associated faulting and shearing. These fault zones are typically wide (up to 200 m) zones of numerous, anastomosing structures and are thought to be part of an early, pervasive tectonic fabric which is seen in all rock types except the late stage mafic dykes. These features show consistent reverse movement with some influence on mineralisation. It is interpreted that there may be a possible generic relationship between the emplacement of mineralised porphyries and these early faults.

A second set of conjugate strike slip faults exist and are dominated by a series of northwest trending sinistral faults with steep to moderate dips to the west which overprint the earliest structures. Conjugate to these northwest trending faults are a series of northeast trending dextral faults with moderate to steep dips to both the north and south. Overprinting relationships between these two faults are unclear suggesting that they formed contemporaneously, as a conjugate set. These faults are observed in mapping and in geophysical images.

Late-stage faulting is characterised by normal, east-west trending faults with steep dips to both the north and south. Another set of northeast trending normal faults have also been identified. Both these faults cross cut many of the older structures throughout the Project area.

7.2.3 Surficial geology

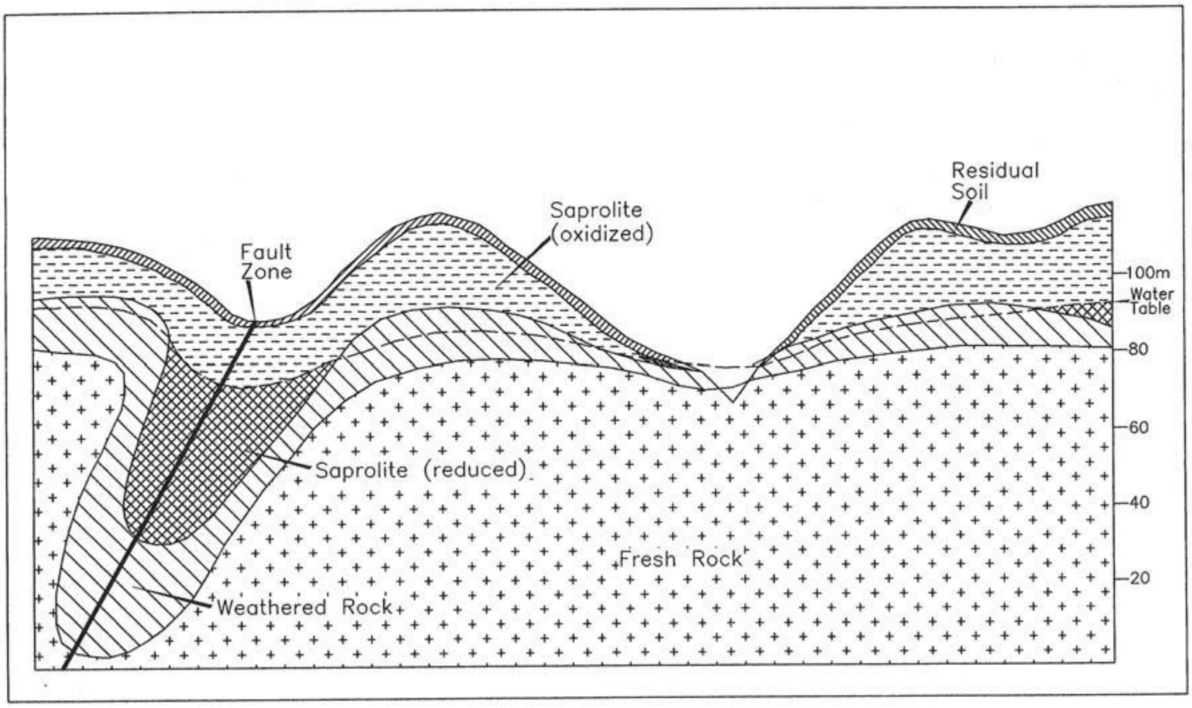
Most of the property area is covered by a thin layer (1cm to 10 cm) of black organic material which overlies a thin layer (<1 m) of residual soil. This in turn overlies a 1 m to 20 m thick layer of orange to white (oxidised) or green (reduced) saprolite (Figure 7-5). The depth of the saprolite layer is controlled by lithology, elevation (hill or creek bottom) and proximity to major faults. The saprolite layer is deeper over andesite relative to the intrusive rocks with the transition to relatively fresh rock occurring within metres over andesite, to tens of metres over the intrusives (AMEC, 2007).

Saprolite and strongly weathered zones (saprock) may be several tens of metres thick (occasionally up to 100 m) in the vicinity of large fault zones. Fresh, unoxidised rocks are only found at the base of large creeks.

Table 7-1 Description of the lithological units of the Cobre Panamá Project (adapted from Rose *et al*, 2012)

Lithology	Logging Code	Description
Andesite	**ANDS**	Light green, chlorite altered, fine-grained andesite that is weakly magnetic and comprises mostly of massive flow units that are bounded by fragmental flows. Lesser lapilli tuffs and volcaniclastic interbeds have also been noted. The andesites have not been subdivided into individual flows.
Granodiorite	**GRDR**	Mottled white-grey lithology with medium-grained equigranular interlocking crystals of white feldspar, hornblende and quartz. The granodiorite also contains subrounded to subangular, centimetric fine-grained mafic xenoliths and occasional apilite dykes. White clay alteration can produce a porphyritic texture by altering the groundmass surrounding the feldspar crystals, producing a pseudo-porphyritic texture and sometimes causing it to be logged as the feldspar-hornblende-quartz porphyry.
Feldspar-Hornblende-Quartz Porphyry	**FQHP**	Light grey porphyritic lithology with a crowed porphyry texture. There is not always a large difference in the grainsize of the groundmass and the phenocrysts. The porphyritic texture is best described as weak. The groundmass is light grey, fine-grained, and contains feldspar with subordinate mafics and quartz. Phenocrysts comprise crowded white subhedral to euhedral feldspar (often plagioclase) phenocrysts, lesser clear, glassy anhedral quartz "eyes" and minor hornblende.
Feldspar-Quartz Porphyry	**FQP**	Light grey porphyritic lithology with a light grey, aphanitic and siliceous groundmass that contains crowded white subhedral to anhedral feldspar phenocrysts and clear, glassy anhedral quartz “eyes” and very rarely bi-pyramidal quartz phenocrysts. This lithology has so far been recognised at the Botija and Brazo deposits only.
Feldspar-Hornblende Porphyry	**FHP**	Light green-grey porphyritic lithology, probably of andesitic composition, with a light grey, fine-grained groundmass containing dominant white subhedral to euhedral feldspar (mostly plagioclase) with subordinate dark green hornblende and lesser quartz. This porphyry contains sparse, irregularly shaped, centimetric, medium-grained mafic xenoliths.
Mafic Dykes	**MD**	Dark green, pervasively chlorite altered, fine-grained mafic pyroxene porphyry that occurs as dykes with clear chill margins noted locally.
Saprolite	**USAP, LSAP, SPRC**	At surface, most areas of the property are covered by a thin layer (1 cm to 10 m) of black organic material. This overlies a thin layer of residual soil generally less than 1 m in thickness, which in turn overlies a 1 m to 20 m thick layer of orange to white (oxidised) or green (reduced) saprolite. On average, the saprolite layer is deeper over andesite relative to intrusive rocks. The transition from saprolite to relatively fresh andesite typically takes place over a range of less than a few metres, whereas the transition zone from saprolite to relatively fresh granodiorite may involve several tens of metres. This transition zone contains blocks of unaltered bedrock and has been called “saprock.”

Figure 7-5 Idealised cross-section of surfaced weathering zone. Vertical scale is approximate and slightly exaggerated (from AMEC, 2007)



7.2.4 Alteration

Five types of alteration have been identified across the Cobre Panamá concession. These are described in Table 7-2 and are detailed from the earliest (propylitic A) alteration progressing through to assemblages that likely overlap or occur later in the development of the hydrothermal system.

It is interpreted that the two propylitic alteration assemblages as well as the potassic alteration were formed early which is supported by paragenetic, cross-cutting and mineral texture relationships. These alteration assemblages are associated with chalcopyrite, minor bornite and minor pyrite mineralisation. These early alteration assemblages are also overprinted by phyllic alteration that includes white and green sericite with ubiquitous pyrite and variable quartz veining and silicification. Phyllic alteration is also observed to occur frequently with chalcopyrite mineralisation, by rarely with bornite. Argillic alteration typically occurs within 300 m from surface. White clay found close to oxidised sulphide suggests that it may be supergene in origin, perhaps after an earlier alteration phase.

Table 7-2 Alteration styles at the Cobre deposits (adapted from Rose *et al*, 2012)

Alteration style	Description
Propylitic A alteration	Chlorite dominated with accessory epidote, pyrite, and calcite. It is particularly well developed within the andesite volcanics.
Propylitic B (silica-chlorite) alteration	Exhibits chlorite and silica in approximately equal intensity, resulting in a much harder core than other alteration types. In porphyry lithologies chlorite generally affects ferro-magnesium phenocrysts while silicification appears to primarily affect the groundmass. Pyrite and sericite (often green-coloured) may also be present. Previous workers on the property have referred to this alteration type as silica-chlorite alteration. It appears to occur at depth within the deposits but is found at shallow levels in the peripheral zones.
Potassic alteration	Occurs mainly as potassium feldspar selvedges to quartz ± sulphide veinlets and as irregular patches. Potassium feldspar flooding is rarely seen. Potassic alteration also occurs as fine-grained secondary biotite that alters ferro-magnesium minerals such as hornblende and magmatic biotite. Secondary biotite also occurs in discontinuous veinlets that commonly contain magnetite, chalcopyrite, and rare bornite. The amount of potassium feldspar or secondary biotite alteration is largely determined by the feldspar or biotite abundance in the protolith. Anhydrite veinlets of generally millimetric thickness are commonly associated with potassic alteration, especially in the deeper parts of the deposit. At depths of approximately 200 m from surface and shallower, the anhydrite appears to have hydrated to form gypsum. In addition, millimetric magnetite-only veinlets are uncommonly observed within the potassic alteration zone.
Phyllic alteration	Occurs as sericite alteration of all rock-forming silicate minerals. Silicification and pyrite are also associated with this phase of alteration. Phyllic alteration occurs in the upper 150 m to 200 m of all of the deposits but is very irregular and/or difficult to map through different protoliths and earlier alteration facies. It can occur much deeper when phyllic fluids have been drawn down into permeable structural zones. Phyllic alteration occasionally occurs with chlorite, and both green and white sericite are found within this zone, likely related to distinct alteration events. Sericite is used as a collective term for white phyllosilicate and displays a wide range of grain size from very coarse, granular, millimetric muscovite, to very fine grained “silky” textured. It may be possible to subdivide these variations spatially because they are probably related to distinct alteration events. Phyllic alteration is ubiquitous, with quartz-pyrite veinlets that frequently contain minor chalcopyrite and have a white sericite-altered selvedge.
Argillic alteration	Frequently occurs in the upper parts of the deposits from surface and is therefore largely coincident with the phyllic alteration zone. This can make visual distinction of clay or sericite dominant alteration a challenge. The clay minerals that have been visually identified range in colour from white to buff or light brown; kaolinite, smectite, and illite have been recognised.

7.3 Mineralisation

7.3.1 Supergene mineralisation

Oxidation of sulphides near the surface weathering profile has leached copper from the present-day saprolite. Copper has been weakly and irregularly re-precipitated in the upper zones of the deposits. Secondary sulphides are dominantly chalcocite with minor covellite and rare native copper. These secondary minerals occur as fracture infills, coatings on primary sulphide minerals and disseminations. Where these sulphides have been oxidised, malachite is the main copper oxide mineral.

Notably absent across the majority of the Cobre Panamá deposits is the presence of a significant zone of enrichment. It is interpreted that this is likely due to removal by erosion of a previously well-developed phyllic alteration zone which may have overlain these deposits. Phyllic alteration zones are suitable host rocks for re-precipitation of copper as they can sufficiently neutralise the acidic fluids required for leaching. A well-developed phyllic alteration zone is developed at Brazo, which accompanies a significant secondary copper sulphide mineralisation zone.

7.3.2 Hypogene mineralisation

Hypogene mineralisation within the granodiorite and various porphyry lithologies consists of disseminated sulphides, micro-veinlets, fracture fillings, veinlets and quartz-sulphide stockworks. Veins have been classified in accordance with Gustafson and Hunt (1975) who described in detail copper porphyry mineralisation at El Salvador, Chile, and which is summarised in Table 7-3.

Copper mineralisation occurs as chalcopyrite with lesser bornite. Throughout all deposits the proportion of bornite relative to chalcopyrite appears to increase with depth. Molybdenite is present in quartz “B” veinlets (Gustafson and Hunt, 1975). Pyrite is ubiquitous but the tenor increases in association with phyllic and chlorite-silica alteration compared to other alteration assemblages. Within the phyllic alteration zone, pyrite occurs as disseminations and within “D” veinlets (Gustafson and Hunt, 1975) with quartz. Minor specularite and magnetite mineralisation occurs as dissemination and veinlets in all deposits.

Mineralisation on the contacts between the andesite and feldspar-hornblende-quartz porphyry can reach high copper tenor in zones of biotite hornfels. Chalcopyrite is the dominant sulphide with minor pyrite and rare bornite, occurring in veinlets, blebs and disseminations. This style of mineralisation is often cross-cut by quartz-sulphide veining.

A description of each deposit within the Cobre Panamá concession is provided in the following sections. The location of all the mineralised zones is shown in Figure 7-6.

7.4 Botija

The Botija deposit is located in the northeast area of the Cobre Panamá concession. Botija is hosted in several feldspar-quartz-hornblende porphyry dykes (up to four) which range in thickness from 20 m to 200 m, and which have intruded the granodiorite and andesite host rocks. The porphyry morphology at Botija suggests that it intruded from the north as a series of dykes that fed the central apophysis, where they coalesced (Rose *et al*, 2012). In general, the dip of the more distinct dykes is approximately 70° to the north.

Two irregular, keel shaped andesite roof pendants of approximately 500 m in diameter have been identified at Botija (Rose *et al*, 2012), separated by approximately 300 m and reaching depths of between 200 m to 300 m. A smaller pendant, up to 250 m along strike and extending to a depth of 150 m sits to the north of the deposit.

A geological model demonstrating the distribution of the main rock types was created based on south-north cross sections spaced at between 50 m to 100 m intervals. A typical cross-section is presented in Figure 7-7.

Figure 7-6 Location map of mineralised zones

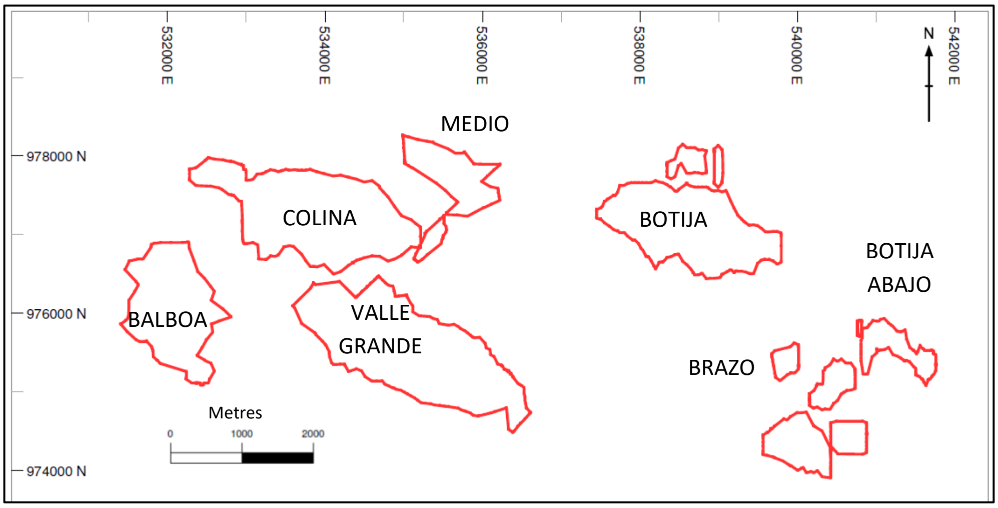
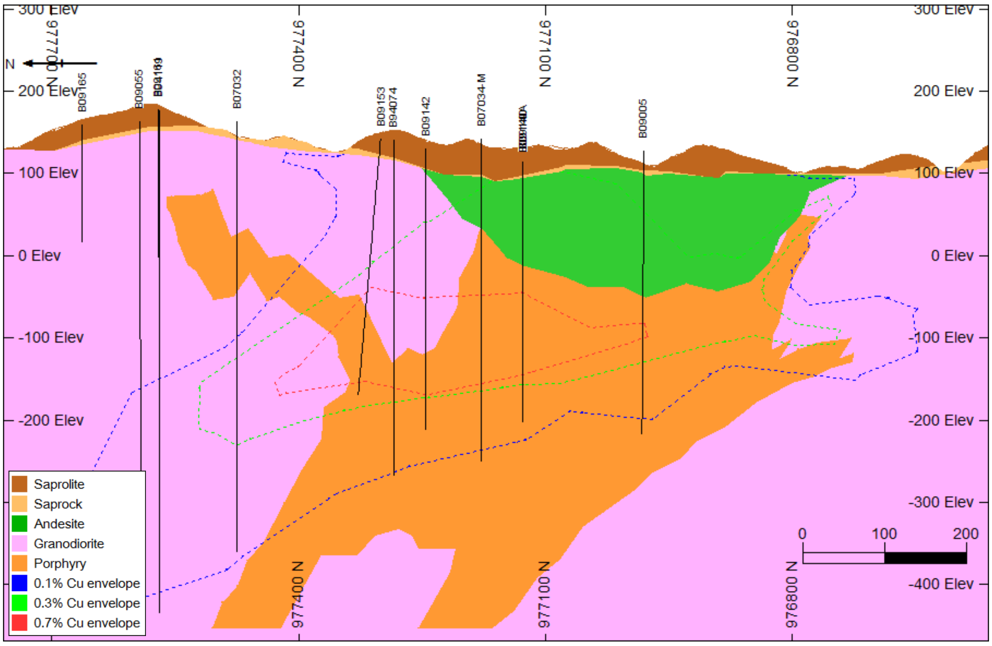


Figure 7-7 Botija south-north geology cross-section at 538140 mN



Since previous studies by Halley (2013) and Model Earth (2013), work continues at Botija in order to improve understanding of the geochemistry, mineralogy and controls on the mineralisation. Apart from the start-up of reverse circulation (RC) grade control drilling at Botija, that now covers the top 45 m of mineralisation, an additional eight diamond drilled holes were completed for metallurgical test work. Results from RC drilling have confirmed the position, size and grade tenor of the initial 45 m of Botija mineralisation. Similarly, results from the metallurgical test work have confirmed lower recoveries within the saprock horizon and consistent recovery behaviour for the fresh styles of mineralisation.

The previous work by Halley (2013) was based on re-assay of almost 1,000 sample pulps from across the deposit using a four-acid digest ICP-MS ultra trace method. Results were then used to validate logging, map fractionation trends in the host porphyries, quantify alteration signatures and pathfinder element haloes. Findings of the study are summarised below:

- Botija, and many of the other deposits at Cobre Panamá, have a sulphide and silicate alteration zonation pattern that is typical of porphyry copper systems worldwide. However, the geometry of the zonation is mushroom-shaped rather than cylindrical.
- The highest copper tenors are associated with an area of potassic and/or strong sericite alteration.
- The central alteration zone is characterised by bornite alteration and is typically located in the thickest part of the porphyry sill complexes.
- Sulphide mineralogy domains can be defined by plotting samples on a Fe:Cu:S ternary plot and then used to validate the geological logging.
- The pathfinder elements (W, Sn, As and Sb) have a similar distribution pattern as seen in other systems but the magnitude of the anomalism is much lower and subdued.
- Elements such as Sc and V are useful in determining the bulk host rock composition, and can be used to validate the logging of mafic (andesite) versus felsic (granodiorite and porphyry) rocks. Geochemically, the granodiorite and porphyry host rocks are difficult to distinguish from one another.
- The andesite compositions vary greatly, suggesting a possible fractionation sequence.
- Dykes are predominantly intermediate in composition and unrelated to the porphyries and granodiorite.

Propylitic alteration is irregular, occurring sporadically across the deposit. Andesite tends to be characterised by Propylitic A alteration. Potassic alteration is widespread in the central part of the deposit and has a general association with porphyry and higher copper grades. Anhydrites (and its supergene product gypsum) are commonly logged at Botija. Phyllic alteration is irregular, occurring mainly in the central area of the deposit near surface and at depth. Argillic alteration is also irregular typically occurring within 250 m of surface.

Detailed field mapping and interpretation of structures at Cobre Panamá was completed by Model Earth Pty Ltd in September 2013. The main focus at Botija was to complete detailed mapping of the deposit and its surroundings to better understand the geology, assist in focussing any exploration and improve future mining (Model Earth, 2013).

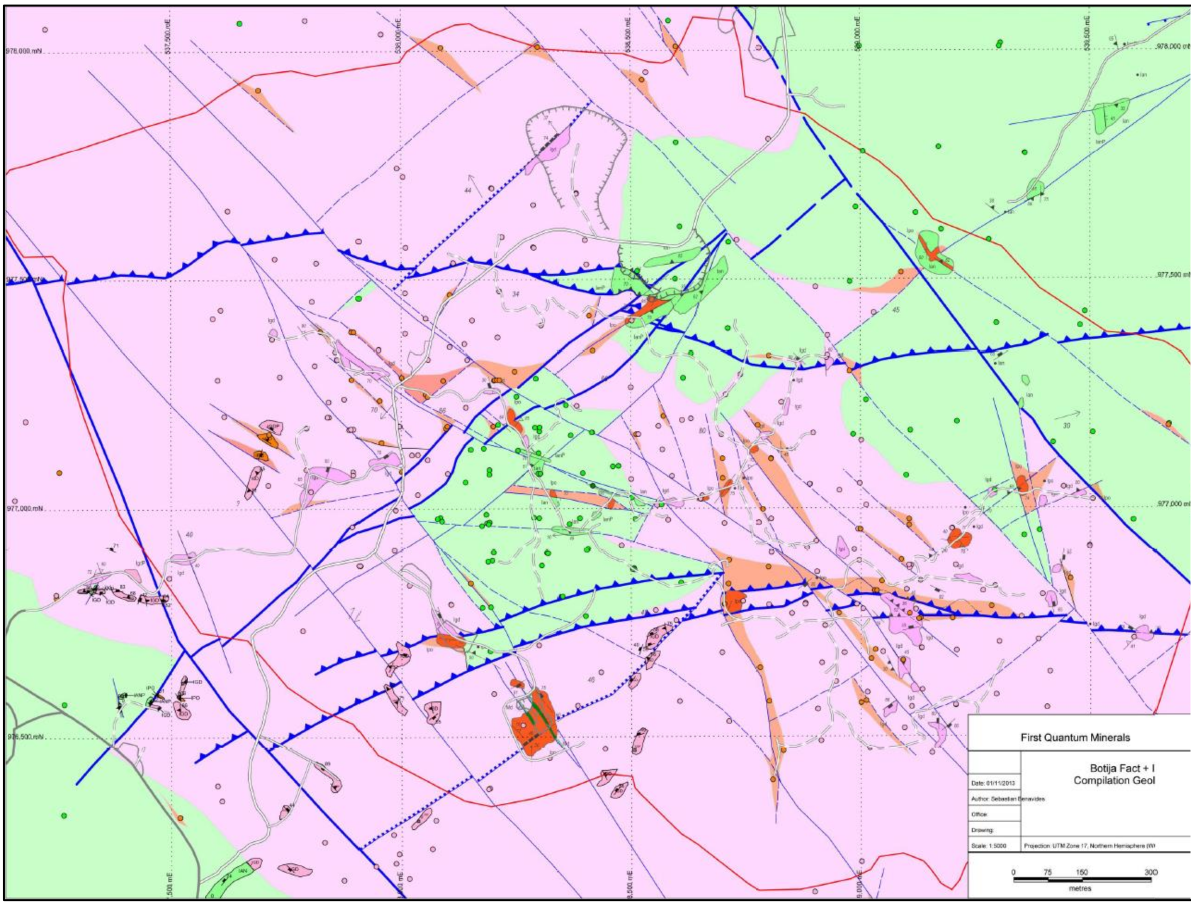
Due to thick vegetation over much of the concession, most of the mapping concentrated on outcrops of fresh rock exposures in gravel pits and quarries, some road cuttings and rarely in creeks and streams. Outcrops of weathered material at surface consisted largely of saprolite and saprock; however, relic textures were used to distinguish the parent rock types.

Mapping identified that structurally, the Botija deposit is characterised by low-angle faulting in both the footwall and hanging wall of the deposit. The most notable is the Botija River Fault in the footwall (Model Earth, 2013). This fault was found to be a wide zone, (~200m), with numerous discrete faults which dip moderately to the north (~30°), all with a reverse sense of movement. A number of northeast and northwest trending strike-slip faults were recorded in the Botija area (Model Earth, 2013). These are responsible for truncations and offsets in lithological contacts and earlier formed structures. The reverse fault in the hangingwall of the deposit seems to be offset in a dextral sense by a northeast trending dextral fault (Model Earth, 2013). All faults identified at Botija are presented in Figure 7-8.

Table 7-3 Vein classification at Cobre Panamá based on the work of Gustafson and Hunt (1975) at El Salvador, Chile (modified from Gustafson and Hunt (1975))

Vein Type	Silicate Assemblage and Texture	Alteration Halo	Structural Style	Sulphide Assemblage and Texture
“A”	Quartz-K-feldspar-anhydrite-sulphide with rare traces of biotite quartz content ranges from 50 to 90 %. Vein minerals are typically fine grained and equigranular and evenly disseminated with the exception of K-feldspar which can occur as bands forming along the edges or centre of the vein.	Halos of K-feldspar are more or less developed adjacent to most veins. These halos may be variable in thickness, especially in K-feldspar altered host rock.	"A" quartz veins are the earliest of all veins, often cross cut by "B" quartz veins. Veins can be typically randomly oriented and discontinuous, commonly segmented and "whispy". Widths range from 1 to 25 mm with strike continuity from centimetres to several metres.	Disseminated chalcopyrite-bornite, with proportions usually similar to the background sulphide; locally, traces of molybdenite.
“B”	Quartz-anhydrite-sulphide, with K-feldspar characteristically absent. The quarts is relatively coarse grained and tends to be elongated perpendicular to the walls, occasionally approaching a "cockscomb" texture. Granular quartz, especially in sheared bands is common. Vein symmetry of sulphides, anhydrite or granularity along centrelines, margins or irregular parallel bands is typical but unevenly developed.	Lack of alteration halo is characteristic. Occasionally faint and irregular bleached halos are present, but most are probably due to superimposed veining.	Younger than "a" type veins and older then "D" veins. "B" veins are characteristically regular and continuous, and tend to have flat attitudes. Widths usually range from 5 to 50 mm with strike continuity in the range of meters to tens of metres.	Molybdenite-chalcopyrite is characteristic. Traces of bornite occur in some, but more commonly minor pyrite occurs in contrast to bornite-chalcopyrite in the surrounding host rock. Sulphides tend to be coarse grained and occupy banding parallel to the vein contacts or cracks perpendicular to them.
“D”	Sulphide-anhydrite with minor quartz (except where superimposed on "B" veins) and occasional carbonate. Anhydrite locally forms coarse crystalline masses and is commonly banded with the sulphides.	Feldspar-destructive halos are characteristic. Proximal Sericite or sericite-chlorite halos may or may not be associated with distal halos of kaolinite-calcite.	"D" veins cross cut all other veins. They are continuous although locally irregular and interlacing but occupy systematic structural patterns. Widths range from 1 to 75 mm with strike continuity from meters to tens of metres.	Pyrite is usually predominant, with chalcopyrite, bornite, enargite, tennantite, sphalerite and galena common. Minor molybdenite and many other sulphides and occur locally. "Reaction" textures are typical.

Figure 7-8 Detailed surface fault interpretation from field mapping and collated with drillhole data



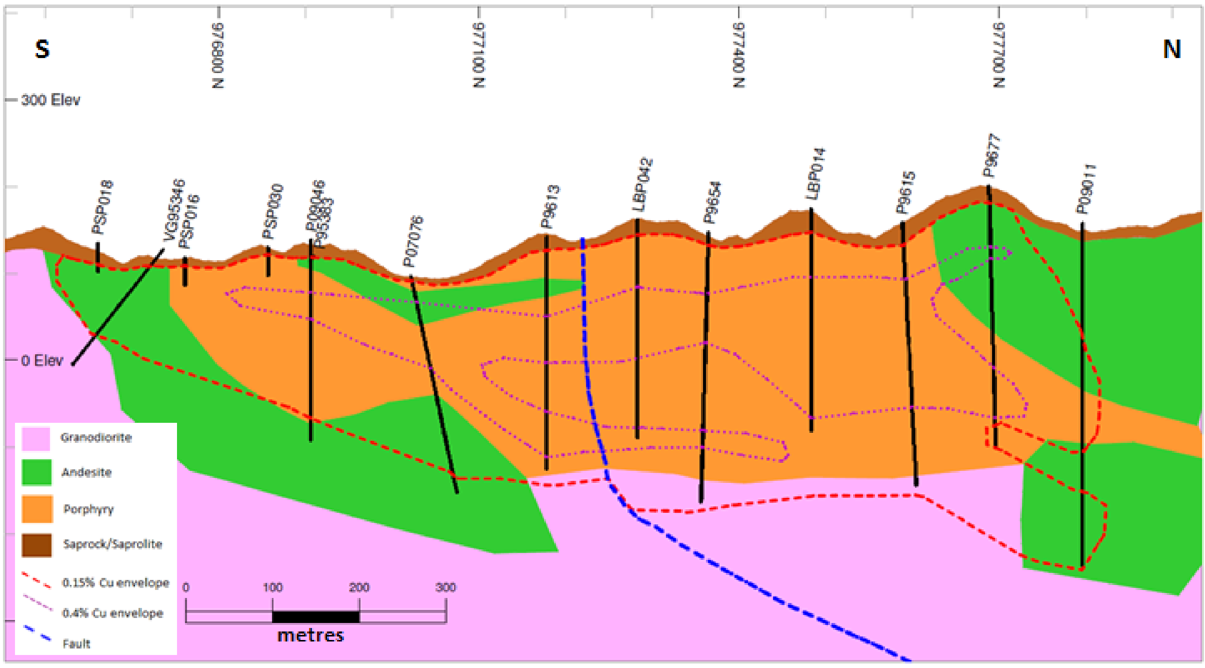
Copper mineralisation at Botija is characterised by a quartz stockwork of "A" and "B" veinlets within the central porphyry dyke complex and throughout the contact zones. This stockwork forms a locus for high grade copper and molybdenum mineralisation (Gustafson and Hunt, 1975) and hosts most of the chalcopyrite and some minor bornite mineralisation. The majority of molybdenite mineralisation is predominantly associated with "B" type veins and is generally located in the lower copper grade zones.

7.5 Colina

The Colina deposit is focused on a 3.0 km long by 1.2 km wide feldspar-quartz-hornblende porphyry sill and dyke complex (lopolith) that trends east-southeast. A geological model demonstrating the distribution of the main rock types was created based on north-south cross sections spaced at 100 m intervals. An example of a cross section through the Colina deposit, demonstrating the geology, faulting and mineralisation is presented in Figure 7-9.

The majority of the feldspar-quartz-hornblende porphyry comprises of 50 m to 200 m thick sills that dip shallowly to the north and are often interconnected by dykes. Both felsic and mafic dykes are logged across the deposit but due to inconsistencies of logging between drilling campaigns, as well as the narrow nature of the dykes (especially the sub-vertical mafic dykes) it is difficult to model their distribution with any great certainty.

Figure 7-9 South-north section along 533,800 mN – Colina deposit

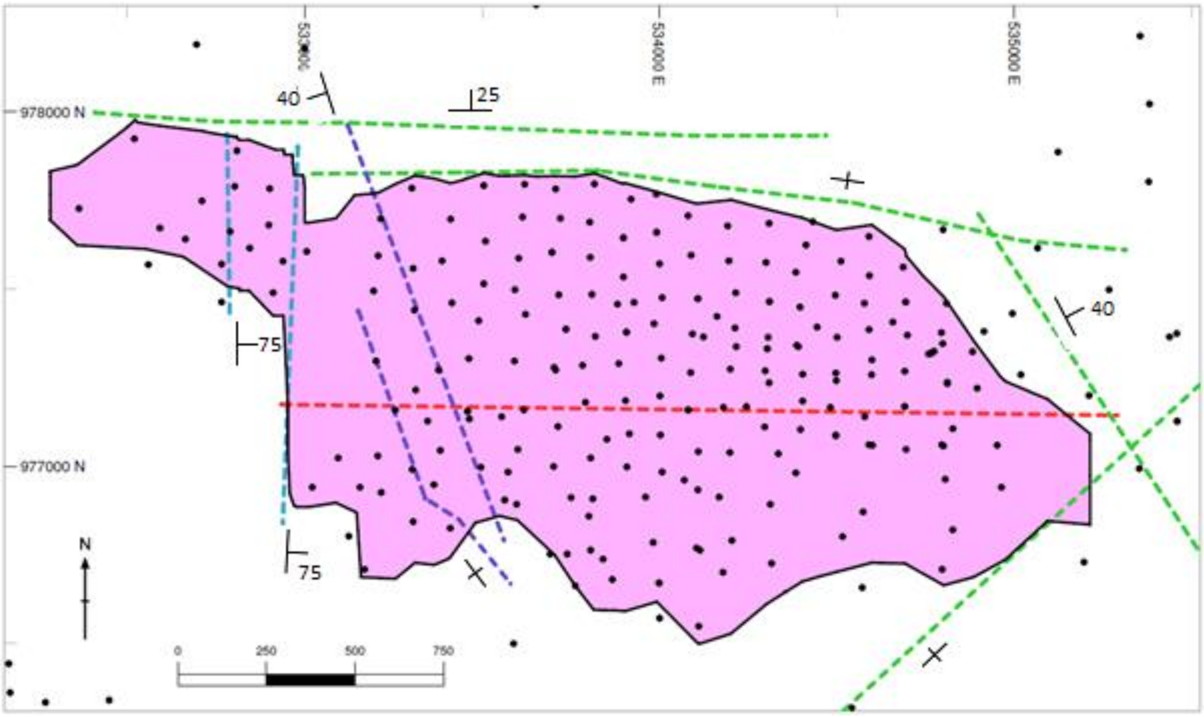


At the base of the porphyry sills a transition to granodiorite often occurs, accompanied by an increase in fracturing and in the general sense, adecline in copper tenor. These contacts are interpreted to correspond to the lower stoped contact and the fracturing is largely due to tensional forces exerted by the sinking granodiorite resulting in tensional failure. This area can have increased frequency of mafic dykes, thought to be a result of the ingress of a highly residual and late-stage fractionated magma. At the northern contact of the dyke, the contact between the granodiorite and andesite is intermixed and the morphology and nature of the contact is complex.

Propylitic alteration generally affects the andesite in the periphery of the Colina deposit and the andesite in the central part is frequently affected by silica-chlorite alteration. Phyllic alteration is patchy and difficult to interpret as a continuous zone. In general, it is not associated with the higher-grade part of the deposit. Potassic alteration is rare at Colina compared with Botija. Anhydrites (and its supergene product gypsum) are also not as commonly logged as at Botija. This is thought to be related to the lack of potassic alteration. Magnetite alteration is common in the western and northern parts of the deposit. Where it is present, potassic alteration is logged as weak potassium feldspar with patchy biotite alteration of mafic minerals.

Several faults have been modelled at Colina based on offsets observed in the geological modelling (Figure 7-10). A thrust fault (red) intersecting the middle of the deposit strikes east-west and dips steeply to the north and offsets the mineralisation and lithology up to 50 m vertically. At least two normal, sub-vertical faults (light blue) have been modelled in the western area of the deposit, offsetting the main part of the deposit from two northwest zones (previously the Fadalito deposit). Faults shown in green (Figure 7-10) were used to truncate the mineralised wireframes and guide the low grade mineralisation interpretation. These faults have a variety of orientations and can be seen in regional geophysics. Two small faults (dark blue in Figure 7-10) have been used to make minor adjustments to the gold domains at Colina. Offsets along these faults were sometimes difficult to distinguish in the lithology and copper mineralisation.

Figure 7-10 Fault locations at Colina based on lithological and mineralisation offsets interpreted from the geological model



Copper mineralisation at Colina appears to be loosely associated with the feldspar-quartz-hornblende porphyry, particularly in the thicker areas were the dykes coalesce and around the upper contact of the sills. The best copper grade intercepts are often associated with intense magnetite, quartz-magnetite and quartz veinlets containing chalcopyrite (Sillitoe, 2013). Pyrite and chalcopyrite with lesser bornite occur mainly in these quartz-rich veinlets, frequently with molybdenite. Vein types have not been systematically mapped at Colina but it is interpreted that they are spatially associated with the contact areas of the porphyry sills and dykes especially in areas of high complexity. Contact metamorphism along the andesite-porphyry boundary is characterised by strongly silicified or biotite-altered zones. Within these zones, pyrite and chalcopyrite are generally found in veinlets, with lesser blebs and disseminations. Molybdenite and magnetite have also been noted, associated with "B" type veinlets (described in Table 7-3).

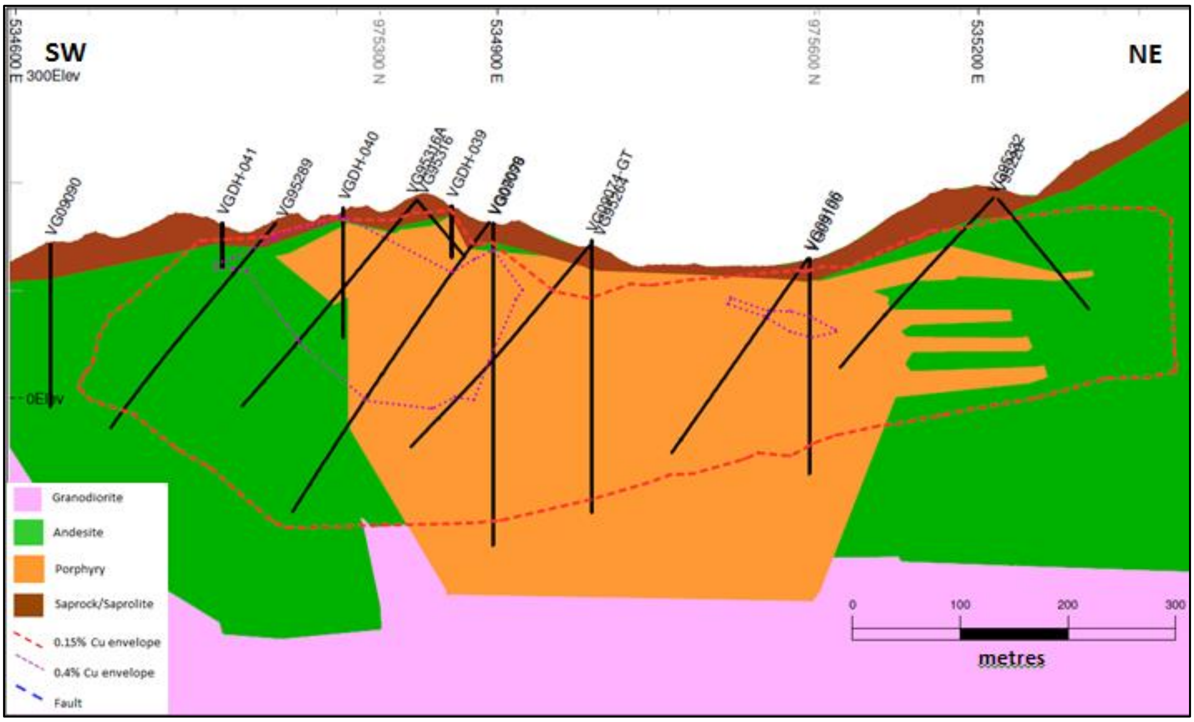
Secondary copper minerals, including chalcocite and covellite, have been observed at Colina, occurring mainly as sooty coatings on chalcopyrite at the base of the saprolite or adjacent to structures penetrating the underlying sulphide domains from surface. Locally, malachite has been observed within zones where total oxidation of the sulphides has occurred.

A large blanket of depleted weathered material (both saprock and saprolite) covers Colina. Within this horizon copper mineralisation is low grade and/or depleted, with no apparent supergene copper enrichment. Localised gold enrichment within this zone has been identified and these zones have been domained out for estimation purposes.

7.6 Valle Grande

The Valle Grande deposit is located to the southeast of Colina and is 3.2 km long and 1 km wide, striking northwest-southeast (Figure 7-6). The deposit is focussed on an irregular feldspar-quartz-hornblende porphyry lopolith. The geological model for Valle Grande was created using sectional interpretations orientated 40° east of north along 100 m section spacings. An oblique cross section of the Valle Grande geological model is presented in Figure 7-11.

Figure 7-11 Oblique section looking northwest through the Valle Grande deposit



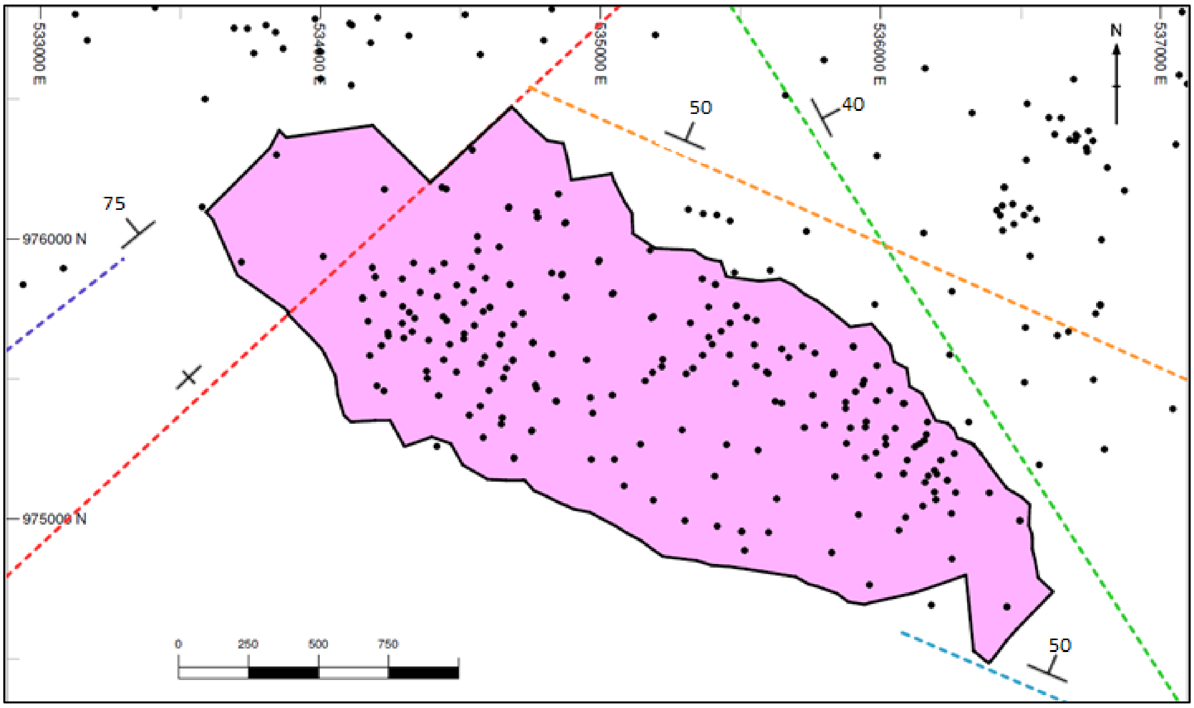
The most commonly logged alteration type logged at Valle Grande is propylitic. This is likely due to the dominance of andesite as the host rock. Silica-chlorite alteration is irregular, often occurring deep within the feldspar-quartz-hornblende porphyry dyke. Potassic alteration distribution is erratic, and often occurs at depths of at least 200 m. The most coherent areas of potassic alteration occur in the central part of the dyke. Phyllic alteration is also irregular and is commonly associated with the contact between the porphyry dykes and the adjacent host rocks. Argillic alteration is most common in the top 150 m of the deposit, often as a continuous zone. Patchy argillic alteration has also been logged at 200 m to 300 m below surface, but does not correlate well between drill sections.

Several faults have been identified at Valle Grande as presented in Figure 7-12. Mineralisation at Valle Grande is displaced by the northeast–southwest fault (red). The northwest-southeast fault (green) and the north northwest-south southeast fault (blue) were used to terminate the mineralisation interpretations.

In general, high grade copper mineralisation at Valle Grande is concentrated along the flanks of an early porphyry lopolith, following the porphyry-andesite contacts. In these areas, quartz-sulphide veinlet densities are high and magmatic-hydrothermal breccias consisting of polymict clasts supported in a matrix of quartz and chalcopyrite are common (Sillitoe, 2013). Conversely, in the central core of the porphyry, vein densities are reduced in conjunction with the copper tenor.

Narrow, post-mineralisation andesite dykes have been logged at Valle Grande with many interpreted to have a sub-vertical orientation.

Figure 7-12 Faults identified at Valle Grande



7.7 Balboa

The Balboa deposit was discovered in late 2010 after testing of a geophysical target located 500 m northwest of the Cuatro Crestas locality. The deposit was drilled on a 100 m by 100 m grid during 2011. A geological plan is presented in Figure 7-13. A geological model was created based on 100 m spaced cross sections throughout the deposit. A northwest-southeast orientated cross section along line D-D' is presented in Figure 7-14 demonstrating the distribution of the different lithologies, fault offsets and the interpreted mineralisation.

Mineralisation at Balboa is dominantly hosted by a feldspar-quartz-hornblende porphyry that intrudes the adjacent andesite at a low to moderate angle, emanating from the north-northwest (Love, 2011). Mineralisation is best developed in the central portion of the porphyry but weakens towards the contacts with the andesite. The porphyry can locally be described as a crowded feldspar porphyry, with variable percentages of feldspar and lesser quartz phenocrysts which range in size from 1 mm to 4 mm.

Figure 7-13 Geological plan of the Balboa deposit (Source: Rose *et al*, 2012)

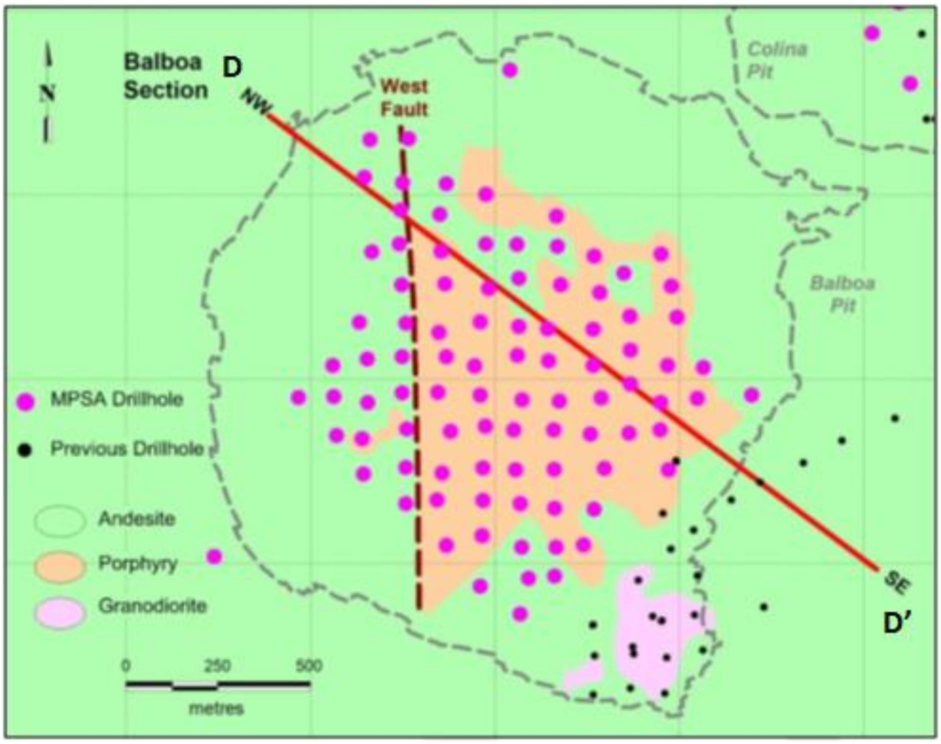
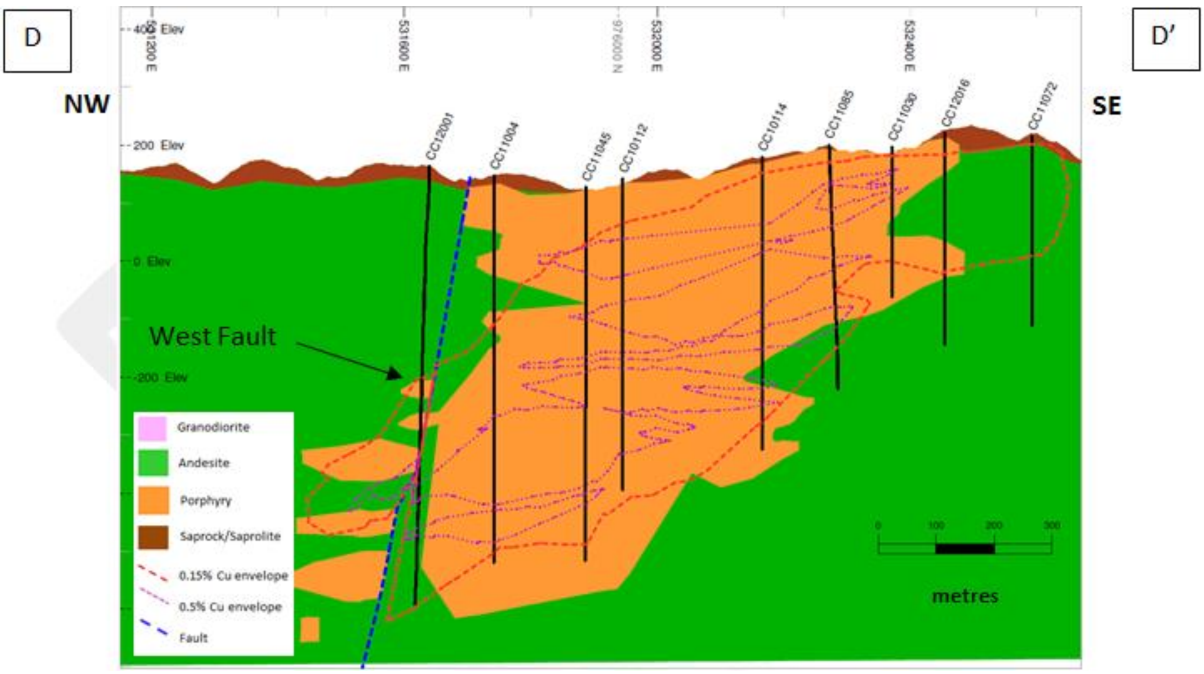


Figure 7-14 Oblique section looking northeast through the Balboa deposit



Near the surface of the central part of the deposit there is a massive, grey, weakly porphyritic unit distinguished by variably silicified and vuggy cavities. Extending to a maximum depth of 150 m below surface, this unit is relatively low grade, with variable pyrite alteration but demonstrates little evidence of oxidation and hence is not thought to be a leach cap. It is locally referred to as porphyry "B" within the logging, and is thought to be a potential lithology cap to the mineralisation.

Andesite rocks throughout Balboa demonstrate variable propylitic alteration. Mineralisation is weakly developed in the andesite and is dominated by pyrite. Local hornfels alteration is present and is characterised by increased biotite and magnetite.

Minor granodiorite is present but at depth. It is characterised as a massive, equigranular and typically unaltered unit with common hornblende crystals.

A variety of intermediate to felsic dykes have intruded Balboa, both during and post mineralisation. These are typically massive, unaltered and contain grey to green feldspar phenocrysts. Orientations with the core are typically low to moderate suggesting that the dykes are steep to moderately dipping. These dykes are found throughout the deposit but are more abundant at depth and to the north of the deposit. Their overall abundance is estimated at 7 % with an average thickness of 6 m but with some reaching 20 m thick.

Alteration at Balboa is currently not well understood as it does not display the typical alteration mineral assemblages seen within other deposits of the Cobre Panamá concession. Potassic alteration at Balboa is extremely limited, even within highly mineralised and stockwork-rich areas. Alteration zones are dominated by pervasive silica and argillic assemblages of variable intensities. The presence of magnetite is a common characteristic of the Balboa deposit compared with Botija where subsequent overprinting alteration phases have reduced the magnetite content. Propylitic assemblages of chlorite and epidote are common throughout all rock types. In summary the alteration pattern at Balboa is described as silicic to argillic, grading out to propylitic. Local phyllic alteration assemblages have also been logged, but do not form consistently across the deposit (Rose *et al*, 2012).

Balboa is cross cut on the western margin by a north-trending fault termed the West Fault (Figure 7-14). The stratigraphy to the west of the fault demonstrates a normal displacement of up to 300 m. In the north-northwest of the deposit this fault intersects the main mineralised keel of the deposit with some of the thickest and best mineralised intercepts are present suggesting that the West Fault may be an important control on mineralisation at Balboa. Other small metre scale faults have also been logged at Balboa but any offset across these structures has not been established.

Mineralisation at Balboa is dominated by chalcopyrite with traces of bornite which occurs as fine grained dissemination within the host porphyry and quartz stockworks. The quartz stockworks form distinctive “A” and “B” type veinlets (Table 7-3) and are present throughout the central, higher grade zones of the deposit. The quartz veins can exhibit vuggy or open space textures but typically the central portion of the veins are filled with chalcopyrite and/or bornite mineralisation. It is the presence of these intense stockwork zones and massive chalcopyrite veins which distinguish Balboa from the other Cobre Panamá deposits.

No significant supergene chalcocite or copper oxide mineralisation is present at Balboa.

7.8 Minor deposits

7.8.1 Medio

Medio is located immediately east-northeast of the Colina deposit and 2 km northwest of the Botija deposit. Mineralisation at Medio was first discovered by the UNDP crews following anomalous molybdenum in stream sampling in 1968. Drilling has delineated a 1.3 km by 800 m area of low to moderate grade porphyry mineralisation. Mineralisation is associated with silicified and sericitised porphyritic intrusive rocks and brecciated andesite volcanics (Rose *et al*, 2012). Copper tenor appears to be strongly correlated to vein and fracture intensity.

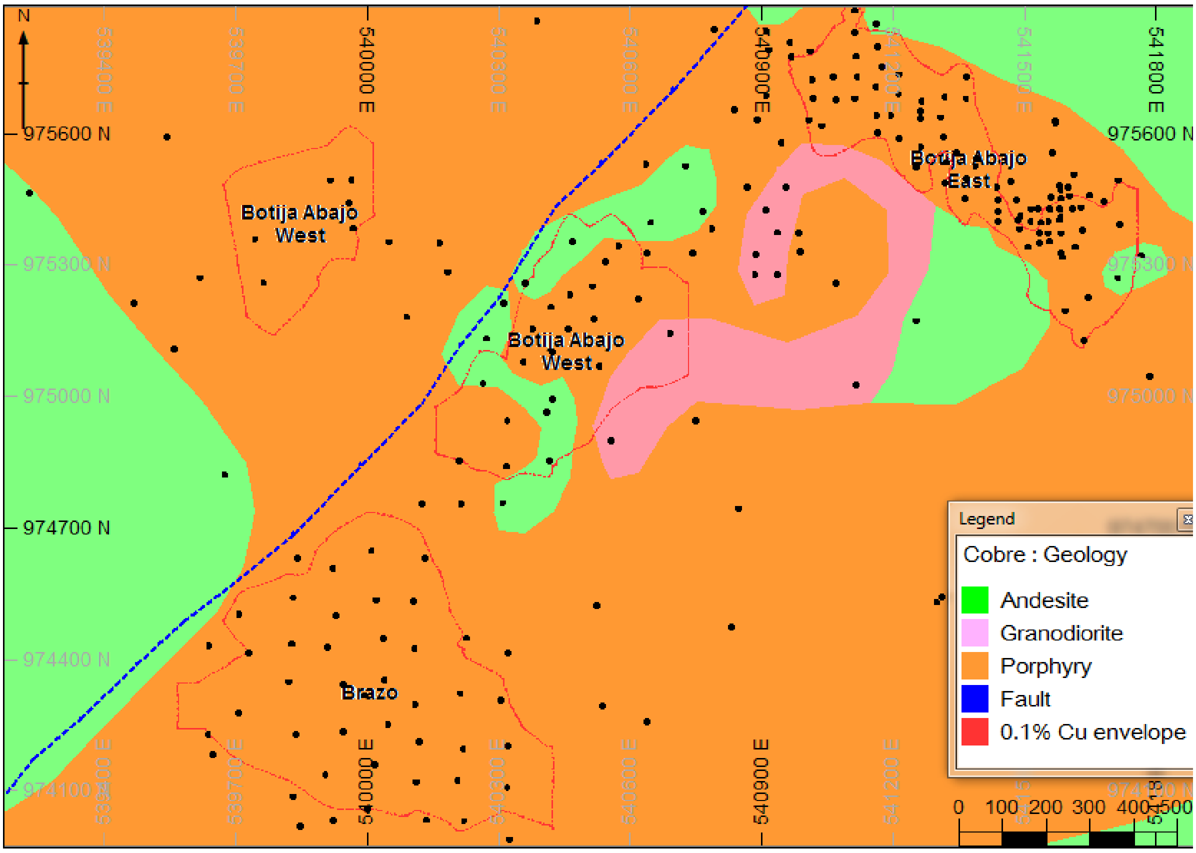
7.8.2 Botija Abajo

Botija-Abajo is approximately 2.5 km southeast of Botija. Drilling, completed mainly by PTC identified two deposit areas, Botija Abajo East and Botija Abajo West (Figure 7-15). Mineralisation is primarily located within feldspar-quartz-porphyry with some mineralisation extending into the andesitic tuffs. Alteration is dominantly argillic (kaolinite, quartz and pyrite with stockworks of quartz and chalcedony).

The eastern area has marked shallow, sub-surface leached copper followed by supergene enriched gold and chalcocite down to depths of 80 m, with hypogene copper to depths of 200 m below surface. Drilling across the eastern deposit has focussed on the shallow mineralisation, with the hypogene copper mineralisation still largely open at depth. The eastern area strikes approximately southeast with a strike length of 650 m and width of 150 m. The western area is dominantly hypogene copper (mainly chalcopyrite) and approximately 500 m by 350 m. Mineralisation extends to at least 150 m below surface.

A major northeast trending fault seperates the east and west deposit areas. Most of the Botija Abajo copper mineralisation remains open at depth.

Figure 7-15 Plan view of the Brazo and Botija Abajo deposits showing drillhole collars and surface geology



7.8.3 Brazo

The Brazo deposit (Figure 7-15) is located approximately 3 km south-southeast of Botija. Copper and gold mineralisation was identified in a feldspar-quartz porphyry with dominant sericite alteration during drilling by Adrian Resources. Pyrite and quartz occur throughout. The Brazo area is characterised mainly by porphyry with very little granodiorite or andesite occurrences. The position and extents of the Brazo deposit were confirmed and developed by additional drilling completed by MPSA. Supergene processes have resulted in some chalcocite development within the first 150 m below surface. Immediately below the supergene zone, chalcopyrite, pyrite and minor bornite occur down to depths of 350 m. The Brazo deposit has an approximate area of 600 m by 700 m and remains open to the east, northeast and at depth.

ITEM 8 DEPOSIT TYPE

The mineralised zones on the Cobre Panamá property are examples of copper-gold-molybdenum porphyry deposits (Lowell and Guilbert, 1970). Common features of a porphyry deposit include:

- Large zones (> 10 km^2) of hydrothermally altered rocks that commonly show a crudely concentric zoned alteration pattern on a deposit scale of a central potassic (K-feldspar) core to peripheral phyllic (quartz-sericite-pyrite), argillic (quartz-illite-pyrite-kaolinite) and propylitic (quartz-chlorite-epidote) altered zones.
- Generally low grade mineralisation consisting of a variety of sulphide mineralisation styles including disseminated, fracture, veinlet and quartz stockworks. Higher grade mineralisation is typically associated with increased densities of mineralised veins and fractures.
- Mineralisation is also typically zoned, with a chalcopyrite-bornite-molybdenite core and peripheral chalcopyrite-pyrite to pyrite domains.
- Enrichment of primary copper mineralisation by late-stage hypogene high sulphidation hydrothermal events can sometimes occur.
- Important geological controls on mineralisation include igneous contacts, cupolas and the uppermost, bifurcating parts of stocks and dyke swarms. Intrusive and hydrothermal breccias and zones of intensely developed fracturing, due to coincident or intersecting multiple mineralised fracture sets, commonly coincide with the highest metal concentrations.

It is the opinion of the QP, David Gray, that there is sufficient evidence from the exploration methods employed and the development thereof to support the characteristics of a mineralised porphyry deposit. The integrity and quality of diamond drilled data used during this Mineral Resource estimate was verified by the issuer through check re-logging and re-sampling programmes. Specifically, historical exploration has:

- used stream sediment and soil geochemical sampling to identify the locality of mineralised porphyry anomalies
- the targets identified from stream and soil geochemical sampling were confirmed and delinetated through diamond drilling, which has:
  - initially defined key deposit extents through a wide spaced grid of holes
  - larger deposits (Botija and Colina) were then infill drilled
  - diamond drilling methods with complete hole sampling were employed in order to maximise definition of the extensive scale porphyry style of mineralisation
  - diamond drill core samples were of an appropriate length and sample mass for the disseminated and veinlet style of porphyry sulphide mineralisation
  - drilled holes were sub vertically oriented and then directed at appropriate angles to otpimise definition of mineralisation shape and structures.
- Sample analysis included a robust set of multi element data in order to identify and define the characteristic altered and zoned behaviour typical to a porphyry deposit.

In addition, the issuer has developed the geology model for the Mineral Resource estimate and has considered the relevant characteristics of a porphyry deposit that is relevant to the domains of mineralisation, by modelling:

- Alteration zones for gypsum and anhydrite.
- Analysis of relative sulphide composition changes combined with grade changes as associated with different porphyry rock types and alteration.
- Each deposit model has considered the zoned nature of the prevailing mineralisation through careful definition of these respective and differently mineralised volumes.
- Domains of mineralisation include high grade copper and gold zones with chalcopyrite and bornite dominated sulphide mineralisation. These zones are similarly associated with higher degrees sericitic alteration.
- Domains of mineralisation have been geostatistically analysed and estimated in order to minimise domain mixing and grade smearing.
- Post-mineralisation intrusives that have no mineralisation have been considered in the volume models.

ITEM 9 EXPLORATION

The following summary information describes the exploration work completed prior to Project acquisition. Whilst no further exploration work has been carried out by the issuer since acquisition, the information is provided in the context of it being relied upon for certain interpretative aspects of the current Mineral Resource estimation update.

9.1 Initial discovery

Copper-gold-molybdenum porphyry style mineralisation was first discovered in Central Panamá during a regional stream sediment survey by a United Nations Development Programme (UNDP) team between 1966 and 1969. Follow up drilling by the UNDP in the 1969 led to the discovery of Botija East, Colina and Valle Grande. Later exploration by several companies has since outlined four large and several smaller deposits in the Cobre Panamá concession. An overview of the history of the Project has been presented in Table 6-1. Significant results of the historical exploration are presented below and additional information on the drill programmes is presented in Item 10.

9.2 Historical regional surveys

Adrian Resources completed soil and auger geochemical sampling across most of the concession between 1992 and 1995. Line spacing was 200 m with more detailed coverage (50 m – 100 m) around the known deposits. An approximate total of 8,000 soil samples were collected during this period at depths of between 5 cm to 20 cm below surface. Analysis was completed by TSL in Saskatoon, Canada for copper, gold and molybdenum. In addition, 3,600 auger samples were taken at depths ranging from 50 cm to 90 cm where anomalous Cu and Mo were detected. Further details on these programmes are presented in McArthur *et al.* (1995).

Between 1990 and 1992, Inmet (then Minnova) implemented reconnaissance-scale rock sampling (890 samples) and grid-based rock (172 samples) and soil (265 samples) sampling of the rock lithogeochemistry. Other sampling regimes included detailed sampling in areas of suspected mineralisation by Adrian and reconnaissance and silt sampling during mapping fieldwork in areas investigated for mine infrastructure locations by Teck. A summary map of the results is presented in Figure 9-1 and shows that near-surface copper mineralisation is present across several areas of the Cobre Panamá concession, including the upper Rio del Medio drainage and north of the Botija deposit.

Geophysical surveys collected over the Cobre Panamá project include a 105.2 km IP survey completed by Arce Geofisco of Lima, Peru, for PTC in 2008. The survey was completed on north-south oriented lines at a 200 m spacing using a pole-pole array with a spacing of 50 m and n=5. The survey demonstrates a well-defined chargeability associated with the Botija deposit and the eastern edge of the Valle Grande deposit with a number of smaller anomalies occurring along the southeastern trend between Botija and Botija Abajo deposits (Figure 9-2).

Figure 9-1 Summary of soil geochemistry over Cobre Panamá (source: Rose *et al*, 2012)

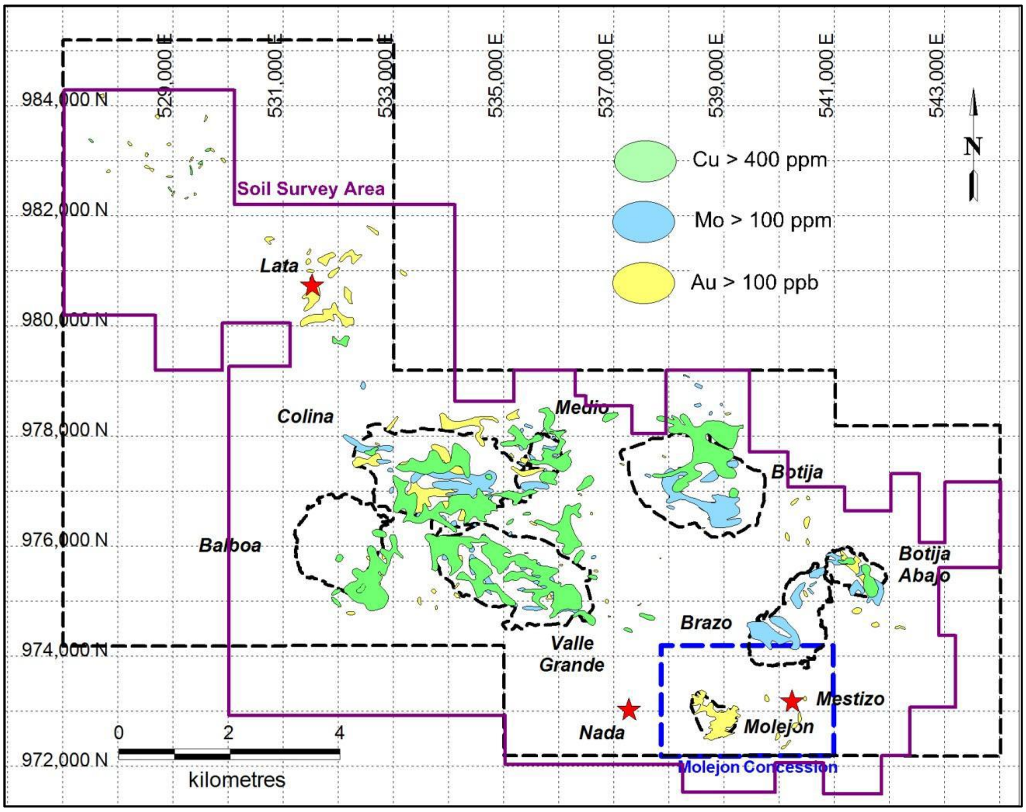
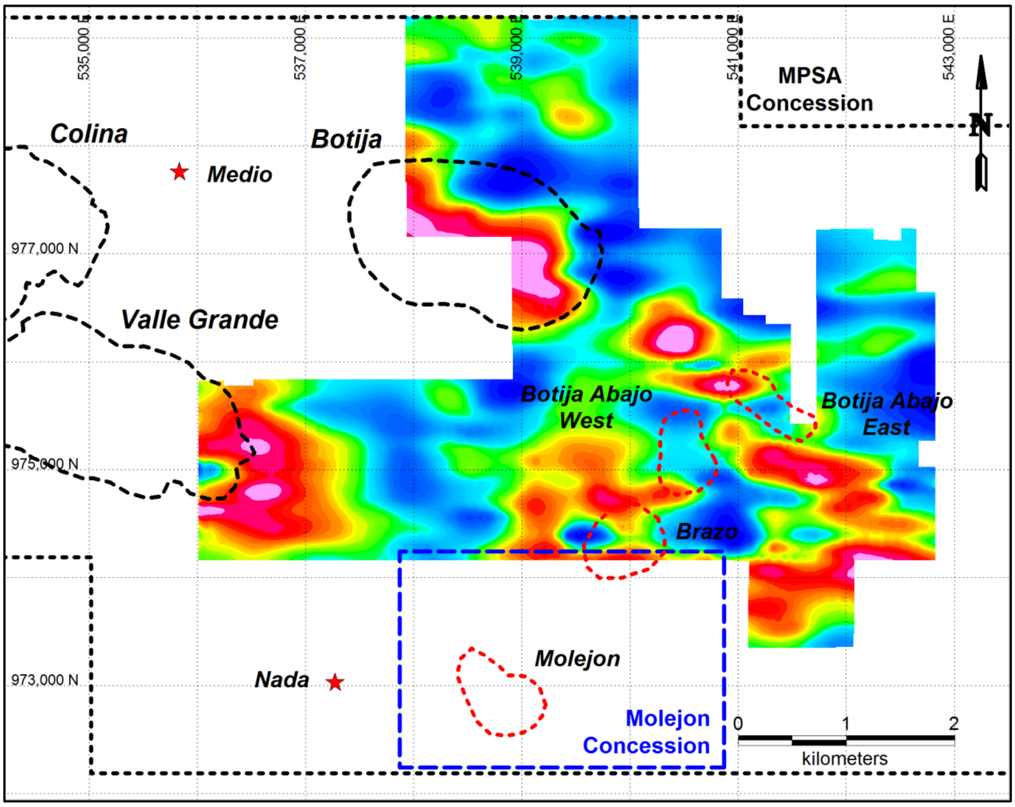


Figure 9-2 Plan of IP Chargeability at -150 m level (source: Rose *et al*, 2012)



ITEM 10 DRILLING

The following information describes the drilling activities carried out prior to Project acquisition as well as by the issuer since acquisition. The information is provided in the context of it being relied upon for the current Mineral Resource estimation update. The issuer has verified the drillhole core logging data with check re-logging and has verified sample analysis with a programme of check assaying.

10.1 Introduction

Since 1968 a number of drill programmes have been conducted over the concession area to test the extent of porphyry copper mineralisation. Details of the drill programmes across the Cobre Panamá Project area are summarised in Table 10-1 and Table 10-2.

Table 10-1 Summary of drilling by operator and area to August 2013

Programme	Years	BOTIJA		COLINA-MEDIO		VALLE GRANDE	
		No of holes	Metres	No of holes	Metres	No of holes	Metres
UNDP	1968-1969	25	1,336.7	25	1,322.6	8	628.9
PMRD	1970-1976	20	5,249.3	30	7,236.2	1	207.1
Adrian	1992-1995	58	17,789.9	49	10,218.7	114	24,185
Teck	1996-1997	47	11,356	74	17,519.1	19	3,081.8
PTC	2006-2008	45	2,229.6	4	268.2	44	2,271.6
MPSA	2007-2018	259	57,520.2	95	27,630.6	79	15,854.7
Total		**454**	**95,481**	**277**	**64,195**	**265**	**46,229**

Programme	BALBOA		BOTIJA ABAJO - BRAZO		OTHER		TOTALS	
	No of holes	Metres	No of holes	Metres	No of holes	Metres	No of holes	Metres
UNDP	-	-	-	-	-	-	58	3,288
PMRD	-	-	-	-	-	-	51	12,693
Adrian	5	669.3	30	5,064.5	140	18,063.8	396	75,991
Teck	-	-	7	600.7	20	2,385.8	167	34,943
PTC	22	3,272.6	193	2,2341	-	-	308	30,383
MPSA	94	49,545.3	65	22,008.5	241	18,918	833	191,478
Total	**121**	**53,487**	**295**	**50,015**	**401**	**39,368**	**1,813**	**348,775**

NOTE: UNDP: United Nations Development Programme, PMRD: Panamá Mineral Resources Development, PTC: Petaquilla Copper, MPSA: Minera Panamá SA

As at February 2018, 1813 holes had been drilled for 348,775 m. Since acquisition, the issuer has drilled 8 diamond drilled holes across the Botija deposit as well as 2,911 by 45 m deep reverse circulation grade control holes across Botija's pre-strip area. End November 2018 was the cut-off date for drilling data used for the Mineral Resource estimate described in Item 14.

10.2 Historical drilling

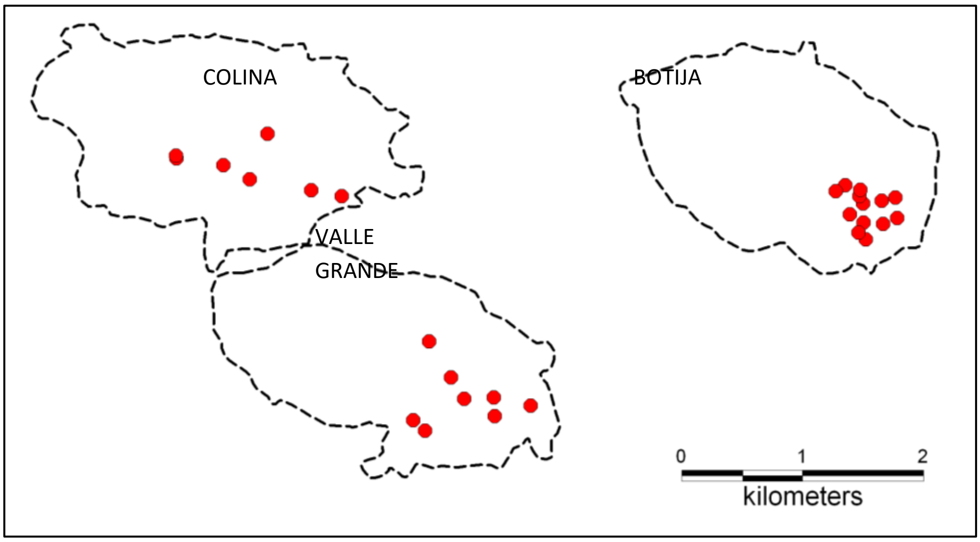
10.2.1 United Nations Development Programme (1967-1969)

From October 1968 to June 1969 the UNDP team drilled 58 holes (3,288 m) at Botija, Colina and Valle Grande. Core recovery averaged 70%. No core exists from this phase of drilling, and geology from the drill holes was used to develop the geological model. The assay information was not used in the estimation process. A map of the UNDP drill hole collars is presented in Figure 10 1.

Table 10-2 Summary of drilling by area (current to August 2013)

Area	Number of Holes	Total Metres
Botija	454	95,481
Colina-Medio	277	64,195
Valle Grande	265	46,229
Balboa	121	53,487
Botija Abajo-Brazo	295	50,015
Port	55	1,248
Tailings	144	7,006
Plant	22	2,850
Botija S dump	21	2,983
Others	159	25,281
Total	**1,813**	**348,775**

Figure 10-1 Plan view of UNDP drill hole collar locations (Source: MPSA, 2014)



10.2.2 Panamá Mineral Resources Development (PMRD)

Between 1970 and 1976, PMRD drilled 51 holes (12,693 m) to test the extent of mineralisation in the main deposits. Holes tested the Botija and Colina deposits using a drill spacing of approximately 200 m. One hole was drilled at Valle Grande. These holes have been re-surveyed. The assay and geological information from these holes is included in the database and they have been used in the geological modelling for the Colina and Botija deposits. A map of the collar locations from this programme is presented in Figure 10-2.

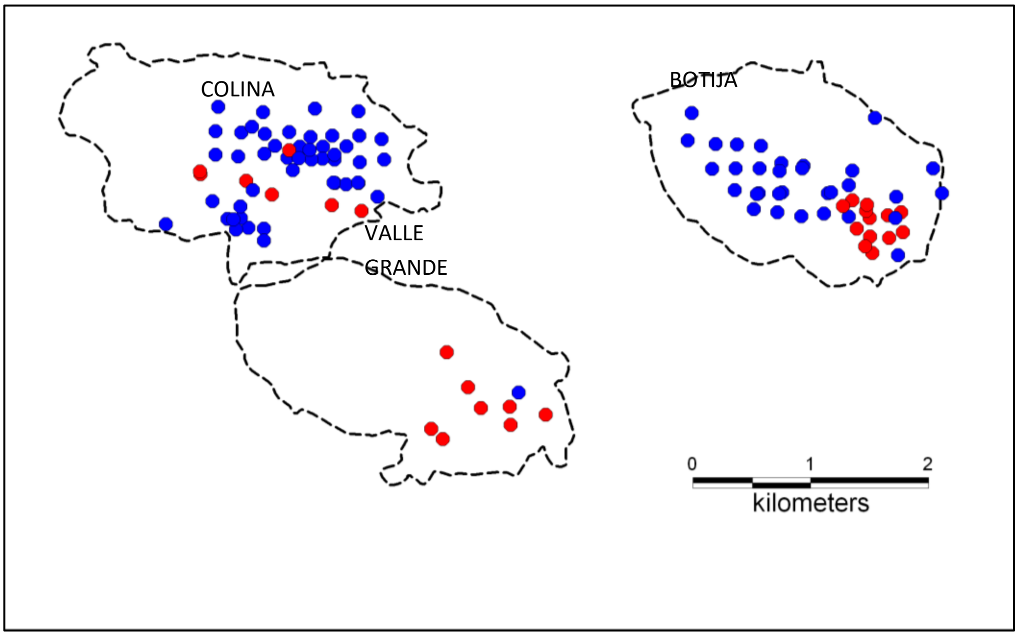
Core recovery averaged 95.5% (PMRD, 1977). Core recovery was poor in weathered intervals but excellent in fresh rock. No drill core, sample rejects or pulps from this drilling campaign remains.

10.2.3 Adrian Resources (1992-1995) (Inmet-Adrian – Georecursos)

Adrian Resources, a Vancouver-based company, ran the project from 1992 to 1995 and drilled a total of 396 drill holes for 75,991 metres (Figure 10-3). Drilling was completed using three F-1000 hydraulic drills and one Longyear 38 by Falcon Drilling of Prince George, B. C. Helicopter transport around site was supplied by Coclesana SA with the main operation based out of the Botija camp.

Core recoveries were generally poor in overburden (20% to 80%) but very good, near 100%, in fresh material. Core diameter was thin-wall B (BTW) or NQ.

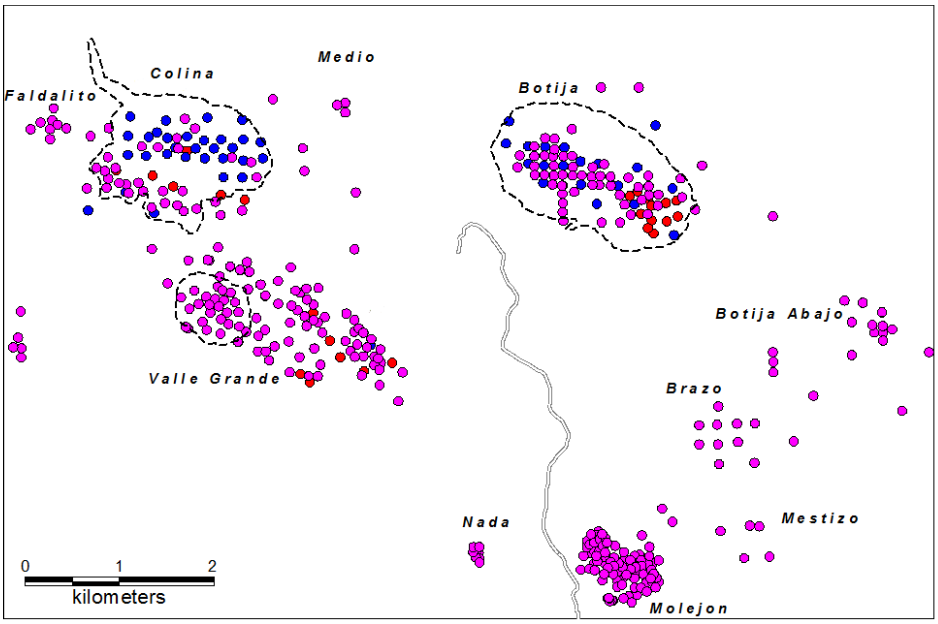
Figure 10-2 Plan view of PMRD (blue) and UNDP (red) collar locations (Source: MPSA, 2014)



At Botija, infill drilling was completed with vertical holes down to a spacing of 100 m. At Colina, vertical holes concentrated on testing the southwest gold zone and the main deposit to a drill spacing of 200 m. Drill spacing at Valle Grande varied from 100 m to 200 m with most of the holes drilled with an azimuth of 220° or 40° and a dip of -50° to intersect a hypothesised northwest-trending structural grain. Successes included the discovery and drillout of the Moléjon Gold zone. Exploration drilling of several smaller targets included Botija Abajo, Brazo and Medio.

Skeleton core from most holes is available and stored at the MPSA New Camp core storage facility. During the period 1998 to 2006, exposure of the core boxes holding the original core to weather and insects destroyed many boxes left at Colina Camp core racks. In 2006 efforts were made to recover the core by placing it in new boxes but some uncertainty remains as to whether cores are in their correct positions in the new boxes.

Figure 10-3 Drilling by Adrian Resources (pink), PMRD (blue) and UNDP (red) (source: MPSA, 2014)

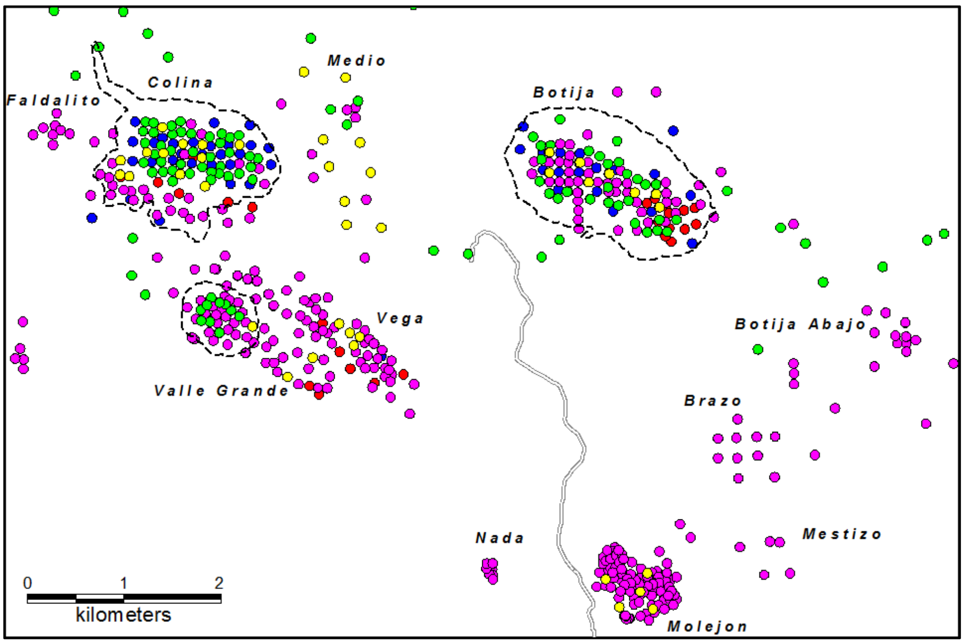


10.2.4 Teck (1996-1997)

Teck completed two phases of drilling across the Cobre Panamá concession as part of an infill drill campaign consisting of 167 holes totalling 34,943 m. Drilling concentrated on the Botija, Colina, Valle Grande and Moléjon deposits. Drilling commenced in 1996, with an initial 124 holes. The campaign continued in 1997, with drilling aimed at collecting metallurgical test samples from both Botija and Colina, as well as testing for higher grade zones at Valle Grande, exploration of Medio and step-out and continued infill drilling at Moléjon.

A plan of the drill collars, with the Teck drilling colour coded by year, is presented in Figure 10-4.

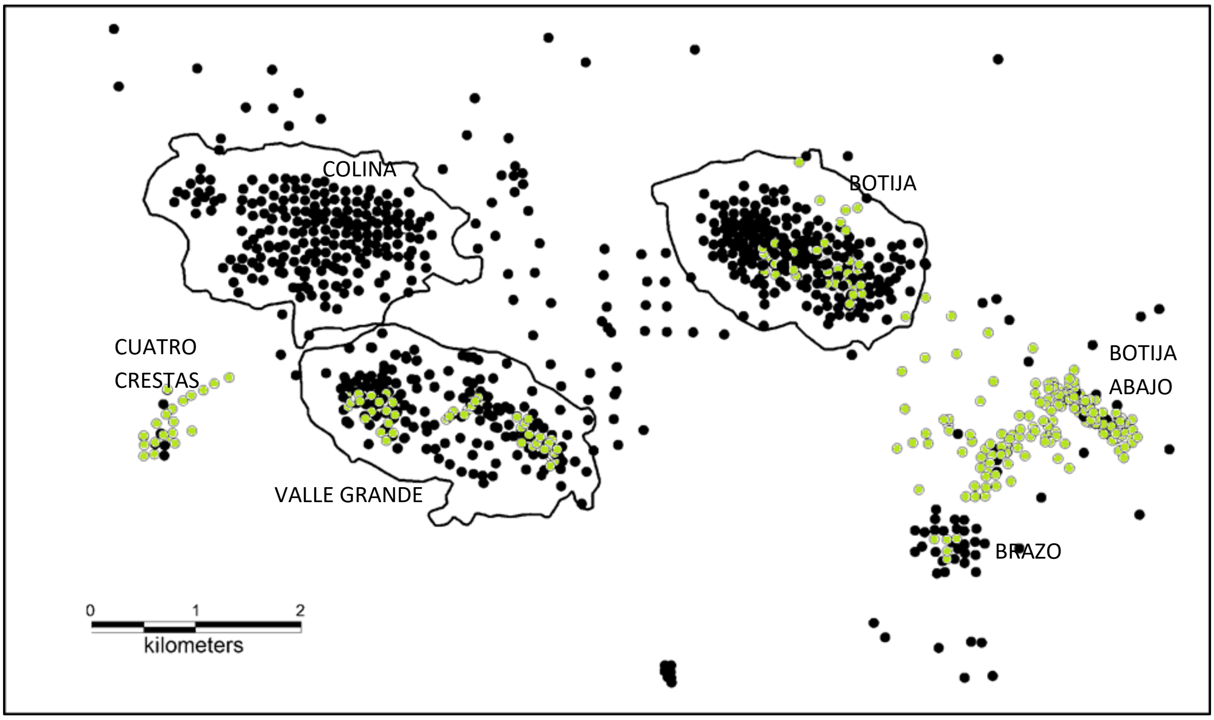
Figure 10-4 Drilling by Teck – 1996 (green) and 1997 (yellow), Adrian (pink), PMRD (blue) and UNDP (red) (Source: MPSA, 2014)



10.2.5 Petaquilla Copper (2006-2008)

From 2006 to 2008 PTC drilled a total of 308 holes for 30,383 m using helicopter-supported Longyear 38 drills. Holes at Botija and Valle Grande assessed the potential for oxide copper mineralisation while drilling at Botija Abajo assessed potential for gold mineralisation. In addition, several exploration targets, including Brazo, Cuatro Cresta and Lata were drilled (Figure 10-5).

Figure 10-5 Drillhole collar map with PTC holes coloured green (source: MPSA, 2014)



10.2.6 MPSA (2007-2018)

During the period October 2007 to August 2013, MPSA drilled a total of 825 HQ holes totalling 188,996 m. Cabo Drilling Panamá Corp. provided the drilling services using a variety of drill machines. Equipment moves were completed using helicopters provided by Heliflight Panamá S.A.

More recently (2017-2018), the 8 diamond holes and 2,911 reverse circulation (RC) holes across Botija were drilled by the AKD International drilling company.

Drilling focussed on several key objectives:

- increase the drill density at Botija, Colina, Valle Grande and Balboa to calculate Indicated Mineral Resources
- collect metallurgical test work samples from Botija (seven holes), Colina (five holes), Valle Grande (four holes) and Brazo (one hole)
- complete geotechnical holes at Botija and Colina
- complete condemnation drilling at possible plant site locations and in the tailings dam area
- start-up of RC grade control drilling across the Botija pre-strip area

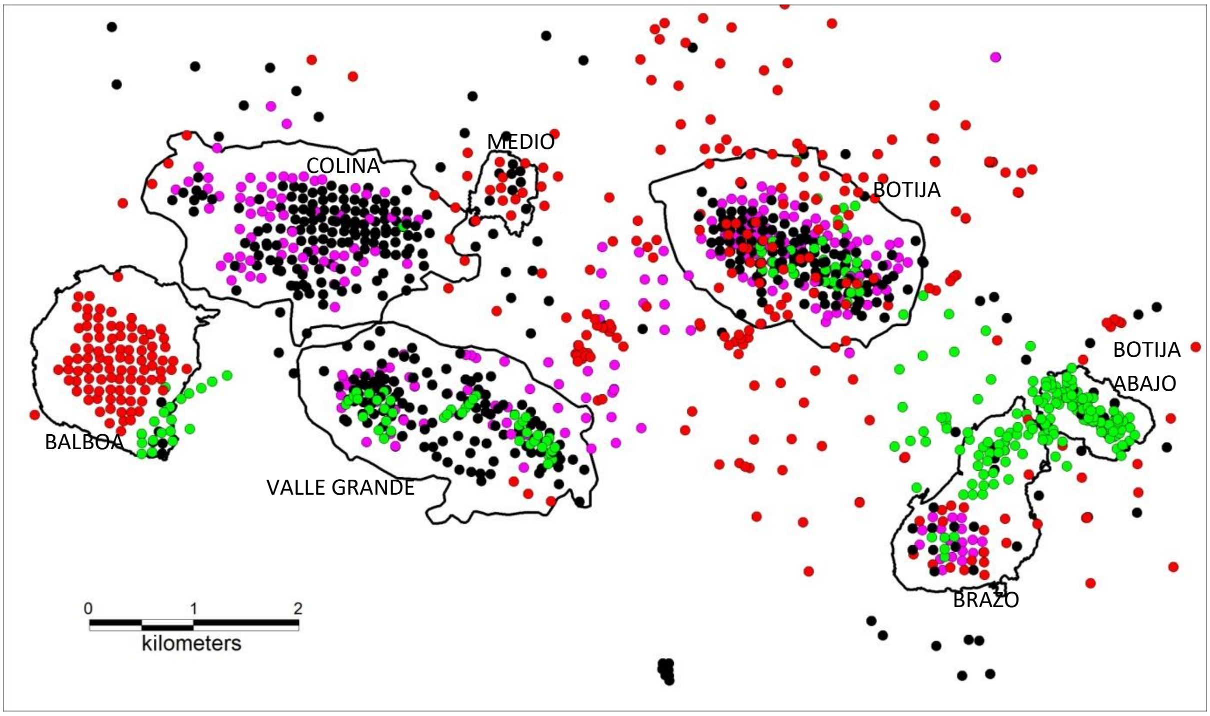
Between 2011 and 2012 the Balboa deposit was discovered and a 94 hole drill out was completed.

Core recovery data from MPSA, Teck and Adrian range from 1% to 100%, with an overall average of 93%. Less than 5% of the sample intervals have recoveries below 50%, whilst over 90% of the data have recoveries above 80%.

A map of the MPSA drill collars over the Cobre Panamá concession is presented in Figure 10-6.

In addition to diamond drilling, MPSA started with RC drilling across the Botija pre-strip area as a pre-cursor to future mining grade control. As at the end of November 2018, 2911 RC holes had been completed. The RC holes have been drilled 45 m vertically on a 15 mE by 15 mN grid.

Figure 10-6 MPSA (red) drill collar positions at Cobre Panamá; PTC (green), Teck (pink) and all other companies (black) are shown for reference (source: MPSA, 2014)



10.3 Collar surveying

Before April 2009, all drillhole collars were surveyed in North American Datum 27 (NAD 27), Canal Zone. Subsequent to this date, all historical collar locations were converted to and any new holes were located using WGS 84, Zone 17N, which is the geographic projection for all engineering work on the Project. WGS 84 coordinates are 19.7 m east and 207.1 m north of NAD 27, Canal Zone in the area of the concession.

10.3.1 Pre-2006 drill programmes

A number of holes drilled by Adrian were located using global positioning system (GPS) surveys conducted using a Trimble 4000 SE instrument and base station. Teck surveyed most holes in the Botija, Colina, Valle Grande and Moléjon deposits by conventional methods (total stations). Locations of many of the drillholes testing regional exploration targets are approximate, located using hip chain and compass traverses from known locations or on hand-help GPS readings.

10.3.2 Petaquilla Copper (2006-2008)

Holes drilled by PTC were surveyed using hand-held GPS. In late 2009, MPSA re-surveyed 46 of the 177 holes drilled by PTC in the Botija Abajo and Brazo area using a Topcon HiPer Lite plus differential GPS system and found no discernible difference between the two pickups. Coordinate conversion discrepancies were noted by FQM database administrators early in 2014, which resulted in a consistent 19 metre north south shift of some collars. All PTC drilled hole collars were re-surveyed by FQM during early 2014 and have been corrected in the database.

10.3.3 MPSA (2007-2013)

Holes drilled by MPSA during 2007 and 2009 were located in the field using a GARMIN GPS-60CSx hand-held GPS unit and were later confirmed using a differential GPS system and base station by contractors GeoTi S.A. All of the MPSA holes have been surveyed in this manner, to an accuracy of 5 cm.

In 2008 GeoTi SA were contracted to re-survey 61 historical drillholes (11% of the database). All collar co-ordinates were found to be within 5 m of the original historical co-ordinates except for one hole, B96-33, which was 10 m southwest of the original location. The large majority of these holes were adjusted southwest of the original survey position.

MPSA checked the location of another 29 holes using a hand-held Garmin GPS 60Csx during the 2007-2009 field programmes. All locations were validated except Botija hole LBB-038, which was located in the field 33 m west and 35 m south of the original historical surveyed collar coordinate. This hole was corrected in the database.

Three drill holes collars at the Medio deposit were discovered in the field to be located in the wrong places. They were re-surveyed by GeoTi SA and corrected in the MPSA database to the correct positions (ME9669, ME9664, and ME97-03).

Since 2009 all MPSA drill holes were surveyed by GeoTi SA using a differential GPS system and base station after the holes were completed.

10.4 Downhole surveying

10.4.1 Historical drilling

There are no records of downhole surveys for holes drilled prior to 1992.

Holes drilled between 1992 and 1997 by Adrian and Teck were surveyed downhole using a Tropari device or acid tests. Vertical holes commonly only had one test near the bottom of the hole as readings rarely deviated more than 1°. Inclined holes normally deviated with depth by several degrees and required two to three tests per hole. Spurious results were discarded. Probable sources of error include the abundance of magnetite in some area and rock types and possible equipment malfunction due to corrosion in the tropical climate.

No downhole surveys were completed on the PTC drilling. Most of the holes are vertical and shallow, and due to the HQ core size, significant deviation is unlikely.

10.4.2 MPSA

Downhole surveys were completed on all geotechnical holes using a Reflex Maxibor II instrument. All resource holes greater than 300 m in depth were surveyed using the Maxibor instrument or the FLEXIT smart-tool single shot at 60 m downhole increments. In 2011, MPSA purchased a REFLEX Gyro E596 downhole surveying instrument to negate the effect of magnetic interference. Holes deeper than 300 m were surveyed at 10 m intervals.

ITEM 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Historical data

11.1.1 UNDP

No record of the sampling preparation, analyses and security used by UNDP remains. No core has been recovered from this phase of drilling. Holes from this phase of drilling were not used to develop the geological or resource model.

11.1.2 PMRD

The core from the PMRD drill programme was split with a mechanical splitter with samples being taken at 2 m intervals downhole. Samples were pulverised and sent for assay at *Direction General de Recursos Minerales* of Panamá where they were analyzed for Cu and Mo using atomic absorption. Thirty duplicate umpire samples were assayed at the Central Research Lab of Mitsui Smelting and Mining Limited for check assaying. The umpire results were slightly higher than the original *Direction General de Recursos Minerales* results, but were reported to be within acceptable limits (Rose *et al,* 2012). The Mitsui results averaged 0.597% Cu and the Panamá results averaged 0.578% Cu.

Two PMRD holes were twinned by Adrian and Teck for grade comparison (Simons, 1996). Holes with adjacent holes were also identified in the western part of Colina and Botija; some differences were noted but generally the data was validated by nearby holes.

11.1.3 Adrian/Teck

Procedures in place during the Adrian/Teck drilling campaigns are summarized below:

- Core was delivered to the core logging and sampling facility at Colina and marked into 1.5 m intervals. Geological logging was completed by Canadian geologists hired by Adrian and included collection of geological data, including lithology, alteration, mineralisation and structure. Geotechnical logging was completed by Panamánian geologists hired by Georecursos and included recovery and RQD values.
- Samples were 1.5 m in length, measured from the collar of the hole. Samples were split using a mechanical splitter with half of the core being archived and the other half crushed and split with a Jones splitter. The archived core is no longer available. A one-eighth split weighing approximately 250 g was taken for each interval for analysis. Rejects were stored on site in plastic bags. Crushing and splitting equipment was cleaned using an air hose between each sample.
- Metallurgical samples averaged 4.5 m in length, but lengths were modified to observe changes in lithology. Metallurgical samples were whole core sampled.
- Analysis was completed at TSL Laboratories in Saskatoon, Canada. Copper analysis was performed by acid digestion of a 0.5 gram sample and analysed by atomic absorption (AA). Molybdenum analysis was performed by acid digestion of a 1 gram sample and analysed by AA. A 30 gram aliquot was used for gold analysis by fire-acid with an AA finish and silver was analysed using acid digestion of a 2 gram sample with AA.

11.1.4 Petaquilla Copper

Sample preparation, transport and analytical procedures for the PTC drilling are summarised below:

- Core samples were transferred into metal trays and dried at 115°C in an electric drying oven for three to six hours. In the event of interrupted power supply, samples were dried in a room heated by wood-fired stoves to approximately 60°C.
- After drying, samples were immediately sent to the crusher (a small BICO jaw crusher) where the entire sample was crushed to approximately a 5 mm to 6 mm particle size. The crusher was cleaned between sample by crushing a small amount of blank basalt, and cleaning with compressed air.
- Using a Jones Splitter or rifle splitter (12 mm openings) a 250 gram aliquot was taken, with the reject material being stored in plastic bags, labelled and stored undercover on site. The sample aliquot was placed in a small, durable plastic bags, labelled and heat sealed. Approximately 60 samples were then placed in large woven plastic (rice) bags which were tied and clearly marked ready for dispatch.
- The sample preparation facility was locked when not operational in order to monitor sample integrity.
- Samples were transported by truck to Panamá City from the PTC office at La Pintada on a twice weekly basis. They were then shipped using FedEx to either SGS Laboratories in Lima, Perú, or ALS Chemex in Vancouver, Canada.
- Samples sent to SGS Laboratories in Lima, Perú were analysed for sequential copper, gold, silver, total copper and molybdenum. Samples sent to ALS Chemex in Vancouver were analyzed for gold using a 50g fire assay with a gravimetric finish and copper, silver, molybdenum and a selection of multi-elements using ICP-MS.

11.2 MPSA

11.2.1 Sample preparation

Samples from diamond core and RC chips were placed within aluminium trays and dried in ovens for 12 hours at 90°C. Once dry, the entire sample (approximately 8 kg) was crushed in a Rocklabs Boyd crusher to a -10 mesh (2 mm) particle size. Sieve tests were conducted regularly (at least twice a day) to ensure that the material was being crushed to the appropriate size. The equipment was cleaned after every sample using high-pressure air and after every tenth sample a coarse blank sample was passed through the crusher.

The crushed sample material was split using a Jones rifle splitter and a 500 g aliquot taken for assay. The aliquot was placed in a small plastic bag which was heat sealed and marked with a bar-coded sample tag. The reject material was returned to the original sample bag and stored on site.

11.2.2 Assaying

The sample aliquots were shipped by air courier to ALS Chemex Lima in Lima, Perú, for analysis. Umpire assay checks and secondary assay work was conducted by Acme Santiago in Santiago, Chile. Both labs have ISO/IEC 17025-2005 certification.

Upon receipt of the samples by ALS Chemex, the bar code on the assay sample bag was scanned. Half of the sample (250 grams) was pulverised using a LM5 pulveriser using low-chrome steel equipment resulting in 85% of material passing 75 µm. Copper assays were conducted using atomic absorption spectrometry (AAS) after a four-acid digestion (HF-HNO_3-$HClO_4$-HCL). Gold analyses were by fire assay and AAS analysis on a 30 gram sample. Silver and molybdenum were included as part of a multi-element ICP-AES analysis. Total sulphur was analyzed using a LECO induction furnace.

Near-surface samples were analysed for sequential copper to estimate the amount of leachable copper. Sulphuric, cyanide and citric-acid soluble copper and residual copper values were reported for each sample.

Residual pulps were stored at either ALS Chemex in Lima, Perú, or at a storage warehouse at FQM's Minera Antares office in Arequipa, Peru. Residual pulps were discarded after 3 years due to oxidation and a reduced ability to repeat original assays. All MPSA drill core is however safely and securely stored in warehouses located on the Cobre Panama site.

11.2.3 Sample security – chain of custody

All assay samples were kept in a locked facility on site until they were ready for shipment. Samples for a given hole were batched once the entire hole had been logged and sampled. Samples were collected into larger bags in batches of approximately 90 samples per bag. Samples to be assayed for sequential copper were batched into bags of 20 to 25 samples. Several times a week, the samples were dispatched by road to a secure warehouse in Penonomé by MPSA staff. While in storage, generally for less than two days, samples were kept under locked conditions until picked up by DHL cargo shipping. DHL then airfreighted the samples to ALS Chemex Laboratory in Lima, Peru.

11.3 QAQC

A detailed review of all the historical and current QAQC practices, QAQC data and historical QAQC reports at Cobre Panamá has been undertaken in order to determine the accuracy, precision and bias present in the drillhole assay data for the Project area.

FQM provided the QAQC data, historical NI 43-101 reports and recent data validation reports in order for this review to be completed.

11.3.1 QAQC programme history

Since 1968 a number of drill campaigns have been conducted over the Cobre Panamá concession area to test the extent of the porphyry copper mineralisation.

The establishment of a rudimentary QAQC system did not occur until the 1970s when PMRD initiated a small programme of umpire check analyses. Thirty duplicate samples were sent to the Central Research Lab of Mitsui Smelting and Mining Limited for check assay. The *Direction General de Recursos Minerales* of Panamá assays were determined to be slightly lower, but within acceptable limits. The Mitsui results averaged 0.597% Cu and the Panamá results averaged 0.578% Cu. Results of this work were deemed satisfactory, (AMEC 2007).

Adrian continued with the check assay programme between 1992 and 1995, sending a small number of check samples to XRAL, Canada for analysis of copper, and XRAL and ALS Vancouver, Canada for analysis of gold. Francois-Bongarcon completed a review of the check pulp assay results for copper and gold (Francois-Bongarcon, 1995). The repeatability of gold assays at XRAL and ALS Vancouver were both very poor. Francois-Bongarcon concluded that the TLS – XRAL gold duplicate samples were not correlated at all, and stated that this was either due to errors at one of the laboratories, or as a consequence of severe errors during assay preparation. He concluded that "QA-QC procedures urgently need to be audited and tightened up, and past problems need to be properly investigated without delay", and recommended further action to determine the cause of the bias. Following this recommendation, Adrian (then Petaquilla Mining Limited) adopted detailed QAQC protocols designed, implemented and monitored by AAT Mining Services, (Behre Dolbear, 2012).

During the period 1996 to 1997 Teck Cominco began to implement the new QAQC sampling procedures, regularly inserting CRM standards into the sample submissions. Teck Cominco also routinely submitted one in fifteen samples for umpire check analysis to ALS Vancouver.

Implementation of the QAQC sampling procedures began in earnest during the PTC drilling programmes undertaken between 2006 and 2009. CRM standards, field duplicates, blanks and coarse crush duplicates were regularly inserted into the assay submissions sent to ALS Lima, ALS Vancouver and SGS Lima for assay. Several programmes of check analysis were also undertaken during this period and included coarse umpire checks and pulp checks. Bruce Davis of BD Resource Consulting Inc. undertook regular reviews of the PTC QAQC data. A detailed examination of all the available PTC QAQC data has been undertaken as a part of the QAQC review.

MPSA has continued to collect and review the QAQC data since the beginning of the FEED programme in October 2007. CRM standards, blanks, field duplicates, coarse crush duplicates and pulp umpire checks have been routinely collected and submitted to ALS Lima and ACME Santiago Chile. Regular reviews of the QAQC data have been undertaken by MPSA personnel and corrections made to the database when an error was identified. In addition, Optiro completed a detailed review of all the available MPSA QAQC data as a part of the QAQC review.

11.3.2 CRM standard performance

A total of twenty eight different CRM standards have been submitted for analysis with drillhole samples collected by PTC and MPSA at Cobre Panamá (Table 11-1).

Detailed analysis of the CRM standard performance has been undertaken by drilling programme, assay laboratory and assay method. Numerous CRM standard swaps and drillhole sample results have been identified in the PTC QAQC data; however, once these errors have been corrected, the CRM standard performance is acceptable. It is evident from the number of errors that the data was not reviewed by PTC during the drilling programme and it is strongly recommended that FQM corrects the errors in the corporate database.

The MPSA CRM standard performance is considerably better than that of PTC, with only a few CRM standard swaps identified. Five copper CRM standards with bias outside accepted limits are present, two CRM standards are in the low grade range, two in the medium grade range and one in the high grade range (Table 11-1).

This indicates the possibility that the low grades are being overstated and the higher grades are being understated in sample batches which have been assayed with these CRM standards; however, there may still be sample swaps present in four of the five CRM standards.

Table 11-1 CRM standards submitted with Cobre Panamá samples

Standard ID	Number
CDN-CGS-11	255
CDN-CGS-12	250
CDN-CGS-16	372
CDN-CGS-18	293
CDN-CGS-19	446
CDN-CGS-22	197
CDN-CGS-23	106
CDN-CGS-24	402
CDN-CGS-27	192
CDN-CGS-28	200
CDN-CGS-3	2
CDN-CM-1	520
CDN-CM-13	136
CDN-CM-15	177
CDN-CM-24	214
CDN-CM-4	216
CDN-CM-8	408
CDN-FCM-4	46
CDN-FCM-5	39
CDN-HLHZ	43
OREAS 152a	404
OREAS 153a	297
OREAS 50Pb	146
OREAS 51P	62
OREAS 52P	8
OREAS 52Pb	129
OREAS 53Pb	116
OREAS 54Pa	142
TOTAL	**5,818**

Table 11-2 CRM Standards extreme bias, Cu%

CRM standard ID	Cu%				Number of standards submitted	Number of potential swaps
	Expected value	Actual mean	Bias	Type		
CDN-CGS-16	0.112	0.116	**3.80%**	LG	342	2
CDN-CGS-19	0.132	0.137	**4.16%**	LG	437	5
CDN-CGS-28	2.089	2.016	**-3.51%**	HG	200	0
OREAS 50 Pb	0.744	0.720	**-3.20%**	MG	67	4
OREAS 54 Pa	1.550	1.498	**-3.39%**	MG	68	1

Detailed review of the CRM standard data has identified minor problems with the labelling of CRM standards; however, this is not considered to be problematic. It is recommended that any CRM standard swaps which have been identified are corrected in the corporate database.

The recent RC grade control drillhole sampling at Botija has included eight CRM standards for Cu, Mo and Au assaying. These include CDN-CGS-26, CDN-CM-29, CDN-CM-25 to 27, OREAS-503c, OREAS-151b, OREAS-601. Anomalous standard behaviour was detected from OREAS-503c and OREAS-601 with a 13% failure of submitted samples. However, the remaining standards performed well within failure limits and the quality of the standard material was believed to have deteriotated.

11.3.3 Blank performance

Blanks have been routinely inserted into the assay sample submissions during the PTC and MPSA drilling programmes in order to monitor the sample preparation process. Blank failures are defined as samples which have an assay result more than five times the practical detection limit in any three of the four main elements, copper, gold, silver and molybdenum, (Table 11-3).

Table 11-3 Blank failure rates

Drilling programme	Number submitted	Number of failure	Failure rate
PTC	179	11	6.0%
MPSA	2,634	15	0.6%

The failure rate of blanks is within acceptable limits for both drilling programmes, indicating that the sample preparation processes in place at the site preparation facility are satisfactory.

RC grade control drillhole sampling inserted blank samples at a rate of 3.7%. Blank sample failure rate was at 0.3% and is well within acceptable limits indicating appropriate control during sample preparation.

11.3.4 Field duplicate performance

Field duplicate samples are inserted into sample submissions in order to determine the precision and bias of drilling assay results. Field duplicates have been routinely collected at Cobre Panamá during the PTC and MPSA diamond and RC drilling programmes.

During the PTC drilling programme, duplicates were collected at the pulverisation stage of the sample preparation process and submitted as a pulp duplicate for assay. Analysis of this data indicates low sample bias with high levels of precision between the paired assays thus confirming that the parent pulverised sample was homogenous (Table 11-4 and Table 11-5).

Table 11-4 PTC pulp field duplicate statistics

Statistic	Original Cu%	Duplicate Cu%	Original Au ppm	Duplicate Au ppm
Mean	0.21	0.21	0.19	0.18
Maximum value	4.6	4.44	40	17.3
Correlation coefficient	0.97		0.91	

Table 11-5 PTC pulp field duplicate precision

Statistic	Cu%	Au ppm
% of Assays Within 5%	89.71	37.93
% of Assays Within 10%	96.81	52.11
% of Assays Within 15%	97.68	63.98

MPSA field duplicates were collected at the primary sampling stage, with the original sample representing half of the drill core and the duplicate sample representing one quarter of the remaining half core. Strictly speaking these samples do not represent field duplicates, since the volume of material available for assay is different; however, these samples have been analysed as field duplicates. The bias present between the two samples is moderately high and the precision lower than that of the PTC pulp duplicates (Table 11-6 and Table 11-7).

Table 11-6 MPSA core field duplicate statistics

Statistic	Original Cu%	Duplicate Cu%	Original Au ppm	Duplicate Au ppm
Mean	0.17	0.17	0.04	0.04
Maximum value	1.87	1.99	1.42	1.27
Correlation coefficient	0.96		0.93	

Table 11-7 MPSA core field duplicate precision

Statistic	Cu%	Au ppm
% of Assays Within 5%	66.74	53.92
% of Assays Within 10%	88.50	78.21
% of Assays Within 15%	94.20	89.54

The MPSA RC grade control sampling has similarly ensured field duplicates of the original chip samples are taken at the rig during drilling. The field duplicate results for the RC sampling have 95% of assays within 10% indicating high levels of precision. Limited to no bias was noted.

11.3.5 Coarse crush duplicates performance

Coarse crush duplicates are collected at the crushing stage of the sample preparation process and have been inserted into the sample submissions of both the PTC and MPSA drilling programmes.

Coarse crush duplicates collected during the PTC drilling programme were submitted to ALS Lima and ALS Vancouver, and display low levels of bias and high precision, indicating that the sample is homogenous (Table 11-8 and Table 11-9).

Table 11-8 PTC coarse crush duplicate statistics – ALS Lima

Statistic	Original Cu%	Duplicate Cu%	Original Au ppm	Duplicate Au ppm
Mean	0.29	0.29	0.14	0.13
Maximum value	1.613	1.74	1.635	1.645
Correlation coefficient	1.00		1.00	

Table 11-9 PTC coarse crush duplicate statistics – ALS Vancouver

Statistic	Original Cu%	Duplicate Cu%	Original Au ppm	Duplicate Au ppm
Mean	0.34	0.33	0.27	0.26
Maximum value	1.43	1.43	1.00	1.00
Correlation coefficient	1.00		1.00	

The determination of the precision between the data pairs has been undertaken, with copper displaying levels of precision which are considered excellent in that 96.4% of the data has better than 10% precision at ALS Lima and 97.9% of the data has better than 10% precision at ALS Vancouver. Gold, however, does not display the same degree of precision, with 88.9% of the data within 10% precision at ALS Lima and 75.0% of the data within 10% precision at ALS Vancouver. The decreased precision in gold is a function of its low grades and the associated sampling errors.

MPSA coarse crush duplicates also display low bias and high degrees of precision (Table 11-10). The degree of precision between the data pairs for copper are considered excellent, in that 98.4% of the data is within 10% precision. Gold, however does not display the same degree of precision, with 74.1% of the data within 10% precision.

Table 11-10 MPSA coarse crush duplicate statistics

Statistic	Original Cu%	Duplicate Cu%	Original Au ppm	Duplicate Au ppm
Mean	0.190	0.191	0.043	0.044
Maximum value	5.63	5.43	2.69	2.81
Correlation coefficient	1.00		0.89	

The results of the coarse crush duplicate analysis indicate that the sample has been adequately homogenised during the crushing stage of the sample preparation process.

11.3.6 Umpire check laboratory performance

Check assaying has been undertaken to some degree during every drilling campaign undertaken at Cobre Panamá. PMRD undertook the first programme of check analysis when it submitted samples to the Central Research Laboratory of Mitsui Smelting and Mining Limited for analysis of copper. The results were deemed to be satisfactory (AMEC, 2007).

Adrian re-submitted a batch of pulp samples to XRAL Canada and a batch to ALS Vancouver for analysis of copper and gold. The results were mixed, with XRAL replicating the TSL copper assays adequately; whereas there was considerable scatter and poor correlations between the TSL vs XRAL gold assays, and the ALS Vancouver vs TSL gold assays. Francois-Bongarcon (1995) raised concerns about the data and recommended the implementation of a detailed QAQC process, which was later undertaken by Teck Cominco and PTC.

In 1996 Teck Cominco submitted one in 15 samples to ALS Vancouver for check assay analysis. Table 11-11 details the averages of the results from the two laboratories. The data suggests that there is a small bias in the copper data, but the bias was considered to be insignificant by AMEC (2007). Gold appears to be biased low at TSL relative to ALS Vancouver. Silver is biased somewhat high relative to ALS Vancouver, but the bias is within the expected ranges.

Table 11-11 1996 check assay data summary, after AMEC (2007)

Element	Number	TSL mean grade	ALS mean grade	Difference
Cu%	1,563	0.56	0.57	1.8%
Au ppm	1,113	0.106	0.098	-7.5%
Ag ppm	1,087	2.0	2.1	5.0%
MoS ppm	1,038	218	240	10.1%

During the PTC drilling programme two batches of umpire check analysis were undertaken. A total of 1,111 samples were re-sampled and re-submitted to an umpire laboratory for check analysis of copper and gold. One of the programmes replicated the original assay method used for copper, whilst the other utilised a different assay method for copper. Both programmes delivered moderate to high levels of correlation between the paired samples for copper; however, the gold results were considerably worse. This is considered to be a function of the low grade nature of the gold mineralisation at Cobre Panamá or the differences in the detection limits applied by the different laboratories, and is not considered problematic.

MPSA routinely submits umpire check samples to a secondary laboratory for analysis and therefore the dataset available for analysis was very large. Results indicate that the correlations between the two datasets are very high for copper and moderately high for gold. The precision of copper is very high, with 98.3% of the data within 10% precision for copper; however, gold is seen to be less precise, with 57.9% of the data within 10% precision.

11.3.7 Twinned drillhole analysis

Twinned drillhole analysis has been undertaken several times during the life of the Cobre Panamá Project. The initial review of twinned drillholes was undertaken by Teck in 1998 with a view to comparing the Teck drilling to the earlier PMRD drilling. The results of this analysis were reported by Simons (1998) in the Feasibility Study NI 43-101 report. Simons concluded that the methodology used to derive the mean grade had not been well documented in the report, with the authors determining that the analysis was 'inconclusive'.

In 2007 AMEC compared ten sets of twinned drillholes at Colina and Botija. This programme compared the results from the 1996 and 1997 Teck drill campaigns with earlier drill campaigns. The authors reviewed the twin data using downhole grade profile plots and summary statistics. In conjunction with other QAQC data and information, AMEC recommended that the UNDP drilling data not be used for the purpose of resource estimation and that additional twinned drillhole drilling be undertaken to cover all the historical drilling campaigns.

In 2010 Bruce Davis of BD Resource Consulting Inc. undertook a twinned drillhole study on behalf of MPSA. Davis reviewed ten pairs of drillholes that were within 15 m of each other and drilled to a reasonable depth below the saprolite cover at Colina and Botija. Davis (2010) states in conclusion: *"The correspondence of the QQ-plot and grade profile results suggests that the twin holes are giving similar information about the spatial distribution of grade within the deposits. There is no indication that one set of drilling is giving significantly different information from another set. There are a few discrepancies among the various combinations of drill holes and metals, but there is no conclusive evidence that the drilling from a particular campaign is not appropriate. The twin drillhole results confirm the comparisons of declustered grades by drilling campaign which indicated no significant differences exist among those campaigns. All the historical information has been vetted by QC programmes, comparison of twin drillhole results, and the comparison of declustered grade distributions over common areas. In my opinion, all drilling information assembled for building the resource model is useful."*

11.3.8 Additional QAQC work

Numerous programmes of additional QAQC work have been undertaken at Cobre Panamá over the life of the project.

An independent programme of check pulp duplicate sampling was undertaken in 2011 by Jeffrey Jaacks of Geochemical Applications International LTD, on behalf of Freeport McMoran Exploration, Geochemical Applications International Inc., (Freeport). This work was part of a due diligence project undertaken on the Cobre Panamá project on behalf of Freeport. Thirty drillholes located in Botija, Colina and Valle Grande were selected for re-analysis and review. A total of 700 pulp samples were submitted for re-analysis to ALS La Serena, Chile for copper, gold, silver and molybdenum analysis. QAQC samples were inserted into the sample submissions for each batch of forty samples and included a minimum of one high grade CRM standard, one low grade CRM standard, one blank and one set of pulp duplicates, (Jaacks and Candia, 2011). Jaacks and Candia concluded: *"Copper check analyses show good reproducibility with previous Inmet drill programme analyses. Ninety-five percent of the original and check copper analyses are within ±10% of one another. Bias between the two sets of data is less than 3 percent. The mean grade for the original set of analyses was 0.321 % Cu. The* *mean grade for the check analyses is 0.331 %Cu. The check analysis programme validates earlier copper analyses and there is no significant bias between the two sets of data. There is no significant difference between the standard deviation or the mean grade of the original versus check analysis data".*

Behre Dolbear completed a NI 43-101 technical report on the Botija Abajo Deposit in 2012, on behalf of Petaquilla Minerals LTD (PML), formerly known as Adrian Resources. A total of 45 samples were submitted to ALS Chemex for analysis of copper, gold and an additional 33 elements; however the assaying methodologies utilised at ALS Chemex were different for the check samples. Gold was assayed using a 50g sample by fire assay with gravimetric finish. Copper and other elements were assayed using an aqua regia digest with an ICP analysis. The assaying methodologies utilised in the check sampling programme are considered to be of better quality than those utilised in the assaying of the original samples. The result of the check core duplicate sampling was *"a very good correlation between PML ALS Chemex core samples taken by the authors and assayed at ALS Chemex Laboratories".*

FQM undertook a check sampling and assaying programme after concerns were raised about the lack of comprehensive QAQC data for the 1995-1996 drilled Adrian/Teck drillholes. The remaining core from drillholes completed at this time had been stored outside and were exposed to the elements. As a function of this, the core trays had deteriorated and the remaining core had spilled onto the ground. MPSA collected, re-labelled and placed the core into new trays for as many of the drillholes as possible. The check sampling used the original sample lengths and sampled the remaining half core, as if it were a field duplicate. Sample preparation was undertaken on site and samples dispatched to ALS Lima, Peru for analysis. ALS Lima undertook four acid digest with atomic absorption (AA) final for copper, fire assay analysis and AA final for gold, and ICP-AES for molybdenum and silver. QAQC samples were inserted blind into the check assay sample submissions. Five CRM standards, five field duplicates and ten blanks were submitted with the 168 duplicate samples. Gray (2013) concludes: *"Accepting that the most likely reason for imperfect correlation and sub-standard field duplicate precision is due to incorrect interval labelling during salvage of the skeleton core, results were encouraging and demonstrate similar low and high values across similar intervals for both duplicate values. Sample assay results from the original Adrian and Teck campaigns are accepted as reasonable and representative of the true metal grades and are able to be repeated at another reputable laboratory given the correct duplicate sample intervals".*

11.3.9 Conclusions

Numerous programmes of QAQC sampling have been undertaken at Cobre Panamá by previous owners and MPSA. Whilst a systemised programme of QAQC sampling was not fully implemented until 2006, numerous programmes of check analysis were undertaken to compare each programme of drilling to historic drilling undertaken by previous owners. Similarly, routine review of the QAQC data and results did not occur until the MPSA drilling programmes. Reviews and corrections of any errors identified are currently completed on a quarterly basis.

MPSA is currently importing and validating all the Cobre Panamá drillhole data into a corporate database, including all the historic QAQC data collected over the life of the project. Errors identified during this QAQC review will be investigated and corrected in the corporate database.

The copper QAQC results indicate that:

- the assaying laboratories are reporting assays to acceptable levels of accuracy
- standard failure rates are within acceptable levels
- blank samples indicate that the sample preparation process is operating successfully and that failure rates are low
- field duplicate assays display low bias and moderate degrees of precision
- coarse crush duplicates display low bias and high degrees of precision
- umpire check samples display low bias and moderate degrees of precision
- twinned drillholes display correlations between assays which are considered acceptable.

The gold QAQC results indicate that:

- the assaying laboratories are reporting assays to acceptable levels of accuracy
- standard failure rates are within acceptable levels
- blank samples indicate that the sample preparation process is operating successfully and that failure rates are low
- field duplicate assays display moderate bias and moderate degrees of precision
- coarse crush duplicates display moderate bias and moderate degrees of precision
- umpire check samples display moderate bias and moderate degrees of precision
- twinned drillholes display correlations between assays which are lower than copper, although still considered acceptable.

It is considered that the QAQC results reviewed for this Technical Report indicate that the Cobre Panamá drillhole assays are suitable for Mineral Resource estimation.

ITEM 12 DATA VERIFICATION

12.1 Historical data

MPSA has carried out a number of programmes of checks on historical (i.e. pre-MPSA) data, including re-logging, re-assaying, full database validations and collar location verfications. In particular, verification of collar coordinates highlighted issues relating to grid conversions for the PTC holes. The issues appear to highlight a constant offset for a number of hole collars, which have been corrected. Given the magnitude of the offset (~18 m) relative to the drillhole and the Mineral Resource block spacing, the Mineral Resource estimate will not be materially affected and the corrected collars will be incorporated into the next update. MPSA also undertook a check assaying programme from core skeletons, as described in Item 11. This demonstrated the integrity of the pre-MPSA data but the loss of this drill core will remain a minor limitation for future refencing or check sampling. It has not materially affected the Mineral Resource estimate.

12.2 Data verification by the QP

During respective site visits by David Gray (QP) and Ian Glacken (contributing author), a number of collar positions, both MPSA and pre-MPSA, were field checked using a handheld GPS. These collar locations were found to match the database locations within the accuracy of the GPS instrument.

In addition to this collar checking, the QP also supervised the checking of a number of holes and a reasonable selection of original assay certificates against the database, and found no errors. The QP has also supervised the validation of all drillhole database data for duplicates, gaps and overlaps. No limitations were noted for the quality of the database data.

The QP has also verified the porphyry deposit type and style of mineralisation through investigation of drill core and observation at the available outcropping mineralisation in road cuttings and the various on-site quarries and excavations. No limitations were noted.

12.3 Data verification prior to Mineral Resource estimation

As part of the Mineral Resource estimation process, a number of data validation checks were made upon the data. These included:

- Visual investigation and checks of the relative magnitudes of downhole survey data were completed in order to identify improperly recorded downhole survey values. No value corrections were required.
- The geology and assay dataset was examined for sample overlaps and/or gaps in downhole logging data, with overlaps and duplication identified and removed.
- Assay data for total copper, gold, molybdenum and silver were interrogated for values that were out of expected limits.
- The dataset was examined for sample overlaps and/or gaps in downhole survey, sampling and geological logging data, with minor problems resolved.

ITEM 13 MINERAL PROCESSING AND METALLURGICAL TESTING

The following information is largely reproduced from Rose et al (2013), with an update on confirmatory work provided by Robert Stone (QP).

13.1 Metallurgical testwork

Metallurgical test work has been undertaken on the Cobre Panamá Project from 1968, commensurate with the various levels of preliminary feasibility and prefeasibility studies that were completed in 1977, 1979 and 1994; as well as feasibility studies in 1994 (updated 1995), 1996 and 1998.

In 1997 an extensive programme of metallurgical testing was designed to confirm earlier studies on the metallurgical response of the Botija and Colina material. Much of the testing was conducted by Lakefield Research Ltd. (Lakefield) in Lakefield, Ontario, Canada. Work included grinding, flotation, dewatering and mineralogical testing. In addition, further testing was completed by G&T Metallurgical Services Ltd (G&T) in Kamloops, British Columbia, Canada. This included locked-cycle flotation testwork and modal analysis to assist in defining grind requirements for both rougher and cleaner flotation. Copper-molybdenum separation using differential flotation was conducted by International Metallurgical and Environmental (IME), in Kelowna, British Columbia, Canada.

All testwork prior to (and including) 1997 was based on large composite samples, for which the results, especially for flotation testing, could not be used for interpreting the variability of the response across and between the different deposits. Between 2008 and 2009, a total of 16 fit-for-purpose holes were drilled across the Botija, Colina and Valle Grande ore bodies in order to provide additional insight into the variability within each deposit. Hole locations were selected to cover an even spread along the major axes of the two deposits and to penetrate the major combinations of lithology and alteration identified in the 1998 geological model and to replicate ore arsisings predicted during the early operating years. Sample preparation, flotation testing and testing of flotation products were primarily completed at G&T. SGS Mineral Services, in Lakefield, Ontario, Canada, and Philips Enterprises LLC, In Golden, Colorado, USA, conducted much of the grinding testwork.

SGS Mineral Services in Lakefield, Ontario, Canada were hired to conduct an investigation into the variability of the flotation response of samples at Botija and Colina (SGS, 2012) between May and November 2011. Composite samples were prepared from drill core and based on rock type blends that varied by copper grade. Botija composites were taken from the infill drilling programme in the area of the proposed Botija starter pit and with grades varying from 0.3%, 0.4%, 0.5% and 0.8% Cu. Colina composites were sourced from samples in cold storage at G&T laboratories in Kamloops with grades of 0.3%, 0.5% and 0.6% Cu. Locked cycle testing was carried out to confirm simplification of a cleaner flotation. A revised reagent protocol was tested to confirm concentrate grade and metal recovery.

Between 2011 and 2012, further flotation and grinding studies on core samples from the Balboa and Brazo deposits were conducted by Hazen Research in Golden, Colorado, USA (Hazen Research, Schultz, 2012a, b, c, d, e; Reeves, 2012). To represent Balboa, 27 composites for drill core with a total weight of 4,487 kg were selected and to represent Brazo, 17 composites totalling 2,165 kg were selected. Composites were crushed, coned, quartered and split to produce subsamples for grindability, laboratory flotation work, mineralogical characterization, quantitative head chemical analysis, as well as whole rock analysis. This testwork provided substantial advances in the knowledge and understanding of the following:

- a comprehensive suite of grindability parameters resulting in new throughput estimates
- additional flotation response data for estimating concentrate production and operating costs
- additional geological data
- sample materials for marketing purposes
- additional design data for solid-liquid separation, regrinding and pipeline design
- a reduction in copper cleaning stages, while maintaining concentrate grade at improved metal recovery levels; and
- additional floatation response data for estimation copper, gold and molybdenum metal recovery from the Balboa and Brazo deposits.

13.1.1 Grindability testwork

A large amount of grindability data was collected in 2009 to supplement earlier work. This data confirmed preliminary assumptions of throughput rates and indicated that there were benefits to adding a pebble-crushing circuit to each grinding line.

In 2011 and 2012, samples of Balboa and Brazo were provided to produce comminution data for these two deposits. Work included obtaining data for Bond Ball Mill work index, Bond Rod Mill work index, Bond Abrasion index, JK Drop weight and SMC evaluation.

Balboa Bond Work Index varied from 12.4 to 18.0 kWh/t, the Rod Mill work Index varied from 13.1 kWh/t to 19.6 kWh/t, while the abrasion index varied from 0.0760 g to 0.4129 g. The JK drop weight A x b index varied from 34.5 to 87.2.

A fine primary grind in the range of 75 µm to 100 µm improves copper and gold recovery for all composites tested from Balboa. Hazen Research conducted a generic evaluation of adding power to the grinding circuit and was able to conclude that a P_{80} of less than 100 µm appears to be beneficial to cash flow for a wide range of copper prices and ore grades.

Brazo Bond Work Index varied from 8.0 kWh/t to 14.6 kWh/t, the Rod Mill Work Index varied from 8.7 kWh/t to 16.4 kWh/t, and the Abrasion Index varied from 0.0922 g to 0.3729 g. The JK drop weight A x b index varied from 44.0 to 87.9.

13.1.2 Confirmatory flotation testwork

Confirmatory batch laboratory flotation testwork was conducted during 2014 by ALS Metallurgy in Perth, Western Australia (ALS Metallurgy, Steele, 2014). The laboratory flotation testwork was conducted on a production composite sample prepared from core that represents the first two years of mining from the Botija deposit. The sample was previously prepared by Hazen Research in Golden, Colorado , USA for pilot plant testwork conducted in 2012 (Hazen Research, Schultz, 2013). The aim of the testwork was to confirm or optimise the following flotation circuit design parameters:

- Rougher flotation response to primary grind size variations, pulp density variations, pulp pH variations and collector type variations.
- Cleaner flotation response to regrind size variations.

The rougher grind optimisation testwork data indicates that the flotation of the copper and molybdenum minerals are grind sensitive and copper and molybdenum recovery and flotation kinetics generally improve with finer grind sizes. Furthermore, the molybdenum recovery decreases with grind sizes finer than 125 µm indicating that the optimum grind P_{80} for recovering molybdenum in the bulk rougher flotation circuit is 125 µm. However, the improvements in recovery of copper and molybdenum with a decrease in the current grind P_{80} of 180 µm are not large enough to economically justify the finer grind sizes.

A decrease in pulp pH from 9.5 to a natural pH of 7.8 had no effect on the selective flotation of copper minerals in the rougher circuit. However, the molybdenum recovery in the rougher circuit improves with the lower pH. This confirms previous testwork results conducted by Lakefield Research (Lakefield, 1997) and SGS (SGS, 2012) which indicated that raising the pH in the roughers did not significantly impact the final copper concentrate grade or recovery.

Decreasing the pulp density in the rougher flotation circuit has no effect on the selective flotation of copper minerals but does improve the grade recovery relationship of molybdenum. The improvement in molybdenum flotation warrants a decrease in the design feed density of the rougher flotation circuit from 35% w/w to 30% w/w solids.

Changes to the collector and promoter types showed no significant improvements to the flotation performance of copper and molybdenum in the rougher circuit.

Selective flotation of copper in the cleaner circuits is grind sensitive as the grade recovery curve for copper typically is improved with a finer regrind size. Although the final concentrate grade continuously increases with a finer regrind size, copper recovery does starts decreasing with a regrind P_{80} size finer than 35 µm. The flotation rate of molybdenum decreases with a decrease in regrind size in the cleaner circuits.

13.2 Recovery projections

Based on the testwork described above, variable processing recovery relationships were determined for copper and gold, whilst fixed recovery values were determined for molybdenum and silver. The design recoveries vary for each deposit, as summarised in Table 13-1.

Table 13-1 Cobre Panamá process recovery relationships and values

Deposit	Recovery			
	Cu (%)	Mo (%)	Au (%)	Ag (%)
Botija	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Colina	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Medio	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Valle Grande	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775))-4)	52.0%	MIN(80,MAX(0,(15.993*LOG(auppm))+92.138))	47.3%
Balboa	MIN(96,((2.4142*LOG(cutpct))+92.655))	55.0%	MAX(0,MIN(80,(7.6009*LOG(auppm))+85.198))	40.0%
Botija Abajo	6.6135*Ln(Cu%) + 92.953	55.0%	50.0%	30.0%
Brazo	6.6135*Ln(Cu%) + 92.953	55.0%	50.0%	30.0%

During 2018 a confirmatory geometallurgical testwork programme was commenced using grade control samples to validate the above processing recovery relationships, especially in regard to the ore from initial mining horizons. At this time, however, an update to the processing recovery relationships is not warranted.

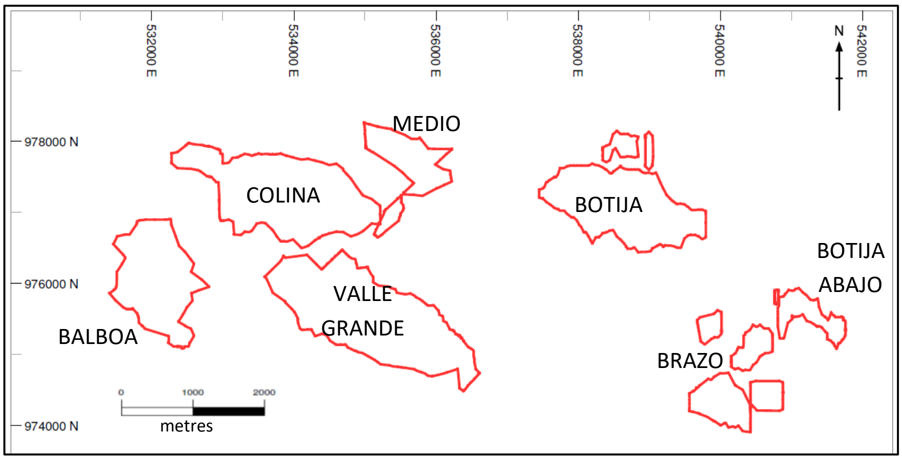
ITEM 14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

Mineral Resource estimates have been generated for seven porphyry copper deposits identified within the Cobre Panamá Project area: these are the Botija, Colina, Valle Grande, Balboa, Medio, Botija Abajo, and Brazo deposits. The Mineral Resource estimates were prepared under the supervision of David Gray (QP), with the assistance of Optiro and FQM staff.

This Item documents Mineral Resource estimates for the four most well-defined porphyry deposits (Botija, Colina, and Valle Grande and Balboa) in addition to the three lesser-drilled deposits (Medio, Botija Abajo and Brazo). The locations of these deposit areas are shown in Figure 14-1. The estimates were developed during the period from November 2013 to January 2014 from 3D block models based on geostatistical applications using commercial mining software, CAE Studio 3 (Datamine), developed by CAE Mining, as well as in-house software developed by Optiro for the post-processing. The Botija block model estimate was updated in December 2018 with the added reverse circulation drilling data.

Figure 14-1 Location of the Botija, Colina, Valle Grande, Balboa, Medio, Botija Abajo, and Brazo deposits



14.2 Data for Mineral Resource modelling

14.2.1 Topographic data

Detailed topographical data was provided by FQM in the form of a CAE Studio format triangulated surface generated from point data at a spacing of 10 mE by 10 mN. This surface was generated from detailed LIDAR information obtained by MPSA during April 2009. The surface initially provided by FQM did not cover the western 770 m of the Balboa deposit area and a second surface file generated from point data at a spacing of 15 mE by 15 mN was combined with the detailed topographic data to provide complete coverage over all seven deposits. At Botija, additional topographic data was collected by MPSA in order to reflect the level of stripping as at end December 2018.

14.2.2 Drillhole data

As detailed in Item 10, diamond drilling has been carried out since the late 1960s. All geological data available from the diamond drilling and surface mapping has been used to generate a 3D geological model for the Cobre Panamá project. Assay data used for the Mineral Resource estimates are from diamond drilling campaigns undertaken by Adrian and Teck in the early to mid 1990s, PTC between 2006 and 2008 and, by MPSA since 2007. At Botija, from late 2017, additional reverse circulation drilling and sampling was completed within the pre-strip areas as a pre-cursor to planned grade control drilling.

Apart from Botija, no additional drilling has been completed at the respective deposits. As such Mineral Resource estimates are unchanged for all deposits except Botija.

The majority of drillholes at Balboa, Botija, Colina, Medio, Botija Abajo and Brazo are vertical with some inclined drillholes at Botija, in the south-eastern area of Balboa and the eastern area of Botija Abajo. Drillhole spacing is generally 100 m throughout the deposits. The majority of the drillholes at Valle Grande are inclined to the southwest or the northeast and are at a spacing of approximately 100 m. Reverse circulation drilling at Botija consists of 45 m deep holes drilled at a 15 mE by 15 mN grid.

Exported text (csv) files were sourced directly from the Datashed database, containing the collar and downhole survey data, geological logging data and assay data. All survey, logging and assay data were validated prior to desurveying and drillhole collar elevations were elevated to the topographical surface, for consistency. At Balboa, Colina and Medio, Valle Grande, Botija, Botija Abajo, 83%, 88%, 90%, 83% and 88% (respectively) of the surveyed collar elevations are within 2 m of the topographical surface. At Brazo, only 35% of the surveyed collar elevations are within 2 m of the topographical surface due to a constant shift in coordinate translation of the PTC holes collar coordinates. The shift was determined to be 19 m in the north-south direction and was deemed to have minimal impact on resource estimates of the much larger parent block sizes. FQM has, however, subsequently re-surveyed these collar coordinates and updated the database for future Mineral Resource estimates.

Data was extracted for the Botija, Colina, Valle Grande, Balboa, Medio, Botija Abajo, and Brazo deposit areas and was filtered based on parent company. The number of drillholes and total drilled length used in the Mineral Resource estimates are listed in Table 14-1.

Table 14-1 Number of holes and metres drilled for each deposit area

Deposit	Number of drillholes	Metres
Colina and Medio	277	63,952
Botija	454	95,481
Botija RC drilling	2911	130,995
Botija Abajo and Brazo	295	50,015
Valle Grande	265	46,229
Balboa	121	53,487
Total	**4,315**	**437,679**

Geological logging data imported into the database includes lithological codes, alteration codes, visually estimated pyrite, chalcopyrite, bornite and chalcocite percentages and vein density. RC chip logging was restricted to lithology.

The assay database comes from four drilling campaigns: PMRD between 1970 and 1976, Teck and Adrian in the early-mid 1990s, PTC between 2006 and 2008 and MPSA in 2007-2009. Only assay data from holes drilled after 1990 was used for block grade estimation. The earlier PMRD drillholes were assayed for copper and molybdenum, but not gold or silver. Data from these drillholes was used to assist in the mineralisation interpretation, but was not included for the block grade estimation. The Teck and Adrian, PTC and MPSA samples were analysed for copper, gold, molybdenum and silver.

Assay data was in the form of a series of text (csv) files, and was reformatted before being imported into CAE Mining software. Values below detection limits for the drill data used for the resource estimate were handled as follows:

- PMRD – copper values of -1 and 0 were set to absent and copper values of -1 were set to 0.0005; molybdenum values of -0.59 were set to absent.
- Teck – copper values of 0 set to absent; silver values of -99 and 0 set to absent and -1 set to 0.05; molybdenum values of 0 and -2 were set to absent and values of -1 were set to 0.5; gold values of -99 set to absent and gold values of -1 were set to 0.0025.
- Adrian – copper, gold, and molybdenum values of 0 were set to absent; copper values of -1 and -0.01 were set to 0.005; silver values of -11 were set to absent and silver values of -1 and -0.2 were set to 0.1; molybdenum values of -2 and -1 were set to 0.5; gold values of -1 and -0.005 were set to 0.0025.
- PTC – copper values of 0 were set to absent and values of -1 were set to 0.0005; silver values of -99 were set to absent and silver values of -1 were set to 0.1; molybdenum values of -99 were set to absent and values of -1 were set to 0.5; gold values of -99 were set to absent and values of -1 were set to 0.0025.
- MPSA - copper values of -0.1 and -0.001 were set to 0.005; silver values of -1 and -0.2 were set to 0.1; molybdenum values of -10 and -1 were set to 0.5; gold values of -0.005 were set to 0.0025.
- FEED0709 - copper values of -1 were set to absent; copper values of -0.01 were set to 0.005 and of -0.001 were set to 0.0025; silver values of -1 and -0.2 were set to 0.1; molybdenum values of -1 were set to 0.5; gold values of -1 and -0.005 were set to 0.0025.

The top sections of two twin drillholes (VG95288A and VG95329) at Valle Grande were not assayed where they were adjacent to drillholes with assays; for these sections the copper, molybdenum, silver and gold values were set to absent.

14.3 Data validation

A series of data validations were completed prior to de-surveying the drillhole data into a three dimensional format. These included:

- Visual investigation and checks of the relative magnitudes of downhole survey data were completed in order to identify improperly recorded downhole survey values. No value corrections were required.
- The geology and assay dataset was examined for sample overlaps and/or gaps in downhole logging data, with overlaps and duplication identified and removed.

- Assay data for total copper, gold, molybdenum and silver were interrogated for values that were outside of expected limits.
- The dataset was examined for sample overlaps and/or gaps in downhole survey, sampling and geological logging data, with minor problems resolved.

14.4 Geological and mineralisation models

The Cobre Panamá Project area hosts seven deposits that contain significant amounts of copper and minor molybdenum, gold, and silver resulting from mineralisation related to the intrusion of porphyritic rocks into pre-existing host rocks of andesitic and granodioritic composition. The deposits all occur within an area measuring approximately 10 km east-west by 4 km north-south.

14.4.1 Lithology

As discussed in Item 7, 3D interpretations of the distribution of the rock types at each of the seven deposits were completed from drillhole and mapping information. These interpretations were used to code the Mineral Resource blocks as granodiorite, porphyry, andesite, saprock and saprolite.

A number of post-mineral (barren) dykes have been identified in the drillhole logging, which are generally less than 3 m wide and of variable strike. Together with the current drillhole spacing, it was not possible to create reliable 3D models of the dykes. Statistical analysis of the drillhole data was undertaken to determine adjustment factors for each of the deposits, based on the percentage of the drillhole intersections that were within dyke material. For Valle Grande, vertical and inclined drillholes both yielded similar (approximately 2% dyke material) percentages of intersections that are within dyke material, suggesting that percentages determined in this way were representative of dyke volumes relative to host rock volumes.

Mining is expected to be on a scale of 25 mE by 10 mN on 15 m bench heights; at this scale it will not be possible to selectively mine the mineralised material and to exclude the barren dyke material. Volume adjustment factors, as listed in Table 14-2, were applied to the metal content and the reported Mineral Resource estimate has thus accounted for the barren dyke material.

Table 14-2 Factors applied to metal content to account for barren dyke material

Deposit	Percentage dyke material	Factor applied to metal content
Botija	1.9%	0.981
Balboa	7%	0.93
Colina	1.7%	0.983
Valle Grande	2%	0.98
Medio	1%	0.99
Botija Abajo	0%	1
Brazo	0%	1

14.4.2 Alteration

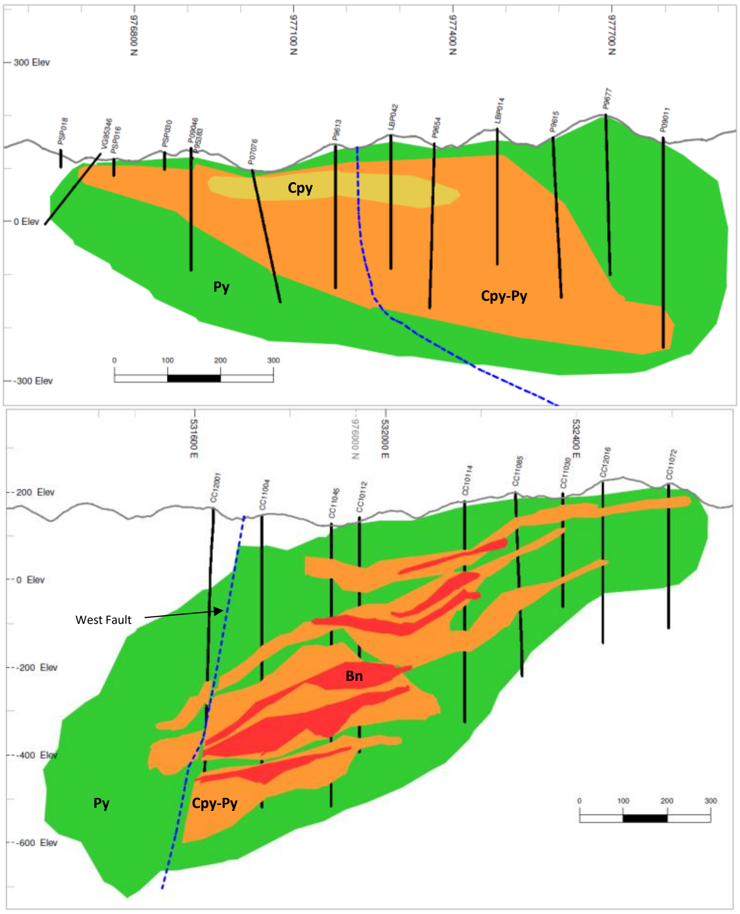
As discussed in Item 7, the database includes geological logging of alteration types. The alteration intersections were reviewed to determine if a 3D interpretation of alteration types could be generated. The distribution of the logged alteration types is spatially erratic and it was therefore not possible to develop a 3D model of the alteration. Optiro recommends that the geological logging of the alteration is reviewed, in conjunction with the geochemical data, to determine if it is possible to develop a robust interpretation of the alteration for future Mineral Resource estimates.

14.4.3 Sulphides

Hypogene copper mineralisation consists primarily of chalcopyrite and minor bornite. A poorly developed mixed zone of oxide copper minerals and secondary copper minerals such as chalcocite and covellite exists in the saprock zone, overlying the hypogene mineralisation.

Three dimensional models were developed to encompass areas that are respectively dominantly pyrite; mixed chalcopyrite and pyrite; chalcopyrite; and bornite. At Botija, Balboa, Colina, Valle Grande and Medio the chalcopyrite domain is surrounded by a mixed chalcopyrite and pyrite domain and both of these domains are encompassed by the pyrite domain. At Balboa and Botija a mixed bornite and chalcopyrite domain was defined that is encompassed by the chalcopyrite-rich domain. Botija Abajo and Brazo did not have sufficiently close drill data to confidently define zones of continuous sulphide groupings. An example of the sulphide domains at Colina and Balboa are presented in Figure 14-2, where Cpy = chalcopyrite, Py = pyrite and Bn = bornite.

Figure 14-2 Sulphide domains as defined at Colina (top) and Balboa (bottom)



14.4.4 Fault zones

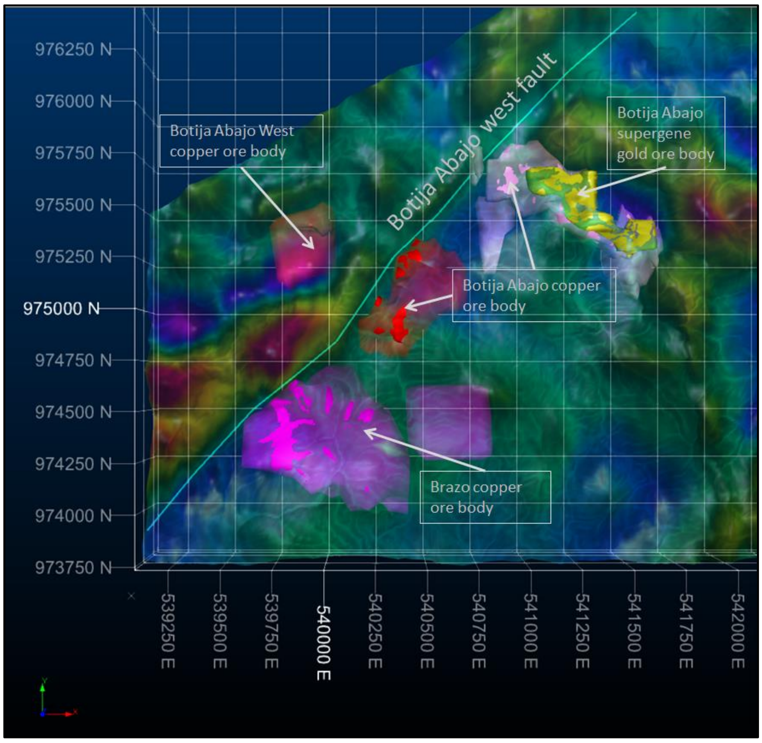
As discussed in Item 7, faults have been modelled at Colina, Medio, Valle Grande, Balboa, Botija Abajo and Brazo based on offsets observed in the geological and mineralisation modelling. While field mapping and logging data at Botija suggests the presence of faulting, no major fault offsets controlling mineralisation were observed. Continued FQM study through detailed logging, mapping, and RC drilling, together with 3D modelling using collated data sets will facilitate a more confident fault model for future Botija estimates.

At Colina several fault orientations were observed. The most significant fault is an east-west trending thrust fault, dipping to the north. A hard boundary at the fault was used for estimation of both the low and medium grade copper domains. Two sub-vertical normal faults offset the mineralisation to the west (Faldalito prospect) and were also treated as hard boundaries (Figure 14-4). Smaller scale faults have been used to truncate the small supergene gold domains but do not extend at depth and were hence treated as soft domains for estimation. Bounding faults to the north and west of the Colina deposit were used to truncate the overall mineralisation envelope. At Medio, a large, sub-vertical strike slip fault was used to truncate and offset the mineralisation.

At Valle Grande the mineralisation is offset along the northeast-southwest fault illustrated in Figure 7-9 and a hard boundary was applied at this fault for grade interpolation. Balboa is cross cut on the western margin by a north-trending fault termed the West Fault (Figure 7-13) and for grade interpolation a hard boundary was applied at this fault.

A relatively minor volume of mineralisation along the western extents of Botija Abajo (Botija Abajo West) is offset by a well developed northeast striking sub-vertical fault, identified from a strong linear feature in the airborne magnetic imagery (Figure 14-3).

Figure 14-3 3D plan view of the northeast trending fault at Botija Abajo relative to the main copper ore body positions



14.4.5 Gypsum/Anhydrite front

Geological logging data and strontium sample assay data were used to develop a surface that constrained the extent of a gypsum/anhydrite front at depth at Botija, Balboa, Colina, Valle Grande and Medio. During weathering, the strontium contained in plagioclase feldspars is typically reduced to below 100 ppm and as a result often coincides with the gypsum/anhydrite front. The presence of anhydrite as opposed to weathered gypsum will decrease milling throughput and increase power consumption. Moreover, weathered gypsum zones will reduce rock strength and may impact on slope stability. Accordingly, the model was coded above or below this surface. There was insufficient drill data at Botija Abajo and Brazo to permit confident delineation of the gypsum anhydrite front for these deposit areas.

14.4.6 Mineralisation interpretation

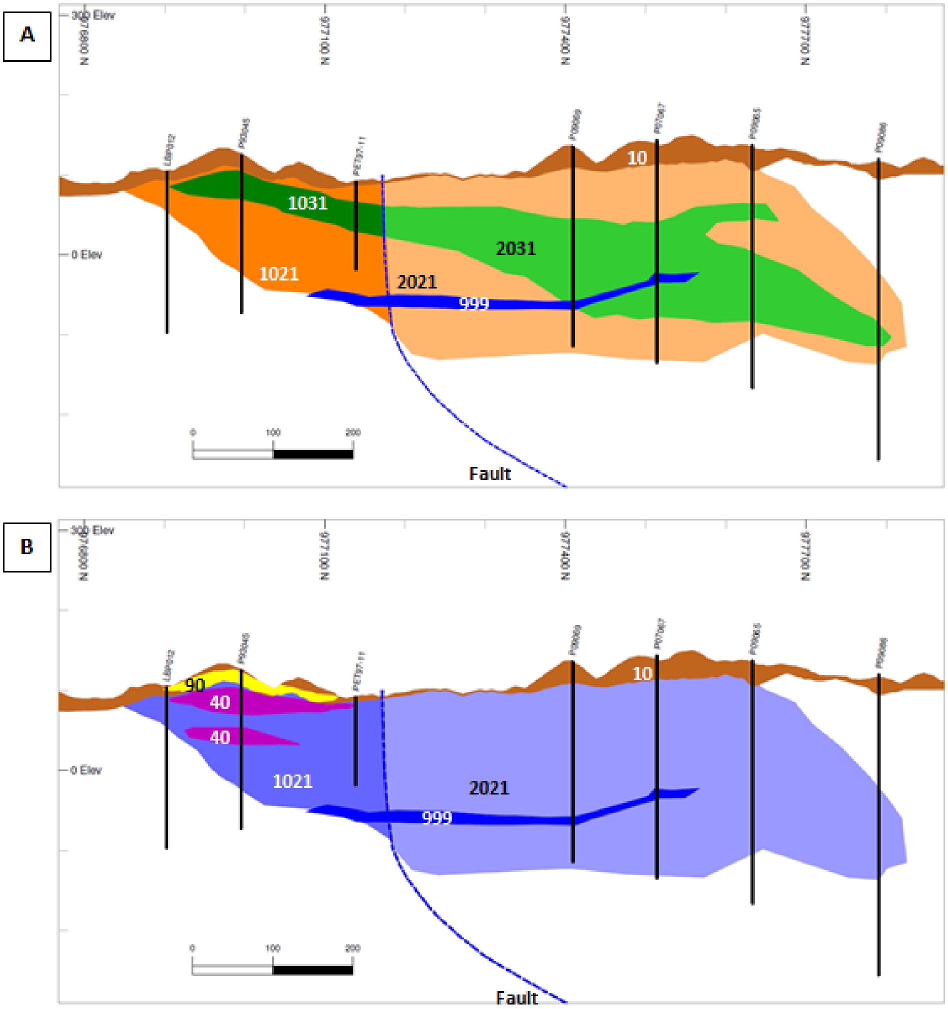
Mineralisation at the Cobre Project tends to occur within the porphyritic rocks and extends into the contact area of the adjacent plutonic/volcanic rocks. For each deposit, examination of the drillhole intersections and grade inflections in log-probability plots was used to select nominal cut-off grades used for the interpretation of low grade and medium grade copper domains. Additional gold domains were defined at Balboa, Colina, Botija Abajo and Brazo.

At Balboa, copper mineralisation is dominantly hosted by a feldspar-quartz-hornblende porphyry that intrudes the adjacent andesite and mineralisation is best developed in the central portion of the porphyry. Nominal cut-off grades of 0.15% copper and 0.4% copper were used to model the low grade and the medium grade domains respectively. In addition, a depleted copper/supergene gold cap was interpreted to constrain 'higher' gold grades and depleted copper grades. The medium grade domain was not applicable for molybdenum and a soft boundary was applied between the low and medium grade domains for estimation of molybdenum grades. The low grade and medium grade domains were used for grade estimation of copper, gold and silver. Mineralised domains are illustrated in Figure 14-4.

Copper mineralisation at Colina tends to be associated with the feldspar-quartz-hornblende porphyry, particularly in the thicker areas were the dykes coalesce and around the upper contact of the sills. Nominal cut-off grades of 0.15% Cu and 0.4% Cu were used to model the low grade and the medium grade domains respectively. Gold domains were interpreted based on a nominal 0.3 ppm cut-off grade. Internal waste domains were defined where there were significant intervals of material below a nominal cut-off grade of 0.15 % Cu. Mineralised domains for both copper and gold are illustrated in Figure 14-4.

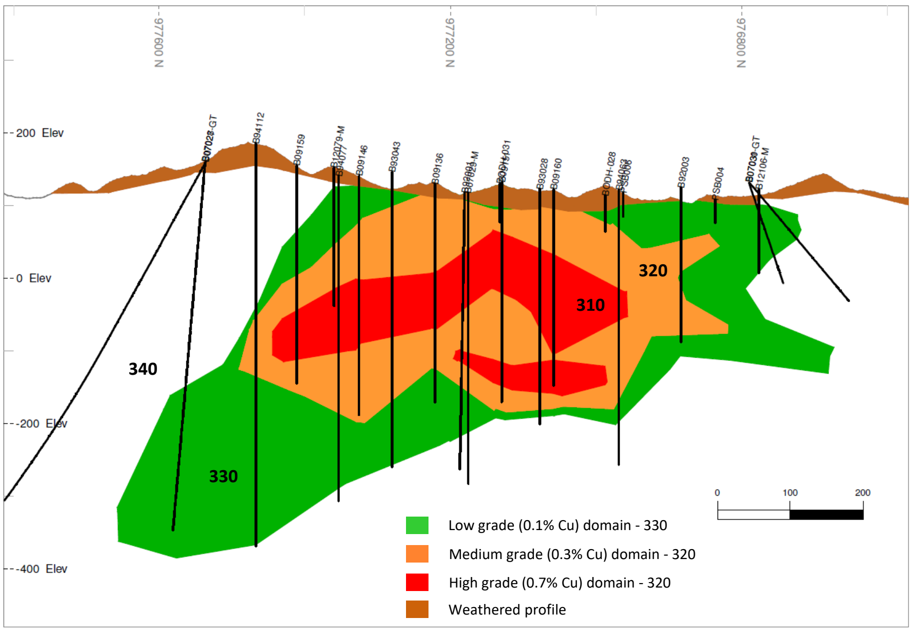
Copper mineralisation at Botija was based on nominal copper cut-off grades of 0.15% for low grade, 0.3% for medium grade and 0.7% for high grade mineralisation (Figure 14-5). Gold, molybdenum and silver were each estimated within the same copper domains due to reasonable to good correlations with copper grades. Mineralisation at Botija favours the early, intra-mineral and late stage porphyries. In addition, the mid-southern portion of Botija is characterised by a sub-surface andesitic cap, which has some lower grade mineralisation along its lower contact with the underlying mineralised porphyry.

Figure 14-4 Copper (A) and Gold (B) domains at Colina (south-north section along 533,520 mE)



Domain codes; Depleted zone (10), Waste zone (999), Low grade copper and gold (south of fault – 1021, north of fault – 2021), Medium grade copper (south of fault – 1031, north of fault – 2031), High grade hypogene gold – 40, High grade supergene gold – 90

Figure 14-5 Example of domains at Botija (north-south section along 538090 mE)



Copper mineralisation at Medio was interpreted based on a nominal copper cut-off grade of 0.15% for the low grade mineralisation and 0.5% for the medium grade domains. Fault zones have been interpreted at Medio. Due to the scarcity of data, soft boundary conditions were applied at Medio for grade estimation.

At Valle Grande, copper mineralisation is generally concentrated along the flanks of the porphyry lopolith, following the porphyry-andesite contacts. Nominal copper cut-off grades of 0.15% and 0.5% were used to model the low grade and the medium grade domains respectively. Mineralised domains for Valle Grande are illustrated in Figure 7-11. At Botija Abajo and Brazo, gold domains were defined separately to the copper domains. The northern body of the Botija Abajo mineralisation has evidence of supergene gold developing within a narrow horizon in upper leached copper horizon. Copper mineralisation at Botija Abajo and Brazo used a nominal 0.15% copper cut-off. Molybdenum and silver mineralisation were grouped with the copper domains for estimation.

The mineralisation domain codes developed for each deposit are listed in Table 14-3.

14.5 Data preparation for modelling

The de-surveyed assay drillhole data were selected within the mineralisation wireframes and each sample interval was coded with a mineralisation domain code for estimation (Table 14-3). The coded drillhole data was exported for compositing, statistical and geostatistical analysis, and grade estimation.

14.5.1 Data compositing

The distribution of input sample lengths guided the selection of a 1.5 m composite sample length (Figure 14-6). Approximately 80% of the data has a sample length within a few centimetres of 1.5 m. All data was composited to intervals of 1.5 m within the mineralisation domains, per element, and excluded the waste material from mafic and felsic dykes. This has ensured that the sample intervals provide good resolution across domain boundaries, as well as honouring the original sample lengths.

14.6 Statistical analysis

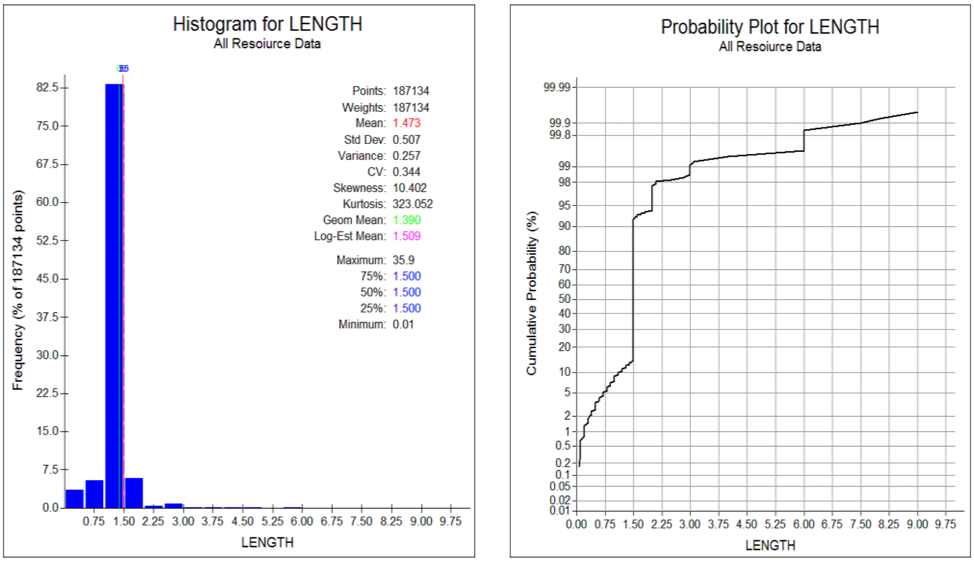
Statistical analysis of the data, including spatial statistics, was carried out using Snowden Supervisor software. Data distributions for copper, gold, molybdenum and silver were investigated by domain with histograms and probability plots and statistical parameters for the composited copper, gold, molybdenum and silver grades within each of the final domains are summarised in Table 14-4.

The objective of the mineralisation interpretations was to separate mixed populations and reduce internal variability, thereby assisting with spatial analysis and providing a more robust estimate. Molybdenum and silver histograms showed some evidence of possible detection limit artefacts associated with the different generations of drilling.

Table 14-3 Domain codes for the drillhole data and block model

Deposit	Element	Domain description	Domain code
Botija	Cu,Au,Mo,Ag	Ultra low grade halo mineralisation Low grade Medium grade High grade	340 330 320 310
Balboa		Low grade west of West Fault	21
		Medium grade, west of West Fault	22
		Low grade, east of West Fault	31
		Medium grade, east of West Fault	32
		Near surface supergene, depleted copper, enriched gold	50
Colina	Cu,Mo	Main low grade, north fault block	2021
		Main low grade, south fault block	1021
		Mid-west low grade	22
		West low grade	23
		Main medium grade, north fault block	2031
		Main medium grade, south fault block	1031
		Mid-west medium grade	32
		West medium grade	33
		Depleted zone	10
		Internal waste	999
	Au,Ag	High Grade – Hypogene	40
		High Grade – Supergene	90
Valle Grande		Low grade, west of NE-SW Fault	1
		Low grade, east of NE-SW Fault	2
		Medium grade	3
Medio	Cu, Mo, Ag	Low Grade - East	1
		Low Grade - Main	2
		Medium Grade - Main	3
		Low Grade - West	4
Botija Abajo, Brazo	Cu,Mo,Ag	Medium grade, Brazo	321
		Medium grade, Botija Abajo	322
		Medium grade, Botija Abajo	324
		Low grade, Brazo	331
		Low grade, Botija Abajo	332
		Low grade, Botija Abajo West	333
		Low grade, Botija Abajo	334
		Ultra low grade halo mineralisation	340
	Au	Medium grade gold, Botija Abajo	322
		Low grade gold, Brazo	331
		Low grade gold, Botija Abajo	332
		Botija Abajo West	333
		Low grade gold, Botija Abajo	334
		Supergene gold, Botija Abajo	335

Figure 14-6 Histogram and cumulative distribution of sample lengths highlighting the dominant 1.5 m sample length



Statistical analysis indicates that the selected mineralised domains are well defined, with a minimal degree of mixing. Coefficients of variation (CV) for each deposit and per domain are mostly less than one for copper, indicating that the domains have captured areas of similar (low) sample variability. CVs for gold, molybdenum and silver are mostly moderate. Where the CV is high and outlier grades were identified, top-cuts were applied (as discussed below). Sample values for all metals and domains show reasonably well constrained histograms, with limited evidence for domain mixing.

14.6.1 Determination of top-cuts

Top-cuts were used to define the maximum reasonable metal grade for a composite sample value within a given domain. If the grade of a sample exceeded this value, the grade was reset to the top-cut value. Top-cuts for the seven deposits were established by investigating the mean and coefficient of variation and histograms and log-scale probability plots of assay data by domain. The top-cuts applied to the data for resource estimation are listed in Table 14-5. The selected top-cuts have a marginal effect on the mean value per domain, but do reduce the risk of excessively high value composites distorting block estimates, particularly in areas of low data support.

14.6.2 Boundary analysis

A series of contact profiles were generated to evaluate the change in copper grades across domain boundaries at Botija, Botija Abajo, Brazo, Balboa, Colina and Valle Grande and the change in gold grades across boundaries at Balboa and Valle Grande (Figure 14-7). Sharp grades are present and so hard boundaries were employed across these contacts, in order to limit smearing and dilution.

Figure 14-7 Boundary analysis at Balboa, Colina and Valle Grande

Table 14-4 Summary statistics for copper %, gold g/t, molybdenum g/t and silver g/t per domain

Domain	Copper			Gold			Molybdenum			Silver		
	Number	Mean	CV	Number	Mean	CV	Number	Mean	CV	Number	Mean	CV
Botija												
310	2,624	0.89	0.32	2,345	0.21	0.72	2,608	134.77	1.21	2,208	2.26	0.58
320	7,135	0.56	0.44	6,013	0.12	0.73	7,110	106.07	1.27	4,099	1.60	0.71
330	26,610	0.22	0.66	18,777	0.05	1.17	26,604	49.99	1.65	8,631	0.92	0.84
335	8,345	0.13	0.76	6,560	0.03	1.87	8,346	25.16	1.77	4,052	0.57	0.97
340	6,761	0.05	1.42	6,290	0.02	2.45	6,763	11.61	2.82	6,228	0.36	1.75
Balboa												
21	1,702	0.16	0.74	1,702	0.03	2.09	1,947	21.19	3.53	1,702	1.03	1.76
22	245	0.64	1.12	245	0.32	2.44				1,702	1.03	1.76
31	12,149	0.20	0.86	12,149	0.04	2.69	17,717	19.85	2.30	12,149	0.96	1.20
32	5,568	0.64	0.78	5,568	0.19	2.40				5,568	2.40	1.06
50	126	0.07	0.52	126	0.55	1.36	126	12.74	1.70	126	1.11	1.84
Colina												
10	2,010	0.10	1.83	1685	0.06	1.33	2,010	41.6	1.95	1,686	1.49	11.6
21	12,790	0.31	0.64	17,673	0.06	2.51	12,792	59.3	2.01	17,771	1.65	1.77
22	374	0.28	0.60	455	0.06	0.87	374	47.6	1.40	455	1.59	0.76
23	939	0.31	0.60	1,034	0.08	0.93	939	30.9	1.83	1,034	1.44	0.76
31	8,524	0.62	0.53	17,673	0.06	2.51	8,528	97	1.45	-	-	-
32	132	0.80	0.58	-	-	-	132	108.4	2.50	-	-	-
33	140	0.60	0.53	-	-	-	140	17	1.74	-	-	-
40	-	-	-	481	0.39	0.76	-	-	-	482	2.5	0.70
90	-	-	-	190	0.51	0.66	-	-	-	190	1.11	1.62
999	316	0.06	1.42	247	0.02	1.88	312	16	2.41	247	0.39	1.12
Valle Grande												
1	285	0.20	0.48	285	0.043	1.49	285	43.7	1.15	285	0.34	0.82
2	14,514	0.29	0.95	14,475	0.04	3.36	14,514	61.1	1.92	14,389	1.26	1.86
3	5,037	0.68	0.77	5,004	0.084	1.31	5,037	95.8	1.60	4,967	2.19	0.69
Medio												
1	443	0.21	0.55	443	0.031	0.93	443	33.10	1.61	443	1.26	1.31
2	1523	0.22	0.75	1,524	0.032	1.14	1524	39.33	1.93	1,524	0.90	1.08
3	148	0.83	0.54	148	0.083	0.99	148	68.20	2.43	148	2.65	0.64
4	542	0.36	1.26	542	0.033	2.82	-	-	-	542	1.27	0.95
Botija Abajo												
322	29,096	0.57	2.31	29,035	0.09	0.88	29,057	52.8	1.10	29,057	0.73	1.05
324	29,096	0.72	1.10				29,057	24.3	0.32	29,057	2.84	0.81
332	29,096	0.26	1.50	29,035	0.04	0.60	29,057	43.8	1.12	29,057	0.33	1.05
333	29,096	0.29	1.14	29,035	0.09	0.31	29,057	70.6	0.48	29,057	1.38	0.65
334	29,096	0.19	1.46	29,035	0.30	0.60	29,057	25.4	0.84	29,057	1.26	0.84
335				29,035	0.75	0.80						
340	29,096	0.05	0.64	29,035	0.03	0.26	29,057	13.1	0.40	29,057	0.43	0.35
Brazo												
321	29,096	0.56	1.82				29,057	45.70	0.86	29,057	0.91	0.93
331	29,096	0.21	1.32	29,035	0.18	1.07	29,057	38.87	0.64	29,057	0.67	0.61

Table 14-5 Top-cuts applied per domain

Deposit	Domain	Copper (%)	Gold (ppm)	Molybdenum (ppm)	Silver (ppm)
Botija	310	5.0	1.8	2000	20.0
	320	5.0	1.2	2000	18.0
	330	1.0	0.8	1800	18.0
	340	1.0	0.8	1600	18.0
Balboa	21	None	0.7	600	20.0
	22	None	4.0	600	11.4
	31	None	1.0	600	20.0
	32	4.0	2.0	600	30.0
	50	None	4.0	None	10.0
Colina	10	1.5	0.5	550	10
	21	1.5	0.8	800	14.0
	22 and 23	1.0	0.45	350	6.0
	31	3.0	NA	1,200	NA
	32 and 33	1.8	NA	600	NA
	40	NA	1.5	NA	9.0
	90	NA	1.7	NA	7.5
	999	None	0.25	None	None
Valle Grande	1 and 2	4.0	1.1	1,900	15.0
	3	7.0	1.4	1,900	15.0
Medio	1 and 2	1.4	0.18	800	8
	3	2.0	0.18	800	8
	4	1.4	0.18	800	8
Brazo	321	3.0	None	570	15
	331	1.4	2.1	810	23
Botija Abajo	322	1.7	0.77	350	8
	324	4	None	810	35
	332	1.6	0.62	300	4
	333	2.1	5.8	1500	19
	334	1.3	4	300	20
	335	None	5	None	None
	340	1.2	None	300	24

14.6.3 Metal correlations – self organizing feature maps

Self-organising feature maps were generated using the copper, gold, molybdenum and silver data from Balboa, Colina, Valle Grande and Medio to examine the relative metal associations within the deposits as well their relative differences.

Self-organising feature maps (SOFM) are a pattern recognition technique based on neural networks. The training phase of a SOFM consists of several iterations over the input patterns to gradually adjust the synaptic weights in the lattice according to a well-defined rule. In this way, after repeated iterations, the SOFM becomes tuned to the statistical regularities present in the input patterns, developing the ability to create internal representations for assessing features of the input data. Patterns are commonly analysed by colouring each neuron in the lattice with the components of the weight vector. A set of maps is produced which allow examining features that tend to be distributed in a similar way. These features may be used for gaining insight into the raw data and their groupings: for instance, they can be used for domain definition, correlation analysis and pattern identification.

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At Botija (Figure 14-8), gold, silver and molybdenum have different patterns to copper suggesting poor relationships. Despite this, there is some correlation of a more dominant high grade copper population with higher grade gold and silver. There is also a higher grade molybdenum grouping, which has some correlation with a separate moderate to higher grade copper. A high grade gold grouping has little to no correlation with copper, silver or molybdenum and is likely to be associated with a weak supergene or leach process.

At the Balboa, Colina and Valle Grande deposits copper, gold, molybdenum and silver grades per composited sample from the drillhole database were used as a four dimensional input vector to a SOFM. The self-organising feature maps obtained are shown in Figure 14-9 to Figure 14-12 and the following observations are made:

- The mineralisation relationships at all four deposits are different and that for all of the deposits there is no relationship between the molybdenum mineralisation and the other three metals (copper, gold and silver).
- At Balboa there is a poor relationship between the copper, gold and silver mineralisation.
- At Colina there are similarities in the gold and silver maps; these show no relationship to the copper. This supports the definition of additional gold domains at Colina which were also applied for silver estimation.
- At Valle Grande there is a poor relationship between gold and silver; these show no relationship to copper. Domain definition based on a low copper cut-off grade will capture the elevated silver and molybdenum mineralisation and the majority of the gold mineralisation. An elevated gold dataset is evident that is not associated with copper or silver mineralisation; examination of the drillhole data indicated that this was within the saprolite domain and that there was insufficient data to generate a separate sub-domain. This elevated gold mineralisation was capture in the background model.
- At Medio there are similarities between the copper and silver mineralisation and no relationship between the copper and gold mineralisation. There is insufficient data at Medio to develop mineralised domains based on gold. Domain definition based on a low copper cut-off grade will capture the elevated gold, silver and molybdenum mineralisation.

Figure 14-8 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Botija; hot colours signify high values and vice versa

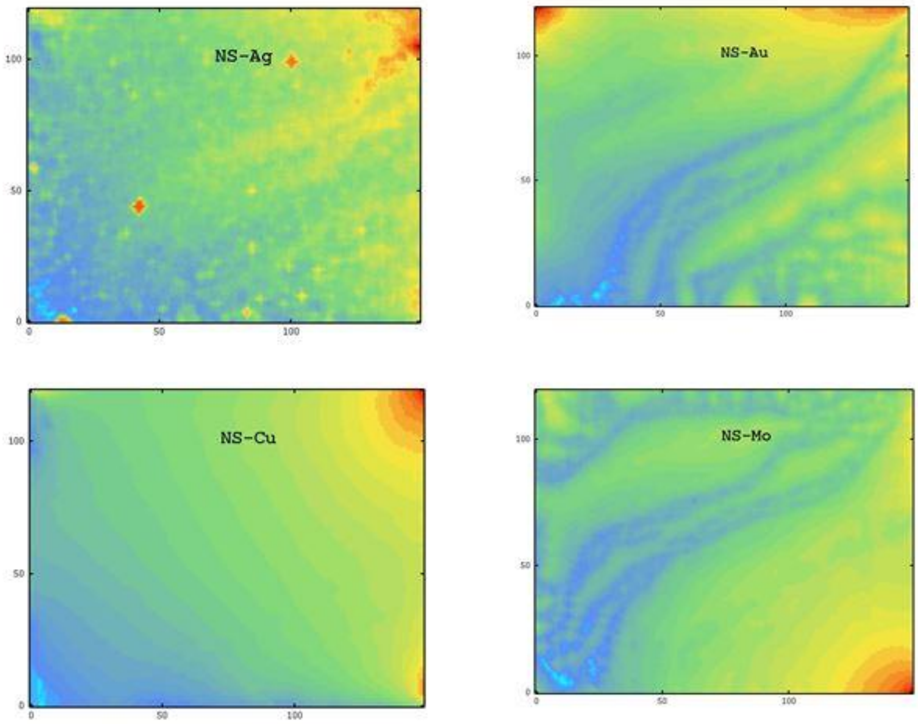


Figure 14-9 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Balboa; hot colours signify high values and vice versa

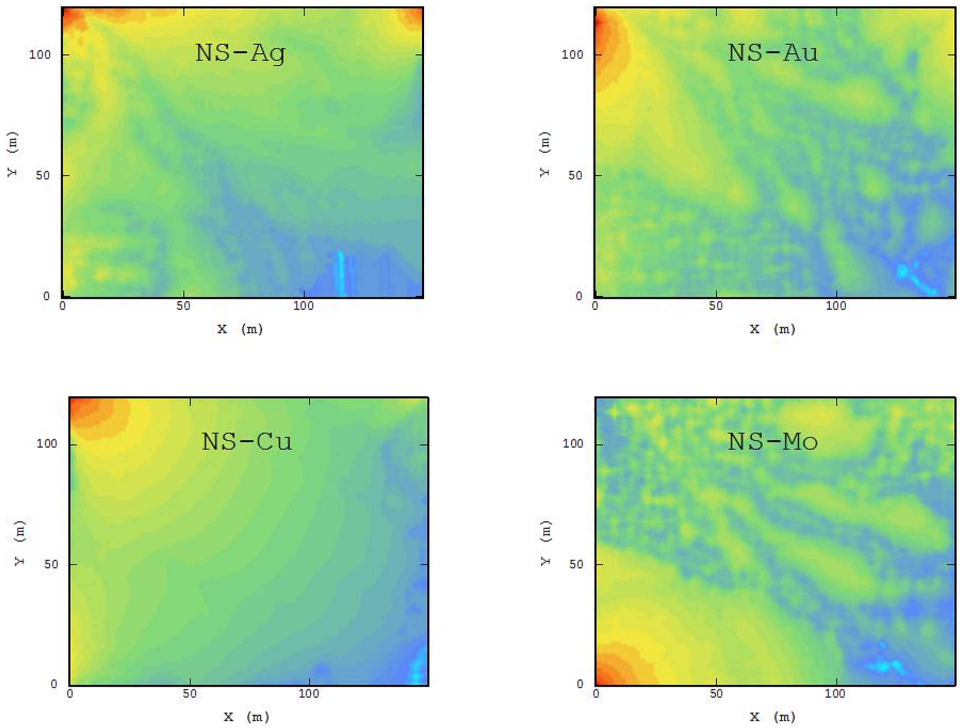


Figure 14-10 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Colina; hot colours signify high values and vice versa

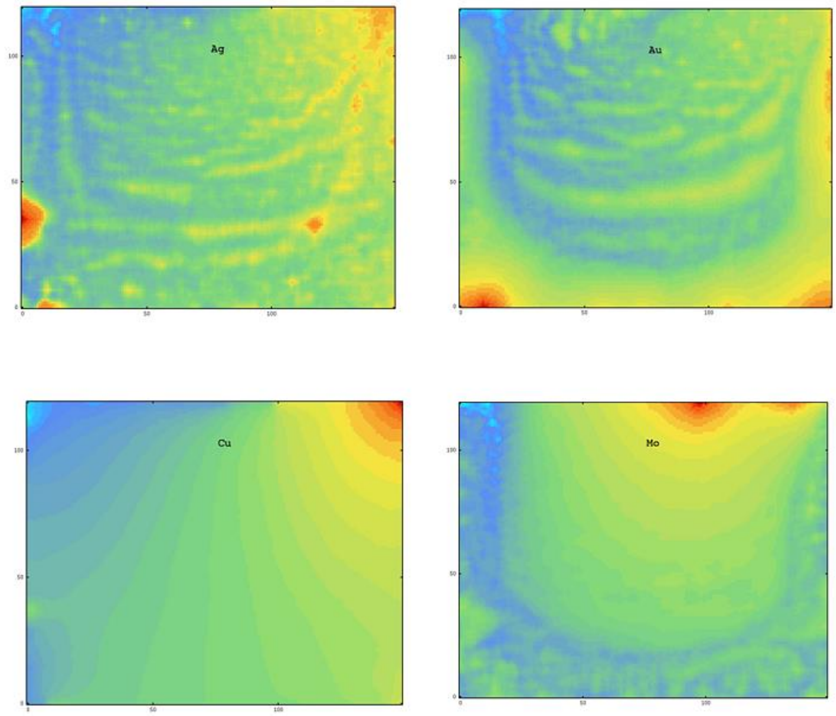


Figure 14-11 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Valle Grande; hot colours signify high values and vice versa

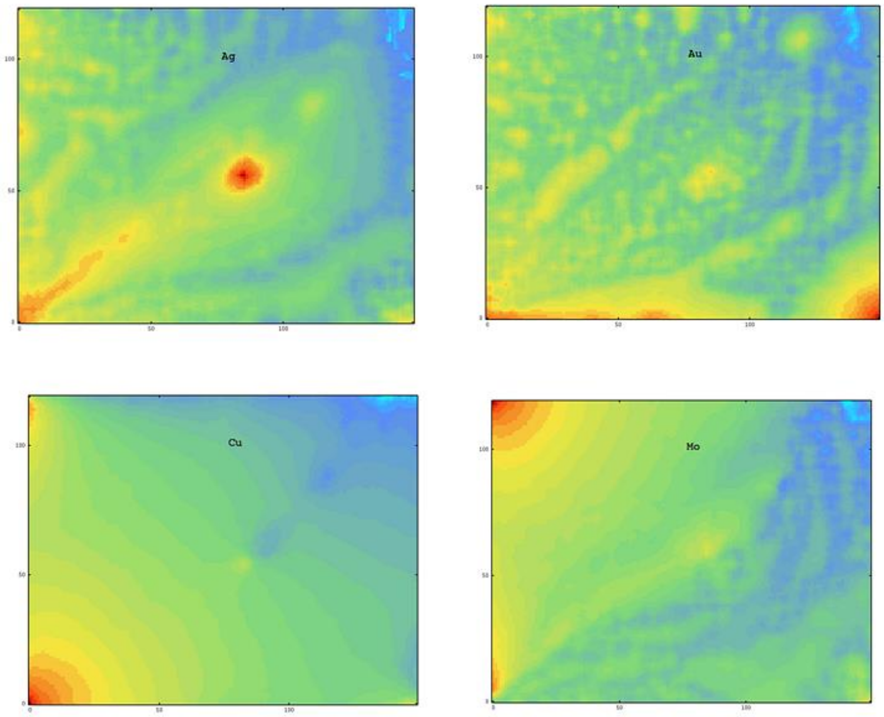
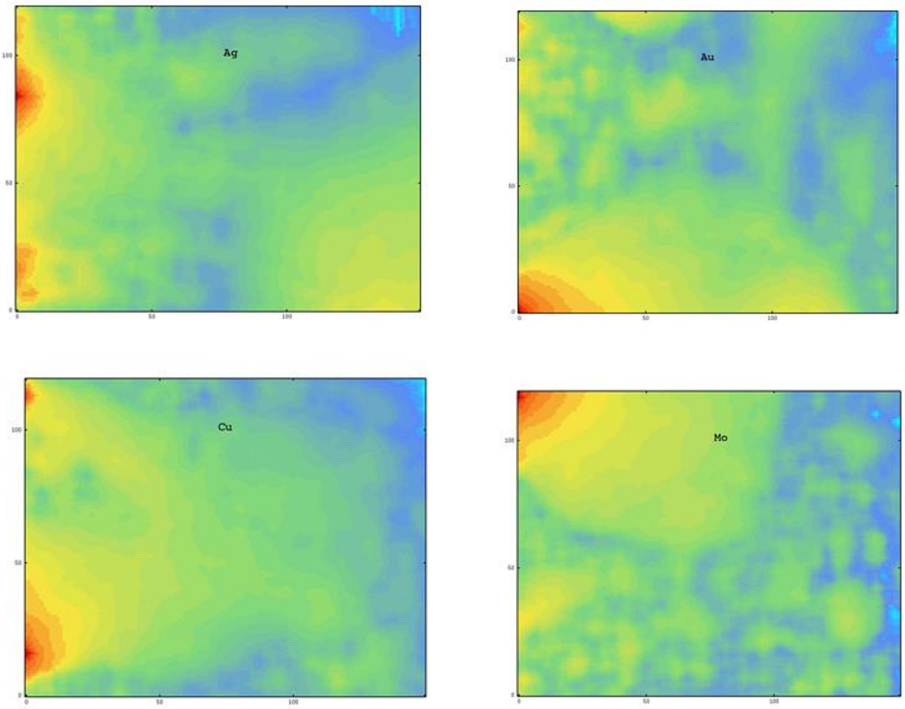


Figure 14-12 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Medio; hot colours signify high values and vice versa



14.6.4 Bivariate element relationships

The correlation coefficient between copper and gold (0.78) is good for Botija mineralisation. Silver is also reasonably well correlated with copper (0.71). There is an average to good correlation between copper and molybdenum (0.65), with molybdenum sometimes showing higher grades in the lower copper grade areas.

At Botija Abajo and Brazo, copper, molybdenum and silver are spatially coincident in similar volumes of the host rock despite both having poor correlations (<0.3) with copper. Correlation of copper and gold at Botija Abajo and Brazo is average (0.54), but gold mineralisation is grouped into distinctly different spatial volumes when compared to copper. The northern portion of Botija Abajo is noted for having a supergene gold horizon that has developed within the near surface leached copper zone. Accordingly, gold in these deposits was defined and estimated using separate volumes from copper.

At Balboa, correlation coefficients indicate a strong relationship (0.81) between gold and silver within the low grade western domain (21). There is no correlation between the other metal combinations. Correlation coefficients indicate a moderate to good relationship (0.56) between Cu and Ag within the low grade eastern domain (31). There is no correlation between the other metal combinations. Within the medium grade domains there are moderate correlations (of 0.56 to 0.77) between copper and gold in domains 22 and 32, copper and silver in domains 22 and 32, and gold and silver in domain 32. There is no correlation between molybdenum and the other metals. Within the depleted copper domain (50) there is no correlation between copper, gold, silver or molybdenum.

At Colina there is poor to moderate correlation between copper, gold, silver or molybdenum within the low and medium grade domains. Correlation coefficients are all less than 0.45. Within the gold domains there is a moderate correlation between copper and gold (0.58) in domain 40 and between copper and silver (0.61) in domain 90.

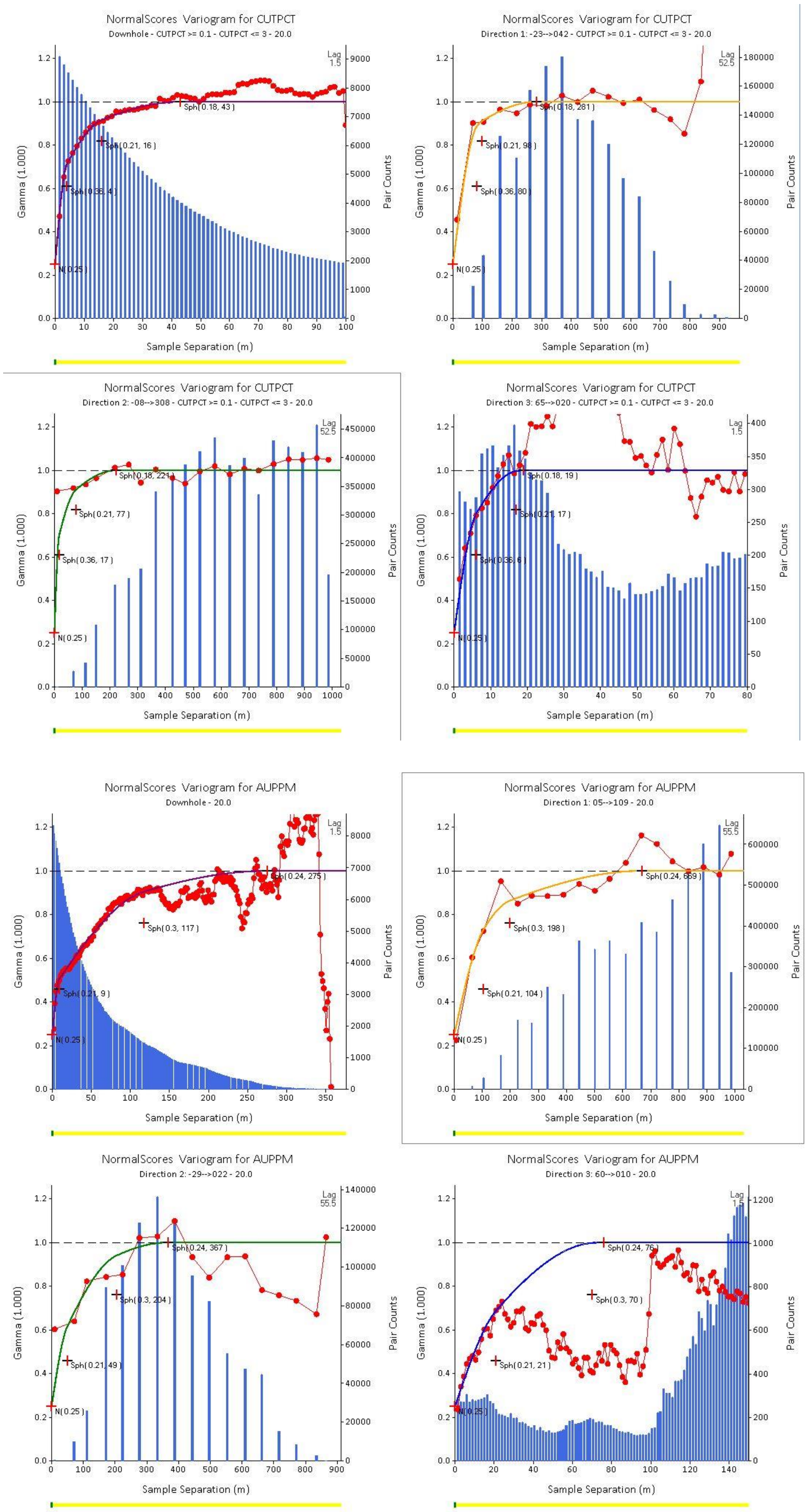
At Valle Grande there is no correlation between copper, gold, silver or molybdenum within the low grade domains. Within the medium grade domain there is a moderate to poor correlation between copper and gold (0.53) and copper and silver (0.69).

Variograms, representing 3D grade continuity, were generated for copper, gold, molybdenum and silver using composite data located within each of the mineralised domains at Botija, Balboa, Colina, Valle Grande, Medio, Botija Abajo and Brazo. The following methodology was applied:

- data was declustered prior to variogram modelling so as to remove the effect of closely spaced samples
- the principal axes of anisotropy were determined using variogram fans based on normal scores variograms
- directional normal scores variograms were calculated for each of the principal axes of anisotropy
- downhole normal scores variograms were modelled for each domain to determine the normal scores nugget effect
- variogram models were determined for each of the principal axes of anisotropy using the nugget effect from the downhole variogram
- the variogram parameters were standardised to a sill of one for copper, gold, molybdenum and silver
- the variogram models were back-transformed to the original distribution using a Gaussian anamorphosis and used to guide search parameters and complete ordinary kriging estimation
- the variogram parameters for copper and gold were standardised to the population variance for each domain to permit post-processing of the copper and gold panel estimates to SMU estimates.

At Botija, variograms for most elements have similar nugget values, ranging from 0.25 (for low grade copper) to 0.35 (for low grade silver). Ranges of grade continuity were well defined from variography and had clearly different anisotropy and orientations per metal. The longest ranges of copper grade continuity (200 m to 300 m) were in the approximate east-west direction. Gold tended to have longer ranges (greater than 400 m) of continuity and was more strongly anisotropic than copper. Molybdenum and silver variogram models were similar to copper. An example of Botija variography (for Domain 31) is shown in Figure 14-13.

Figure 14-13 Variograms and models for Domain 330 at Botija for copper (top) and gold (bottom)



At Colina, nugget values for each element were similar, ranging from 7% to 28% of the total variability. For both copper and gold, the longest ranges of continuity within the main low grade domains were oriented at between 1° and -7° to the east with a range of 130 m (copper) and 540 m (gold). For the medium grade copper domain, the longest ranges of continuity was oriented down-dip at -2° to the west with a range of 180 m. The high grade gold domain had a longest range of continuity of 250 m oriented towards the east. In most domains, direction of anisotropy for molybdenum and silver followed the respective copper and gold domains but the ranges were significantly different; longer for molybdenum (between 185 m and 580 m) and shorter (between 100 m to 255 m) for silver. For domains with insufficient samples to generate reasonably structured variograms, e.g. domains at the western most part of the ore body, variograms from adjacent domains were applied. Typical variograms from Colina for Domains 31 (copper) and 21 (gold) are depicted in Figure 14-14.

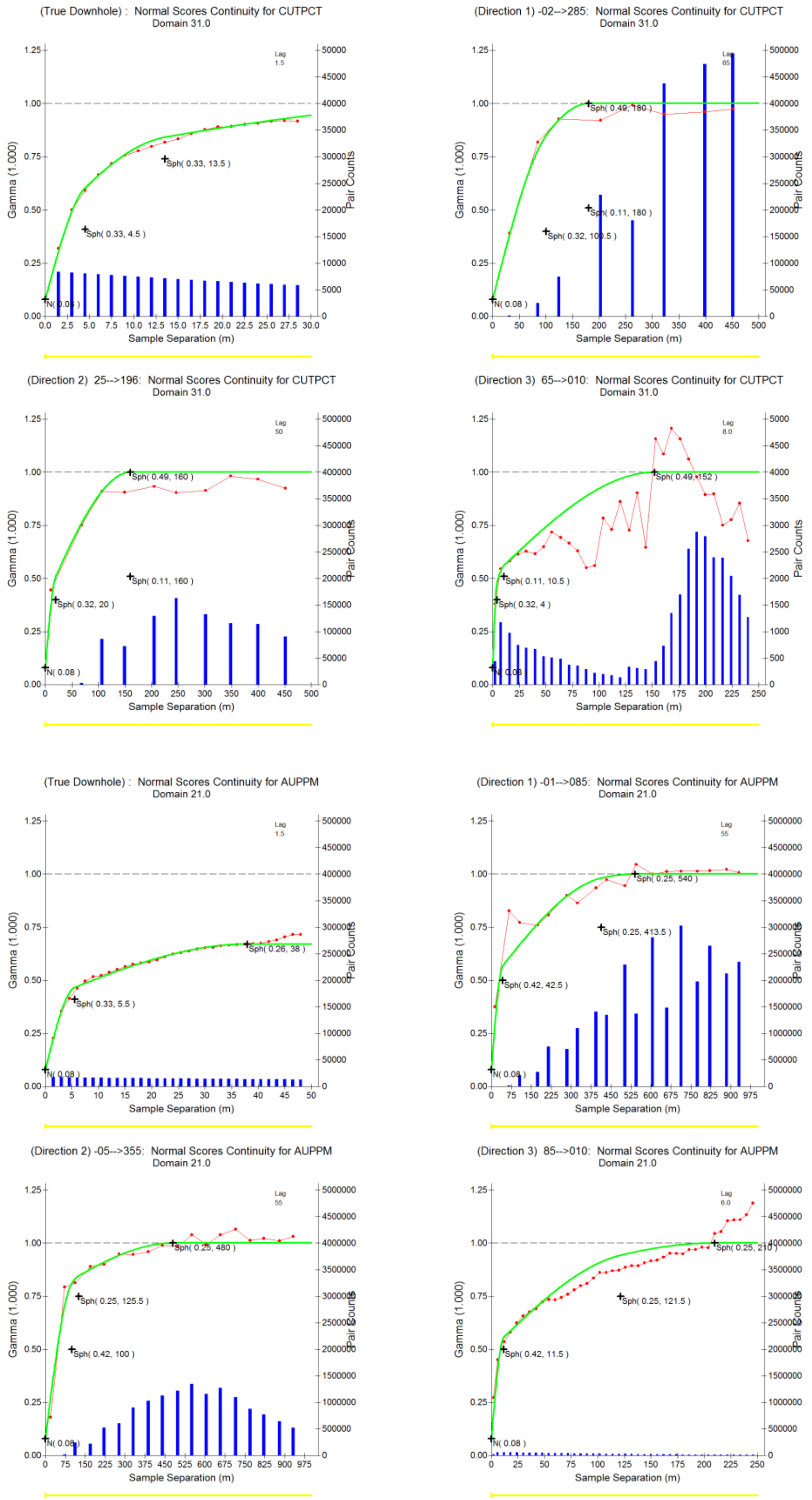
At Balboa, the western domains (21 and 22) had insufficient samples to generate reasonably structured variograms, and so variograms generated for the eastern domains (31 and 32) were applied to the western domain. For domain 50 there were insufficient samples to generate directional variograms. Omni-directional variograms were generated, the overall geological orientation being used to define the directions and the ranges were factored such that direction 1 was twice the omni-directional range, direction 2 was the omni-directional range and direction 3 was half the omni-directional range. This domain was classified as Inferred.

Variogram analysis at Valle Grande indicates that the copper, silver and molybdenum mineralisation within the low grade domain have similar orientations and anisotropic ratios. The longest ranges of continuity are oriented down-dip at -20º to -30º to the southwest and copper, silver and molybdenum have long ranges of 300 m, 230 m and 310 m, respectively. Copper, silver and molybdenum have long ranges of 215 m, 85.5 m and 270 m, respectively, oriented at 0º to -4º to the northwest. The ranges of continuity perpendicular to the dip direction are 22 m for copper, 65 m for silver and 67 m for molybdenum.

For the gold mineralisation the longest range of continuity, of 235 m, is oriented at -5º to the northwest, and the down dip and perpendicular ranges are 110 m and 51 m respectively. Within the medium grade domain the longest ranges of continuity are oriented along plunge at -3º to -6º to the northwest; copper has a long range of 260 m and silver has a long range of 210 m. For gold the longest range of continuity (280 m) is also oriented northwest-southeast, except that it plunges at -6º to the southeast.

The mineralisation continuity dips to the southwest at -14º to -39º to the southwest with long ranges of 100 m, 265 m and 180 m for copper, gold and silver respectively. The shortest ranges of continuity (perpendicular to the dip) are 27 m for copper, 75 m for gold and 46 m for molybdenum. For the molybdenum mineralisation the longest range of continuity, of 270 m, is oriented down-dip at -25º to the southwest, and the along strike (northwest) and perpendicular ranges are 160 m and 46 m respectively. Nugget effects are low and range from 7% to 16% of the total variability.

Figure 14-14 Variograms and models from Colina for copper (Domain 31, top) and gold (Domain 21, bottom)



Variograms generated for copper, gold, molybdenum and silver mineralisation at Medio were poorly defined, particularly within the medium grade domain. Within the low grade domain the longest ranges of mineralisation continuity (260 m, 280 m and 330 m for copper gold and silver respectively) were interpreted along -5° to -19° to the southeast. The copper, gold and silver mineralisation was interpreted to dip at -9° to -23° to the southwest and long ranges of 160 m, 240 m and 150 m were modelled for copper, gold and silver respectively.

Variogram models for the perpendicular direction had short ranges of 6 m to 19 m and a long range zonal component was modelled with ranges of 130 m to 220 m. Variogram ranges for the medium grade domain are much shorter with ranges of 80 m to 100 m to the west, 45 to 80 m to the north and 8 m to 30 m in the perpendicular direction. Molybdenum data was combined for the low and medium grade domains. Ranges of 380 m to the southeast, 130 m at -10º to the southwest and 160 m in the perpendicular direction were modelled. Nugget effects are low to moderate and range from 14% to 24% of the total variability. Variograms across the respective deposit areas at Botija Abajo and Brazo also had nugget values ranging from 0.09 (Botija Abajo north high grade copper) up to 0.31 (Botija Abajo low grade molybdenum). Most nugget values were close to 0.2 and were clearly defined from downhole variography. In contrast, directions of anisotropy and their respective ranges were poorly defined, resulting in the need to use isotropic variography per element and domain. Short ranges of 47 m were obtained for copper mineralisation in the Botija Abajo north area. In contrast, molybdenum had a long range of influence of 320 m in the Brazo area low grade domain. Variogram ranges for most of the domains and respective metals at Botija Abajo and Brazo was between 80 m and 250 m.

14.7 Kriging neighbourhood analysis

A detailed kriging neighbourhood analysis (KNA) was undertaken at Botija and Valle Grande to determine the optimal block size, each ellipse dimensions, minimum and maximum numbers of samples to be used for grade estimation and the discretisation parameters. The optimal parameters established from this study were applied to the estimates for Colina, Balboa, Botija Abajo and Brazo.

KNA was completed in CAE Studio and using Supervisor's KNA analysis tools. These analyses used the variogram parameters determined for copper within the high, medium and low grade domains. A series of estimates were run with varying block sizes and the kriging efficiency (KE) and slope of regression (RS) values were calculated. Once the block size was selected a second series of estimates were run using a range of sample numbers and discretisation parameters.

Block configurations varying between 50 m and 100 m in the northing axis (Y) and easting axis (X) for bench heights of 10 m, 15 m and 20 m were tested. The results (Figure 14-15) indicate that the kriging efficiency and regression slope results are not sensitive to the block size, with overall small decreases in kriging efficiency and regression slope with increasing block size. A block size of 50 mE by 50 mN (approximately half the drillhole spacing at Botija, Balboa, Colina and Valle Grande) was selected to provide local definition of the grade variability, and a block height of 15 m was selected to accommodate the expected scale of mining.

The influence of the number of samples informing a block estimate was similarly tested. Block size was set to 50 mE by 50 mN by 15 mRL, and the sample numbers were varied between 6 and 60. Based on the results of this analysis (Figure 14-16), the minimum and maximum numbers of samples were selected to be 24 and 44.

The influence of using different search ellipsoids was investigated for the selected block size and sample numbers. A search with the same dimensions as the variogram ranges and dimensions that were a half, two times and three times the variogram ranges were investigated. The KNA results indicated no sensitivity to the search ellipse dimensions and so the search ellipse was set to the variogram ranges.

The influence of the discretisation parameters on the block estimate was also tested. For this analysis, the block size was set to 50 mE by 50 mN by 15 mRL, the sample numbers were set to a minimum of 24 and a maximum of 44 and the search ellipse dimensions were set to the variogram ranges. The discretisation was varied between 2 and 5 for each of X and Y and between 2 and 4 for Z. Based on the results of this analysis the discretisation was set to 5 X by 5 Y by 4 Z for grade estimation.

14.8 Block model

A 3D prototype block model was developed that covers the entire Cobre Panamá project area. Details of this model are included in Table 14-6. Individual block models were generated for each deposit. Parent block dimensions were 50 mE by 50 mN by 15 mRL; this dimension was supported by the KNA study which demonstrated little change in the kriging efficiency or slope of regression (a measure of bias) from this block size to larger block sizes.

For Botija, the parent block dimension was set to 60 mE by 60 mN by 30 mRL. Within the closer spaced RC drilled volumes, a smaller parent block size of 7.5 mE by 7.5 mN by 7.5 mRL was used with grade estimated directly into these SMU scaled blocks. As such, within the diamond drilled volumes, block estimates were controlled by the parent block dimension and then post processed using localised uniform conditioning into the smaller (7.5 mE by 7.5 mN by 7.5 mRL) SMU blocks.

For the other deposits, parent blocks were sub-celled to the selective mining unit (SMU) of 10 mE by 25 mN by 15 mRL at the topographical surface and for post-processing of the grade estimate (as discussed below).

Table 14-6 Block model dimensions

Balboa, Colina, Valle Grande, Medio, Botija Abajo and Brazo	Block model extents		Number of parent blocks	Parent block size (m)	Number of SMUs	SMU block size (m)
	Minimum	Maximum				
Easting	530,750	542,750	240	50	1,200	10
Northing	973,700	979,200	110	50	220	25
Elevation	-795	470	85	15	85	15
Botija	Block model extents		Number of parent blocks	Parent block size (m)	Number of SMUs	SMU block size (m)
	Minimum	Maximum				
Easting	530,735	542,735	200	60	1,600	7.5
Northing	973,690	979,210	92	60	736	7.5
Elevation	-795	495	43	30	172	7.5

14.9 Grade estimation

14.9.1 Ordinary Kriging interpolation

Grades for copper, gold, molybdenum and silver were estimated using Ordinary Kriging (OK) into parent blocks within each of the mineralised domains. OK was deemed to be appropriate interpolation technique owing to the near normal (Gaussian) data distributions and the limited degree of domain grade population mixing. Estimation parameters for kriging were based on variography, geological continuity and the average spatial distribution of data.

Estimation into parent blocks used a discretisation of 5 (X points) by 5 (Y points) by 3 or 4 (Z points) to better represent estimated block volumes. Each domain was estimated separately and hard boundaries were applied between domains in most cases.

Figure 14-15 Kriging Neighbourhood Analysis to optimise block size

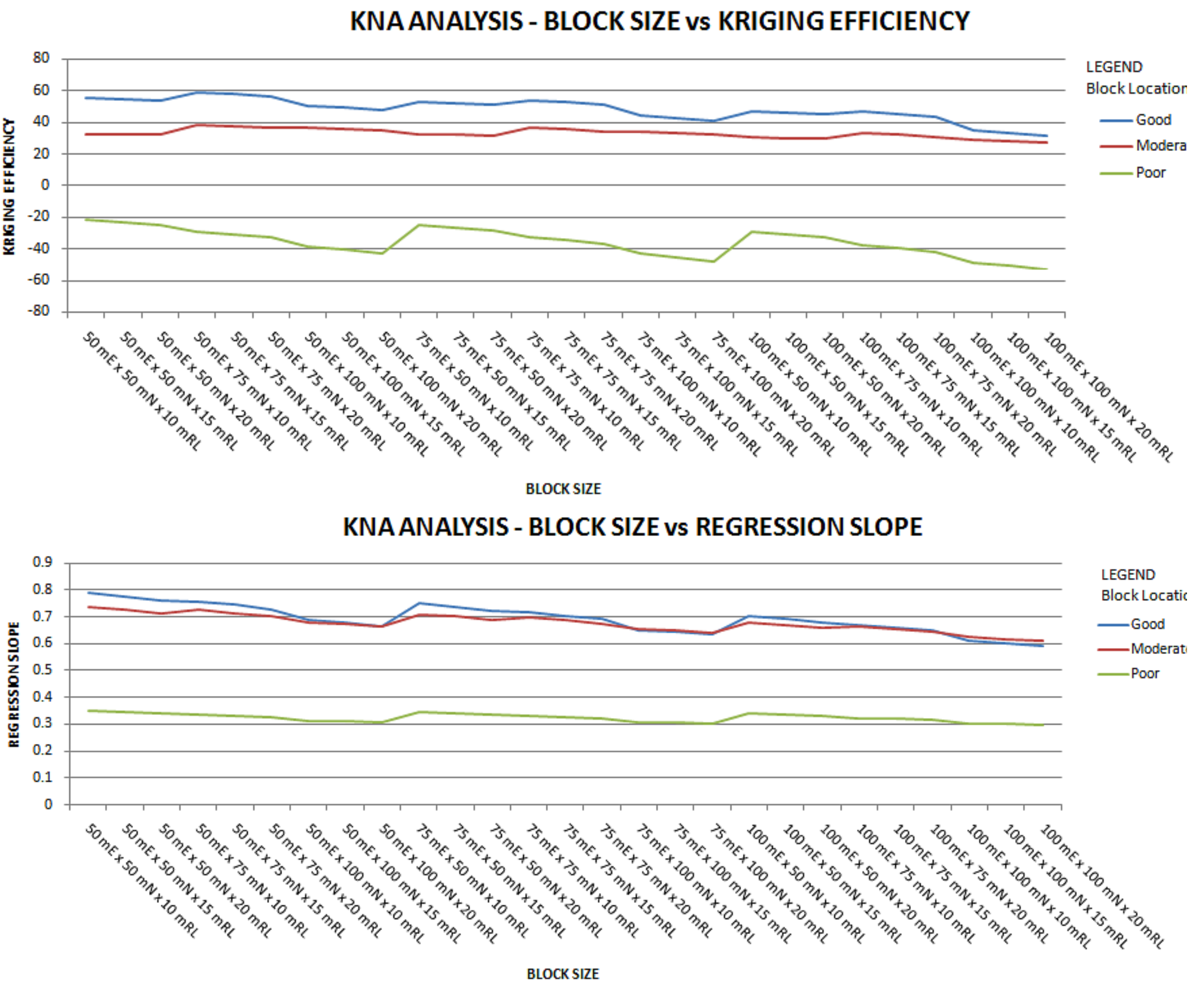
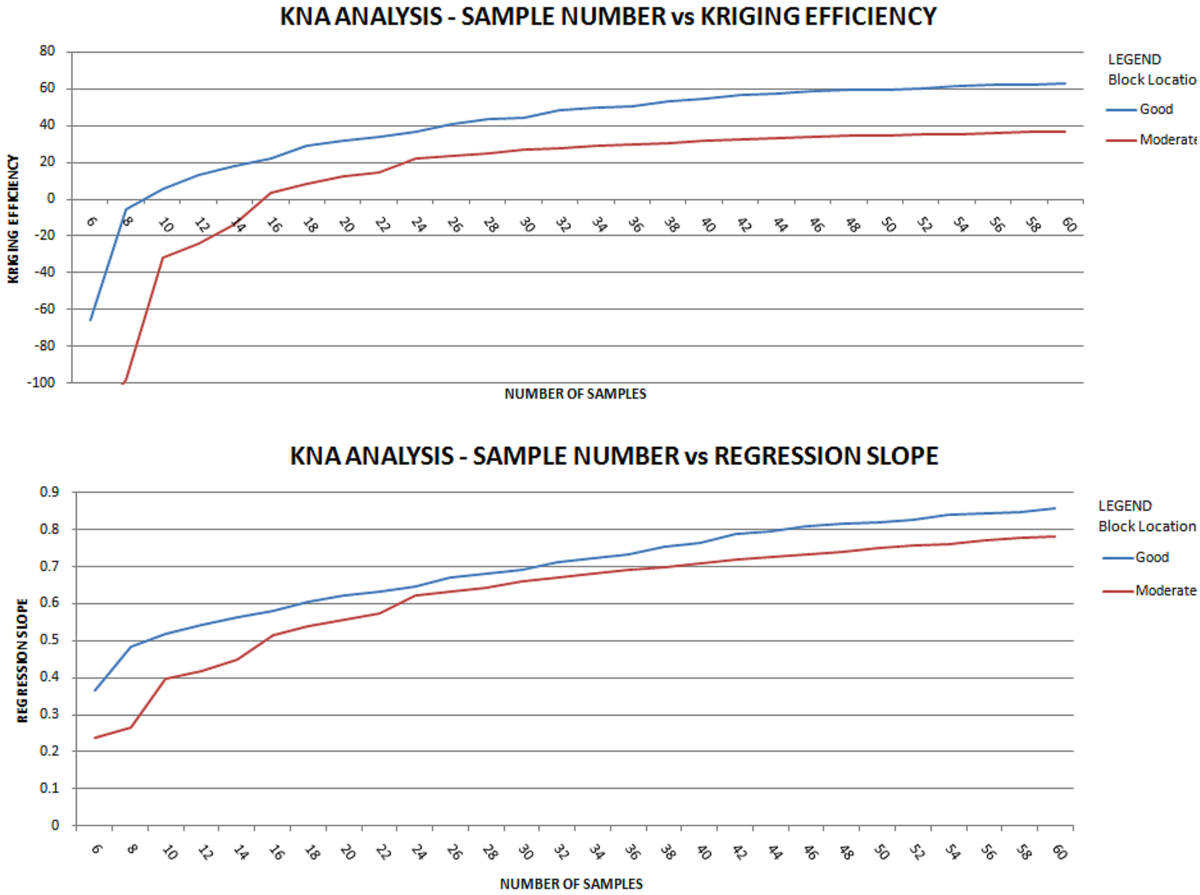


Figure 14-16 Kriging Neighbourhood Analysis to optimise sample numbers



Except for Balboa, the first pass search radii for mineralised domains were set to the respective metal variogram ranges and a minimum of 24 samples was required for a single block estimate and a maximum of 44 samples in order to limit grade smoothing. The dimensions of the second search were set to twice the first, and the minimum number of samples was reduced to 16. The dimensions of the third search were set to twice the second and the minimum number of samples was reduced to 6 for Balboa, Colina/Medio and Valle Grande and 12 for Botija, Botija Abajo and Brazo. Most ore blocks (greater than 89%, 83%, 91% and 92% at Botija, Colina, Valle Grande and Medio respectively) were estimated within the first search radius.

At Balboa, the first pass search radii for low and medium grade mineralised domains were set to the gold variogram ranges, as the gold variogram ranges were significantly longer than the copper variogram ranges. For the supergene gold domain the first pass search radii were set to 150 m by 150 m by 90 m. For each mineralised domain, a minimum of eight samples were required for a single block estimate and a maximum of 48 samples in order to limit grade smoothing. In addition a maximum of six samples per drillhole were used for block grade estimation. The dimensions of the second search were set to twice the first, and the dimensions of the third search were set to twice the second search radii. Most blocks (98%) were estimated within the first search radius.

14.10 Model validation

Validation of the OK block grades included the following processes:

- visual comparisons of drillholes and estimated block grades
- checks to ensure that only blocks significantly distal to the drillholes remained without grade estimates
- statistical comparison of mean composite grades and block model grades
- examination of trend plots of the input data and estimated block grades

Visual validation of the block model was carried out by examining cross-section, long-section and plan views of the drillhole data and the estimated block grades. Examples of the cross-sections are included in Figure 14-18 through to Figure 14-20**.** These indicate good correlation of the estimated block grades with the input drillhole data.

The block estimates were statistically validated against the informing composites. The mean estimated copper, gold, molybdenum and silver grades were compared to the top-cut and declustered input data means. Validation statistics are included in Table 14-7. For this data the drillhole composites were declustered and top-cut for comparison with the block model. Cell declustering was applied using a 50 x 50 x 1.5 m declustering grid.

Grade trend profiles were constructed in order to assess any global bias, average grade conformance and to detect any obvious estimation issues. The copper, gold, molybdenum and silver trend plots were examined in the easting, northing and elevation directions. The validation plots indicate that there is good correlation between the input grades and the block grades.

Based upon the summary statistics, visual validations and trend plots, the OK estimates, per deposit, are consistent with the drillhole composites, and are believed to constitute a reasonable representation of the respective domains mineralisation.

Figure 14-17 North-South cross section of Botija along 538,140 mE demonstrating the correlation between the drillhole data and the ordinary kriged (OK) model

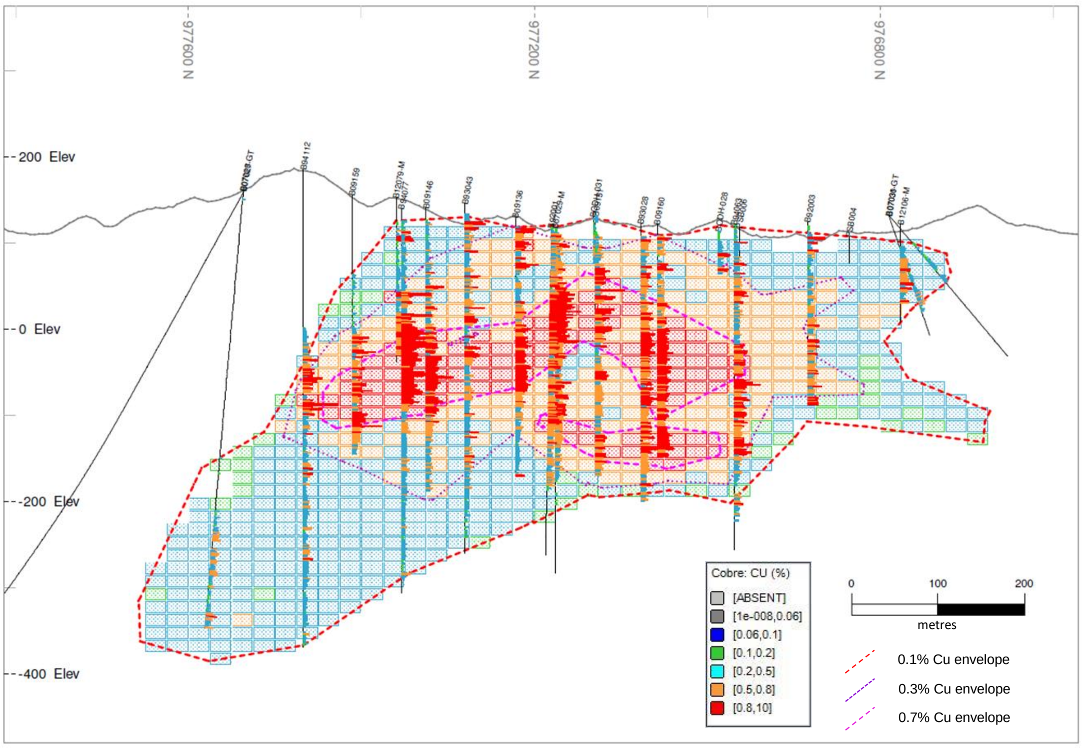


Figure 14-18 South-North cross section of Colina along 533,800 mE demonstrating the correlation between the drillhole data and the ordinary kriged (OK) model

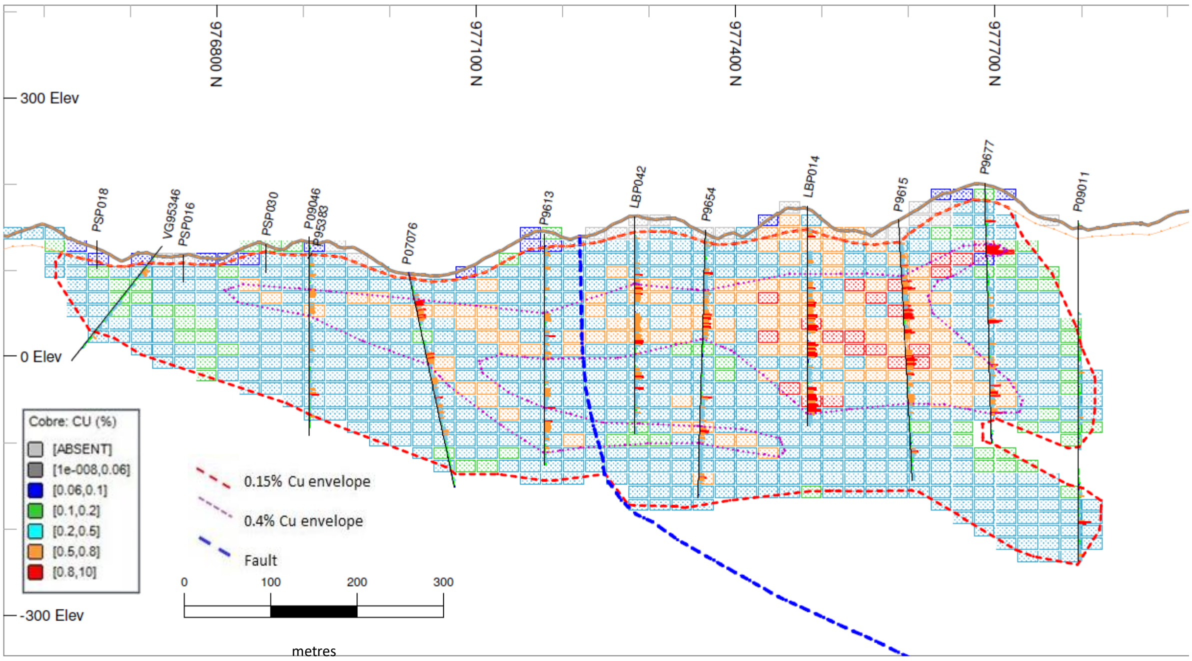


Figure 14-19 Oblique cross section looking northwest of Valle Grande demonstrating the correlation between the drillhole data and the ordinary kriged (OK) model

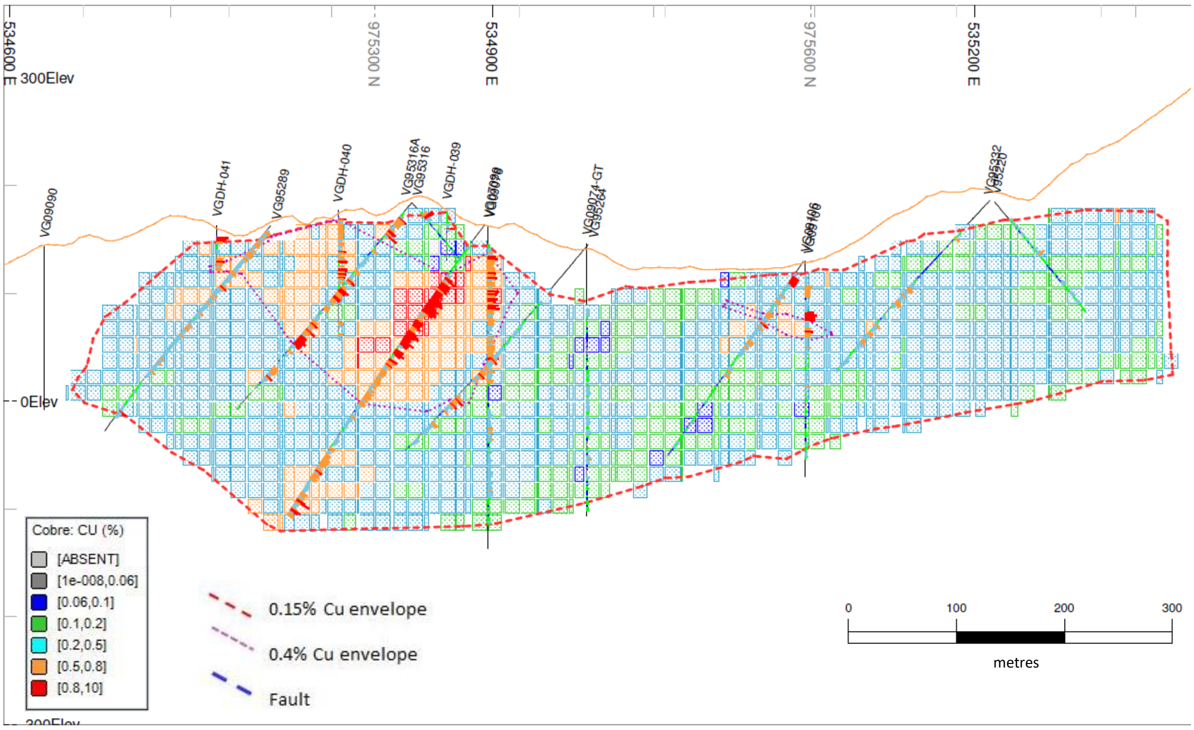
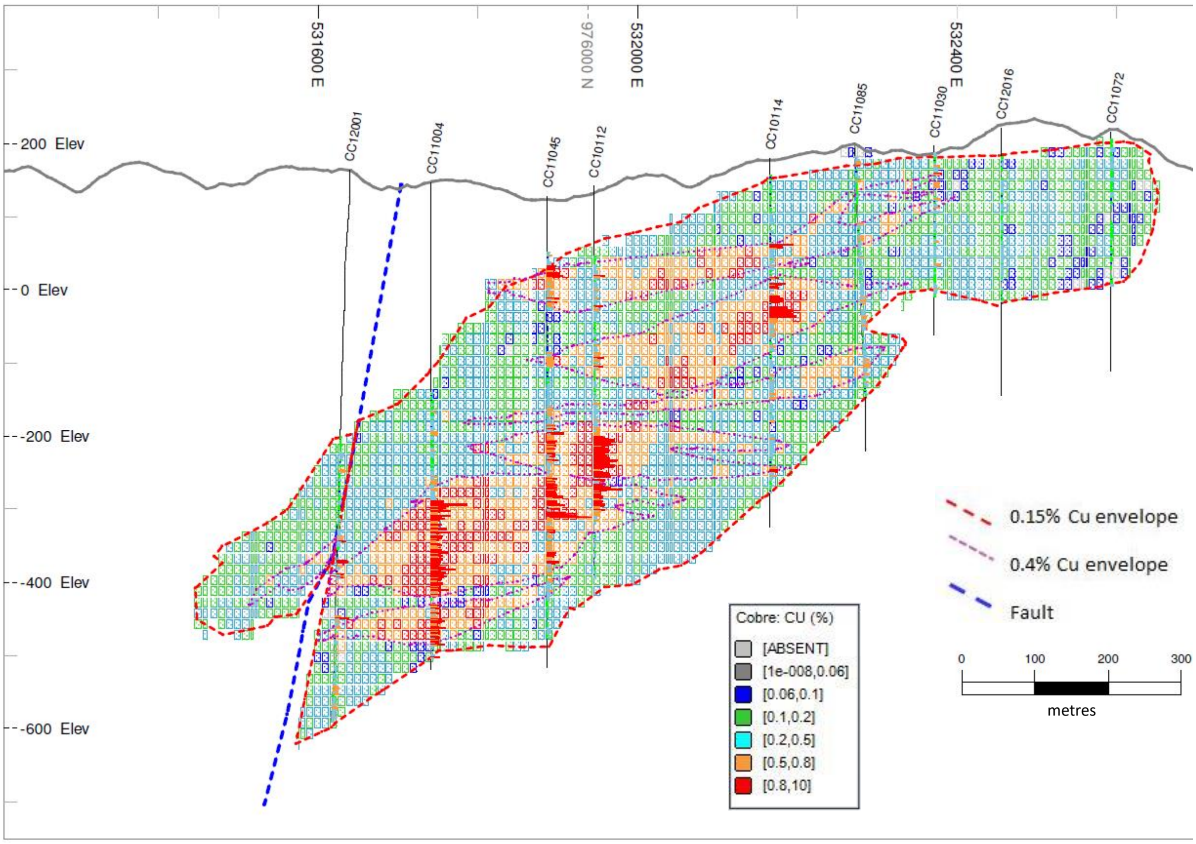


Figure 14-20 Oblique cross section looking northeast through the Balboa deposit demonstrating the correlation between the drillhole data and the ordinary kriged (OK) model



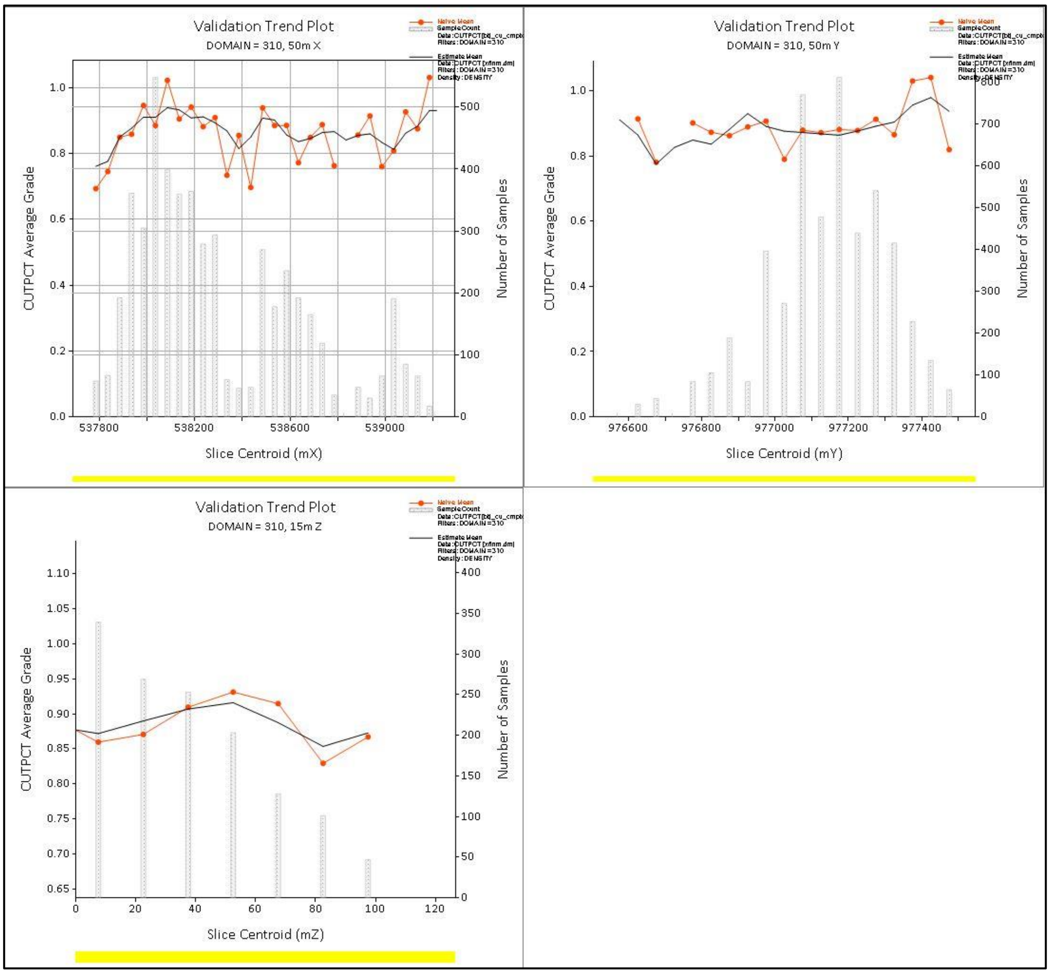
Grade trend profiles were constructed in order to assess any global bias, average grade conformance and to detect any obvious estimation issues. The copper, gold, molybdenum and silver trend plots were examined in the easting, northing and elevation directions. The validation plots indicate that there is good correlation between the input grades and the block grades. An example of trend validation plots for copper in Domain 301 at Botija is presented in Figure 14-21.

Based upon the summary statistics, visual validations and trend plots, the OK estimates are consistent with the drillhole composites, and are believed to constitute a reasonable representation of the respective domains mineralisation.

Table 14-7 Validation statistics for the main domains of each deposit

Botija												
	Copper			Gold			Molybdenum			Silver		
Domain	**310**	**320**	**330**	**310**	**320**	**330**	**310**	**320**	**330**	**310**	**320**	**330**
Composites	0.89	0.56	0.26	0.21	0.11	0.05	133	109	47	2.25	1.56	0.92
Declustered DH data	0.89	0.56	0.26	0.21	0.11	0.05	134	106	47	2.26	1.56	0.92
OK estimate	0.88	0.56	0.26	0.22	0.11	0.05	112	104	55	2.31	1.5	0.91
% Difference	0.62	-0.21	0.3	6.17	-0.59	-1.85	-16.1	-1.9	18.7	2.26	-3.51	-1.27
Balboa												
	Copper			Gold			Molybdenum			Silver		
Domain	**31**	**32**		**31**	**32**		**21 & 22**	**31 & 32**		**31**	**32**	
Composites	0.2	0.64		0.04	0.173		20	20		0.959	2.38	
Declustered DH data	0.2	0.62		0.039	0.155		19	20		0.948	2.325	
OK estimate	0.21	0.62		0.042	0.159		21	22		0.951	2.351	
% Difference	5.6	0.8		7.7	2.6		6.9	13		0.3	1.1	
Colina												
	Copper			Gold			Molybdenum			Silver		
Domain	**21**	**31**		**21**	**40**		**21**	**31**		**21**	**40**	
Composites	0.3	0.62		0.057	0.387		59	97		1.65	2.51	
Declustered DH data	0.3	0.61		0.057	0.394		57	95		1.61	2.44	
OK estimate	0.3	0.59		0.056	0.371		58	94		1.58	2.39	
% Difference	-1.1	-3.5		-2.2	-5.9		1.4	-1.6		-1.3	-2.2	
Valle Grande												
	Copper			Gold			Molybdenum			Silver		
Domain	**1 & 2**	**3**		**1 & 2**	**3**		**1 & 2**	**3**		**1 & 2**	**3**	
Composites	0.29	0.68		0.039	0.084		61	96		1.22	2.19	
Declustered DH data	0.29	0.68		0.039	0.087		60	93		1.21	2.19	
OK estimate	0.28	0.65		0.035	0.079		57	89		1.18	2.15	
% Difference	-3.5	-4		-10.6	-10.8		-6.3	-4.5		-3	-1.6	
Medio												
	Copper			Gold			Molybdenum			Silver		
Domain	**1**	**2**		**1**	**2**		**1**	**2**		**1**	**2**	
Composites	0.21	0.22		0.03	0.031		33	39		1.2	0.89	
Declustered DH data	0.21	0.22		0.03	0.031		33	39		1.21	0.89	
OK estimate	0.21	0.21		0.03	0.036		33	39		1.23	1.04	
% Difference	0.1	-4.8		-0.2	16.1		0.1	-0.1		1.9	16.1	
Botija Abajo												
	Copper			Gold			Molybdenum			Silver		
Domain	**322**	**324**	**332**	**322**		**332**	**322**	**324**	**332**	**322**	**324**	**332**
Composites	0.57	0.72	0.26	0.09		0.04	52.8	24.3	43.8	0.73	2.84	0.33
Declustered DH data	0.57	0.65	0.26	0.09		0.04	55.1	21.2	44	0.74	2.63	0.34
OK estimate	0.58	0.7	0.25	0.08		0.03	48.6	16	54.8	0.79	2.8	0.35
% Difference	1.05	7.54	-2.69	-7.16		-17.44	-11.81	-24.42	24.55	5.93	6.46	4.13
Botija Abajo (continued)												
	Copper			Gold			Molybdenum			Silver		
Domain	**333**	**334**		**333**	**334**	**335**	**333**	**334**		**333**	**334**	
Composites	0.29	0.19		0.09	0.3	0.75	70.6	25.4		1.38	1.26	
Declustered DH data	0.29	0.18		0.09	0.26	0.58	72.8	26.5		1.42	1.23	
OK estimate	0.31	0.19		0.07	0.26	0.28	64.2	24.4		1.53	1.29	
% Difference	5.17	4.44		-23.01	0	-52.24	-11.85	-8.03		7.75	4.72	
Brazo												
	Copper			Gold			Molybdenum			Silver		
Domain	**321**	**331**			**331**		**321**	**331**		**321**	**331**	
Composites	0.56	0.21			0.18		45.7	38.9		0.91	0.67	
Declustered DH data	0.56	0.2			0.18		45.7	38.6		0.91	0.68	
OK estimate	0.58	0.2			0.18		43.3	32.3		0.98	0.7	
% Difference	3.2	0			2.23		-5.23	-16.31		8.13	2.94	

Figure 14-21 Example of validation trend plot for copper in Botija Domain 310



14.10.1 Post-processing by Localized Uniform Conditioning

A localised uniform conditioning estimate (LUC) was generated for copper and gold at the Botija, Colina, Valle Grande, Balboa and Medio deposits using Optiro's proprietary LUC software. LUC was not completed for Botija Abajo and Brazo due to wider-spaced drillhole data and relatively poor variography.

Uniform conditioning (UC) is a geostatistical technique which is used to assess recoverable resources inside a panel by using the estimated panel grade. UC provides an estimate of the proportion of SMUs inside the panel that are above a cut-off grade and their corresponding average grade; however, it does not provide information regarding the spatial distribution of SMU grades within the panel. LUC is a post-processing technique that spatially locates the UC estimates of individual SMUs and thus generates more detail from a panel scale UC estimate to assist with open pit optimisation and Mineral Reserve estimation.

Within the LUC process, ordinary kriging is used to estimate grades into the individual SMU blocks to determine a grade ranking for each of the SMU locations within each panel. Once the SMUs are ranked for each panel, the UC derived metal and tonnage curves are divided into equal proportions based on the number of SMUs in the panel. The grades of these equal proportions are then calculated and assigned to the SMUs in ranked order. The product of the LUC process is a model comprising SMU size blocks with grades assigned within each panel based on the local grade trends revealed by the drilling data. The grades of the SMU size blocks within a panel have a variance that is compatible with the SMU support scale and collectively, the metal contained by SMUs within each panel is identical to the original metal content of the panel. LUC provides grades at SMU scale block sizes, and as such provides a good representation of grade variability that may be encountered per panel.

For Balboa, Colina, Valle Grande, Medio, Botija Abajo and Brazo, the OK estimates of the panels (50 mE by 50 mN by 15 mRL) and LUC cell size, as well as composited drillhole data files and variogram models were used as input to LUC. The LUC estimate was based on grade variability at a scale of 10 mE by 12.5 mN by 7.5 mRL to provide a discretisation of 5X by 2Y by 2Z; these estimates were re-blocked to the SMU block size of 10 mE by 25mN by 15 mRL. It is important to note that the local accuracy of these LUC SMU block grades is poor and should not be relied upon spatially. Results were exported from the LUC process and merged with the OK panel grade estimates.

Similarly, for Botija, the OK estimates of the panels (60 mE by 60 mN by 30 mRL) and LUC cell size, as well as composited drillhole data files and variogram models were used as input to LUC. The LUC estimate was based on grade variability at a scale of 7.5 mE by 7.5 mN by 7.5 mRL to provide a discretisation of 5X by 5Y by 2Z; these estimates were re-blocked to the SMU block size of 7.5 mE by 7.5 mN by 15 mRL. It is important to note that the local accuracy of these LUC SMU block grades is poor and should not be relied upon spatially. Results were exported from the LUC process and merged with the OK panel grade estimates.

The LUC models were validated by:

- visual comparisons of drillholes, OK panel grades and SMU block grades
- checking that the average SMU grades within each parent block matched the parent block grade
- confirming that metal correlations in the SMU blocks were comparable with the correlations observed in the drillhole data
- checking that the contained copper and gold metal at a zero cut-off grade is the same for both the OK panel models and the LUC models.

14.11 Background model

For each deposit model area a background model was generated to capture all mineralisation that was not included in the interpretation of the mineralised domains. A categorical indicator model was developed, using the drillhole intersections external to the mineralised domains, to identify blocks that have a probability of ≥0.6 of containing mineralisation of ≥0.03% copper. Grades were estimated into these low grade mineralised blocks using sample intervals of ≥0.03% copper that were not included in the mineralised domain interpretations. Blocks with a probability of <0.6 of containing mineralisation of ≥0.03% copper were identified as waste blocks and sample intervals with <0.03% copper were used to estimate waste block grades.

14.12 Density estimates in the block model

Density was estimated into the panel scale blocks using ordinary kriging at Botija. For the other deposits, bulk density sample data was not available for most of the MPSA drillhole samples and as a result did not provide good spatial coverage for each deposit. Average density values were therefore determined based upon the rock type and were assigned to the resource model. The density values are listed in Table 14-8. A total of 3,708 sample density values were available for use in the Cobre deposit estimates. Due to the lack of saprolite density measurements, a conservative value of 1.5 t/m^3 was used.

Table 14-8 Assigned average density values per material type

Lithology domain	Description	Density (t\m^3)
100	Granodiorite	2.60
200	Porphyry	2.64
300	Andesite	2.72
500	Saprock	2.54
600	Saprolite	1.50
999	Unknown – external to extents of drilling	2.65

14.13 Mineral Resource classification and reporting

The Mineral Resource estimates for the Botija, Colina, Balboa, Valle Grande, Medio, Botija Abajo and Brazo deposits have been classified and reported in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014), and also comply with the guidelines of the JORC Code (JORC, 2012). Table 1 criteria of the JORC Code and supporting comments are listed in Appendix A.

Classification of the Mineral Resources was primarily based on confidence in drillhole data, geological continuity, and the quality of the resulting kriged estimates. Geological confidence is supported by diamond drill core and logging data and a good understanding of the local and regional geology. Confidence in the kriged estimate is associated with drillhole coverage, analytical data integrity, kriging efficiency and regression slope values.

Wireframe models were used to define areas with high confidence at Botija: Mineral Resources within these areas were classified as Measured (Figure 14-22). Areas with moderate confidence were defined at Botija, Colina, Balboa, Valle Grande, Medio, Botija Abajo and Brazo: these areas were classified as Indicated. Mineral resources outside these areas were classified as Inferred.

Volume adjustment factors, as listed in Table 14-2 for dilution from dyke intrusives, were applied to the metal content and the reported Mineral Resource estimate has accounted for this barren dyke material. Mineral Resources have been reported at a 0.11% Cu cut-off grade in

Table 14-9. This cut-off grade is considered to be appropriate based on economic input mining parameters and assumptions derived from deposits of similar type, scale, and location.

Figure 14-22 Mineral Resource classification at Botija, highlighting measured resources within closely drilled areas

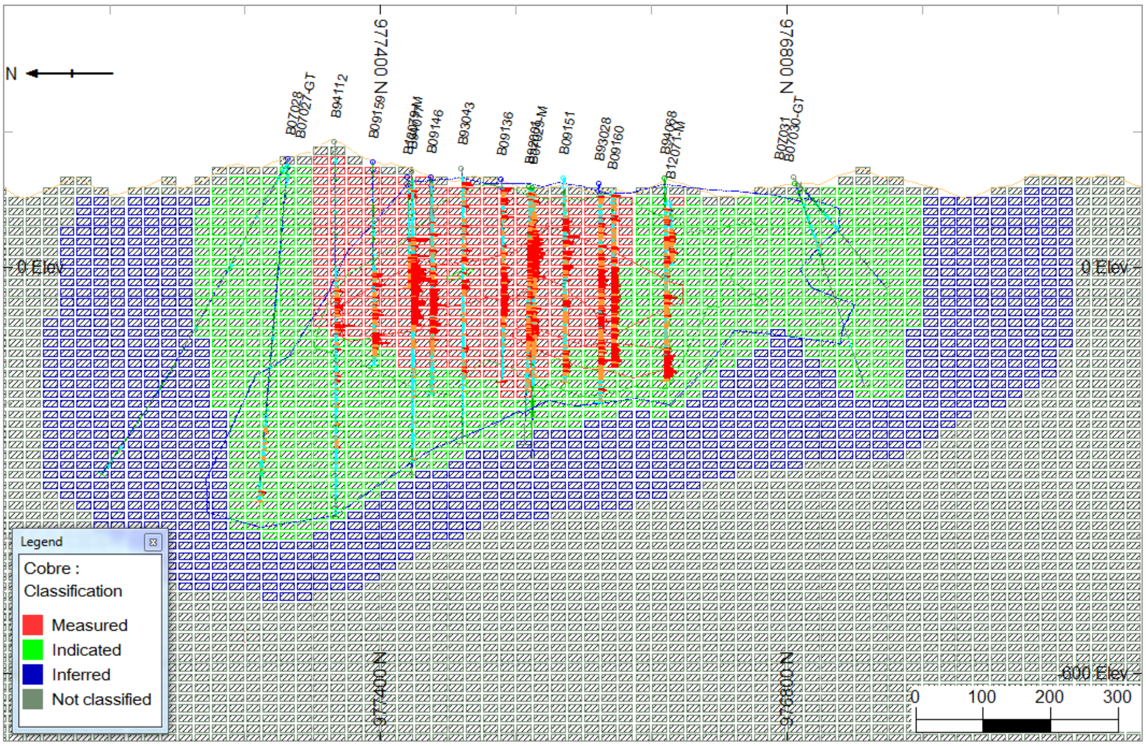


Table 14-9 Cobre Panamá Mineral Resource statement, at 31st December 2018 above a 0.15% Copper Cut-off Grade. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources are inclusive of Mineral Reserves

Deposit	Category	Tonnes (Mt)	TCu (%)	Mo (%)	Au (g/t)	Ag (g/t)
Botija	Measured	310	0.47	0.008	0.11	1.44
Botija	Indicated	660	0.36	0.007	0.07	1.13
Colina	Indicated	1,032	0.39	0.007	0.06	1.58
Medio	Indicated	63	0.28	0.004	0.03	0.96
Valle Grande	Indicated	602	0.36	0.006	0.04	1.37
Balboa	Indicated	647	0.35	0.002	0.08	1.37
Botija Abajo	Indicated	114	0.31	0.004	0.06	0.93
Brazo	Indicated	228	0.36	0.004	0.05	0.81
Total Meas. plus Ind.		**3,657**	**0.37**	**0.006**	**0.07**	**1.34**
Botija	Inferred	198	0.23	0.004	0.05	0.87
Colina	Inferred	125	0.26	0.006	0.05	1.20
Medio	Inferred	189	0.25	0.005	0.03	1.25
Valle Grande	Inferred	363	0.29	0.005	0.03	1.14
Balboa	Inferred	79	0.23	0.003	0.04	0.96
Botija Abajo	Inferred	67	0.27	0.005	0.06	1.25
Brazo	Inferred	76	0.21	0.003	0.01	0.73
Total Inferred		**1,097**	**0.26**	**0.005**	**0.04**	**1.09**

To ensure that the reported resource exhibits reasonable prospects for eventual economic extraction, the Mineral Resource statement (Table 14-9) was guided by the determination of a copper cut-off grade that was derived from a pit shell generated from all metal grades from blocks classified in the Measured and Indicated categories. This pit shell assumed a metal price of $3.00/lb copper, $13.50/lb molybdenum, $1,200/troy ounce gold and $16.00/troy ounce silver and total operating costs of $9.30/t.

There are no known factors related to environmental, permitting, legal, title, taxation, socioeconomic, marketing, or political issues that are believed to materially affect the Mineral Resource.

14.14 Mineral Resource estimate comparisons

Apart from Botija, the respective deposit estimates of Mineral Resources for Cobre Panamá have not changed since the previous June 2015 estimates. The updated Botija estimate took into consideration the following changes since the previous (June 2015) estimate:

1. drillhole additions from infill metallurgical drilling at Botija (~8 holes),
2. addition of 2,911 reverse circulation grade control holes
3. continued geological mapping and 3D geology modelling
4. estimation into a smaller block size as per the selective mining unit (SMU)

The Botija deposit changes, as compared to the June 2015 estimates generally relate to reductions in ore tonnages associated with the pre-stripping. The marginal increase in copper grade is due to the reduced SMU block size from 15mN by 15mE by 15mRL to 7.5mN by 7.5mE by 15m RL. The Inferred Mineral Resource tonnages for 2018 has increased by 23% when compared to June 2015. The increase is due to an improved understanding of geological and grade continuity as associated with the 2,911 RC drilled holes.

A detailed comparison of the 2018 and 2015 estimates is provided in Table 14-10.

Table 14-10 Comparison of Mineral Resource estimate results as at December 2018 and the previous results as at June 2015; copper cut-off grade was 0.15%

Deposit	Category	Tonnes (Mt)			TCu (%)			Mo (%)			Au (g/t)			Ag (g/t)		
		Dec-18	Jun-15	%Var	Dec-18	Jun-15	%Var	Dec-18	Jun-15	%Var	Dec-18	Jun-15	%Var	Dec-18	Jun-15	%Var
Botija	**Measured**	310	336	-8%	0.47	0.46	3%	0.008	0.008	0%	0.11	0.10	9%	1.44	1.35	6%
Botija	**Indicated**	660	672	-2%	0.36	0.35	2%	0.007	0.007	0%	0.07	0.06	16%	1.13	1.08	5%
Total Meas. plus Ind.		**970**	**1,008**	**-4%**	**0.39**	**0.39**	**2%**	**0.007**	**0.007**	**0%**	**0.08**	**0.07**	**12%**	**1.23**	**1.17**	5%
Botija	**Inferred**	198	152	23%	0.23	0.23	0%	0.004	0.004	0%	0.05	0.05	0%	0.87	0.78	10%

ITEM 15 MINERAL RESERVE ESTIMATE

15.1 Introduction

Detailed technical information provided under this item relates specifically to the Mineral Reserve estimate completed to date and based on the Mineral Resource models and estimates as reported in Item 14.

As part of the estimation process, the original pit optimisation work completed for the 2015 Technical Report (FQM, July 2015) was reviewed. Along with more recent detailed pit designs completed by FQM personnel, this review and design work was overseen and supervised by Michael Lawlor (QP) of FQM.

To conform with NI 43-101 standards, the Mineral Reserve estimate is derived from Measured and Indicated Resources only. The Measured and Indicated Mineral Resource estimates as listed in Table 14-9 are reported inclusive of the Mineral Reserve.

15.2 Methodology

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate followed a conventional approach, commencing with open pit optimisation techniques incorporating economic parameters and other “modifying” factors.

The ultimate (optimal) pit outlines (shells) were used to create practical and detailed open pit designs accounting for the siting of in-pit crushers, batters, berms and haul roads.

These pit designs then provided the bench by bench ore and waste mining inventories for the detailed production schedule that demonstrates viable open pit mining. This schedule, which in turn provides the physical basis for cash flow modelling, is described in Item 16.

15.3 Mine planning models

A mine planning model was produced from each of the Datamine Mineral Resource models described in Item 14. For the purposes of the optimisation software, each mine planning model (except for Botija) was regularised to 10 m x 25 m x 15 m (x, y z); the Botija planning model was regularised to 20 m x 50 m x 15 m (x, y z).

For subsequent production scheduling (Item 16), the planning model was regularised to a smallest mining unit (SMU) block size, suitable for the scale of proposed ultra class primary mining equipment, ie to dimensions of 100 m x 100 m x 15 m (x, y,z).

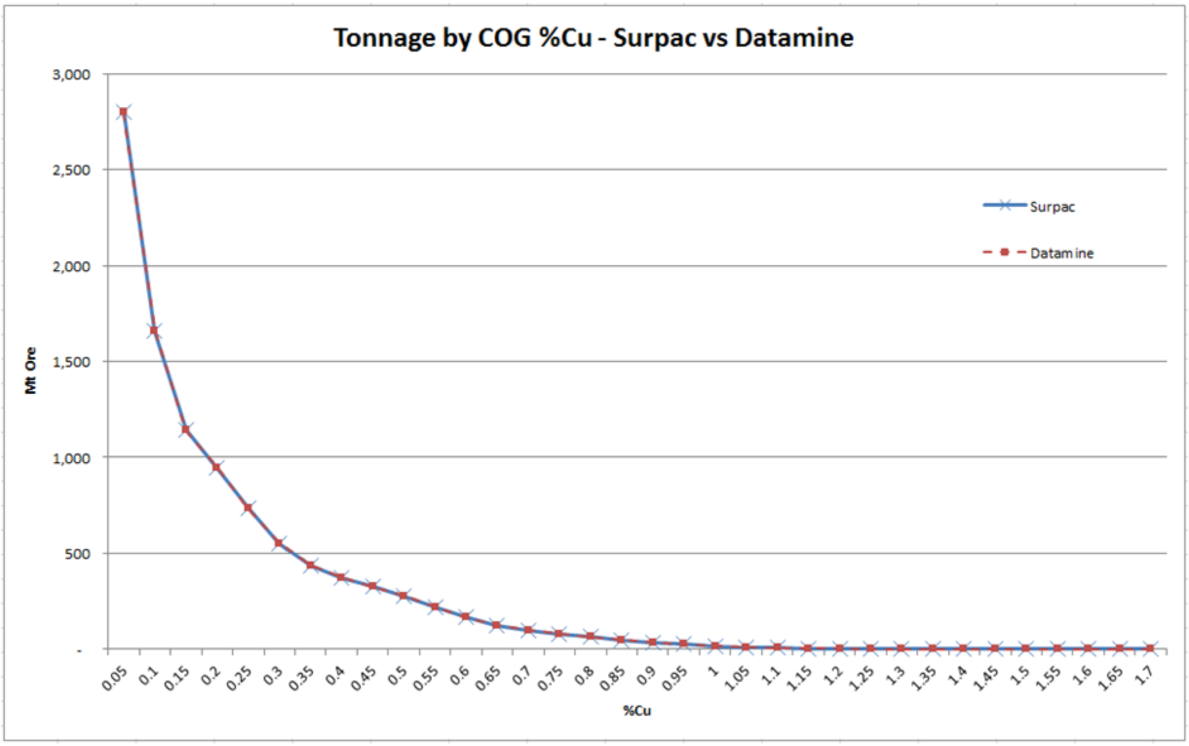
Each of the Surpac mine planning models was validated against its corresponding regularised Datamine model, before proceeding. An example of one such validation, for Botija, is shown in Figure 15-1. As with this particular validation plot, all other model comparisons were essentially perfect.

15.4 Pit Optimisation

The original pit optimisation work completed for the 2015 Technical Report (FQM, July 2015) was not updated for this Technical Report. Optimisation sensitivity analyses that were carried out in 2015 indicated that processing recovery and copper metal price were the most sensitive optimisation variables. There has been no material change to these particular variables since that time. Whilst further geotechnical work has been completed since 2015, specifically in relation to the Botija Pit, the revised overall slope angles remain within the selected optimal shell outlines. Mine planning and engineering work is on-going at this time, however, in respect of in-pit crusher siting and the impact that particular siting scenarios will have on overall ultimate pit slope design. An iterative approach is required between ultimate designs and optimisations to arrive at an acceptable final design.

Operating costs have been updated since 2015, as have metal costs (ie, treatment and refining charges (TCRCs, plus royalties). Item 21 provides a description of these updates, whilst a commentary on the impact of these to the original pit shell selection is provided in Item 15.4.5.

Figure 15-1 Botija Pit validation – Mineral Resource to mine planning models



15.4.1 Optimisation methodology

Conventional Whittle Four-X software was used to determine optimal pit shells for each of the various deposits.

All mined ore will be processed in a conventional sulphide flotation plant, with differential processes to produce separate copper and molybdenum concentrates. The copper concentrate will contain gold and silver.

Against this background, optimisations were completed on a maximum net return (NR) basis, and with recoveries to metal in concentrate based on different variable and fixed relationships for each deposit. In general:

Net return = revenue from recovered metals – metal costs
where metal costs = the costs associated with treating and transporting the concentrate plus the royalties payable

A mining block will be considered as 'ore' when the net return is greater than the processing cost.

15.4.2 Optimisation input parameters

Metal prices

The optimisation inputs for long term metal prices were as follows:

- copper = $3.00/lb ($6,615/t)
- molybdenum = $13.50/lb ($29,762/t)
- gold = $1,200/oz
- silver = $16.00/oz

Metal recoveries

The input processing recovery projections were the same as those listed in Item 13 (Table 15-1).

Table 15-1 Cobre Panamá process recovery relationships and values

Deposit	Recovery			
	Cu (%)	Mo (%)	Au (%)	Ag (%)
Botija	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Colina	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Medio	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775)))	55.0%	MIN(80,MAX(0,(15.993*LOG(Auppm))+92.138))	47.3%
Valle Grande	MAX(0,MIN(96,((5.8287*LOG(%Cu))+95.775))-4)	52.0%	MIN(80,MAX(0,(15.993*LOG(auppm))+92.138))	47.3%
Balboa	MIN(96,((2.4142*LOG(cutpct))+92.655))	55.0%	MAX(0,MIN(80,(7.6009*LOG(auppm))+85.198))	40.0%
Botija Abajo	6.6135*Ln(Cu%) + 92.953	55.0%	50.0%	30.0%
Brazo	6.6135*Ln(Cu%) + 92.953	55.0%	50.0%	30.0%

The recovery from stockpiled saprock ore was reduced by 20% to simulate the potential leaching of copper metal from material that may be stockpiled for up to thirty years before processing.

Pit slope design criteria

Pit optimisation input included overall slope design angles as listed in Table 15-2. The geotechnical engineering basis for these design angles is outlined in Item 16.

Table 15-2 Cobre Panamá overall slope angles for pit optimisation

Pit	Location	Sector / Wall Height	Overall Slope Angle (degrees)
Botija	North Wall	1	41.8
		2	42.0
	South Wall River Diversion Diversion Sectors	3	40.3
		4	38.4
		5	40.3
		6	41.0
		7	41.0
	West Wall	8	41.1
		9	41.9
		10	42.0
		11	41.1
Colina	SE, SW	1	39.5
	NW, Pit Bottom	2	42.5
	N, E, S	3	45.6
	SSW	4	45.6
VG	All		39.5
Low RQD	All		-
BABr	W		41.0
	E		29.0
	All Remaining		34.0
Balboa	All		42.0

Operating costs

Since the Project will be mill constrained, the process operating costs were input as the sum of the fixed and variable costs. These costs are the same for each deposit (details are outlined in Item 21):

- operating cost = \$4.20/t
- fixed cost (equivalent G&A cost in variable terms) = \$1.25/t
- total processing cost = \$5.45/t

Variable mining costs comprising drill, blast, load and haul costs, on a bench by bench basis, were derived by MPSA. These costs were estimated from first principles using haul profiles and productivity estimates related to preliminary mine designs, production schedule, proposed equipment fleet and concept ore/waste haulage destinations. There were two cost components:

- a base mining cost, comprising drill, blast, load and haul costs up to a reference bench elevation, and
- an incremental mining cost, comprising haulage beyond the reference bench level, and to respective ore tipping and waste dumping destinations.

For the ore hauls, there were three reference elevations at 120 mRL, 50 mRL, and -60 mRL, catering for notional in-pit crushing elevations. For the waste hauls, there was one reference elevation, at 120 mRL. The base ore mining cost was \$1.77/t and the base waste mining cost was \$1.98/t. There were two incremental mining costs, ie \$0.035/t/bench for material hauled up to the reference elevation and \$0.023/t/bench for material hauled down to the reference bench. In these particular estimates, the haulage costs component took no account of potential future savings from trolley-assisted haulage.

From the above, the algorithm used for the Whittle input of depth incremented mining costs was:

- ore mining (\$/t) = 1.77 + 0.035 x (Bench RL – Reference RL) for up hauls
- ore mining (\$/t) = 1.77 + 0.023 x (Bench RL – Reference RL) for down hauls
- waste mining (\$/t) = 1.98 + 0.035 x (Bench RL – Reference RL) for up hauls
- waste mining (\$/t) = 1.98 + 0.023 x (Bench RL – Reference RL) for down hauls

The resulting weighted average variable mining costs were \$2.20/t for ore and \$1.92/t for waste. Further details of these mining costs are outlined in Item 21. Item 21 also explains the basis for the additional stockpiled ore reclaim cost, which although not included in the optimisation, is considered in the production scheduling process (Item 16).

The operating cost of \$4.20/t included a 4.5% allowance for sustaining capital costs (\$0.18/t). The G&A cost of \$1.25/t included the same sustaining cost allowance rate (\$0.05/t). An additional allowance for sustaining capital for mining equipment was included, equivalent to \$0.26/t mined.

Metal costs

In addition to royalties, metal costs for each of the copper and molybdenum concentrates comprise:

- concentrate transport charges
- concentrate refining charges
- payable rates for each metal recovered into concentrate

Table 15-3 lists the metal costs adopted for the pit optimisation, inclusive of charges for gold and silver in concentrate.

Table 15-3 Cobre Panamá metal costs for pit optimisation

Copper concentrate charges	
Cu con grade, %	25.0%
Cu con overland freight, $/dmt	$0.00
Cu con ocean freight, $/dmt	$40.00
Cu con treatment, $/dmt	$70.00
Cu refining, $/lb payable	$0.07
Cu payable, %	96.43%
TCRCs, $/lb Cu payable	**$0.277**
Molybdenum concentrate charges	
Mo con grade, %	52.0%
Mo con overland freight, $/dmt	$0.00
Mo con ocean freight, $/dmt	$90.00
Mo con treatment, $/dmt	$0.00
Cu removal charge, $/lb	$0.00
Mo refining, $/lb payable	$0.00
Mo payable, %	86.20%
TCRCs, $/lb Mo payable	**$0.091**
Gold in concentrate charges	
Au refining, $/oz payable	$5.50
Au payable, %	92.00%
TCRCs, $/lb Au payable	**$5.060**
Silver in concentrate charges	
Ag refining, $/oz payable	$0.40
Ag payable, %	90.00%
TCRCs, $/lb Ag payable	**$0.360**

The Panamánian government levies a royalty on each metal recovered[5,6]. From the above table, therefore, the all-up metal costs adopted for pit optimisation were:

- the copper metal cost is:
  = metal price x royalty[7] + Cu metal cost
  = $3.00 x 5% + $0.277 = $0.43/lb Cu
- the molybdenum metal cost is:
  = metal price x royalty + Mo metal cost
  = $13.50 x 5% + $0.091 = $0.77/lb Mo
- the gold metal cost is:
  = metal price x royalty + Au refining cost x Au payable
  = $1,200 x 5% + $5.50 x 92% = $65.06/oz Au ($948.77/lb Au)
- the silver metal cost is:
  = metal price x royalty + Ag refining cost x Ag payable
  = $16.00 x 5% + $0.40 x 90% = $1.16/oz Ag ($16.92/lb Ag)

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[5] As governed by Law No. 9, 1997, MPSA is expected to pay a 2% royalty on "Negotiable Gross Production" which is defined as "the gross amount received from the buyer due to the sale (of concentrates) after deduction of all smelting costs, penalties and other deductions, and after deducting all transportation costs and insurance…incurred in their transfer from the mine to the smelter".

[6] A tax of 3 Panamánian Balboas (US$3.00) per hectare per year for the total concession area will also apply, but was not adopted for pit optimisation input

[7] In the optimisations for simplicity, and based on the rates set out in the Mining Code, a 5% royalty rate was adopted for all four metals.

Mining dilution and recovery factors

Geological losses were built into the regularised mine planning models to account for the presence of unmineralised dykes. These losses could be considered as "planned dilution". In the Whittle optimisation inputs, "unplanned dilution" and mining recovery factors were included to emulate practical mining losses. In the absence of operational reconciliation information, the selected factors (Table 15-4) are considered to be reasonable for bulk mining of large orebodies.

Table 15-4 Dilution and recovery factors, Whittle optimisation

MR model	Unadjusted inventory		Dilution	Diluted inventory		Recovery	Recovered inventory	
	Tonnes (Mt)	Grade (%Cu)	Factor (%)	Tonnes (Mt)	Grade (%Cu)	Factor (%)	Tonnes (Mt)	Grade (%Cu)
Botija	1,010	0.37	2	1,030	0.36	98	1,010	0.36
Colina & Medio	1,079	0.37	2	1,101	0.37	98	1,079	0.37
Valle Grande	603	0.35	2	615	0.34	98	603	0.34
Balboa	522	0.31	2	533	0.30	98	522	0.30
BABR	263	0.37	2	268	0.36	98	263	0.36
TOTAL	**3,477**	**0.36**		**3,546**	**0.35**		**3,477**	**0.35**

Net return

To avoid confusion, the NR (ie, net return = recovery * (revenue – metal costs)) must be expressed in units of metal grade. Since the metal grades are in % terms, this is $/10kg. In other words, the $/lb costs must be multiplied by 2,204.62 and divided by 100.

Table 15-5 lists the notional NR values, based on overall average recoveries and model grades.

15.4.3 Marginal cut-off grades

Whittle optimisation software uses the following simplified formula to calculate the marginal cut-off grade as listed in Table 15-6.

Marginal COG = (PROCOST x MINDIL)/(NR)

where PROCOST is the sum of the processing cost plus the ore mining cost differential, and MINDIL is the mining dilution factor

Table 15-5 Cobre Panamá net return values

	Botija, Colina, Medio	Valle Grande	Balboa	BABR
	All Rock Types	All Rock Types	all Rock Types	all Rock Types
Processing Parameters:				
Mill throughput, tpd	202,740	202,740	202,740	202,740
Cu recovery, %	90.8%	86.4%	90.3%	87.3%
Mo recovery, %	55.0%	53.0%	55.0%	55.0%
Au recovery, %	54.2%	46.5%	68.6%	50.0%
Ag recovery, %	47.3%	47.3%	50.0%	30.0%
Average Grades:				
Cu, %	0.37	0.35	0.31	0.37
Mo,ppm	66.93	65.95	15.65	40.48
Au , ppm	0.07	0.05	0.07	0.07
Ag, ppm	1.35	1.37	1.30	0.84
Price less Metal Costs:				
Cu Metal Price, $/lb	3.00	3.00	3.00	3.00
Cu Metal Cost, $/lb	0.43	0.43	0.43	0.43
Cu Net Return, $/lb	2.57	2.57	2.57	2.57
Cu Net Return, $/lb (recovered)	2.34	2.22	2.32	2.25
Mo Metal Price, $/lb	13.50	13.50	13.50	13.50
Mo Metal Cost, $/lb	0.77	0.77	0.77	0.77
Mo Net Return, $/lb	12.73	12.73	12.73	12.73
Mo Net Return, $/lb (recovered)	7.00	6.75	7.00	7.00
CuEq Net Return, $/lb (recovered)	0.08	0.04	0.01	0.02
Au Metal Price, $/oz	1,200.00	1,200.00	1,200.00	1,200.00
Au Metal Cost, $/oz	65.06	65.06	65.06	65.06
Au Net Return, $/oz	1,134.94	1,134.94	1,134.94	1,134.94
Au Net Return, $/lb	16,550.83	16,550.83	16,550.83	16,550.83
Au Net Return, $/lb (recovered)	8,974.87	7,691.04	11,356.76	8,275.42
CuEq Net Return, $/lb (recovered)	0.10	0.03	0.10	0.04
Ag Metal Price, $/oz	16.00	16.00	16.00	16.00
Ag Metal Cost, $/oz	1.16	1.16	1.16	1.16
Ag Net Return, $/oz	14.84	14.84	14.84	14.84
Ag Net Return, $/lb	216.41	216.41	216.41	216.41
Ag Net Return, $/lb (recovered)	102.36	102.36	108.21	64.92
CuEq Net Return, $/lb (recovered)	0.02	0.01	0.01	0.00
Total Net Return, $/lb (recovered)	2.53	2.30	2.45	2.32
Total Net Return, $/10kg (recovered)	**55.83**	**50.67**	**53.95**	**51.06**

Table 15-6 Cobre Panamá marginal cut-off grades

	Botija, Colina, Medio	Valle Grande	Balboa	BABR
	All Rock Types	All Rock Types	all Rock Types	all Rock Types
Marginal Cut-Off Grade:				
PROCOST, $/t ore	5.45	5.45	5.45	5.45
MINDIL	**1.02**	**1.02**	**1.02**	**1.02**
TOTAL NET RETURN, $/10kg	55.83	50.67	53.95	51.06
C/O GRADE, Cu%	**0.10**	**0.11**	**0.10**	**0.11**

15.4.4 Optimisation results

Figures 15-2 to 15-6 show the graphical results of pit optimisation. The optimal pit shells were selected on a maximum net return (undiscounted) basis. Table 15-7 lists the inventories and cashflows from each of the selected optimal shells. These cashflows do not include capital, depreciation or taxes.

Figure 15-2 Botija Pit optimisation results

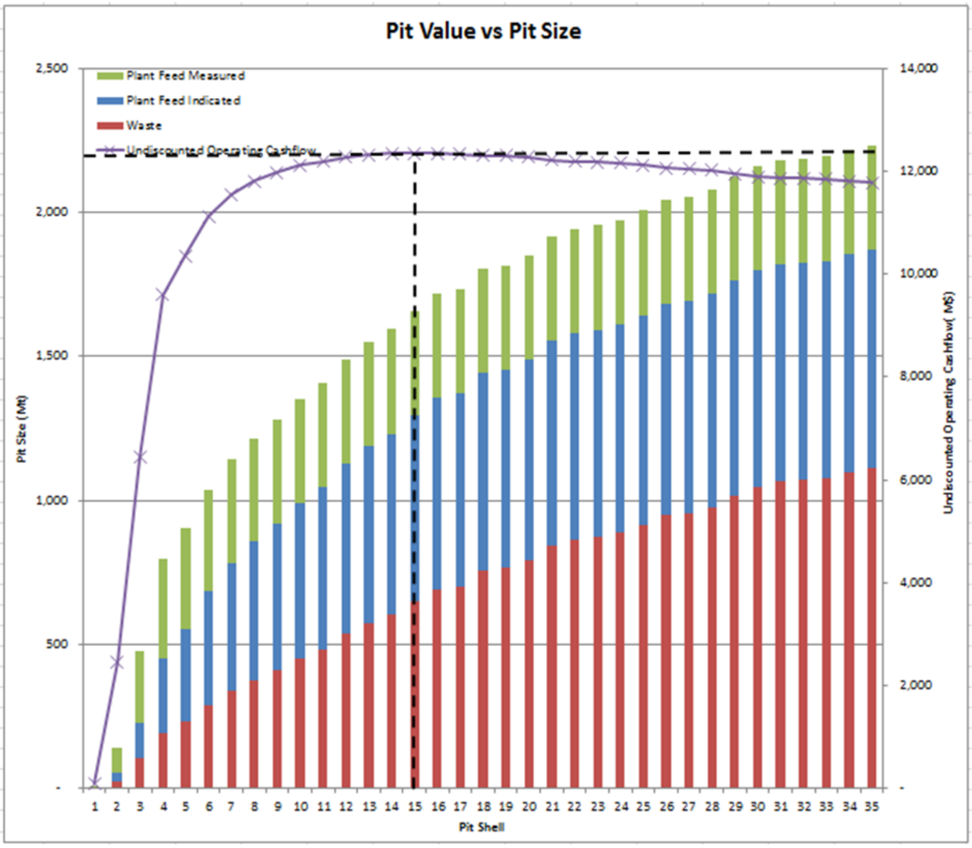


Figure 15-3 Colina/Medio Pit optimisation results

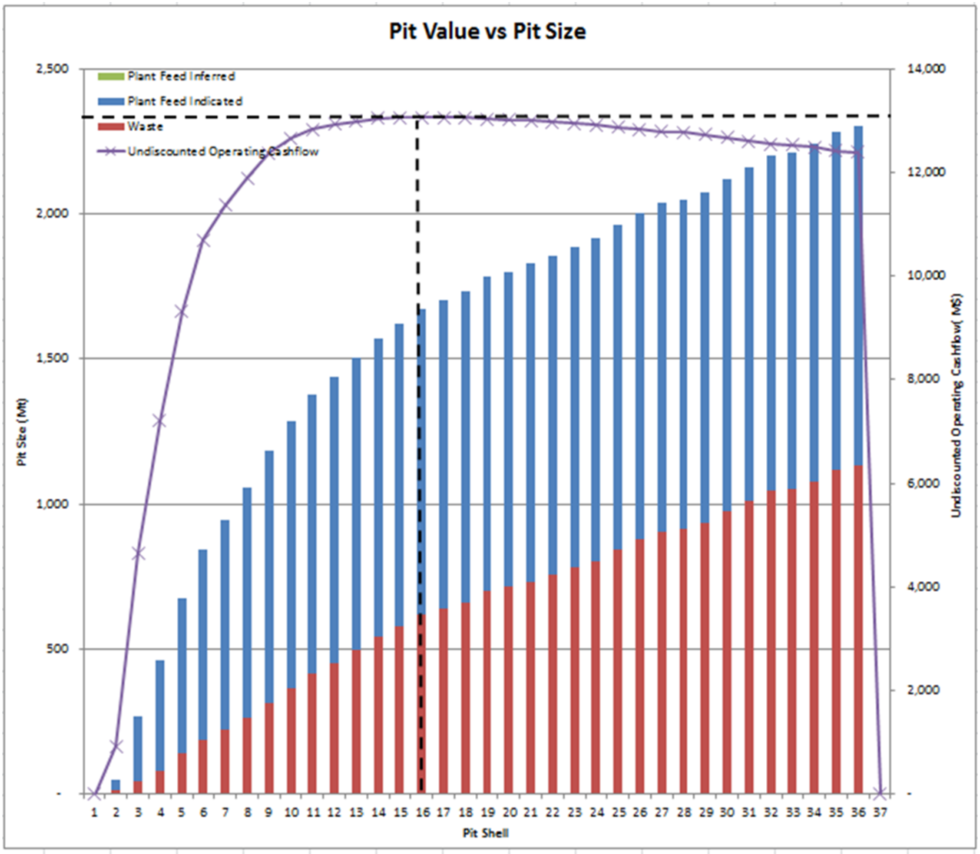


Figure 15-4 Valle Grande Pit optimisation results

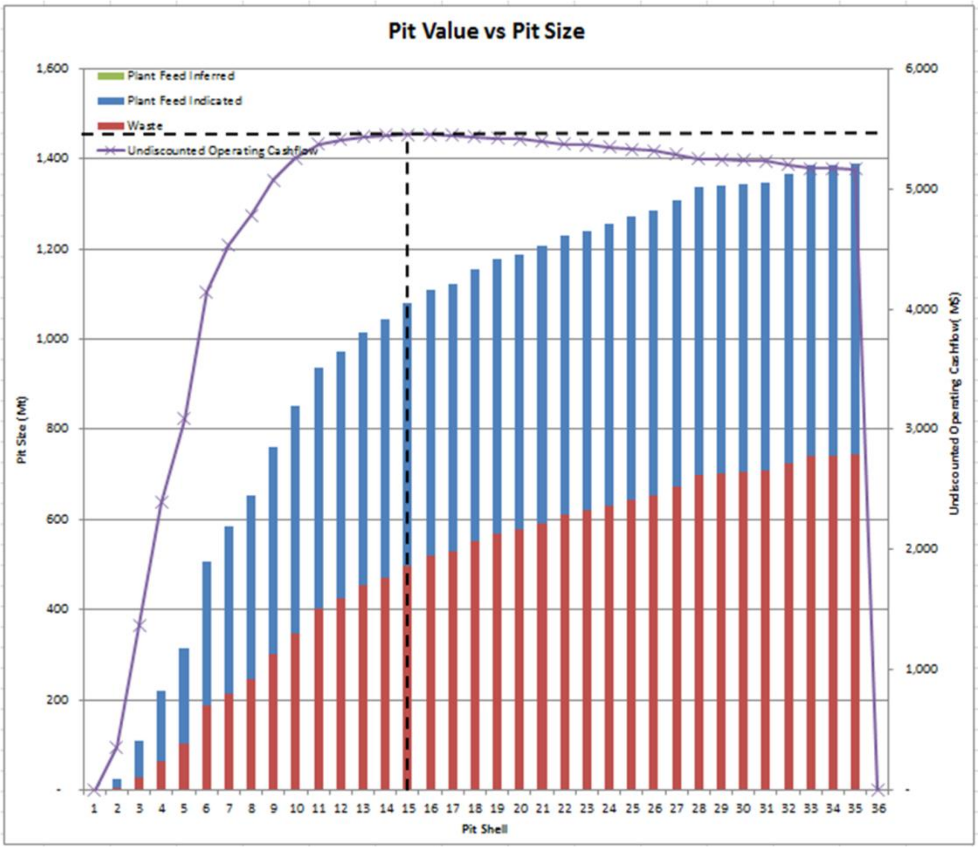


Figure 15-5 Balboa Pit optimisation results

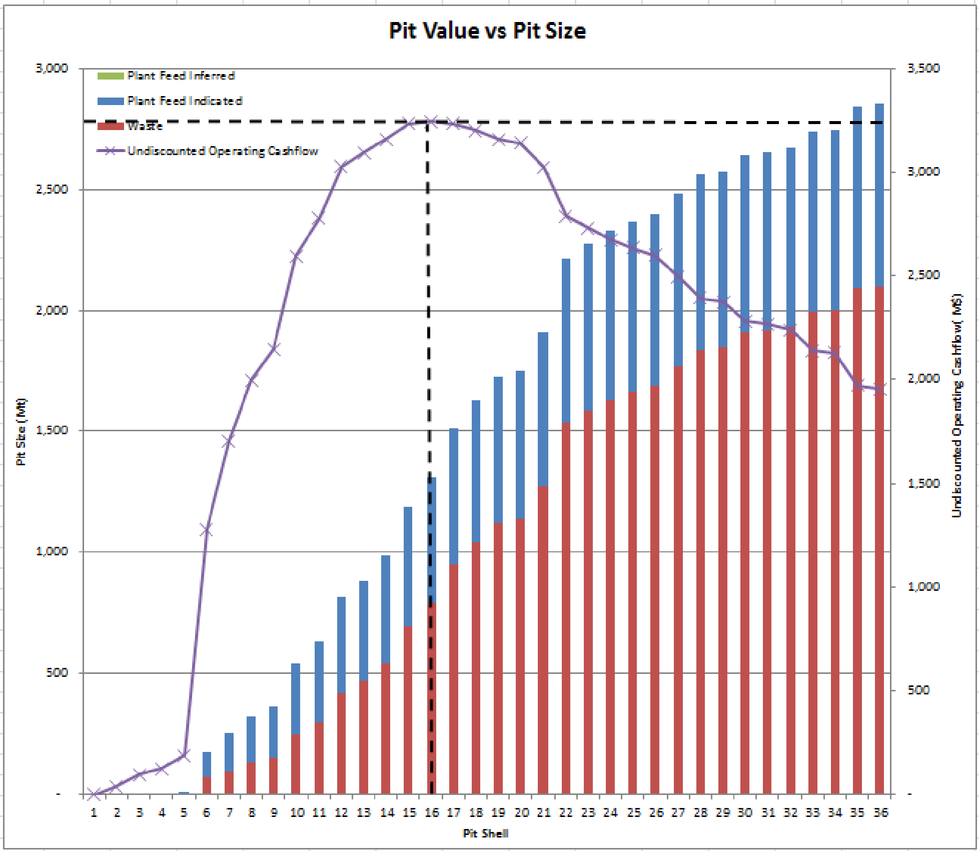


Figure 15-6 Botija Abajo, Brazo (BABR) Pit optimisation results

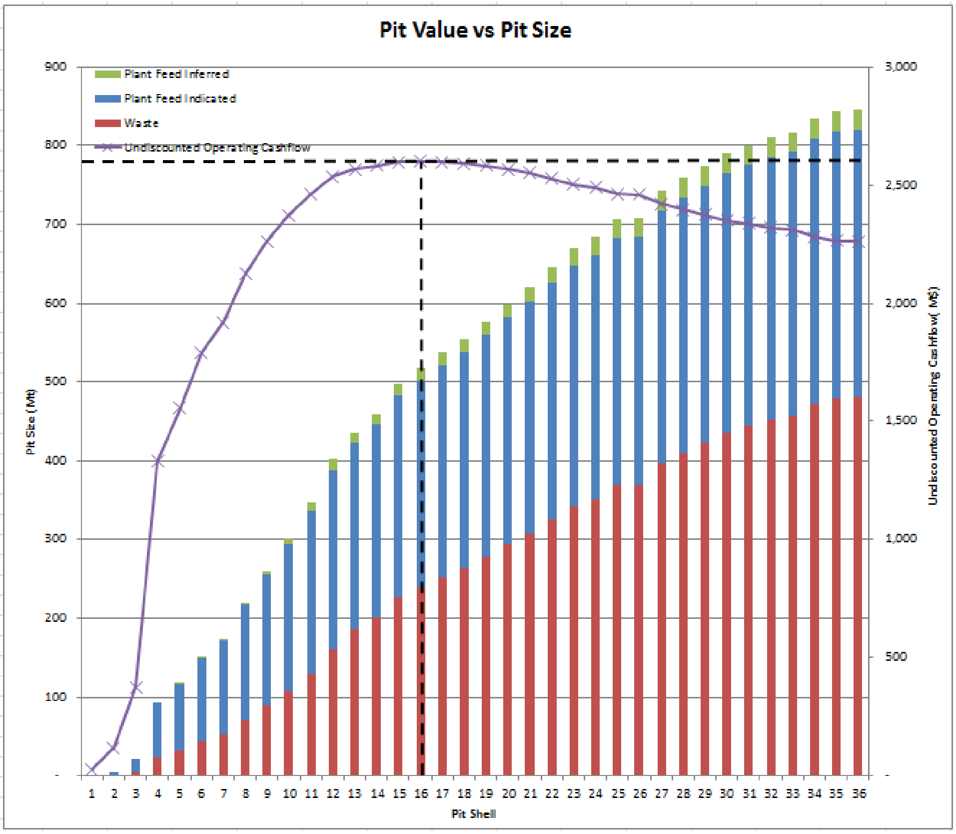


Table 15-7 Cobre Panamá optimal pit shell inventories

Ultimate Pits		Botija	Colina & Medio	ValleGrande	Balboa	BABR	TOTAL
Shell No		**15**	**16**	**15**	**16**	**16**	
Plant Feed	Mt	1,009.6	1,056.1	582.2	522.2	262.7	**3,432.8**
Feed Grade	%TCu	0.36	0.37	0.35	0.30	0.36	**0.35**
Cu Metal Insitu	kt	3,644.7	3,894.3	2,017.8	1,587.6	953.5	**12,098.0**
Waste	Mt	648.2	617.2	498.5	790.7	239.0	**2,793.6**
Total Mined	Mt	1,657.8	1,673.3	1,080.7	1,312.9	501.7	**6,226.4**
Strip Ratio		0.64	0.58	0.86	1.51	0.91	**0.81**
Recovered Cu	kt	3,309.4	3,534.6	1,742.2	1,431.2	833.4	**10,850.9**
Recovered Mo	kt	37.7	37.1	20.0	4.5	5.7	**105.1**
Recovered Au	koz	1,341.6	1,111.5	390.5	812.0	274.1	**3,929.6**
Recovered Ag	koz	17,310.9	24,616.9	11,960.8	8,459.3	2,073.3	**64,421.3**
Undisc. Cashflow	**M$**	**12,343.6**	**13,060.3**	**5,448.9**	**3,244.3**	**2,598.1**	**36,695.2**

15.4.5 Optimisation sensitivity

Sensitivity to changed metal prices

A sensitivity analysis was completed, post cashflow modelling (Item 22), to assess the impact of updated long term (concensus forecast) copper, molybdenum, gold and silver prices as follows:

- gold = \$3.07/lb (\$6,772/t)
- molybdenum = \$8.83/lb (\$19,467/t)
- gold = \$1,310/oz
- silver = \$18.87/oz

The results as shown in Table 15-8 indicate minimal impact to the total net return due to these pricing updates (yellow highlighted information). The marginal cut-off grade is unchanged.

Table 15-8 Cobre Panamá optimisation sensitivity analyses

	Opt'n	Sensitivity analyses			
Copper					
TCRCs, $/lb Cu payable	$0.28	$0.28	$0.28	$0.28	$0.34
Metal price, $/lb	$3.00	$3.07	$3.00	$3.00	$3.00
royalty rate, %	5.0%	5.0%	5.0%	5.0%	2.0%
Cu metal cost, $/lb Cu payable	**$0.43**	**$0.43**	**$0.43**	**$0.43**	**$0.40**
Net return, $/lb (recovered)	**$2.34**	**$2.40**	**$2.34**	**$2.34**	**$2.36**
Molybdenum					
TCRCs, $/lb Mo payable	$0.09	$0.09	$0.09	$0.09	$1.33
Metal price, $/lb	$13.50	$8.83	$13.50	$13.50	$13.50
royalty rate, %	5.0%	5.0%	5.0%	5.0%	2.0%
Mo metal cost, $/lb Mo payable	**$0.77**	**$0.53**	**$0.77**	**$0.77**	**$1.60**
Cu_{eq} Net Return, $/lb (recovered)	**$0.08**	**$0.05**	**$0.08**	**$0.08**	**$0.07**
Gold					
TCRCs, $/lb Au payable	$5.06	$5.06	$5.06	$5.06	$4.59
Metal price, $/oz	$1,200	$1,310	$1,200	$1,200	$1,200
royalty rate, %	5.0%	5.0%	5.0%	5.0%	2.0%
Au metal cost, $/oz Au payable	**$65.060**	**$70.560**	**$65.060**	**$65.060**	**$28.590**
Au metal cost, $/lb Au payable	**$948.77**	**$1,028.98**	**$948.77**	**$948.77**	**$416.93**
Cu_{eq} Net Return, $/lb (recovered)	**$0.10**	**$0.11**	**$0.10**	**$0.10**	**$0.10**
Silver					
TCRCs, $/lb Ag payable	$0.36	$0.36	$0.36	$0.36	$0.40
Metal price, $/oz	$16.00	$18.87	$16.00	$16.00	$16.00
royalty rate, %	5.0%	5.0%	5.0%	5.0%	2.0%
Au metal cost, $/oz Ag payable	**$1.160**	**$1.304**	**$1.160**	**$1.160**	**$0.718**
Ag metal cost, $/lb Ag payable	**$16.92**	**$19.01**	**$16.92**	**$16.92**	**$10.46**
Cu_{eq} Net Return, $/lb (recovered)	**$0.02**	**$0.02**	**$0.02**	**$0.02**	**$0.02**
Total Net Return, $/lb (recovered)	**$2.53**	**$2.58**	**$2.53**	**$2.53**	**$2.56**
Total Net Return, $/10kg (recovered)	**$55.83**	**$56.85**	**$55.83**	**$55.83**	**$56.37**
Total Net Return, $/10kg (unrecovered)	**$61.50**	**$62.63**	**$61.50**	**$61.50**	**$62.10**
Marginal cut-off grade					
PROCOST, $/t ore	$5.45	$5.45	$4.10	$4.60	$5.45
MINDIL	$1.02	$1.02	$1.02	$1.02	$1.02
TOTAL NET RETURN, $/10kg	**$61.50**	**$62.63**	**$61.50**	**$61.50**	**$62.10**
Recovery	90.8%	90.8%	90.8%	90.8%	90.8%
C/O Equivalent grade, %	**0.10**	**0.10**	**0.07**	**0.08**	**0.10**

Sensitivity to changed mining cost estimates

More detailed mining cost estimates were developed subsequent to the optimisations described herein. Rather than being based on generic haulage profiles for a single open pit, these estimates were produced with the benefit of comprehensive haul profiles within completed, individual open pit and waste dump designs. Item 21 provides a commentary on these cost updates.

An optimisation sensitivity analysis using this new information showed marginal differences to the optimal shells selected for the pit designs and Mineral Reserves process (with the exception of BABR, which as discussed in Item 16, is not mined until after 2040).

Sensitivity to revised G&A, processing and metal cost estimates

In the same way that mining costs were revised subsequent to the optimisations, G&A costs, processing costs and metal costs were also reviewed and updated for the cashflow model. Item 21 provides a commentary on these particular cost updates whilst Table 15-8 shows the impact on the marginal cut-off grade due to these changes (green highlighted information). In this table, PROCOST is the sum of the G&A and processing costs. Relative to the $5.45/t ore adopted in the optimisation, PROCOST for the cashflow model is shown as a range of $4.10/t ore and $4.60/t ore. Table 15-8 also shows the revised TCRCs and royalty charges that were adopted for the cashflow model (blue highlighted information).

The overall impact is that the lower marginal cut-off grade due to the operating cost changes suggests some conservatism in the selection of the optimal pit shell. The changes to the metal costs (TCRCs and royalties) have minimal impact on the total net return and the marginal cut-off grade.

15.5 Detailed pit designs

For the 2015 Technical Report (FQM, July 2015), the pit design approach was to produce a series of phased pit designs using ultimate and selected intermediate pit optimisation shells. The approach has changed in the lead-up to this Technical Report, to now reflect a mining progression through each pit according to wide terraced benches, rather than by push back ‘phases’ which emulate intermediate pit optimisation shells. Further information on this change of mining progression is provided in Item 16.

The ultimate pit designs continue to be based on the corresponding selected ultimate pit shells, and according to the design and planning parameters listed below.

15.5.1 Design and planning parameters

Table 15-9 lists the detailed pit slope design criteria adopted during design for each of the Cobre Panamá pits. This table expands upon the information in Table 15-2[8].

The following parameters relate to the design of the ultimate pits, allowing for inclusion of IPCC into the layouts:

- benches (interval between berms) are mined to a height of 15 m or 30 m in ore and waste
- truck ramp width = 3.5 x truck width plus bund = 37.5 m
- truck / dual conveyor ramp width = 55 m (refer to schematic in Figure 16-2)
- all waste ramps = 55 m (allowing for future trolley-assisted haulage, refer to schematic in Figure 16-3)[9]
- maximum conveyor incline angle = 1 : 3.7 = 15°
- maximum haul ramp gradient = 1 : 10 = approximately 6°

Figures 15-7 to 15-11 show the ultimate pit designs produced from respective pit shell outlines. The Botija Pit design has changed since that adopted for the 2015 Technical Report (FQM, July 2015). The new design shown in Figure 15-7 includes a modified north wall which has reduced the waste volume due to a revised positioning of the ultimate in-pit coveyor route.

[8] Item 16.2.2 provides further information on pit slope design criteria, together with the findings from geotechnical reviews carried out after the original pit optimisations had been completed.

[9] The future trolley-assist system will require wider up-haul ramps to cater for the catenary wire towers and triple traffic lanes, so have been designed at minimum 55 m width, with a 1:10 gradient.

Table 15-9 Cobre Panamá pit slope design criteria

Pit	Location	Sector / Wall Height	IR Angle including Drainage * (degrees)	Drainage Catch Bench Width (m)	Stack Height (m)	IR Angle within Stack (degrees)	Bench Height (m)	Bench Face Angle (degrees)	Catch Bench Width (m)
Botija	North Wall	1	43.2	18.0	90	46.0	15	70	9
		2	43.2	18.0	90	46.0		70	9
	East Wall	3	43.2	18.0	90	46.0		70	9
		4	43.2	18.0	90	46.0		70	9
	South Wall	5	43.2	18.0	90	46.0		70	9
	West Wall	13	43.2	18.0	90	46.0		70	9
		14	43.2	18.0	90	46.0		70	9
Colina	SE, SW	1	39.5	24	90	43.1	Double Benching 30 meters	55	11
	NW, Pit Bottom	2	42.5	24	90	46.6		60	11
	N, E, S	3	45.6	24	90	50.2		65	11
	SSW	4	45.6	24	90	50.2		65	11
VG	All		39.5	24	90	43.1	30	55	11
Low RQD	All		-	24	60	36.2	15	55	10
BABr	W		41	-	-	44.0	30	63	16
	E		29	-	-	31.5	15	55	14
	All Remaining		34	-	-	38.0	15	55	9
Balboa	All		42	-	-	45.0	30	68	18
Saprolite	Varies by wall height (m) Non- River Diversion Areas	65	-	-	-	32.5	Benching 5 meters	68	5.8
		60	-	-	-	30.0		68	6.6
		55	-	-	-	34.0		68	5.4
		50	-	-	-	35.0		68	5.1
		45	-	-	-	36.0		68	4.9
		40	-	-	-	37.5		68	4.5
		35	-	-	-	39.5		68	4.0
		30	-	-	-	40.2		68	3.9
		25	-	-	-	40.2		68	3.9
		20	-	-	-	40.2		68	3.9
	Varies by wall height (m) River Diversion Areas	65	-	-	-	29.5	Benching 5 meters	68	6.8
		60	-	-	-	30.0		68	6.6
		55	-	-	-	31.0		68	6.3
		50	-	-	-	32.0		68	6.0
		45	-	-	-	33.0		68	5.7
		40	-	-	-	34.0		68	5.4
		35	-	-	-	36.0		68	4.9
		30	-	-	-	38.5		68	4.3
		25	-	-	-	40.2		68	3.9
		20	-	-	-	40.2		68	3.9

Figure 15-7 Botija ultimate pit design

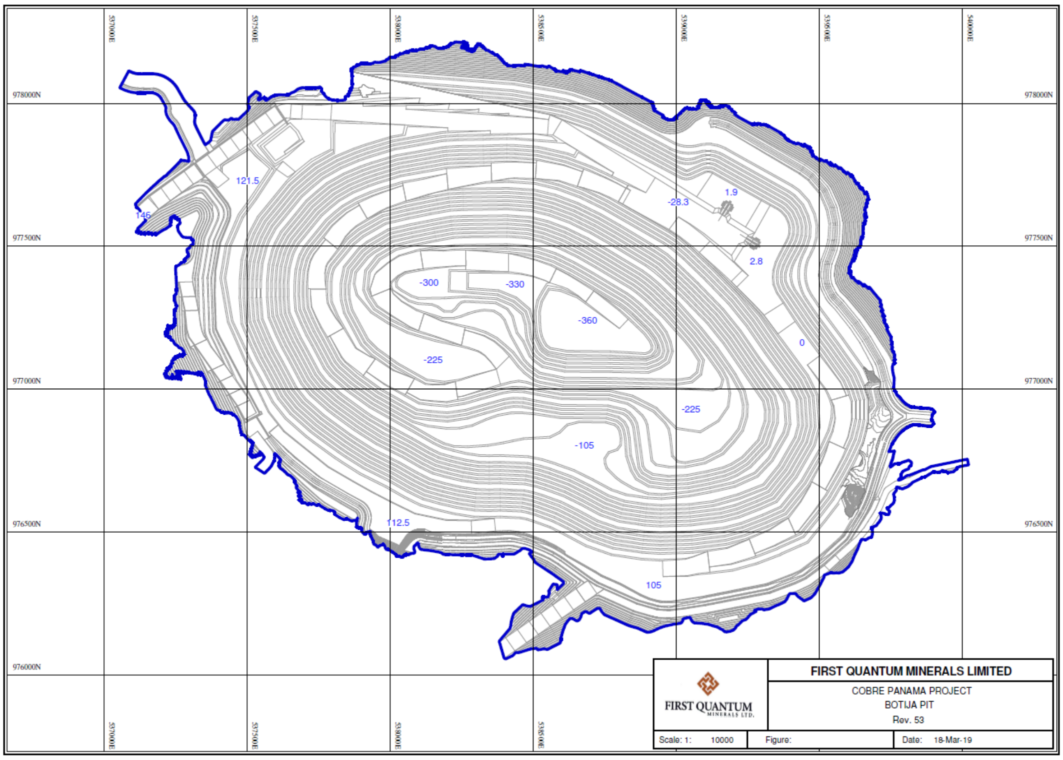


Figure 15-8 Colina ultimate pit design

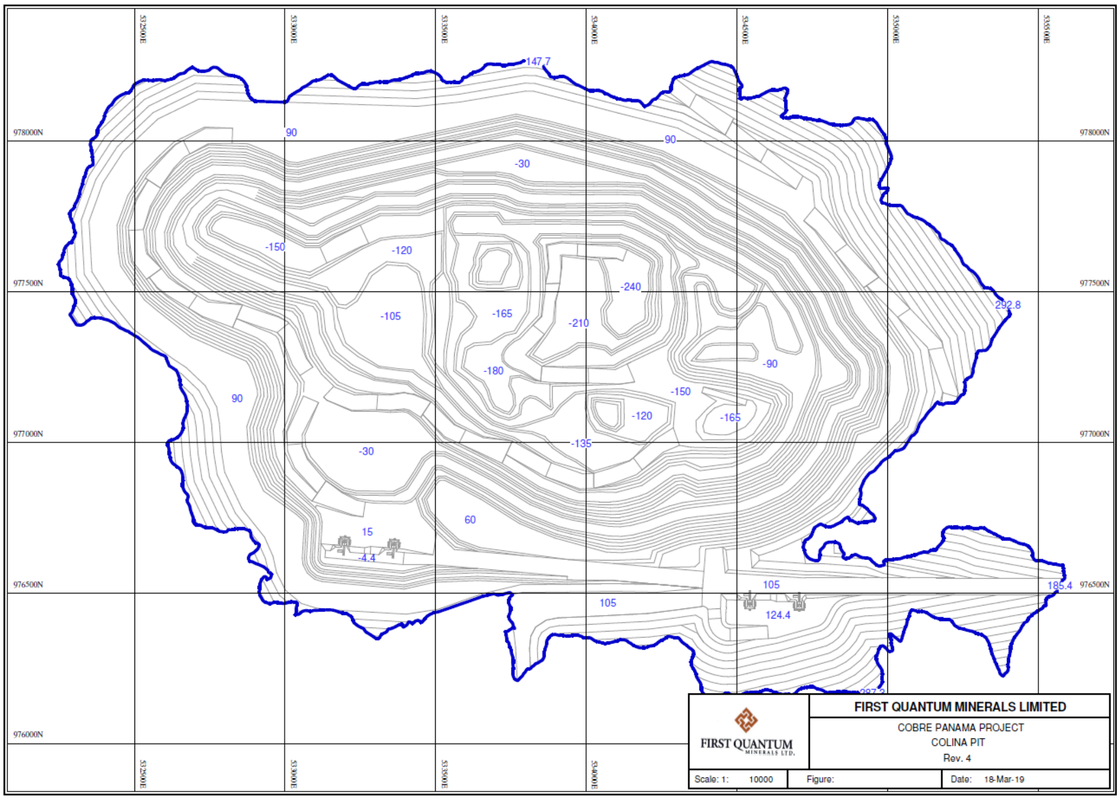


Figure 15-9 Valle Grande ultimate pit design

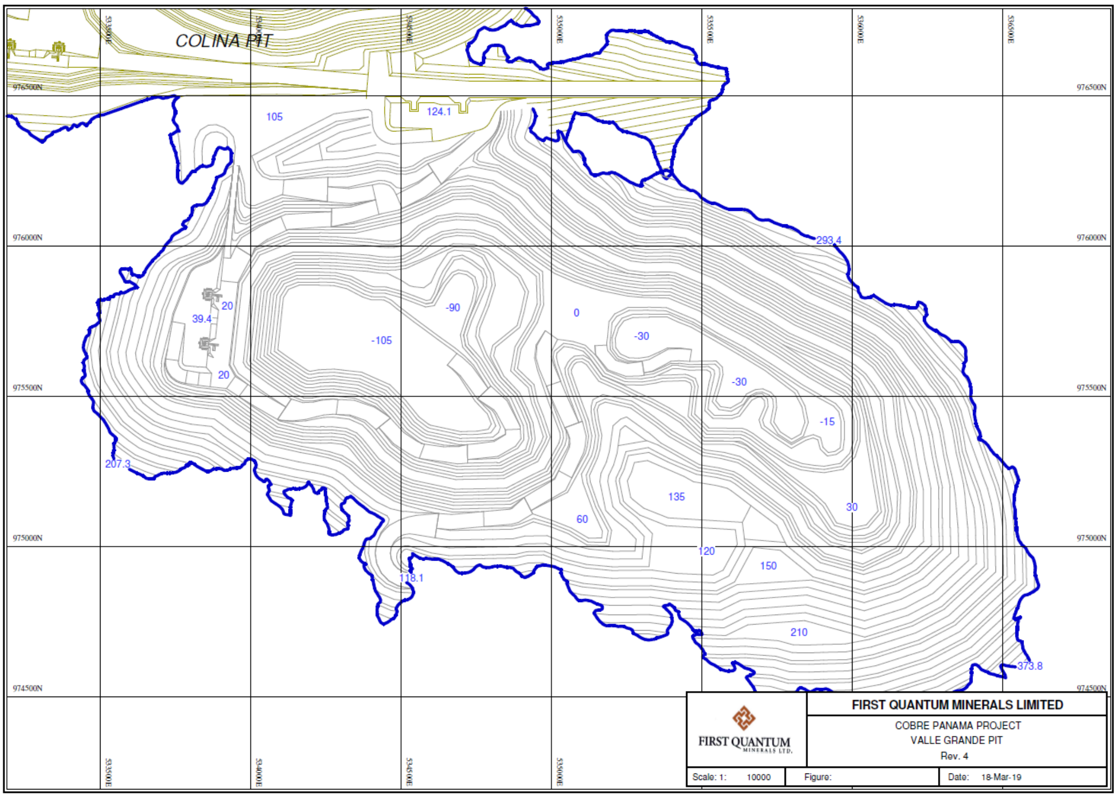


Figure 15-10 Balboa ultimate pit design

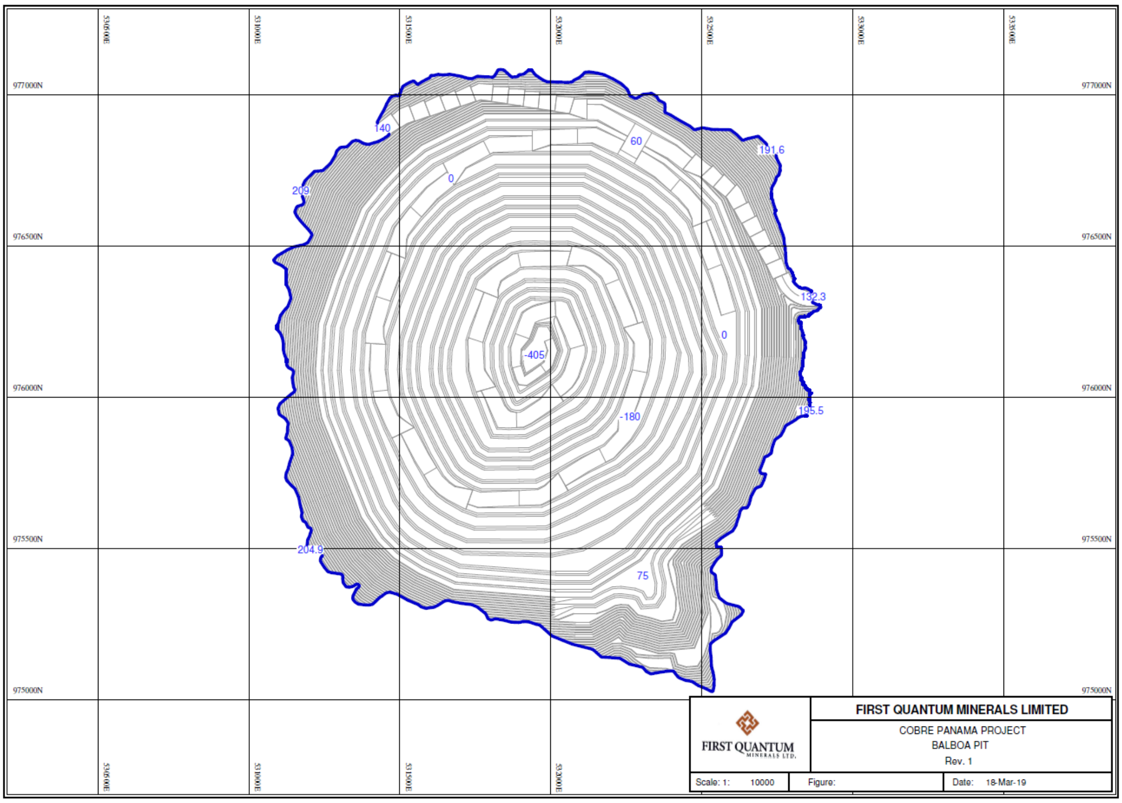
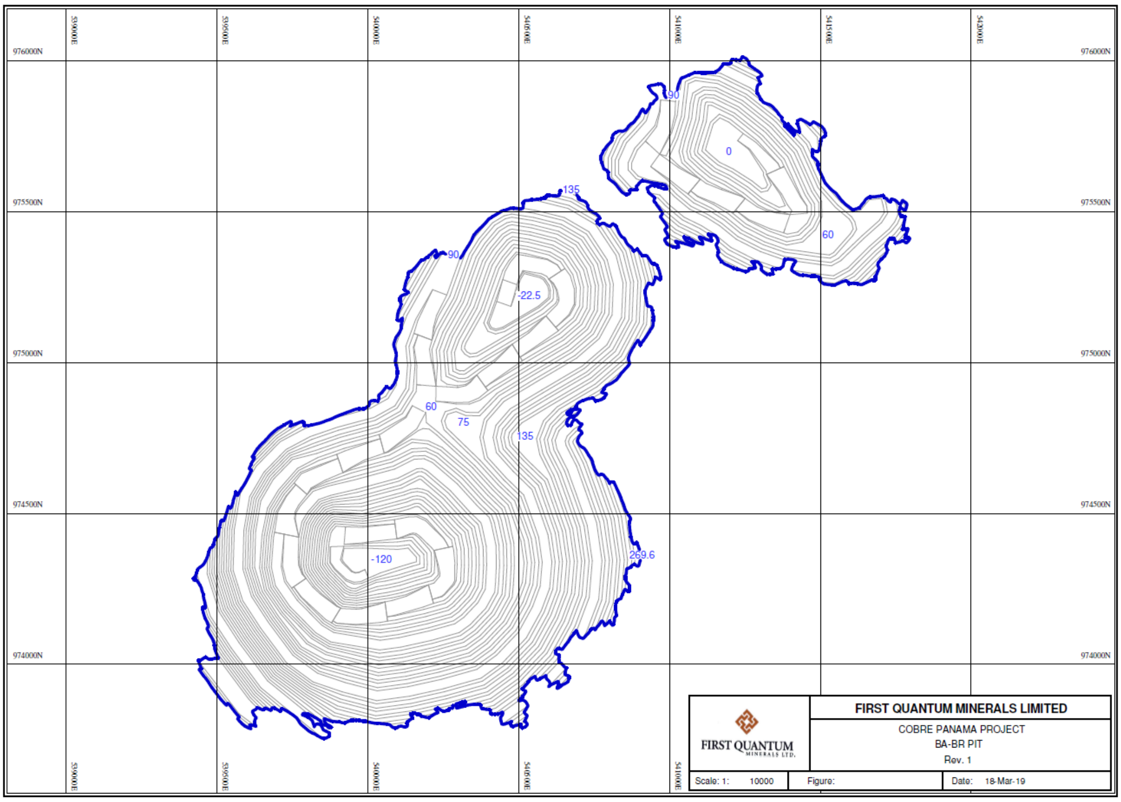


Figure 15-11 BABR ultimate pit design



15.5.2 Design efficiency

Table 15-10 provides a validation comparison between the ultimate pit designs and the shells upon which those designs were based.

Table 15-10 Validation between pit shell and design

	Unit	Botija			Colina & Medio			Valle Grande			Balboa			BABR		
		Shell	Design	Valid'n	Shell	Design	Valid'n	Shell	Design	Valid'n	Shell	Design	Valid'n	Shell	Design	Valid'n
Ore Mined	Mt	1,009.6	988.7	98%	1,056.1	1,051.6	100%	582.2	591.4	102%	522.2	530.2	102%	262.7	244.2	93%
Insitu Cu metal	kt	3,644.7	3,926.7	108%	3,894.3	3,958.1	102%	2,017.8	2,076.1	103%	1,587.6	1,621.2	102%	953.5	916.8	96%
Waste	Mt	648.2	713.9	110%	617.2	737.3	119%	498.5	543.8	109%	790.7	803.2	102%	239.0	256.9	107%
Total Mined	**Mt**	**1,657.8**	**1,702.7**	**103%**	**1,673.3**	**1,788.9**	**107%**	**1,080.7**	**1,135.2**	**105%**	**1,312.9**	**1,333.4**	**102%**	**501.7**	**501.2**	**100%**

In an overall sense, for each pit, there is a reasonable validation between pit designs and the selected ultimate shells. For the Botija Pit, the ultimate pit design has changed since that reported in the 2015 Technical Report (FQM, July 2015) and whilst the overall validation remains reasonable, there are localised areas of the pit design which are outside of the original pit shell.

Item 16.2.3 describes mine planning and engineering work that is currently in progress, and which will be considered for future Botija Pit optimisations. Iterative revisions to the pit optimisations and designs will continue as further engineering work proceeds in relation to future, ultimate IPCC siting scenarios for Botija and Colina.

15.5.3 Mine site layout

Figure 15-12 shows how the ultimate pits relate to each other. Additionally, this figure shows:

- the location of the process plant site and overland ore conveying routes
- the proposed locations of future IPC positions at Botija, Colina and Valle Grande
- the waste dumps associated with the pits, located and sized to allow for expansion, if and when Inferred Resources are converted to Mineral Reserve status

15.6 Mineral Reserve statement

As at the end of December 2018, the total Mineral Reserve inclusive of stockpile inventory is estimated as 3,147.1 million tonnes at 0.38% TCu, 59.36 ppm Mo, 0.07 g/t Au and 1.37 g/t Ag. The estimate is entirely within the Measured and Indicated Mineral Resource estimate reported in Table 14-9. A breakdown by pit and classification is provided in Table 15-11. Table 15-12 lists the additional Mineral Reserve inventory that is available on stockpiles which have accumulated during the pre-strip mining.

The reported Mineral Reserve is based on an economic cut-off grade which accounts for a longer-term copper metal price projection of $3.00/lb ($6,615/t). The inventory reflects the ultimate pit designs and the mining production schedule described in Item 16.3.

Figure 15-12 Ultimate pit designs and waste dumps

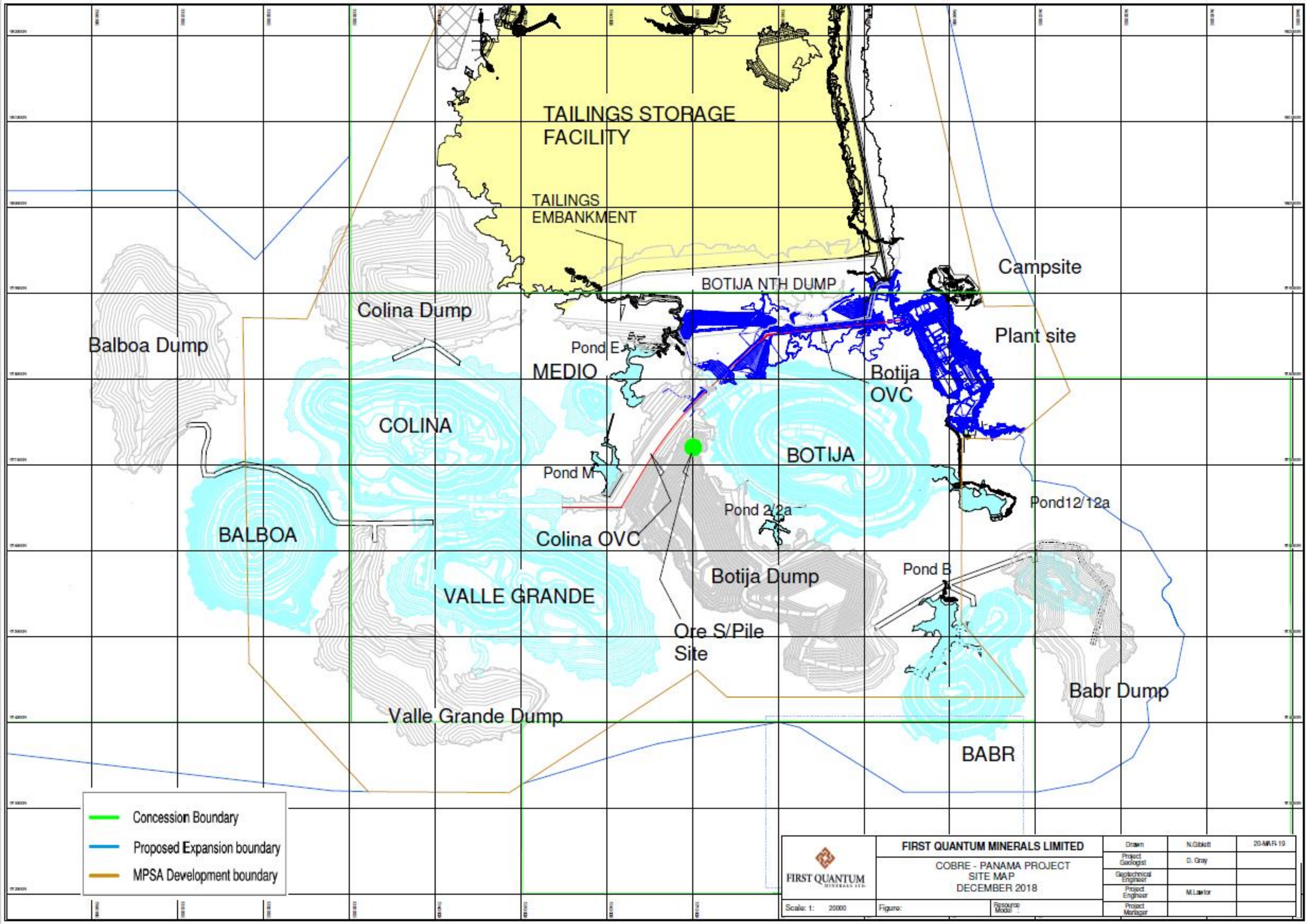


Table 15-11 Cobre Panamá Project In-pit Mineral Reserve statement, at December 2018

MINERAL RESERVE AT 31st DECEMBER 2018 (Cu = $3.00/lb, Mo = $13.50/lb, Au = $1,200/toz, Ag = $16.00/toz)											
		Insitu Mining Inventory									
Pit	Class	Mtonnes	TCu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	TCu metal (kt)	Mo metal (kt)	Au metal (koz)	Ag metal (koz)
BOTIJA	Proved	323.2	0.45	75.98	0.11	1.43	1,467.0	24.6	1,106.3	14,886.7
	Probable	641.4	0.35	68.28	0.08	1.13	2,220.2	43.8	1,512.4	23,357.7
	Total P+P	**964.6**	**0.38**	**70.86**	**0.09**	**1.23**	**3,687.2**	**68.4**	**2,669.6**	**38,188.5**
COLINA &	Proved									
MEDIO	Probable	981.3	0.39	66.98	0.06	1.61	3,870.5	65.7	1,986.8	50,646.8
	Total P+P	**981.3**	**0.39**	**66.98**	**0.06**	**1.61**	**3,870.5**	**65.7**	**1,986.8**	**50,646.8**
VALLE GRANDE	Proved									
	Probable	541.1	0.37	67.43	0.05	1.42	2,016.0	36.5	805.2	24,637.3
	Total P+P	**541.1**	**0.37**	**67.43**	**0.05**	**1.42**	**2,016.0**	**36.5**	**805.2**	**24,637.3**
BALBOA	Proved									
	Probable	437.1	0.35	16.10	0.08	1.36	1,509.0	7.0	1,126.9	19,168.2
	Total P+P	**437.1**	**0.35**	**16.10**	**0.08**	**1.36**	**1,509.0**	**7.0**	**1,126.9**	**19,168.2**
BABR	Proved									
	Probable	219.7	0.40	41.31	0.07	0.87	882.1	9.1	527.5	6,163.9
	Total P+P	**219.7**	**0.40**	**41.31**	**0.07**	**0.87**	**882.1**	**9.1**	**527.5**	**6,163.9**
TOTAL	Proved	323.2	0.45	75.98	0.11	1.43	1,467.0	24.6	1,106.3	14,886.7
	Probable	2,820.5	0.37	57.48	0.07	1.37	10,497.7	162.1	6,009.6	123,918.1
	Total P+P	**3,143.7**	**0.38**	**59.38**	**0.07**	**1.37**	**11,964.7**	**186.7**	**7,116.0**	**138,804.8**

Table 15-12 Cobre Panamá Project Stockpile Mineral Reserve statement, at December 2018

MINERAL RESERVE AT 31st DECEMBER 2018 (Cu = $3.00/lb, Mo = $13.50/lb, Au = $1,200/toz, Ag = $16.00/toz)										
		Stockpile Inventory								
Pit	Class	Mtonnes	TCu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	TCu metal (kt)	Mo metal (kt)	Au metal (koz)	Ag metal (koz)
BOTIJA	Proved									
	Probable	3.3	0.22	43.12	0.04	1.18	7.4	0.1	4.5	126.9
	Total P+P	**3.3**	**0.22**	**43.12**	**0.04**	**1.18**	**7.4**	**0.1**	**4.5**	**126.9**

The in-pit Mineral Reserve listed in Table 15-11 represents mining recovery of 86% and 88% of the Measured plus Indicated Mineral Resource tonnage and insitu metal estimate, respectively. In total, 10% of the combined Mining inventory tonnage is classified as Proven with the remainder classified as Probable.

15.7 Mineral Reserve estimate comparisons

Reflecting the updates to the Mineral Resource model and the comparisons made between the 2018 and 2015 estimates (Item 14.14 and Table 14-10), there are commensurate differences between the respective Mineral Reserve estimates for each pit.

A comparison between the 2018 and 2015 estimates is provided in Table 15-13. Aside from the modelling related differences described in Item 14 and applicable to the Botija deposit, the change in Mineral Reserve tonnes for Botija and the other pits is partly attributable to a redefinition of ore material types.

Mineral Reserve differences for the Botija Pit are also in part attributable to losses incurred during pre-strip mining, offset against changes due to redefinition of ore material types. Further information on the redefinition of ore types is provided in Item 16.1.5.

Table 15-13 Comparison of Mineral Reserve estimate results as at December 2018 and the previous results as at 2015

Pit	Class	Mtonnes			%Cu			TCu metal (kt)		
		2018	2015	%Var	2018	2015	%Var	2018	2015	%Var
BOTIJA	Proved	323.2	345.6	-6%	0.45	0.45	1%	1,465.3	1,550.2	-5%
	Probable	644.7	603.5	7%	0.35	0.35	-2%	2,225.1	2,124.5	5%
	Total P+P	**967.9**	**949.1**	**2%**	**0.39**	**0.39**	**1%**	**3,690.4**	**3,674.4**	**0%**
COLINA & MEDIO	Proved	0.0	0.0	0%	0.00	0.00	0%			0%
	Probable	981.3	1,009.9	-3%	0.39	0.39	2%	3,870.5	3,898.8	-1%
	Total P+P	**981.3**	**1,009.9**	**-3%**	**0.39**	**0.39**	**2%**	**3,870.5**	**3,898.8**	**-1%**
VALLE GRANDE	Proved	0.0	0.0	0%	0.00	0.00	0%			0%
	Probable	541.1	566.0	-4%	0.37	0.36	4%	2,016.0	2,035.9	-1%
	Total P+P	**541.1**	**566.0**	**-4%**	**0.37**	**0.36**	**4%**	**2,016.0**	**2,035.9**	**-1%**
BALBOA	Proved	0.0	0.0	0%	0.00	0.00	0%			0%
	Probable	437.1	437.1	0%	0.35	0.35	0%	1,509.0	1,509.0	0%
	Total P+P	**437.1**	**437.1**	**0%**	**0.35**	**0.35**	**0%**	**1,509.0**	**1,509.0**	**0%**
BABR	Proved	0.0	0.0	0%	0.00	0.00	0%			0%
	Probable	219.7	220.5	0%	0.40	0.40	0%	882.1	882.5	0%
	Total P+P	**219.7**	**220.5**	**0%**	**0.40**	**0.40**	**0%**	**882.1**	**882.5**	**0%**
TOTAL	Proved	323.2	345.6	-6%	0.45	0.449	1%	1,465.3	1,550.2	-5%
	Probable	2,823.9	2,836.9	0%	0.37	0.368	1%	10,502.6	10,450.7	0%
	Total P+P	**3,147.1**	**3,182.5**	**-1%**	**0.38**	**0.38**	**1%**	**11,967.9**	**12,000.9**	**0%**

ITEM 16 MINING METHODS

16.1 Mining overview

Each of the Cobre Panamá deposits is amenable to large scale, conventional open pit mining methods comprising of typical drill and blast, shovel and haulage truck techniques.

Mining is proceeding in a sequence from an initial starter pit at Botija, supplying pre-strip development waste for site infrastructure construction and ore for process plant commissioning. Production will subsequently be ramped up for full-scale ore processing, with the open pits expanded and deepened.

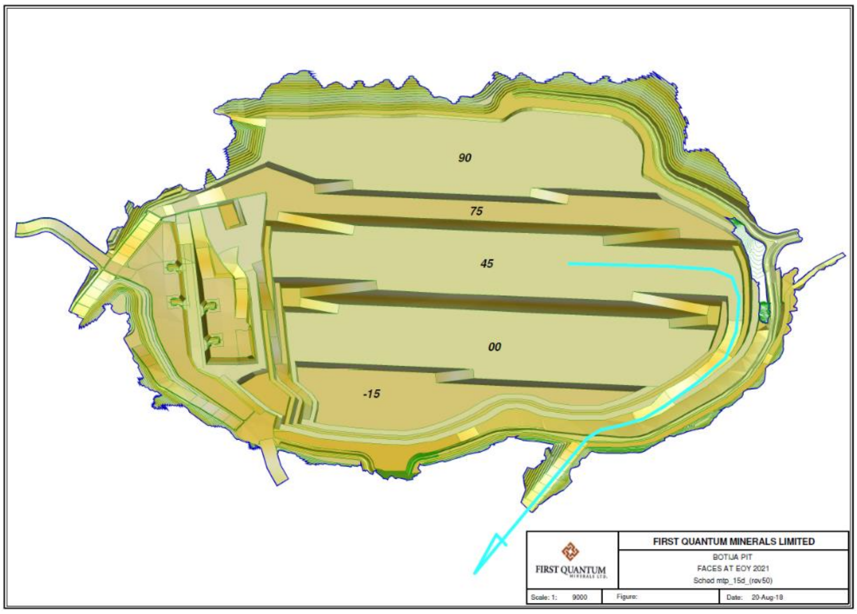
Building upon the technologies developed at other FQM operations, the Project features in-pit crushing and conveying (IPCC). Blasted ore will be hauled to IPCC installations strategically located within the open pits. These installations will be near surface at the outset, but will be moved deeper into the pits as mining proceeds over time. In-pit conveyors will be extended to suit and these will converge on surface at a central transfer station discharging to a permanent overland conveyor connecting to the plant site.

16.1.1 Terrace mining

As mentioned in Item 15, the mine development approach for the open pits has changed from one which was based on phased pits and cutbacks towards an ultimate perimeter, to one which is now based on a terraced mining layout. This change comes about due to experience elsewhere at Company operations, where the deployment and utilisation of ultra class mining equipment has been improved by operating on broad terraces. A terrace layout in the Cobre Panamá environment also provides an improved means of managing rainfall inundation of the pits.

Figure 16-1 shows an example of the terrace layout devised for the Botija Pit. At this stage, conceptual terrace layouts have been prepared for the other pits and incorporated into the life of mine production scheduling. Future mine planning work will consider modifications and potential enhancements to these terrace layouts.

Figure 16-1 Scheduled Botija Pit terrace layout, at end 2021



16.1.2 In-pit crushing and conveying of ore (IPCC)

The primary crushing circuit will comprise up to five semi-mobile, independent, gyratory crushers (3 x ThyssenKrupp KB 63 x 89 and 2 x ThyssenKrupp KB 63 x 130) operating in open circuit. Each crusher will be positioned in-pit and remote from the plant area, and crushed ore will be transported to the plant by an overland conveyor.

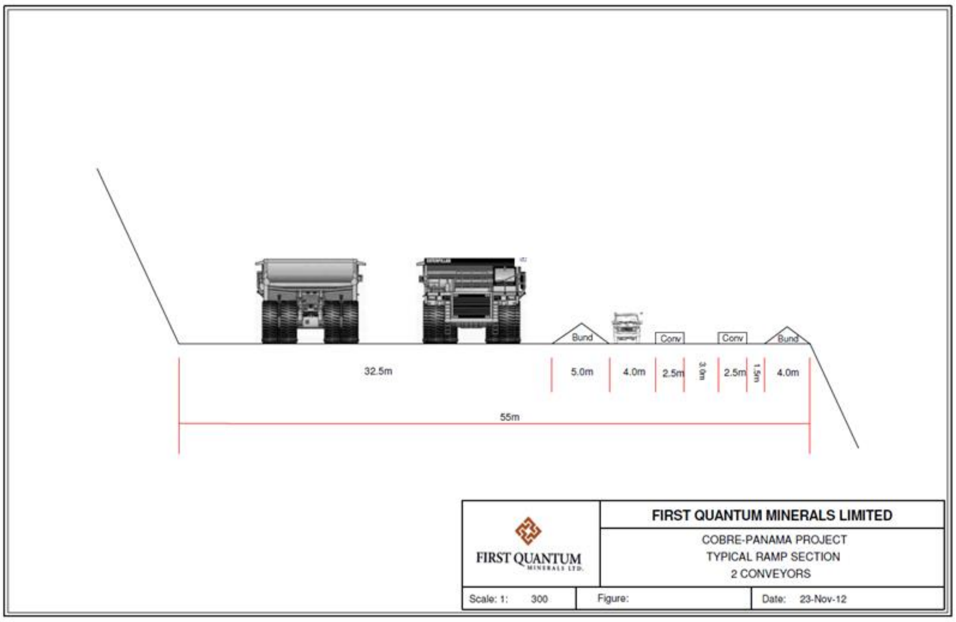
For the purposes of considering crusher relocations in the mine production schedule (Item 16.3), crusher productivity was considered on the basis of the parameters shown in Table 16-1. Under these circumstances, the combined capacity of the five crushers is approximately 130 Mtpa and is in excess of the ultimate milling rate.

Table 16-1 In-pit crusher capacity

ThyssenKrupp IPC		KB 63 x 89	KB 63 x 130	Comments
Nominal productivity	tph	3,600	5,040	for each crusher
Availability	%	96%	96%	routine crusher maintenance
Planned productivity	tph	3,456	4,838	
Utilisation	%	75%	75%	18 hrs/day (discontinuous feed)
Daily capacity	tpd	62,208	87,091	
Operational downtime	days/yr	24	24	adjacent blasting, belt maintenance
Annual capacity	**Mtpa**	**21.2**	**29.7**	

The inpit haul road arrangement to accommodate a dual conveyor easement is shown in Figure 16-1.

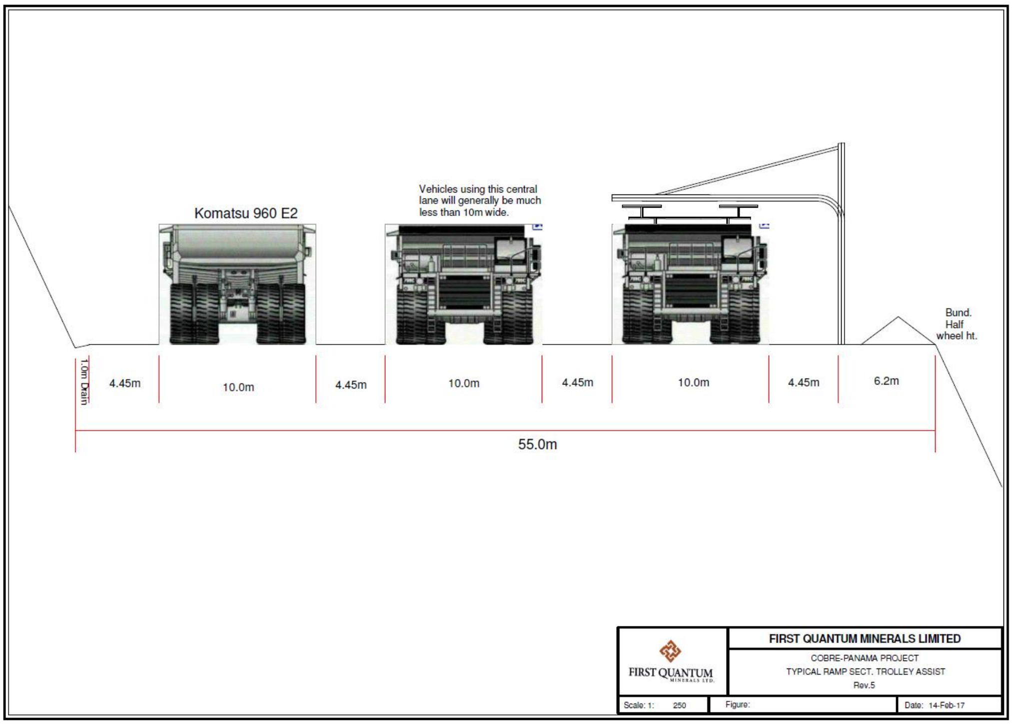
Figure 16-2 Schematic cross section across a conveyor ramp



16.1.3 Trolley-assisted haulage

Trolley-assisted haulage is a concept that is being adopted during the early life of operations. The primary truck haulage fleet is being delivered "trolley-assist ready" (TA). Additional pit ramp width has been included in the detailed pit phase designs to allow for the physical placement of transformers and catenary wire poles (Figure 16-3). In places, these ramps could be extended onto the waste dumps.

Figure 16-3 Schematic cross section across a trolley-assisted haulage ramp



16.1.4 Waste dumping

The planned waste dumps (referred to as “waste rock storage facilities”, WRSF) are located surrounding the various pits, wherever space dictates, and in areas that have been largely sterilised by exploratory drilling.

The dumps have been designed such that they do not encroach upon the ultimate limits of the Measured+Indicated+Inferred resource pit shells, nor cross any major drainage paths. The dump profiles have been designed with a 32° batter angle, a 30 m batter height, 26 m width berms, and minimum 55 m wide ramps at 1:10 gradient. The overall angle of each ultimate dump slope is approximately 22°.

16.1.5 Ore grade ranges, stockpiling and reclaiming

The life of mine (LOM) production schedule described in Item 16.3 requires a stockpile building and reclaim strategy in order to balance the direct crusher feed from the pits and maintain a reasonable overall feed tonnage and grade profile. Because of the IPCC concept, however, it makes sense to minimise surface stockpile building and reclaim, and hence the scheduling process is an iterative one to ensure that this is possible whilst also maintaining an orderly and practical ore and waste mining sequence.

Table 16-2 lists the grade criteria adopted for mine production scheduling, for ore stockpiling and for waste categorisation. The lithology types are 100 = saprolite, 400 = andesite, 500 = porphyry, 600 = hornblende porphyry, 700 = felsic porphyry, 800 = granodiorite and 900 = felsic granodiorite.

Table 16-2 Stockpile grade ranges

Ore/Waste classification			
Current descriptor	**Resource Class**	**Lithology**	**Approx. Av. [1] %Cu COG**
ORE			
Saprock Ore (SAPR_O)	Meas+Ind	> 100	>= 0.20
Fresh High Grade Ore (HIGH)	Meas+Ind	> 100 <= 900	>= 0.28
Fresh Medium Grade Ore (MED)	Meas+Ind	> 100 <= 900	>= 0.20 < 0.28
Fresh Low Grade Ore (LOW)	Meas+Ind	> 100 <= 900	>= 0.15 < 0.20
WASTE			
Mineralised Waste (MARG)	Meas+Ind	> 100 <= 900	>= 0.11 < 0.15
Saprolite (SAP)	Meas+Ind+Inf	100	n/a
NAF Saprock Waste (SAPR_N)	Meas+Ind+Inf	> 100	< 0.20
PAF Saprock Wate (SAPR_P)	Meas+Ind+Inf	> 100	< 0.20
NAF Fresh Waste (NAF)	Meas+Ind+Inf	> 100 <= 900	< 0.11
PAF Fresh Waste (PAF)	Meas+Ind+Inf	> 100 <= 900	< 0.11

1. Estimated average COG based on block value criteria

The material type and grade range definition listed in Table 16-2 are a change from that described and tabled in the 2015 Technical Report (FQM, July 2015). The definition was previously based on a block by block net return calculation, whereas it is now based on average cut-off grade criteria pertaining to net return calculations. This change was made on the basis of consistency with site practice following the development of the grade control programme and the actual mark-out criteria for digging blocks.

For the IPCC reason described above, high and medium grade ore is preferentially direct fed to the crushers. However, some of this tonnage must be mined and stockpiled and hence, high and medium grade ore stockpiles are considered to be "active" throughout the mine life. Space adjacent to the Botija Pit western crest has been set aside as an initial location for these (refer to Figure 15-12). From the production schedule in Table 16-5, the maximum size of these stockpiles is 11.4 Mt, reached in 2019. The maximum reclaim of 10.0 Mt from these piles occurs in 2020 when there will be four crushers in the Botija Pit. This 10 Mt represents 12% of the crusher feed in that year.

Over the life of the mine, further active stockpiles will need to be positioned at convenient locations adjacent to or within convenient a distance to near-surface crusher positions.

Long term low grade ore and saprock ore stockpiles are developed over the life of the mine and are not reclaimed until the final years of operations. The low grade is deferred in preference to direct feed of high and medium grade ore, whilst the saprock ore is deferred for reasons of lesser recovery. The location for these long term stockpiles has not been identified explicitly (other than for a Botija Pit long term stockpile site located on the southern waste dump), although it is expected that they would form separate and discrete volumes within the ultimate waste dumps and close to the final near-surface crusher positions.

16.1.6 Grade control

Conventional open pit grade control practices have been put into place, incorporating RC drilling and sampling on a suitably designed drilling pattern and over multiple bench horizons. Blasthole sampling may also be used, as required, to provide additional data for ore block boundary definition. Multi element sample assaying is being carried out on site. A grade control modelling process has been implemented as the basis for designing dig blocks.

16.1.7 Drilling and blasting

Near-surface saprolite material is being mined essentially as free-dig. As and when required, bench development that requires blasting will be blasted on bench heights of between 5 and 10 m and using small diameter blast holes.

Below this horizon, production drilling and blasting will take-place in rock conditions requiring a range of drilling/charging patterns and powder factors. Due to the mix of large and medium sized rotary drills there will be large and medium diameter holes used to blast ore and waste.

Controlled blasting is being undertaken on final walls to prevent blast damage and to maintain wall control. As shown in Table 16-3, the controlled blasting consists of a small diameter presplit hole loaded with a decoupled charge. Adjacent to the presplit line, trim blasting consists of four to six rows of medium sized holes.

Table 16-3 Drilling and blasting parameters

	Production				Development	
Parameter	**Ore**	**Waste**	**Trim**	**Presplit**	**5m Bench**	**10m Bench**
Burden (m)	6.8	8.5	6.0	NA	3.5	4.2
Spacing (m)	7.8	9.8	6.9	1.8	4.0	4.8
Bench Height (m)	15.0	15.0	15.0	15.0	5.0	10.0
Sub-Drill (m)	2.0	2.5	1.0	0.5	0.5	1.0
Stemming Length (m)	5.5	7.5	5.0	2.4 (Air)	2.5	3.0
Blast hole Diameter (mm)	251	311	200	165	115	127
Powder Factor (kg/m^3)	0.85	0.72	0.64	0.50	0.53	0.59

In order to minimise vibration and fly-rock damage, appropriately engineered blast designs are required near in-pit crushers and conveyors. Due to the favourable nature of the geology, it is anticipated that minimal changes to the infrastructure designs will be required to mitigate the risk of vibration and fly-rock.

Powder factors required to date have generally been lower than those in Table 16-3, but are expected to increase with depth. Rock fragmentation characteristics to date have been encouraging.

16.2 Mine planning considerations

The following information relates to the detail that needed to be considered for designing surface layouts and practical mining pits around the optimal pit shell outlines.

16.2.1 Mine design parameters

Basic mine design parameters relating to pit slope parameters and widths of haul roads are described in Item 15. More detailed parameters have had to be considered for IPCC. ThyssenKrupp engineering drawings provided the required excavation and installation dimensions for each semi-mobile crusher installation at the Botija Pit.

16.2.2 Mine geotechnical engineering

The 2015 Technical Report (FQM, July 2015) provided a comprehensive summary of the then current open pit slope design recommendations as provided by consultants Call & Nicholas Inc. (CNI, March 2013, Aprl 2013, June 2014 and May 2015), and AMEC Earth and Environmental (AMEC, February 2010). With the Botija Pit mining pre-strip having progressed since these recommendations were drafted, and with the availability of saprolite, saprock and fresh rock exposures from which to map and interpret geotechnical conditions, a review was subsequently completed by consultant Peter O'Bryan and Associates (POB, October 2018).

This review was focussed on the exposed rock slopes for the Botija Pit south wall, and specifically to the Zone 5 sector shown in Figure 16-4. The POB review resulted in some changes to the earlier specifications, notably the bench height and face angle. Previously these were 30 m and 68°, respectively; the new recommendations are 15 m and 70°, respectively. The berm width was changed from 18 m to 9 m. The original 90 m stack height recommendation was maintained. Consequently, the updated slope design parameters for Botija, and the unchanged parameters for all other pits, are listed in Table 16-4.

Figure 16-4 Updated Botija Pit slope design sectors, 2018

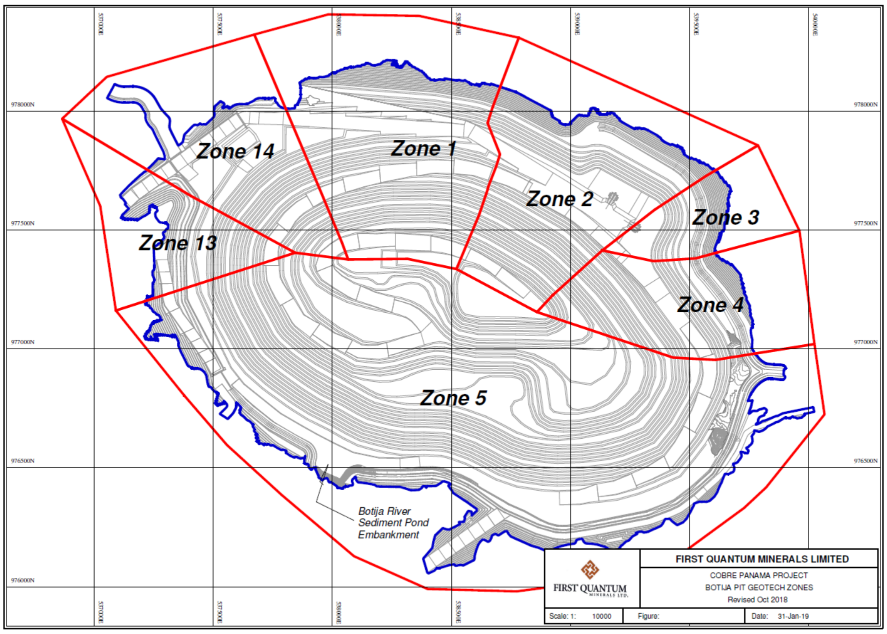


Figure 16-5 shows a cross section through the Botija south wall slope, showing a modest steepening of the overall slope by 1.5°**.** The crest line adjacent to the Botija River diversion channel (Item 16.2.6) has moved towards the north and the channel easement is now wider.

Figure 16-5 Updated Botija Pit slope design, cross section through south wall

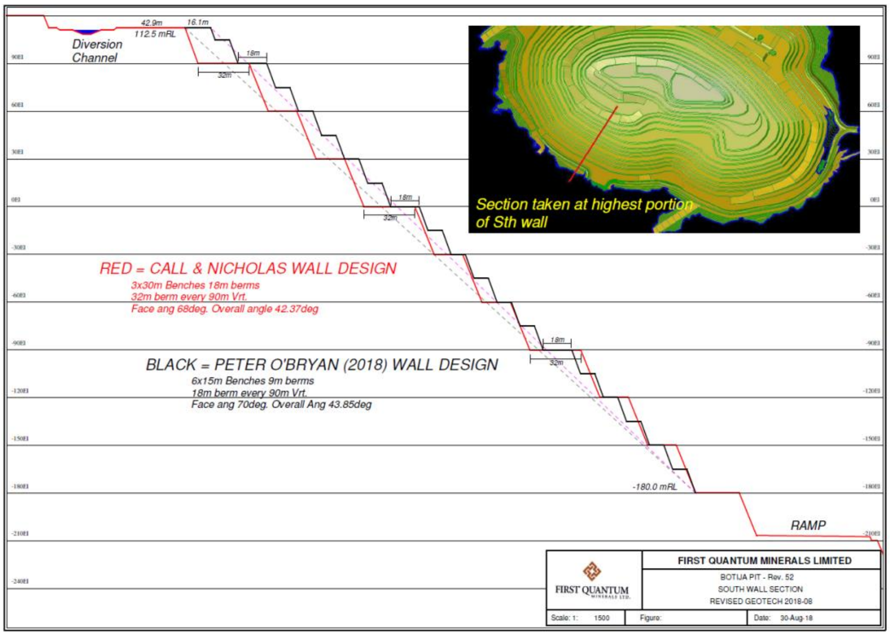


Table 16-4 Updated pit slope design parameters, 2018

Pit	Location	Sector / Wall Height	IR Angle including Drainage * (degrees)	Drainage Catch Bench Width (m)	Stack Height (m)	IR Angle within Stack (degrees)	Bench Height (m)	Bench Face Angle (degrees)	Catch Bench Width (m)
Botija	North Wall	1	43.2	18.0	90	46.0		70	9
		2	43.2	18.0	90	46.0		70	9
	East Wall	3	43.2	18.0	90	46.0		70	9
		4	43.2	18.0	90	46.0		70	9
	South Wall	5	43.2	18.0	90	46.0	15	70	9
	West Wall	13	43.2	18.0	90	46.0		70	9
		14	43.2	18.0	90	46.0		70	9
Colina	SE, SW	1	39.5	24	90	43.1	Double Benching 30 meters	55	11
	NW, Pit Bottom	2	42.5	24	90	46.6		60	11
	N, E, S	3	45.6	24	90	50.2		65	11
	SSW	4	45.6	24	90	50.2		65	11
VG	All		39.5	24	90	43.1	30	55	11
Low RQD	All		-	24	60	36.2	15	55	10
BABr	W		41	-	-	44.0	30	63	16
	E		29	-	-	31.5	15	55	14
	All Remaining		34	-	-	38.0	15	55	9
Balboa	All		42	-	-	45.0	30	68	18
Saprolite	Varies by wall height (m) Non- River Diversion Areas	65	-	-	-	32.5	Benching 5 meters	68	5.8
		60	-	-	-	30.0		68	6.6
		55	-	-	-	34.0		68	5.4
		50	-	-	-	35.0		68	5.1
		45	-	-	-	36.0		68	4.9
		40	-	-	-	37.5		68	4.5
		35	-	-	-	39.5		68	4.0
		30	-	-	-	40.2		68	3.9
		25	-	-	-	40.2		68	3.9
		20	-	-	-	40.2		68	3.9
	Varies by wall height (m) River Diversion Areas	65	-	-	-	29.5	Benching 5 meters	68	6.8
		60	-	-	-	30.0		68	6.6
		55	-	-	-	31.0		68	6.3
		50	-	-	-	32.0		68	6.0
		45	-	-	-	33.0		68	5.7
		40	-	-	-	34.0		68	5.4
		35	-	-	-	36.0		68	4.9
		30	-	-	-	38.5		68	4.3
		25	-	-	-	40.2		68	3.9
		20	-	-	-	40.2		68	3.9

* Drainage benches on 90 m vertical intervals

10 metre wide bench at the base of the saprolite to allow room to contain saprolite slope failures

Pond 2a embankment

Figure 16-4 shows the location of a water retaining embankment located on the crest of the Botija South wall in Zone 5.

Four geotechnical holes and one hydrogeological hole are being drilled in early 2019 to assess ground conditions along strike of the upper Botija Pit south wall. Specifically, the geotechnical holes have the primary objective of detecting potentially adverse structure which could impact on the stability of the Pond 2a embankment and the associated diversion channel. A secondary objective is to assess ground conditions potentially impacting on a proposed south wall ramp.

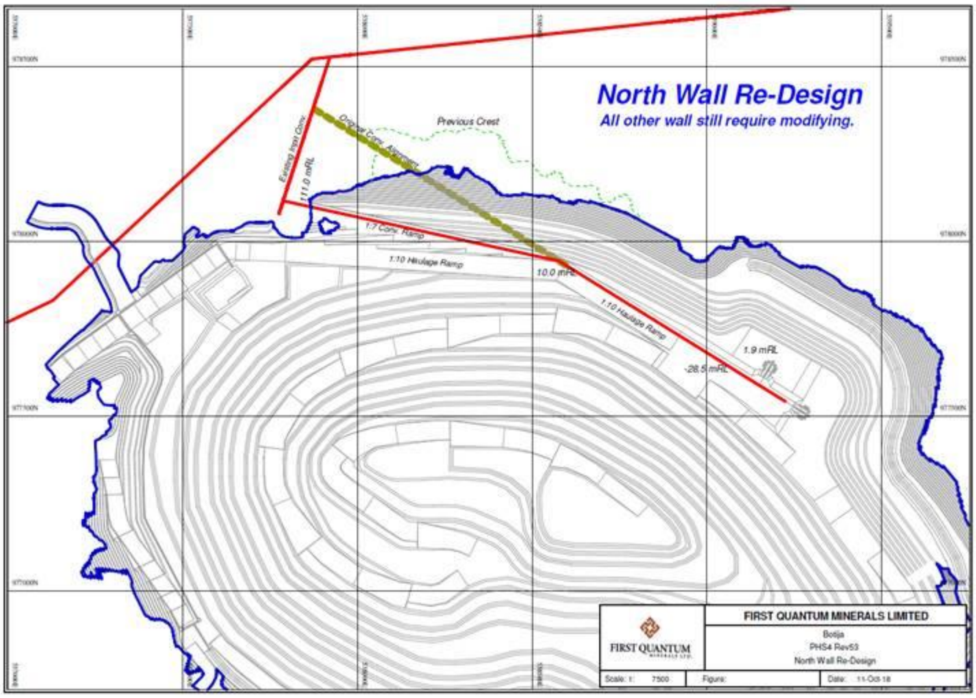
16.2.3 Pit design and related infrastructure updates

Item 15.5 provides a plan showing the latest ultimate pit design for Botija, inclusive of the south wall slope design modifications described above. Other changes made to the previous design were on the north wall, as follows (refer also to the detail in Figure 16-6):

- The position of the ultimate IPC locations has been reoriented and set at an elevation which suits the current terrace mining approach, and the timeframe at which these positions would be reached according to current production scheduling.
- The in-pit conveyor route has been changed, resulting in a reduction in waste stripping on the northern side of the pit.

Except for the north east corner, the revised Botija design remains generally within the optimal pit shell limits. Mine planning continues, however, on considerations for the proposed ultimate north wall siting of the IPCs, with the objective of evaluating the waste mined in the north east corner of the pit, versus an alternative design which would create an ore peninsula extending into the pit.

Figure 16-6 Updated Botija Pit design, future conveyor route on north wall



Subject to a hydrological engineering assessment, there is the potential to further reduce the waste volume in the north east corner of the pit, lying outside of the optimal shell, by removing the designed surface run-off diversion channel that is located above the ultimate crusher position.

An aspect of the Botija Pit design which also remains under review is the positioning of a permanent waste haulage ramp on the south wall. In the current design (Figure 16-7), there is a temporary south wall ramp and a permanent eastern wall ramp, both leading up onto the Botija waste dump [the dump is also undergoing a design review].

In a proposed design (Figure 16-8), the south wall ramp arrangement has been altered to match a modified waste dump layout and includes a permanent south wall ramp. The new dump layout suits operational design changes to incorporate straight-line trolley assist ramps and an excision to accommodate an enlarged sediment control pond. This proposed revised south wall design also remains within the optimal pit shell limits.

Figure 16-7 Current south wall design for the Botija Pit

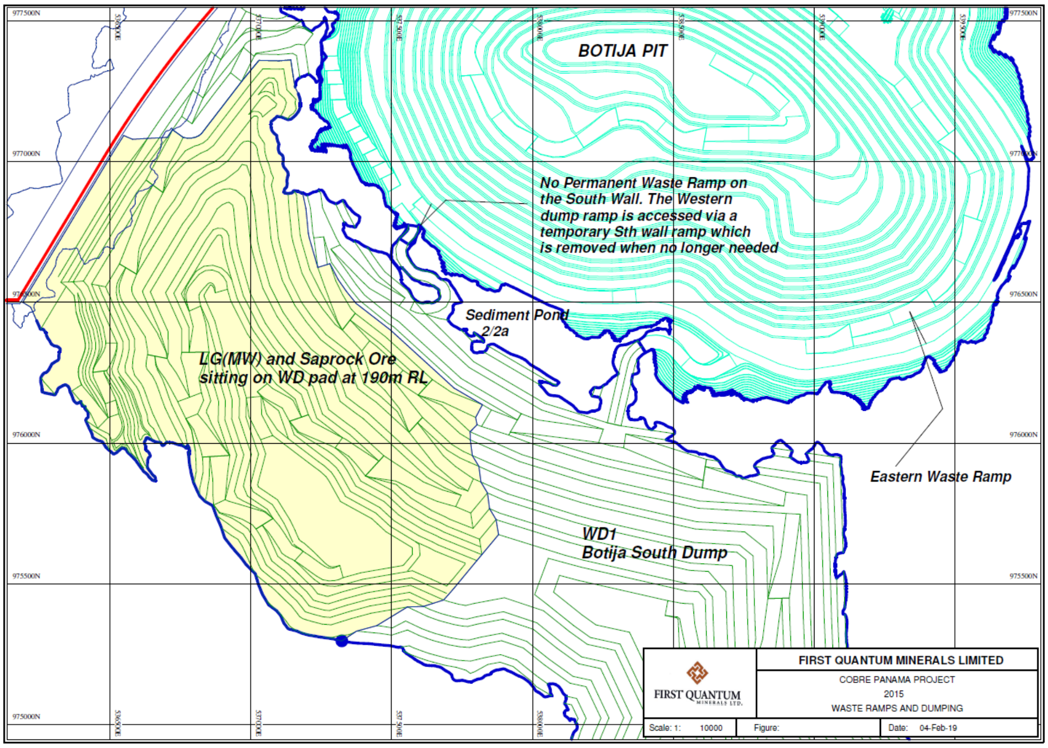
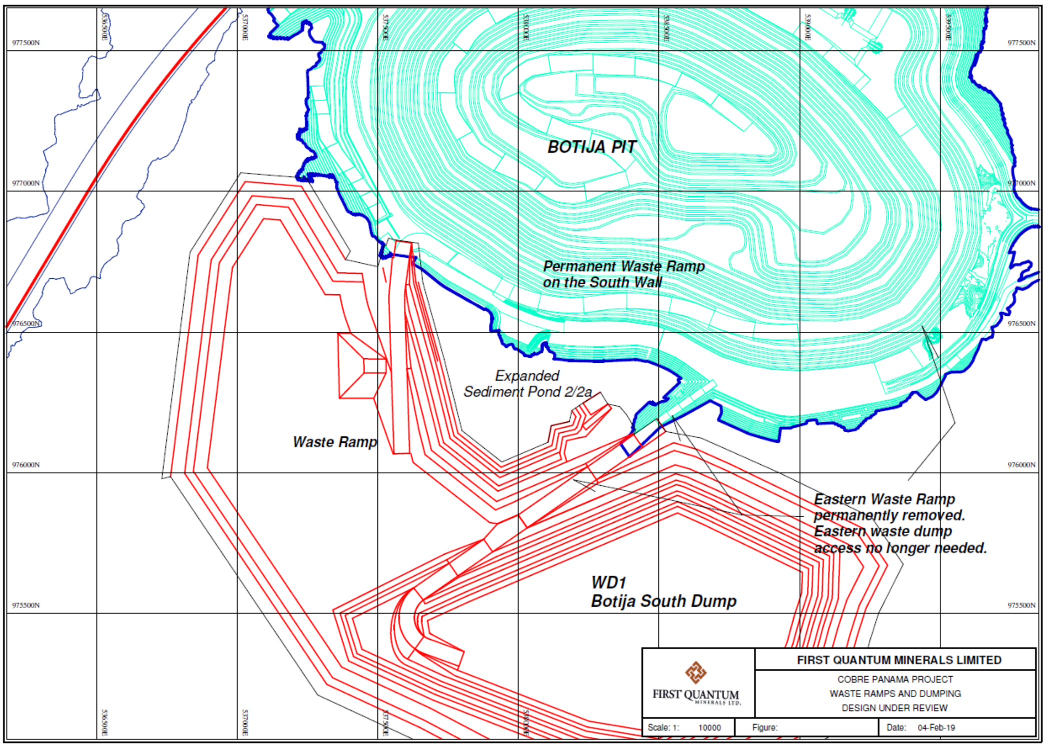
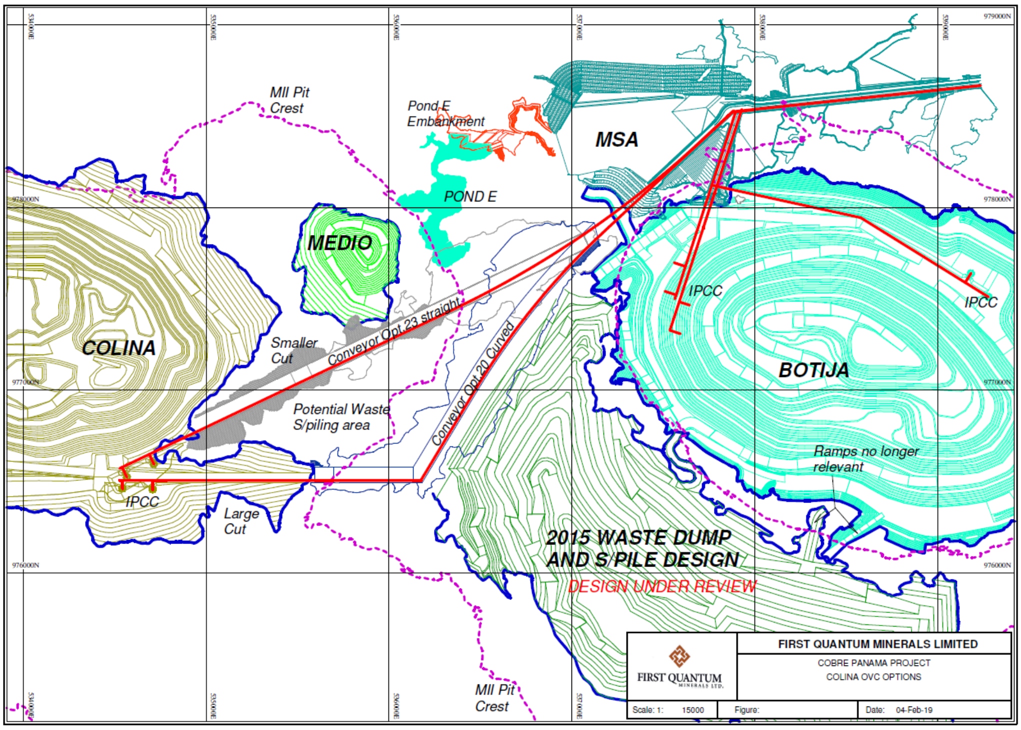


Figure 16-8 Proposed south wall design for the Botija Pit



Although not yet adopted in the current designs for the Colina Pit, concepts have been developed for an alternative overland conveyor route extending from Botija, with potential impacts on the starter pit for Colina, the location of initial IPCs and the surrounding waste dumps. A plan showing concepts currently being evaluated are shown in Figure 16-9.

Figure 16-9 Future Colina Pit access and development concepts



The current overland conveyor design route, extending across to the future Colina starter pit and initial IPC locations, follows a broadly curved path which is intended to skirt the perimeter that could apply if the Colina Pit was ever expanded to include Inferred mineral resource. The disadvantage of this route is the conveyor length, the need for a transfer facility mid-distance, and the need for a deep cut into a ridgeline immediately to the east of Colina.

A proposed new straight-line route is shorter, does not require a transfer facility and necessitates a shallower cut into the ridgeline. Mine planning work continues on the evaluation of this proposal. Trade-off analyses will consider the value of future Inferred mineral resource that would be sterilised if the straight-line conveyor route was adopted.

During 2019, sterilisation drilling is planned to commence on the Colina conveyor options and proposed in-pit crusher locations.

16.2.4 Mine dewatering

A hydrogeological baseline investigation was completed as part of the ESIA (Golder, 2010) process. This involved the drilling and aquifer testing of shallow vertical bores (to 50 m depth) in the vicinity of infrastructure facilities, and drilling/testing of deeper bores within the footprint of the Botija Pit. In addition to collecting static water level readings from exploration drill holes, Golder installed twelve vibrating wire piezometers (VWPs) into the five cored drill holes at Botija. Since the work of Golder, many of the installed piezometers have been destroyed by site construction works and prestrip mining.

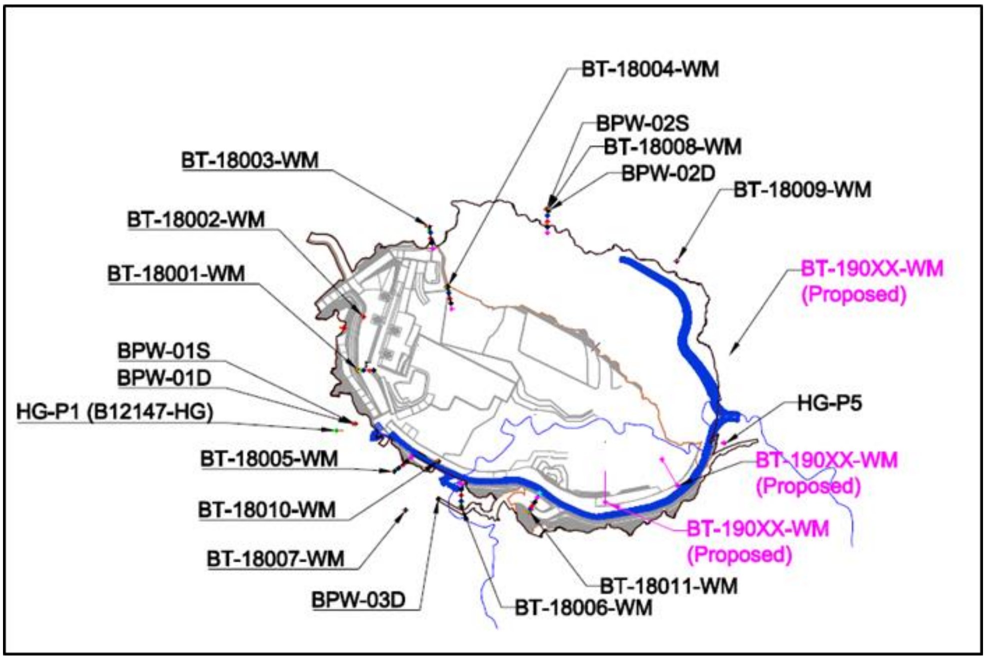
Initial groundwater modelling was completed to assess the impact on regional groundwater levels due to open pit mining and potential seepage from the TMF. For the area adjacent to the Botija Pit, Golder determined that there is moderate hydraulic conductivity in the upper 100 m of the deposit, comprising alluvium, saprolite, saprock and upper bedrock, and a relatively low hydraulic conductivity in the granodiorite and andesite bedrock below 100 m.

Piezometers

Since mid 2018, twelve boreholes ranging from 300 m to 400 m in length have been drilled. Each is equipped with three to five VWPs. Overall, a total of 59 VWPs are installed in these boreholes. Figure 16-10 illustrates the location of the existing and proposed piezometers. The figure also shows the locations of previously installed water-wells (ie, dewatering bores, shown with BPW designations). HG-P1 and HG-P5 are the only two remaining piezometer bores installed by Golder.

The typical hydrographic response of VWPs installed along the western side of the Botija Pit is a strong downward hydraulic gradient and a rapid response to ranfall events; there appears to be no pressure build-up in the pit walls.

Figure 16-10 Map showing the locations of existing and proposed piezometers and water wells for the Botija Pit



Dewatering bores

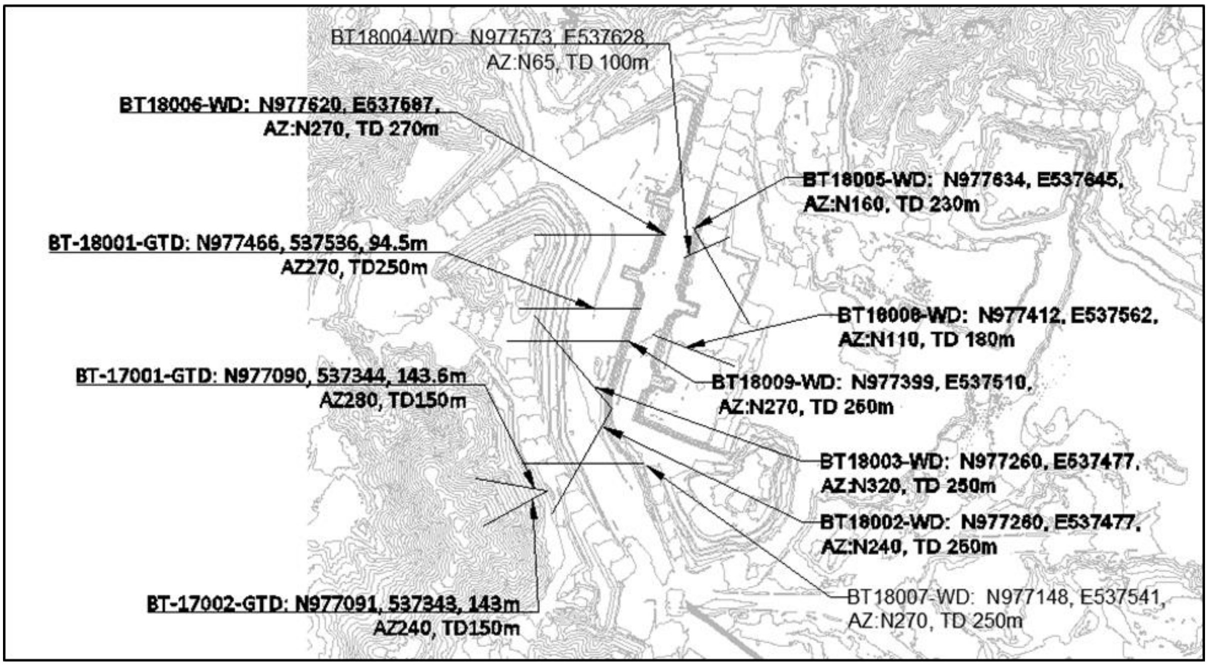
Two previously drilled production bores (BPW-01D and BPW-03D in Figure 16-10) are available for pump testing. There is a significant network of VWPs proximal to these bores, hence new long term tests are planned for 2019, to the extent that submersible pumps and ancillary equipment have been purchased.

Despite this planned pump testing, it is envisaged that vertical dewatering bores will not become the primary means of controlling groundwater inflow to the Botija Pit. Rather, it is expected that several bores only will be required, and these would be located in critical areas such as below sediment pond 2a.

Horizontal drains

A number of horizontal drains were drilled in the Botija starter pit during 2017 to 2018. These drains vary in length from less than 100 m and up to 250 m; their locations are shown in Figure 16-11. The experience of these drains is that they are effective in draining the benches where installed, and especially so after rainfall events. The widespread use of such drains was modelled and simulated by NewFields (2016) when evaluating their effectiveness in producing the drained pit slope conditions upon which the mine geotechnical consultants (CNI) based their recommended pit slope design parameters.

Figure 16-11 Installed horizontal drains, Botija Pit



General observations regarding these horizontal drains are as follows:

- the upper granodiorite is well-jointed and can transmit significant groundwater
- groundwater recharge is transmitted rapidly to the pit
- the drains are effective at capturing water prior to it reaching pit slopes
- as the Botija Pit develops it is likely that a significant annual programme will be required

A further ten to fifteen horizontal drains are proposed to be drilled along the south wall of Botija during 2019.

Summary

Considering the more recent work conducted to date and in relation to current operating horizons, the following is observed in respect of the Botija Pit:

- groundwater flow is localised to the pit; there is no observed regional groundwater flow
- groundwater domain divides can be approximated to surface water divides
- there is virtually no groundwater storage in any of the site rocks; the site consists of low porosity crystalline rocks
- local groundwater recharge flows quickly towards the pit and flows are estimated to be in the range of:
  - peak groundwater inflow = ~150 to 180 L/s
  - average groundwater inflow = ~78 to 85 L/s
  - groundwater base-flow to the pit = ~40 to 70 L/s
- future groundwater inflows into the Botija Pit will increase roughly in proportion to the pit circumference

- there appears to be no evidence for the need of a large scale conventional water well "groundwater dewatering" programme for the Botija Pit, as this is primarily a surface water management project
- this does not preclude the drilling and installation of several test wells to assure that pit-slopes are adequately dewatered/depressurised
- groundwater characteristics below the anhydrite contact are less understood, although it is foreseen that:
    - the anhydrite contact is likely to be a permeability boundary
    - more competent rock is expected below the anhydrite contact
    - deeper depressurisation efforts will likely focus on major structures
- until sufficient VWP monitoring records are available, further numerical modelling is considered to be unwarranted

It is expected that the installation of piezometers, horizontal drains and dewatering test wells will be on-going throughout the life of Botija, particularly at lower mining horizons, and for all of the other pits.

16.2.5 Pit water management

The water that is expected to be piped away from horizontal drains and pumped from dewatering bores is not expected to be material in the context of the significant runoff into the pits from rainfall.

One of the reasons for moving away from the phased mining approach as described in the 2015 Technical Report (FQM, July 2015), was the recognition that the sequence of pit cutbacks led to the formation of constained working areas into which ultra-class mining equipment would be deployed. Moreover, these constrained spaces would have the unintended consequence of creating deep sumps which when inundated, could curtail all production until the water was cleared.

In terms of pit water management, the benefit of a terrace layout was seen to be that a low terrace could be maintained towards one side of each pit, with wide, ascending terraces then providing multiple mining horizons for the ultra class equipment. In the event of a high intensity or prolonged rainfall event, the low terrace could act as a sump and be pumped clear over a duration not impacting on production from the higher terraces.

Since the 2015 Technical Report (FQM, July 2015) the proposed arrangement for staged pumping to surface has changed from the then envisaged concept of vertical lifts at 90 m intervals into intermediate sumps or day tanks positioned on wide berms. Details on the proposed new arrangement are described below.

Short term pit water management

In the more immediate term, during the mining of the Botija Pit starter pit when a low terrace is yet to be developed, the strategy has been to excavate sumps which ensure that the inpit crusher pockets and beltway are never flooded. Figure 16-12 shows the location of sumps B1 and B2 that were designed to protect this pit infrastructure. Sump B3 is a waste quarry mined for construction materials.

The catchments feeding into the B1 and B2 sumps were relatively small during the duration of starter pit mining. The mined capacity of these were 620,500 m^3 and 438,500 m^3, respectively, each with installed pumping capacity sufficient to prevent the crusher pockets and beltway from being flooded.

Figure 16-12 Starter pit drainage sumps, Botija Pit

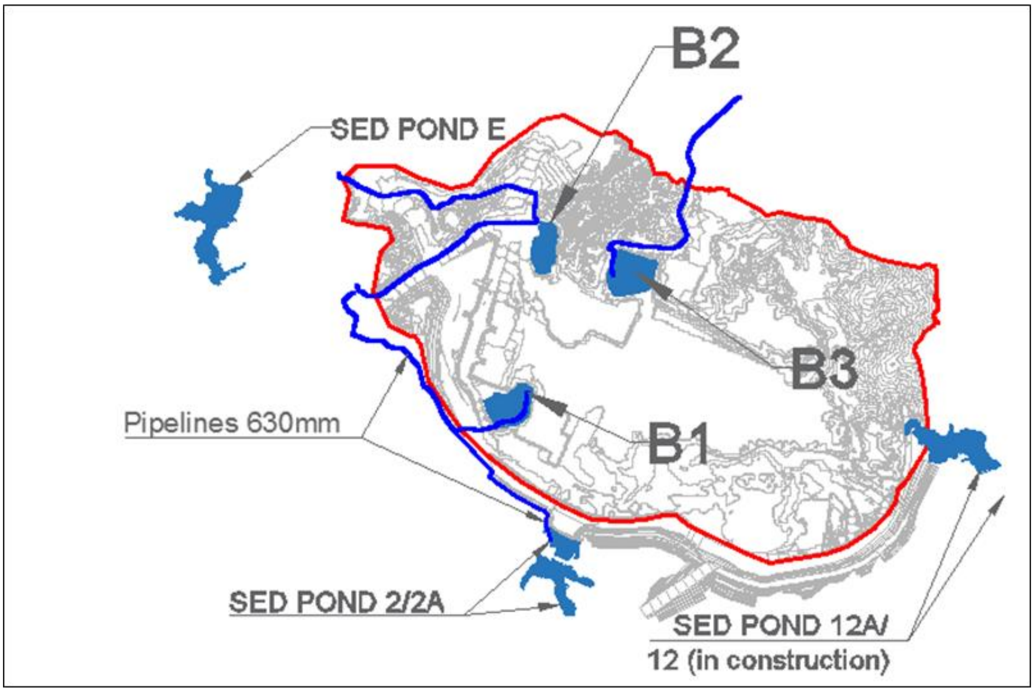


Figure 16-12 also shows pipelines leading away from each sump. Contact water collecting in sumps B1 and B2 has been directed towards the del Medio catchment (sediment Pond E), whereas contact water collecting in sump B3 has been directed northwards into the TMF basin. Contact water from these sources does not find its way into the Botija River. The significance of where pit water is directed to is explained in Item 16.3.5.

Longer term pit water management

Figure 16-13 shows the concept for managing pit water as the starter pit expands out into the initial terraces formation. Features to note are:

- by the end of 2019 the original B1 sump would likely be mined out, to be replaced by new B1 sump(s) where shown on the developing terraces
- the B2 sump at this time would remain in place, continuing to protect the crusher pockets and beltway from rainfall runoff in the northern pit catchment
- the proposed pipelines serving sumps B2 and B3 would be unchanged, whereas new pipelines would be established along ramps on the south wall leading up to surface and then to a discharge point in the del Medio catchment

Figure 16-13 Proposed pit drainage sumps at the end of 2019, Botija Pit

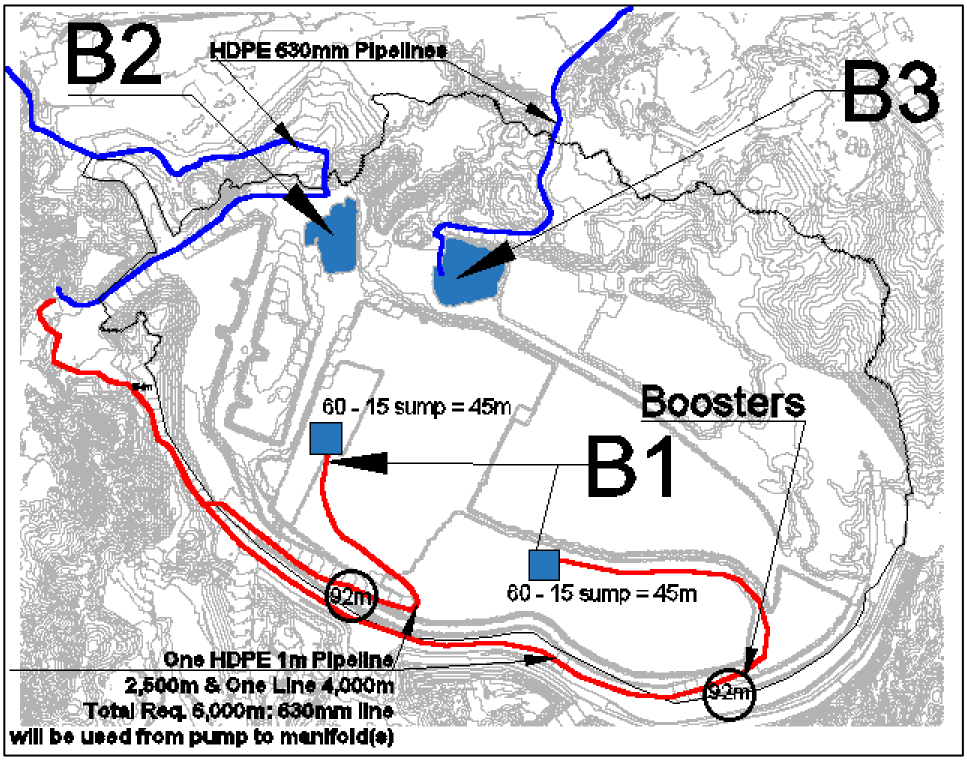


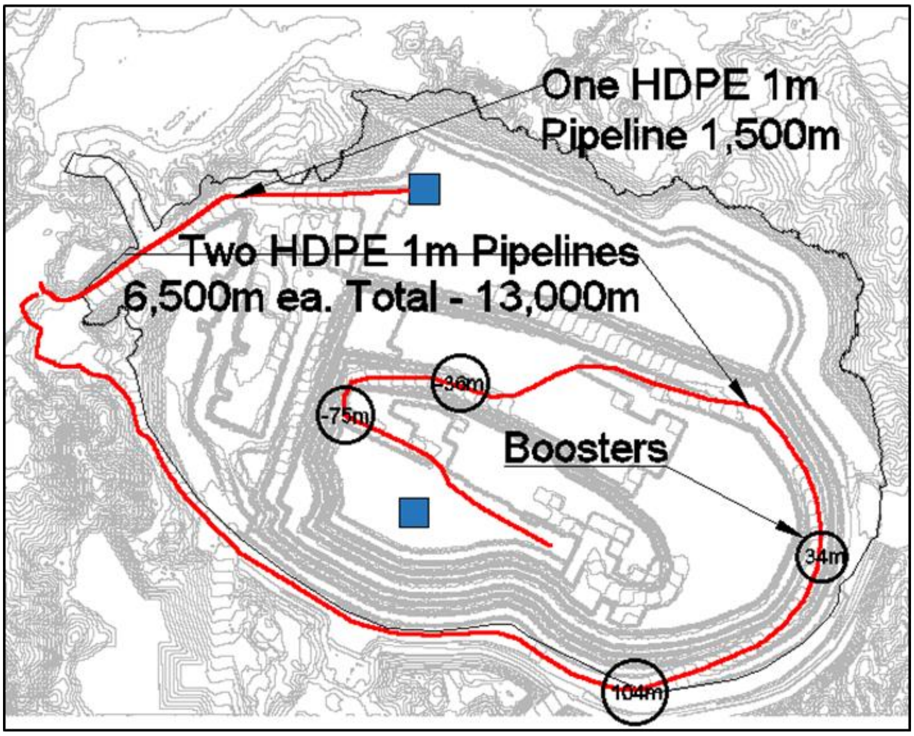
Figure 16-14 shows a notional pit outline at the end of five years, in 2023. At this stage of terrace development:

- sumps B3 and the original B2 would have been mined out
- terraces on the north side of the pit would be formed, and a new B2 sump positioned where shown; a pit sump on this side of the pit would continue to be created as the pit deepens and the northern pipeline would be extended to suit
- the B1 sump would have moved to a lower elevation and the pipelines extended as shown
- this sump would be positioned below the floor of the lowest terrace and in the event that it was overtopped following a high intensity or prolonged rainfall event, the sump pumping rate would need to be sufficient to remove water from the flooded terrace over a timeframe which is deemed to be tolerable
- in the meantime, production faces would remain accessible on higher terraces

The arrangement as shown in Figure 16-14 would be replicated as the pit is deepened, with in-line booster pumps installed to cater for the increasing head. A similar pit water management strategy is envisaged for the other Cobre Panamá pits.

An alternative strategy is also being evaluated involving booster pumping up the Botija Pit south wall, with the advantage of less piping being required.

Figure 16-14 Proposed pit drainage sumps at the end of 2023, Botija Pit



Dewatering sump pumps deployed in the Botija Pit will be both electric (for reliability) and diesel (for flexibility).

16.2.6 Surface water diversion

Diversion of the Botija River is required along the upper south wall of the Botija Pit as shown in Figure 16-15. A 30 m wide diversion channel has been incorporated into the pit design crest below the 105 mRL level.

Figure 16-15 shows the excavation chainage at 50 m increments along the length of this channel. As at the end of December 2018, the channel had been completed between chainages 59 to 46, and earthworks were in progress between chainages 1 to 14, and 31 to 34. Figure 16-16 shows a survey pick-up of progress on the channel, whilst Figure 16-17 is a photograph looking east towards chainages 1 to 14.

Associated with the Botija diversion channel design, is the construction of a sediment and water control pond, and an embankment, behind the pit southern crest. This facility is coincident with the gully of the Botija River (the circled area in Figure 16-15). The purpose of these ponds and embankment is to control the run-off from the upstream waste dump, settle out the solids, allow controlled decant into the diversion channel, and to enable contact water to be captured near to source and then pumped to the del Medio catchment.

Figure 16-15 Diversion channel design, Botija Pit south wall crest

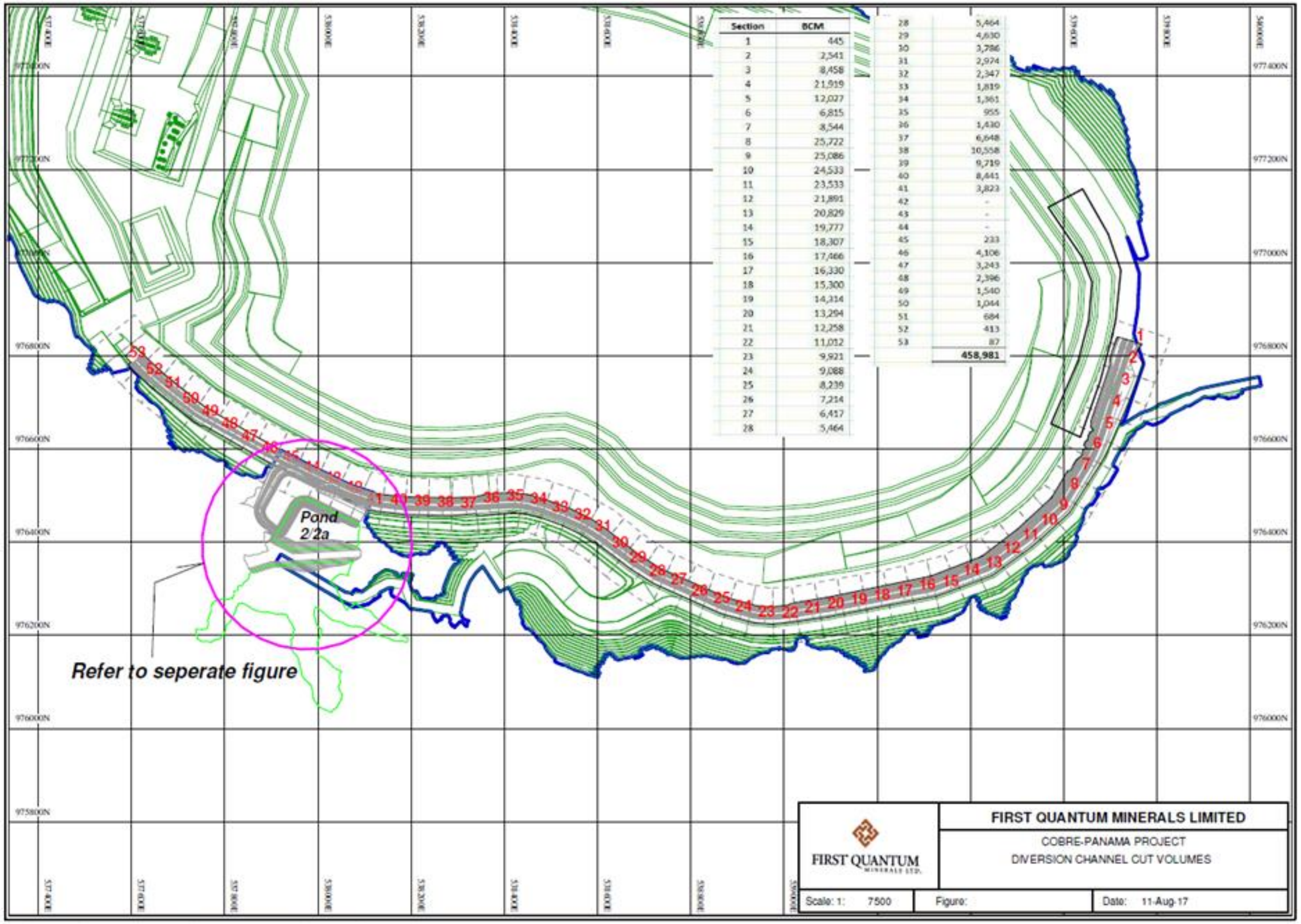


Figure 16-16 Diversion channel progress, Botija Pit south wall crest, as at December 2018

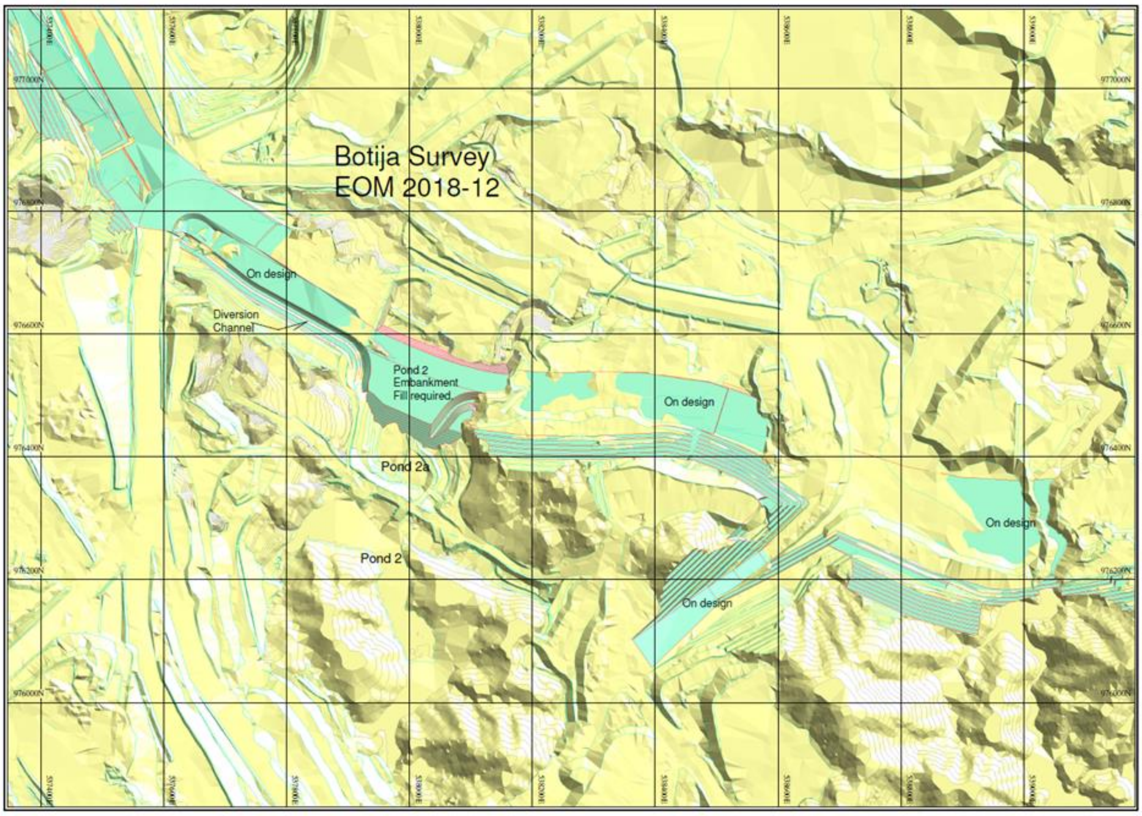
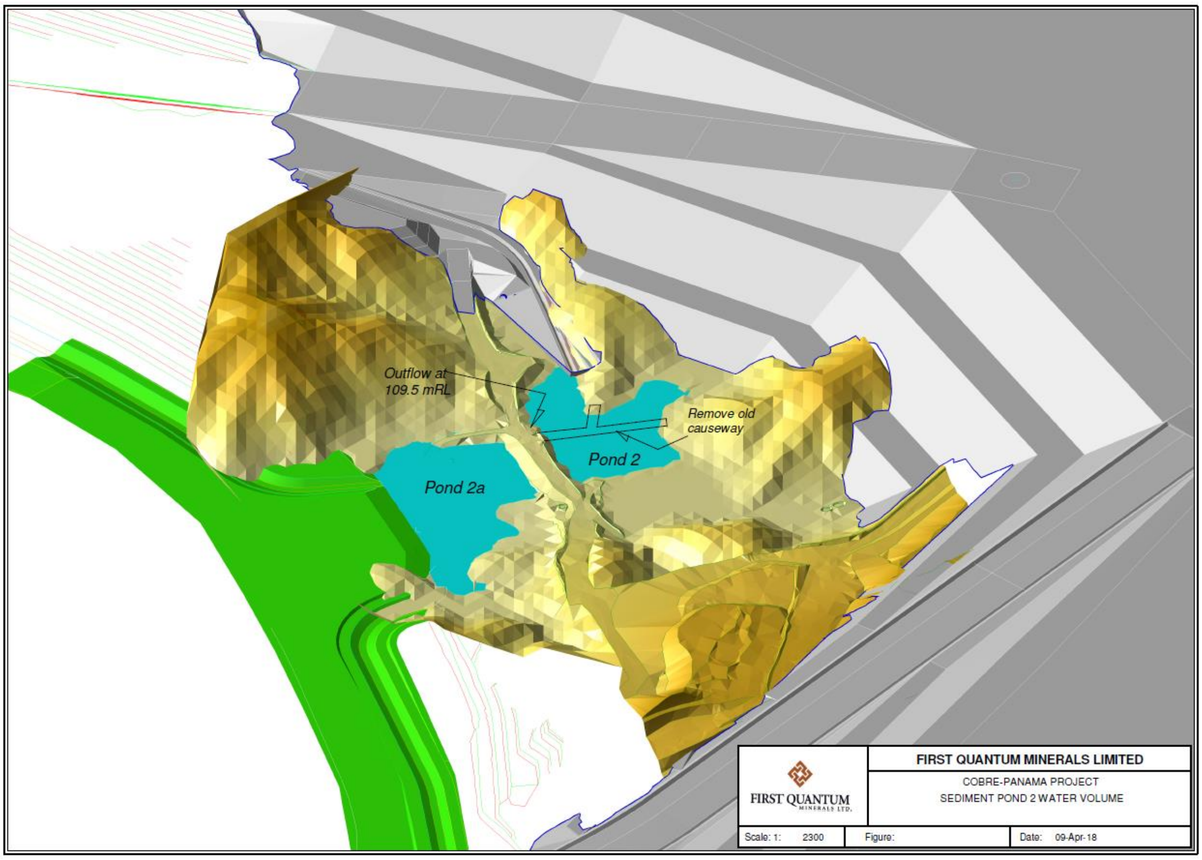


Figure 16-17 View looking east along Botija Pit diversion channel

Construction details differ slightly from the design shown in Figure 16-18, nevertheless there are essentially two ponds. The upper pond receives run-off directly from the waste dump and allows solids to be settled out. Clear water overflows from the upper pond to the lower pond and from there into the diversion channel. Clear water flowing along the western end of the channel also flows into the lower pond. Pumps installed at the lower pond allow water levels and overflow to the eastern channel to be minimised. The pumped water is piped to the del Medio catchment.

Figure 16-18 Botija sediment pond 2 and 2a



In future, diversion of the Petaquilla River will be required along the upper southwest wall of the Colina Pit. Whilst hydrological studies and engineering have not yet been undertaken for the Colina mining area, it is expected that a similar surface water management strategy will be adopted there.

16.3 Mining and processing schedules

With the completion of the detailed ultimate pit designs, detailed life-of-mine (LOM) production scheduling was completed using MineSched software. Scheduling assumptions included:

- minimum mining block size (x, y, z) for Botija = 100 m x 100 m x 15 m
- minimum mining block size for all other pits = 200 m x 125 m x 15 m
- mining flitch height = 15 m
- maximum vertical advance rate = 60 to 75 m/year
- terrace mining with horizontal lag distance of 100 m to 200 m

The schedule level of detail is monthly for 2019 and 2020, quarterly for 2021 to 2023, and annually thereafter.

16.3.1 LOM schedule

Features of the LOM mining and production schedule as listed in Table 16-5 are as follows:

- Mining (ie, the pre-strip period) commenced in July 2015 and processing commences in Q1 2019. From 2015, the Project life is 39 years to 2054.
- The total material to be mined from all pits from the end of 2018, amounts to 6,306.1 Mt (2,436.7 Mbcm), of which 3,143.7 Mt is ore (including high grade ore, medium grade ore, low grade ore and saprock ore) and 3,157.8 Mt is waste (including saprolite, saprock waste and mineralised waste).
- The starting stockpile balance as at the end of 2018 is 3.3 Mt at a grade of 0.22% Cu.
- The direct feed ore is 2,813.1 Mt at a grade of 0.39% Cu and 334.0 Mt at a grade of 0.21% Cu is active and long term stockpile reclaim.
- The direct feed ore from the open pits is 89% of the total plant feed tonnage.
- The total high grade and medium grade ore mined to active stockpiles and reclaimed throughout the Project life is 45.7 Mt at a grade of 0.31% Cu.
- In addition, 239.9 Mt of low grade ore at a grade of 0.17% Cu is mined to stockpile and reclaimed in the final ten years of the Project.
- 48.4 Mt of saprock ore is mined to stockpile from the outset of mining, but not reclaimed (with a degraded recovery assignment) until the final two years of the Project.
- The crusher feed ramps up from 47 Mtpa in 2019 to 85 Mtpa in 2021, and ultimately to 100 Mtpa in 2023 at which rate it remains until 2041.
- The rate drops to 75 Mtpa between 2042 and 2053.
- The average annual copper metal production in the first five years is 310.0 ktpa. For the next five years, when processing at 100 Mtpa, the average annual copper metal production rises to 377.0 ktpa. Thereafter, the annual average is 275.1 ktpa.
- The annual average by-product production is approximately 2,717 tonnes of molybdenum, 107 thousand ounces of gold and 1,701 thousand ounces of silver.
- The overall life of mine strip ratio (tonnes) is 1 : 1.

Figures 16-19 to 16-23 depict the LOM schedule graphical results.

Table 16-5 LOM scheduling – summary of results

Year	Mined Ore (Mt)	Grade (% Cu)	Metal (kt Cu)	Crusher Feed (Mt)	Grade (% Cu)	Metal (kt Cu)	Cu Metal Recovery (%)	Cu Metal Recovery (kt Cu)	HG/MG Stockpiles On (Mt)	HG/MG Stockpiles Off (Mt)	HG/MG Stockpiles Bal. (Mt)	LG Stockpiles On (Mt)	LG Stockpiles Off (Mt)	LG Stockpiles Bal. (Mt)	Saprock Ore Stockpiles On (Mt)	Saprock Ore Stockpiles Off (Mt)	Saprock Ore Stockpiles Bal. (Mt)	Waste (Mt)	Waste Strip Ratio	Ore+Waste (Mbcm)	Ore+Waste (Mt)
<2019	3.3	0.22	7.4						3.3												
2019	76.0	0.31	235.9	47.2	0.38	177.6	81.5	144.8	17.8	9.8	11.4	22.0	5.2	16.9	3.8		3.8	62.7	0.83	57.3	138.7
2020	94.2	0.36	340.8	81.6	0.40	323.1	91.0	293.9	2.6	10.0	4.0	19.8		36.7	0.3		4.1	74.7	0.79	67.9	169.0
2021	101.0	0.36	363.3	85.0	0.40	342.8	91.2	312.6	0.0	3.7	0.3	19.6		56.3			4.1	76.1	0.75	68.2	177.1
2022	109.8	0.40	438.3	90.2	0.45	406.8	91.9	373.7	1.4	1.2	0.5	19.4		75.7			4.1	94.2	0.86	77.8	204.0
2023	124.8	0.41	506.6	100.0	0.46	461.4	92.1	424.9	5.5	3.9	2.2	22.0		97.7	1.0		5.2	79.3	0.64	78.4	204.1
2024	109.1	0.42	455.2	100.0	0.44	436.7	91.9	401.5	0.2		2.3	6.0		103.7	3.0		8.1	100.4	0.92	81.4	209.5
2025	110.9	0.40	446.4	100.0	0.42	420.3	91.6	385.1		1.1	1.2	5.6		109.3	6.4		14.6	98.3	0.89	82.9	209.3
2026	123.8	0.38	466.3	100.0	0.41	412.5	91.2	376.0	1.4		2.6	16.1		125.4	6.3		20.9	85.3	0.69	79.9	209.1
2027	123.4	0.34	417.4	100.0	0.37	370.0	90.5	334.9	0.0	1.2	1.5	18.4		143.8	6.2		27.1	85.1	0.69	82.2	208.4
2028	108.2	0.41	445.1	100.0	0.42	423.5	91.5	387.5	2.3		3.8	3.5		147.2	2.4		29.5	100.9	0.93	80.0	209.1
2029	119.8	0.40	480.9	100.0	0.45	449.3	91.8	412.3	0.2		4.0	18.0		165.2	1.7		31.2	89.3	0.75	80.6	209.1
2030	116.3	0.38	438.6	100.0	0.42	417.3	91.3	380.8		3.3	0.6	18.3		183.5	1.3		32.4	88.0	0.76	79.4	204.3
2031	111.1	0.40	443.8	99.6	0.43	427.1	91.5	390.7		0.6	0.0	11.6		195.1	0.5		32.9	92.2	0.83	78.0	203.2
2032	102.8	0.43	446.0	100.0	0.44	439.5	91.6	402.6	1.8		1.8	0.8		195.9	0.3		33.2	101.4	0.99	78.3	204.2
2033	103.7	0.37	385.5	100.0	0.38	376.9	90.7	341.9	2.2		3.9	0.9		196.9	0.6		33.8	100.1	0.97	77.0	203.8
2034	102.8	0.37	383.8	99.7	0.38	381.6	90.8	346.3		3.9	0.0	6.9		203.7	0.1		33.9	101.4	0.99	76.9	204.2
2035	104.8	0.34	360.9	100.0	0.35	347.2	90.2	313.2	0.2		0.2	0.3		204.0	4.3		38.3	99.4	0.95	77.0	204.2
2036	110.6	0.36	395.1	100.0	0.37	372.2	90.6	337.1	2.0		2.2	7.0		211.0	1.5		39.8	94.2	0.85	77.2	204.7
2037	114.5	0.36	414.7	100.0	0.39	385.1	90.8	349.6	1.0		3.2	10.2		221.2	3.4		43.1	89.6	0.78	77.2	204.2
2038	101.6	0.38	381.6	100.0	0.38	377.1	90.8	342.4		2.0	1.1	0.8		222.0	2.8		45.9	102.6	1.01	78.7	204.2
2039	106.5	0.39	410.9	100.0	0.39	391.0	91.1	356.0	1.5		2.6	4.2		226.2	0.8		46.8	97.7	0.92	79.0	204.2
2040	105.6	0.33	347.7	100.0	0.34	340.4	90.2	307.1		1.2	1.5	6.3		232.5	0.5		47.3	99.1	0.94	80.6	204.7
2041	103.6	0.35	362.0	100.0	0.35	354.3	90.4	320.4	2.1		3.6	1.5		234.0	0.0		47.3	100.5	0.97	80.4	204.2
2042	81.9	0.33	268.4	85.0	0.33	279.6	89.7	250.9		3.4	0.2			234.0	0.3		47.5	122.2	1.49	80.7	204.1
2043	75.5	0.33	249.7	75.0	0.33	247.9	88.7	220.0	0.1		0.3	0.0		234.0	0.4		47.9	129.2	1.71	78.3	204.7
2044	63.1	0.34	214.7	75.0	0.31	233.6	88.1	205.8		0.0	0.3		11.9	222.1	0.0		47.9	130.2	2.06	77.4	193.3
2045	62.0	0.35	214.0	75.0	0.31	235.5	88.5	208.4		0.3	0.0		13.1	209.0	0.4		48.3	131.2	2.12	74.2	193.1
2046	50.7	0.36	180.5	75.0	0.29	219.2	89.2	195.5	0.0		0.0	0.7	25.0	184.7	0.0		48.3	131.7	2.60	72.7	182.5
2047	66.4	0.35	234.5	75.0	0.33	248.0	89.1	220.9		0.0			8.7	176.0	0.1		48.4	136.3	2.05	78.4	202.7
2048	66.3	0.39	256.7	75.0	0.36	270.6	89.4	241.9					8.7	167.3			48.4	109.4	1.65	67.1	175.7
2049	61.0	0.36	219.9	75.0	0.32	242.2	89.1	215.7					14.0	153.3			48.4	65.9	1.08	47.8	127.0
2050	56.7	0.40	229.4	75.0	0.34	258.6	89.5	231.3					18.3	135.0			48.4	37.0	0.65	35.4	93.7
2051	37.4	0.39	147.0	75.0	0.28	207.0	88.6	183.4				0.0	37.6	97.5			48.4	36.8	0.98	28.0	74.2
2052	27.2	0.31	84.7	75.0	0.21	161.0	88.1	141.8					47.8	49.7			48.4	14.6	0.54	15.8	41.8
2053	10.7	0.55	58.8	75.0	0.25	185.9	84.0	156.1					49.7			14.7	33.8	0.8	0.07	4.4	11.4
2054	0.0			33.8	0.33	110.3	71.8	79.2								33.8					
Diluted	**3,147.1**	**0.37**	**11,732.7**	**3,147.1**	**0.37**	**11,732.7**	**90.2**	**10,586.4**	**45.7**	**45.7**		**239.9**	**239.9**		**48.4**	**48.4**		**3,157.8**	**1.00**	**2,436.7**	**6,301.6**

Figure 16-19 LOM scheduling – annual material movement tonnes

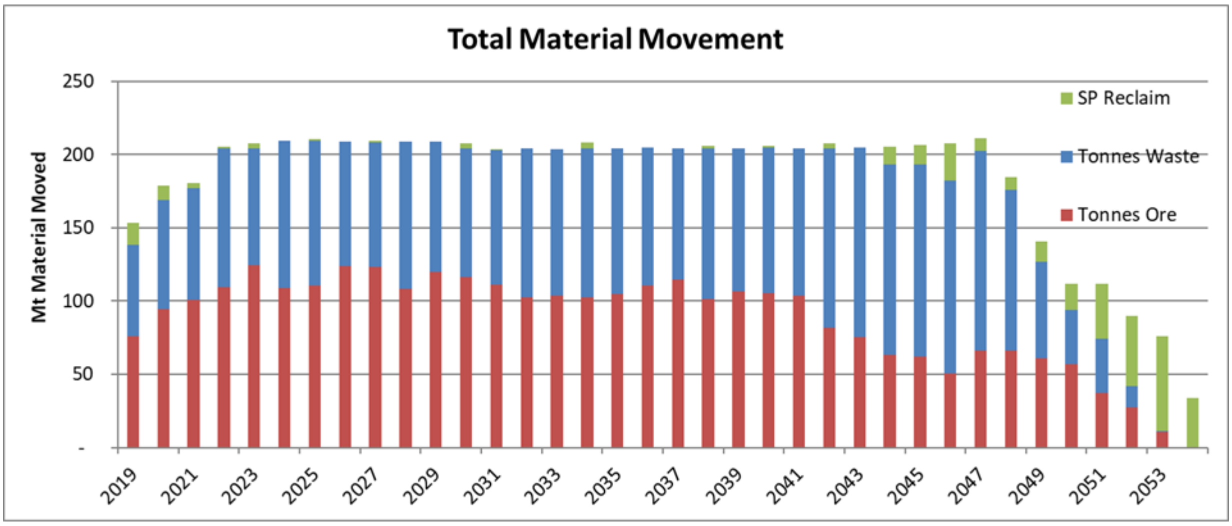


Figure 16-20 LOM scheduling – annual plant feed tonnes

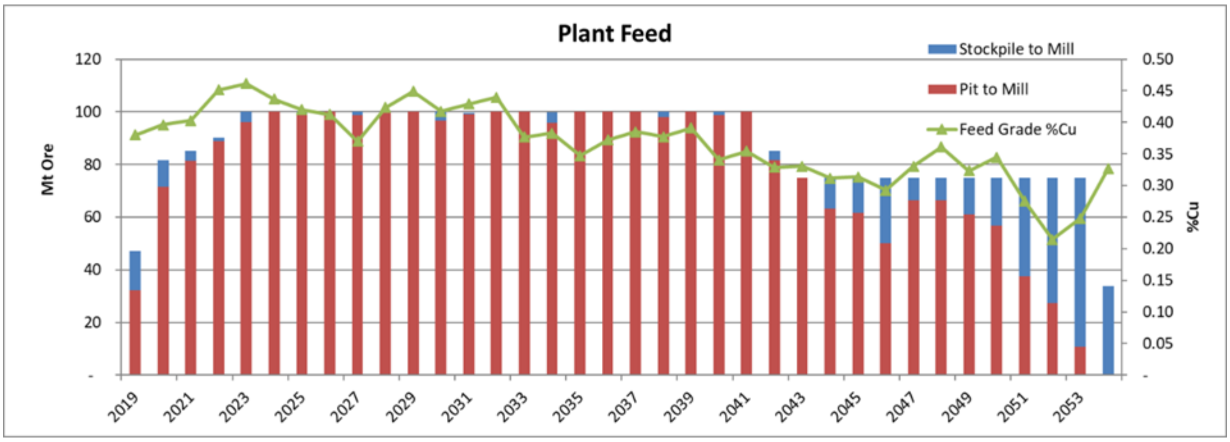


Figure 16-21 LOM scheduling – annual copper metal production

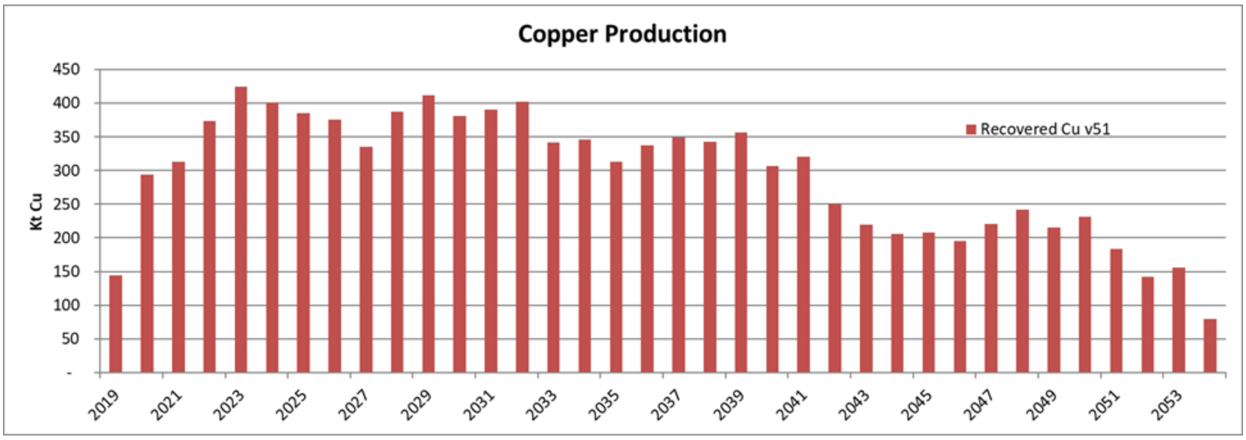


Figure 16-22 LOM scheduling – annual stockpile balance tonnes

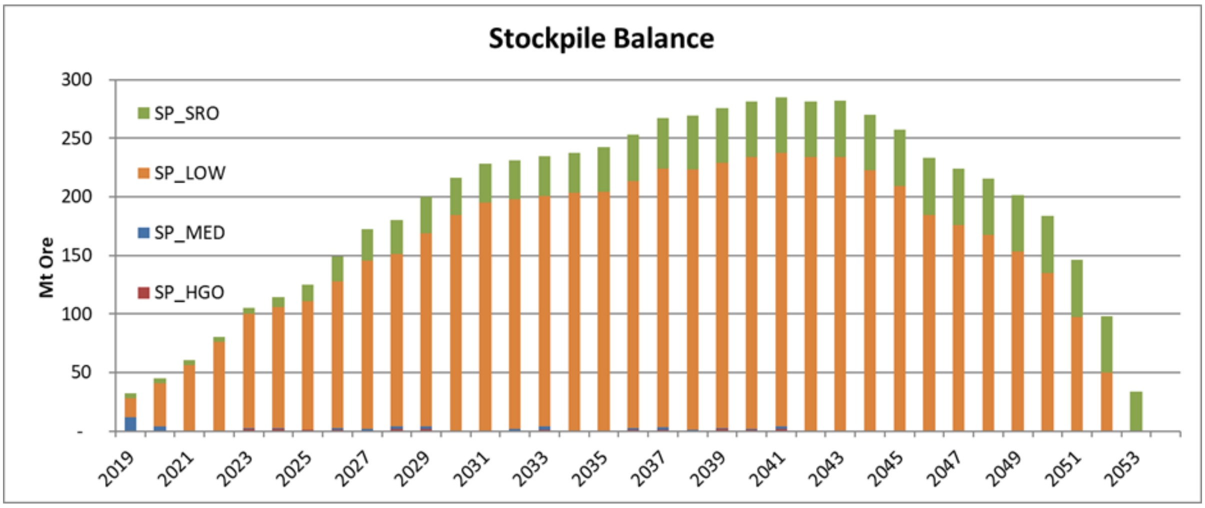
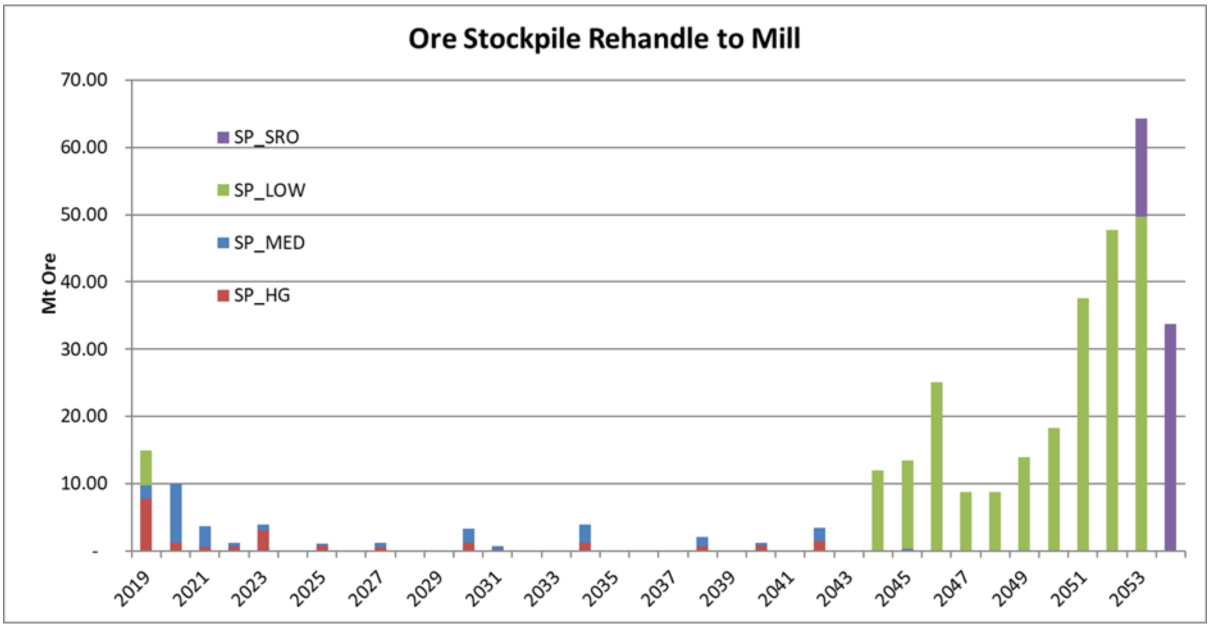


Figure 16-23 LOM scheduling – annual ore stockpile tonnes reclaimed into plant



16.3.2 Mining sequence

The sequencing concept for mining of the several Cobre Panamá pits is dictated by the relative grade distribution across the deposits, the location of innitial mining in close proximity to the processing plant (to minimise initial overland conveying infrastructure), the minimisation of initial waste stripping, the orderly relocation of IPCs, and the achievement of as short a capital pay-back timeframe as possible.

Hence, relative to this concept:

1. The Botija Pit will be mined first, followed by the Colina and Medio Pits. Mining in the Valle Grande and BABR Pits will commence towards the end of Colina mining. The Balboa Pit will be mined last.

2. Prestrip mining commenced in Botija in Q3 2015, so as to provide waste for construction facilities such as the equipment assembly area, overland conveyor road and the MSA. From this time, ore mined in the pre-strip excavations was stockpiled for reclaim at the commencement of processing.
3. As part of the pre-strip, crusher pockets within the Botija boxcut were mined to enable the first IPCs to be available by early 2019, for first ore production in Q1 2019.
4. On the completion of the boxcut, the Botija Pit is being mined using a terrace strategy where the pit has a number of operating benches at different levels so as to enable efficient UC equipment utilisation and an ability to continue mining without being impacted by rainfall inundation.
5. For the majority of the schedule, the Botija Pit mining progression maintains a deepest point of the pit on the southern wall toe, whilst mining advance progresses northwards.
6. Ore mining within the Colina Pit will commence in 2024 after completion of development works for the installation of IPCs in that pit. It is expected that the same terrace mining concept used at Botija will also be used at Colina.
7. The Medio Pit will be mined between 2029 and 2032.
8. The Valle Grande Pit will be mined in two phases from 2032 to 2045.
9. The BABR Pits will be mined in two phases from 2042 to 2051.
10. The Balboa Pit will be mined in two phases from 2038 to 2053.

Figure 16-24 shows the mining sequence and production profile.

16.3.3 IPCC sequence

The IPCC layout plan in Figure 16-25 shows the proposed locations of the crushers. Figure 16-26 shows the number of crushers required each year and their relocations relative to the plan in Figure 16-25.

The Balboa ore would be hauled to the crushers in the Valle Grande Pit and there is assumed to be no crushers at BABR towards the end of the mine life. The IPC installations would be at the opposite end of the Project site, at Colina and Valle Grande, and hence BABR ore would be road hauled to another crusher positioned conceptually on the Botija South dump (adjacent to the Botija Pit). This same crusher could be used on LG and saprock stockpile reclaim.

Figure 16-24 Graph showing mining sequence

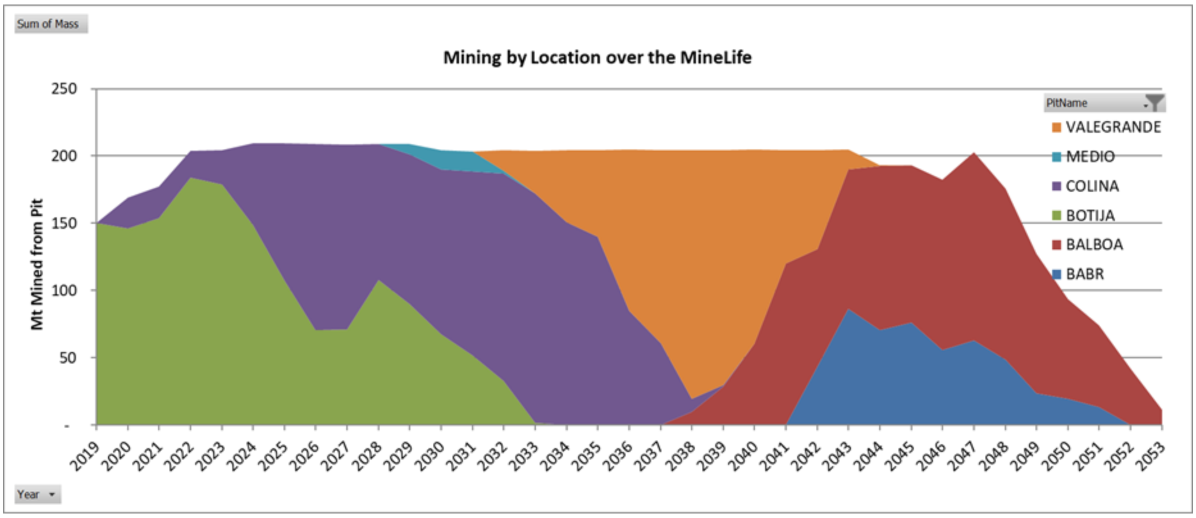


Figure 16-25 IPCC layout plan

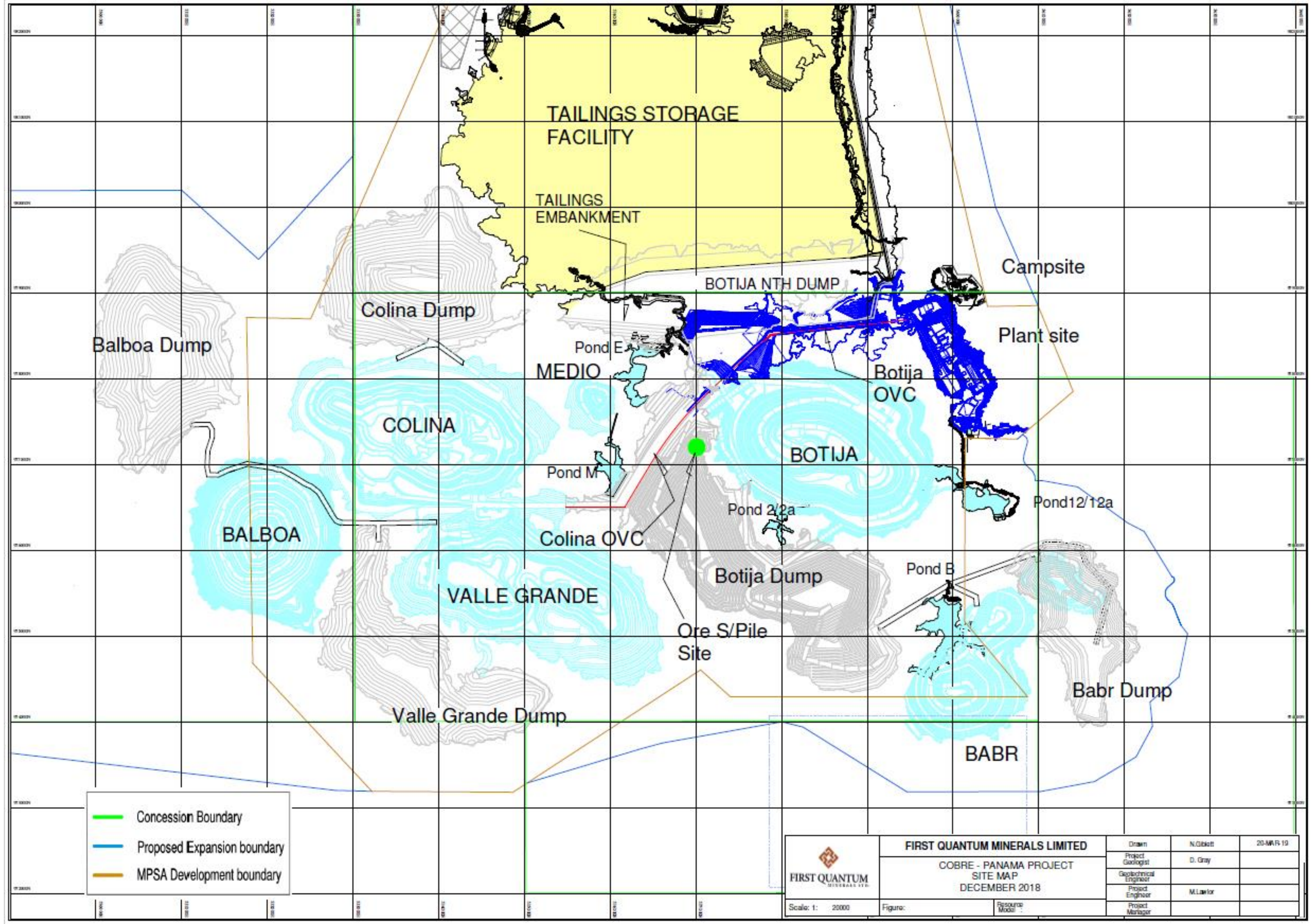
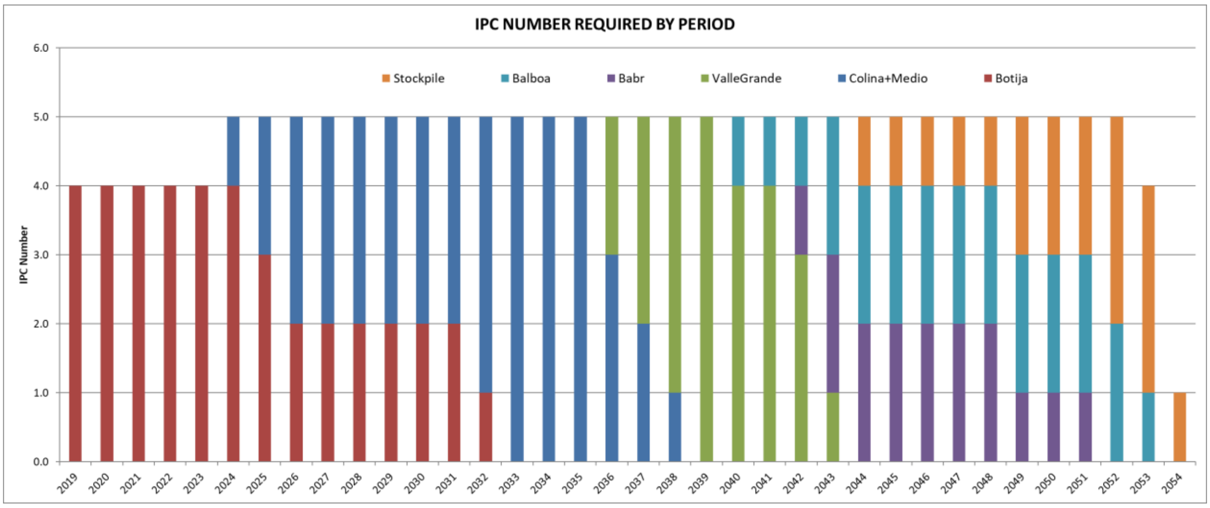


Figure 16-26 In-pit crusher requirements and relocations over time



Low grade and saprock ore that is scheduled to be processed late in the mine life will be stockpiled in a segregated location within an adjacent waste dump. The material is planned to be crushed at surface locations proximal to these long term stockpiles and to the late-mined BABR Pit.

16.3.4 Waste dumping schedule

Table 16-6 shows the waste dumping destinations relative to the site layout plan shown in Figure 15-12. The ultimate waste dump footprints have been designed to minimise clearing and water runoff control requirements. This has been done by designing within water catchment boundaries, as far as practicable. In addition, the designs make allowance for the volume occupied by stockpiled ore material that would be rehandled to the crushers.

Table 16-6 LOM scheduling – waste dump schedule

Pit	Insitu Pit Volumes				Dump Schedule Volumes	Dump Destination Volumes							
	Waste (Mbcm)	Min Wst (Mbcm)	Ore-> S/P (Mbcm)	Total Insitu (Mbcm)	Total Swelled 30% (Mblm)	Botija South (Mlcm)	Botija North (Mlcm)	Colina (Mlcm)	Valle Grande Expit (Mlcm)	BABR (Mlcm)	Balboa (Mlcm)	Bfill (Mlcm)	Total (Mlcm)
BOTIJA	201.7	26.9	70.1	298.7	388.3	299.6	88.7						388.3
COLINA	304.4	20.7	33.9	359.0	466.7			318.3	124.3		24.0		466.7
MEDIO	9.7	0.1	0.6	10.5	13.6			13.6					13.6
VALLE GRANDE	198.2	14.8	10.2	223.2	290.2				77.4		127.2	85.6	290.2
BABR	105.2	9.2	0.5	114.9	149.4	4.2				145.2			149.4
BALBOA	320.9	35.2	5.9	362.1	470.7						167.8	302.9	470.7
TOTAL	**1,140.1**	**107.0**	**121.3**	**1,368.4**	**1,778.9**	**303.9**	**88.7**	**331.9**	**201.7**	**145.2**	**319.0**	**388.4**	**1,778.9**

The general waste dumping strategy is as follows:

Botija Pit:

- In the early years of mining, waste would be hauled to the Botija North Dump and the Botija South Dump (located adjacent to the south west of the pit) until the north dump is completed.
- Thereafter, waste would be hauled to the Botija South Dump.

Colina Pit:

- Waste material would be hauled to the Colina external dump located to the north of the pit until it is full, from which time it would then be hauled to the Valle Grande external dump.

- When the Valle Grande external dump is completed, the remaining Colina waste would be hauled to the Balboa pit external dump.

Medio Pit:

- All waste would be hauled to the Colina external dump.

Valle Grande Pit:

- Valle Grande waste material would be hauled to the Valle Grande external dump located to the south east of the pit until it is full, then to the Balboa external dump until it is full, and finally to the Colina inpit backfill void once that pit is completed.

Balboa Pit:

- Waste material would be hauled to the external Balboa dump located north west of the Balboa pit until it is full.
- The remainder of the waste would then be hauled to the Colina inpit backfill.

BABR Pit:

- Initial material from the smaller BABR Pit would be hauled to the external BABR dump adjacent to the pit, and then the remaining material would be hauled to backfill the initial BABR pit and create the ultimate external dump.
- At the end of the pit life, a small amount of material would be hauled to the Boitija South dump when the BABR dump is full.

16.3.5 Waste segregation

The ESIA (Item 20) included a geochemical evaluation of the mine materials and the development of water quality models based on a site water balance and potential geochemical interactions with the open pits, ore stockpiles, waste dumps and the TMF. Ecometrix (2014) produced a report in which a geochemical characterisation, based on Acid Base Accounting (ABA) principles, is said to identify over 80% of the Botija waste rock as being potentially acid forming (PAG). In the Ecometrix report, the Valle Grande waste is said to be significantly non-PAG (NAG)[10].

The Ecometrix report provided a conceptual water management plan for the operations phase which effectively involves collection of all contact water and the containment/direction of this water into the TMF.

Since the time of the Ecometrix work, the acid rock drainage (ARD) potential for Botija ore and waste types has continued to be reviewed and strategised. Piteau Associates (Piteau, 2018) have periodically reviewed the geochemical sampling, and the static/kinetic testing that has taken place. Piteau has also carried out water quality modelling and projections. Their analysis conclusions reiterate that saprock has the greatest potential for ARD generation, and as such should be segregated from other ore stockpiles and waste types.

[10] Ore stockpiles are also described as being significantly PAG.

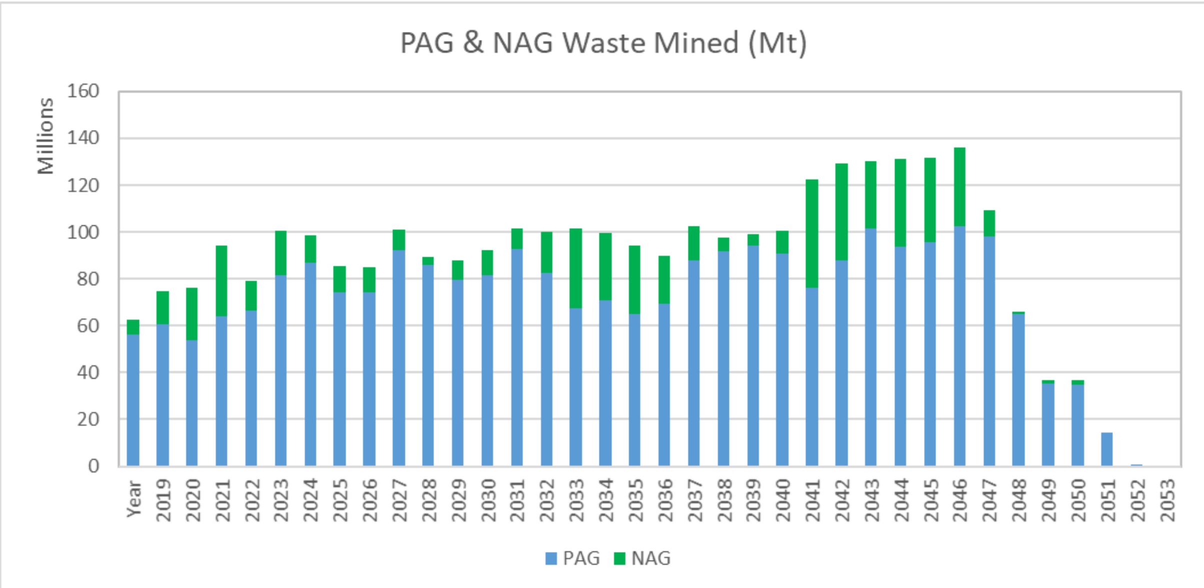
To enable the PAG and NAG volumes to be quantified during the mine planning and scheduling process, criteria based on % sulphur values carried in the Mineral Resource model were used to make the differentiation shown in Table 16-7.

Table 16-7 LOM scheduling – ARD criteria

Ore/Waste classification			
Current descriptor	**Lithology**	**SPCT criteria S% cutoff**	**ARD potential**
ORE			
Saprock Ore (SAPR_O)	> 100	>= 0.10	PAF
Fresh High Grade Ore (HIGH)	> 100 <= 900	n/a	PAF
Fresh Medium Grade Ore (MED)	> 100 <= 900	n/a	PAF
Fresh Low Grade Ore (LOW)	> 100 <= 900	n/a	PAF
WASTE			
Mineralised Waste (MARG)	> 100 <= 900	n/a	PAF
Saprolite (SAP)	100	n/a	NAF
NAF Saprock Waste (SAPR_N)	> 100	< 0.10	NAF
PAF Saprock Wate (SAPR_P)	> 100	>= 0.10	PAF
NAF Fresh Waste (NAF)	> 100 <= 900	< 0.30	NAF
PAF Fresh Waste (PAF)	> 100 <= 900	>= 0.30	PAF

Table 16-8 shows the life of mine waste schedule, in terms of PAG and NAG volumes defined by the % sulphur criteria. Continuing updates to the mine plan and site layout will consider the segregation of PAG and NAG waste and the control of run-off from the waste dumps.

Table 16-8 LOM scheduling – PAG and NAG waste volumes



16.3.6 Saprock ore and waste

Piteau (2018) expresses the view that negligible values of copper will remain in saprock ore that is stockpiled for decades. This is despite the fact that the Mineral Reserves inventory includes long term saprock ore stockpiles, for which reclaim is viable after a 20% reduction in processing recovery that is intended to reflect degradation over time.

Whether saprock ore should be consigned to the waste dump from the outset is an aspect that is considered in the Project economic analysis (Item 22.4).

16.3.7 Mining equipment

Table 16-9 lists the primary mining fleet and support equipment for the Project[11]. The primary mining fleet will comprise the following:

- up to 13 x rotary electric drills capable of drilling 270 mm to 311 mm diameter blastholes
- up to 5 x 63 m^3 P&H 4100XPC-(AC) (or equivalent) electric rope shovels
- up to 45 x Liebherr L284 (or equivalent), 360 t capacity haul trucks

The secondary (ancillary) mining fleet fleet will comprise:

- up to 3 x 350 t weight diesel hydraulic excavators
- up to 4 x 120 t weight diesel hydraulic excavators
- up to 8 x diesel powered drills for mining on small benches and for presplitting
- 40 t articulated trucks (for lesser material movements, saprolite/saprock mining)
- 100 t mechanical drive trucks (for lesser material movements, saprolite/saprock mining)

In addition to the primary equipment listed above, adequate major mining support equipment (track and wheel dozers, graders, service trucks etc) will be in place at Cobre Panamá.

Table 16-9 Mining equipment

Mining Equipment	Manufacturer	Model	2019 to 2028		2029 to 2038		2039 to 2048		2049 to end	
			Average #	Maximum #	Average #	Maximum #	Average #	Maximum #	Average #	Maximum #
Drills										
Development	Furukawa	HCR1500	5	5	5	5	5	5	2	2
Presplit/Trim	Epiroc	D65	3	3	3	3	3	3	2	3
Production	Epiroc	PV271	7	7	7	7	7	7	3	5
Production	Epiroc	PV351/MD6640	5	6	6	6	6	6	3	4
Excavators/Shovels										
Diesel Excavator	Liebherr	9100	4	4	4	4	4	4	1	2
Diesel Excavator	Liebherr	9350	3	3	3	3	3	3	1	1
Electric Shovel	P&H	4100XPC	5	5	5	5	5	5	2	4
Trucks										
40t capacity	Caterpillar	740B	24	24	24	24	23	24	7	12
90t capacity	Caterpillar	777G	18	18	18	18	17	18	6	6
363t capacity	Liebherr	L284	30	35	39	45	44	45	21	36
Ancillary Equipment										
FEL	LeTourneau	L2350	3	3	3	3	3	3	2	2
Track Dozer	Caterpillar	D10	6	6	6	6	6	6	4	5
Track Dozer	Caterpillar	D11	4	4	4	4	4	4	2	3
Wheel Dozer	Caterpillar	834	4	4	4	4	4	4	3	3
Wheel Dozer	Caterpillar	854	3	3	3	3	3	3	3	3
Grader	Caterpillar	16M	7	7	7	7	7	7	4	5
Water Truck	Caterpillar	785	3	3	3	3	3	3	3	3

In addition to the items listed in Table 16-9, other mining related equipment for the Project includes:

- telescopic handlers

[11] Mining equipment manufacturer/model to be read as “or equivalent”.

- crew buses and light vehicles for mine operations and maintenance personnel
- fixed and mobile crushers for blasthole stemming and roadstone production
- front end loaders, wheeled dozers and vibratory compactors for mine road construction and maintenance
- cable tractors, cable trees and cable handling equipment
- 150 t and 400 t lowbed trailers and prime movers
- tool and equipment trucks for maintenace of field mine fleet
- stemming trucks
- mobile mixing units for blasting operations (contractor supply)
- pit shovel motivator
- lighting towers
- diesel and electric sump dewatering pumps

ITEM 17 RECOVERY METHODS

The following information is largely reproduced from Rose et al (2013), with updates provided by Robert Stone (QP) of FQM.

17.1 Mineral processing overview

Ore from the several open pits will be treated in a conventional process plant to produce a copper concentrate which will be pumped to the port, filtered and then loaded onto ships destined for world markets. Additionally, a molybdenum concentrate will be produced which will be filtered and bagged in the process plant before containerisation for export.

Aside from in-pit primary crushing, the processing plant will include conventional facilities, such as:

- crushing (secondary and pebble) and grinding (SAG/ball) to liberate minerals from the ore
- froth flotation to separate most of the copper and molybdenum minerals from minerals of no commercial worth
- differential flotation to separate the copper and molybdenum minerals from each other
- storage of tailings and provision of reclaim water for the process
- removal of water from the products

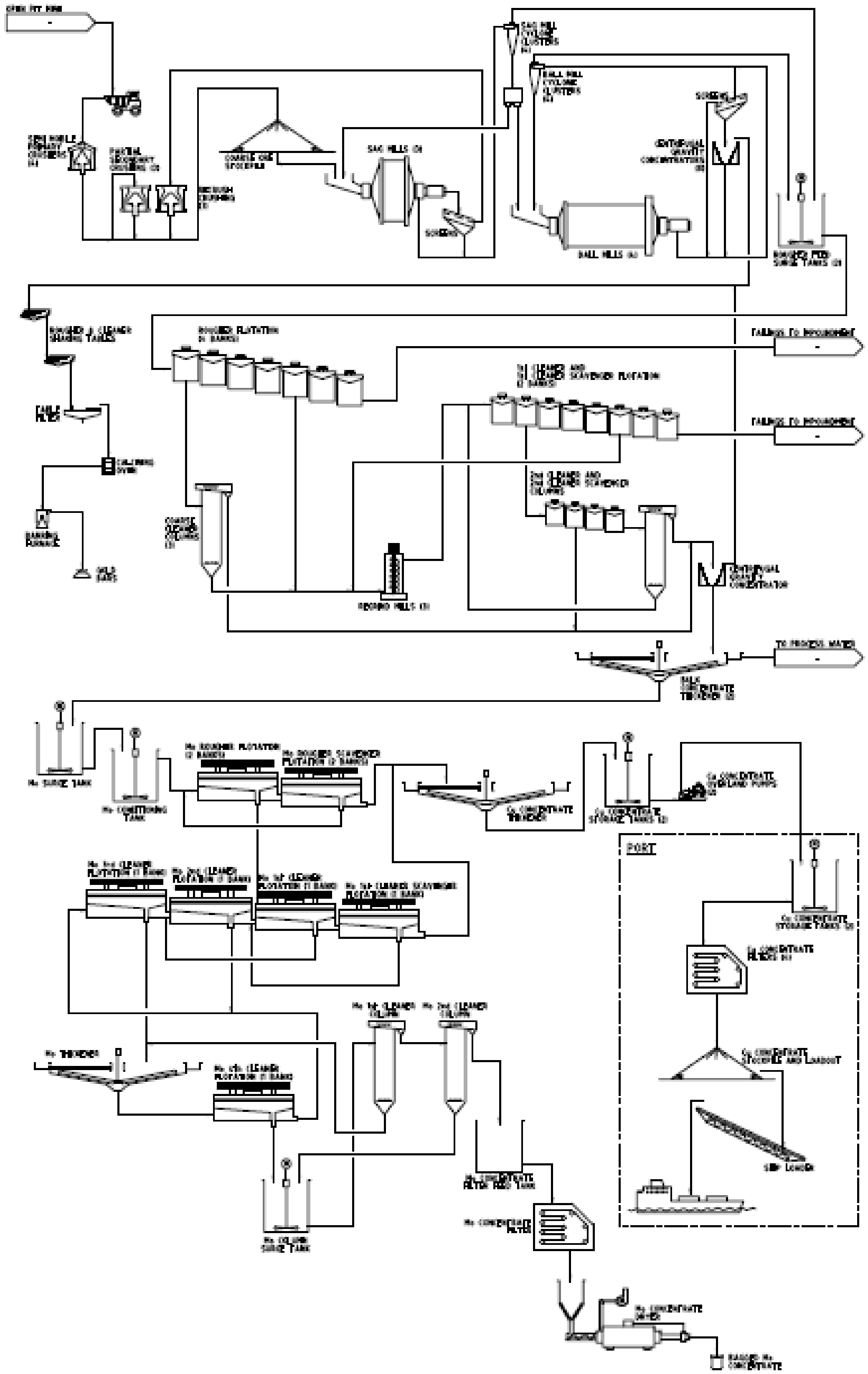
17.2 Process description

in 2019, following start-up, the concentrator will treat nominally 202,000 t/day of ore supplied from the Botija Pit; this rate will increase up to 232,000 t/day by 2020. Eventually, and according to the production schedule listed in Table 16-6, plant feed will be received from Colina, Valle Grande and the other pits. From 2023 onwards, the scheduled concentrator ore throughput will be increased to a nominal 246,575 t/day (where the design capacity will be up to 274,000 t/day allowing for 10% day-to-day fluctuations in throughput). Those crushing, grinding, bulk rougher flotation, water, and air systems which have not been predesigned for the expanded capacity will increase in capacity as required.

The process plant is designed to process ore at a head grade of up to 0.65% Cu and 0.023% Mo. These levels are higher than the highest sustained head grades of 0.46% Cu and 88.40 ppm Mo scheduled to be mined in 2023 and 2029, respectively, but the design provides the flexibility to accommodate a wide range of head grades over the Project life.

A simplified flow diagram is provided in Figure 17-1.

Figure 17-1 Process flow diagram



17.2.1 Crushing and grinding

The initial four primary gyratory crushers located in the Botija Pit are semi-mobile in-pit installations.

Crushed ore will be conveyed out of the pit to a surface transfer point, and thence by dual overland conveyors to where it will discharge into either secondary crusher feed bins or bypass direct via apron feeders to a coarse ore stockpile at the concentrator. Up to four secondary crushers may be installed and ultilised depending on ore characteristics. Provision will be made at the surface transfer point to accept crushed ore from the future mined sources (eg Colina). The coarse ore stockpile will hold a 2½-day supply for the mill, 16 hours of which will be available to the reclaim feeders without the assistance of a bulldozer.

Two trains of six apron feeders feeders and conveyors will draw ore from below the coarse ore stockpile and feed two parallel wet-grinding lines, each consisting of a 28 MW semi-autogenous grinding (SAG) mill and two 16.5 MW ball mills, all equipped with gearless drives. A third train of six apron feeders and conveyors will feed to a third SAG mill linked to the other train of ball mills to maximise their usage and enable maintenance of the treatment rate whilst also being able to operate independently. By the end of 2019, the third train will be equipped with a 22 MW ball mill operating in conjunction with the 28 MW SAG mill, whilst retaining linkage to the other two trains.

The SAG mill circuits will be closed by a combination of trommel screens followed by washing screens; conveyors will deliver screen oversize to pebble crushers via metal removal systems. A dedicated system for the recycling of reject balls is provided.

The pebble crushing circuits will include pebble bins, up to four cone crushers, and a bypass arrangement. Crushed pebbles will return to the SAG mills via the stockpile feed conveyors. The pebble crushing plant is located adjacent to the secondary crushers. A parallel pebble handling circuit provides for standby and direct return of pebbles to stockpile, so as to support crusher and bin maintenance.

Discharge from each SAG mill will be cycloned to recover the finished product whilst unfinished product will be evenly split between two ball-mill circuits. The four ball-mill circuits will be closed by hydrocyclones. The finished product from all cyclones will gravitate to two surge tanks, via in-stream particle and chemical samples, prior to pumping to the flotation area.

Linked to the ball mill circuits will be two gravity gold recovery plants. A proportion of the ball mill discharge will be pumped to the two gravity gold circuits comprising scalping screens and centrifugal gravity concentrators. The centrifugal gravity concentrators will recover the free gold and direct it to a gold plant for upgrading to bullion. Tails from the gravity concentrators will be returned to the milling circuit.

17.2.2 Flotation

Ground slurry will be directed to a flotation circuit where a bulk sulphide concentrate, containing copper, molybdenum, gold and silver values, will be collected and concentrated in a rougher followed by cleaner flotation. A primary high grade concentrate from the first rougher cell will be collected and cleaned directly in columns to produce a final product. The balance of concentrate from the remainder of the rougher cells will be collected, fed into three regrind mills, and then cleaned in two stages of mechanical cells followed by a one column stage to produce a final bulk concentrate.

The rougher and cleaner circuits will be installed to meet ultimate capacity, with no further additions required.

The bulk concentrate will be thickened in conventional thickeners (with no flocculant) and pumped to a differential flotation plant, where copper minerals will be depressed, and molybdenite floated into a molybdenum concentrate.

The flotation circuits are fully equipped with on-stream chemical and particle analysis capability.

17.2.3 Concentrates

Copper/gold concentrate piped from the plant site will be filtered, reclaimed using a mechanical reclaimer and loaded by closed conveyors on to bulk ore carriers. The filtrate water will be treated at the port in a water treatment plant or aternatively pumped through a return pipeline to the TMF. The concentrate will be filtered in automatic filter presses and when dry (8% to 9% moisture), will be stored in a covered building with a capacity of 140,000 t.

The molybdenum concentrate will be filtered, dried, and packaged in containers for shipment to offshore roasters. Tailings from the molybdenum flotation circuit will constitute the copper concentrate, which will be thickened/pumped/piped approximately 25 km to a filter plant at the Punta Rincón port site. If the molybdenum head grade is unsuitable, the molybdenum separation plant can be readily bypassed.

17.2.4 Tailings disposal and process water reclaim

For the first approximate fourteen years of the operation, tailings containing silicate, iron sulphide and other minerals from the rougher and cleaning steps will be deposited into the TMF located north of the mine and plant. The TMF is of centre line/downstream construction.

The majority of the rougher tailings will be processed through cyclones and the coarse fraction used to construct the TMF embankments. The finer rougher tailings fraction, together with the unused cyclone coarser tailings, will be deposited within the TMF on beaches upstream of the embankments. The tailings from the cleaner circuit will be deposited underwater to prevent generation of acid drainage conditions.

After fourteen years of operations, the depleted Botija Pit would receive the cleaner tailings (ie, deposited underwater), whilst the rougher tailings would continue to be deposited into the TMF for a period sufficient to form a cover over the TMF. After that time, the rougher tailings would also be diverted to the depleted Botija Pit and then ultimately to an expanded facility.

The timeframes for the tailings backfill will be reviewed as a function of new and updated mining and processing schedules produced when approaching depletion of the Botija Pit.

Flyash run-off liquor from the power plant will primarily be treated in a water treatment plant at the port, but may also be deposited into the TMF after being pumped up from the port site.

Reclaim water will be pumped from the TMF to a process water tank located to the north of the plant site. A tee off from this line will provide water for the tailings cyclone plant. From the process water tank, water will be either used in the milling plant, or boosted for general in-plant use. Excess water from the TMF will be overflowed via a decant tower and tunnel into the Del Medio River.

17.3 Processing consumables

Table 21-3 (Item 21) provides a list of consumption rates for process plant reagents and power.

The plant is equipped with preparation facilities for all required liquid and solid reagents, including frother, collector, promoter and lime. In addition, a ball charging system is provided in the milling area for feeding balls into the respective mills.

ITEM 18 PROJECT INFRASTRUCTURE

Project infrastructure information that was in the 2015 Technical Report (FQM, July 2015) was summarised in the context of proposed infrastructure and construction works then in progress. Now more than three years later, Project construction is nearing completion and plant start-up is imminent during Q1 2019.

The planning and readiness effort for the operations phase is well underway and key management personnel are in place. The operations readiness plan includes recruitment of staff, development of operating systems, and training of Panamánian employees working in the port, power plant, mine and process plant.

By December 2018, both the port and the first of the generators at the power station were running under full operational employee control. The power transmission line was energised and supplying backfeed power to the Project. Throughout 2019, further handover of commissioned areas in the process plant and power station will continue.

18.1 Project overall layout

Figure 18-1 shows an overall layout plan for the Project, showing the location of the main infrastructure components at the mining and processing sites, and showing the port site to the north. Figure 18-2 is a recent photograph looking towards the south west, with the processing plant in the foreground and the Botija Pit in the background.

18.2 Process plant

Figure 18-3 shows a view over the plant site, as at the end of December 2018.

The processing facilities are arranged broadly along a north-south axis with a low ridge running through the centre. The milling trains are located in a valley on the south side of the ridge with all other facilities to the north. The main plant process areas, except the mills, are located on either side of a major north-south arterial pipe rack. To the north of this area the topography falls away and at a lower level are located the reagents, secondary and pebble crushing areas.

This particular location was selected as being the most suitable from a design and geotechnical perspective. This area allowed both the milling trains and the coarse ore stockpile to be founded on fresh rock.

Located on the east side of the plant area is the Eastern Bypass road. This road originates off the concession to the east and transgresses the plant site linking in to the port and mine access roads.

As at the end of 2018, inching and lining of one of three semi autogenous (SAG) mills and two ball mills was completed. Progress towards completion of other mechanical and structural works was well advanced, as was progress on storage tanks, piping works, electrical installations and instrumentation.

Figure 18-1 Project layout plan

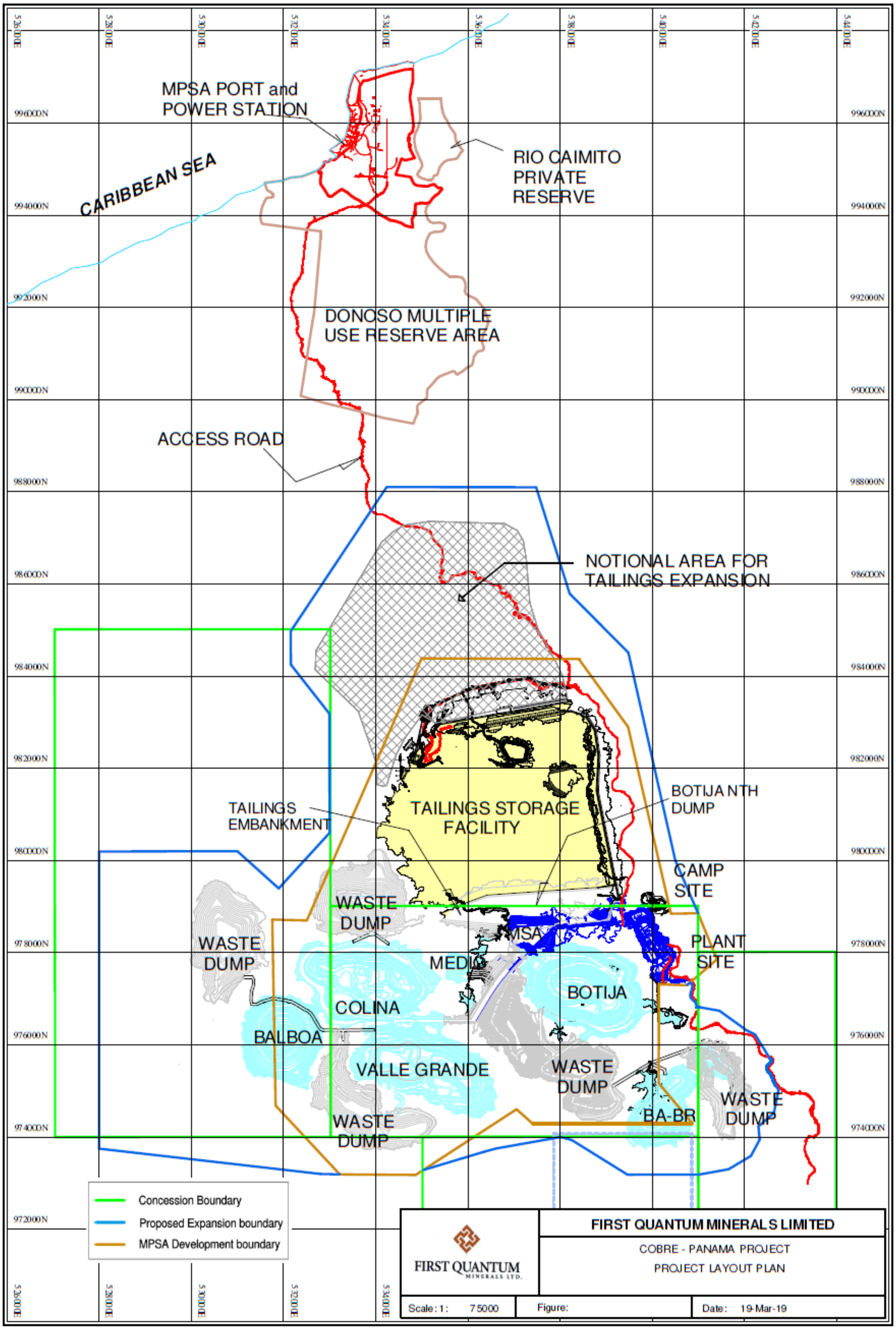


Figure 18-2 View looking SW over the Project processing plant, December 2018

Figure 18-3 Photograph of as-built plant, December 2018

18.3 Tailings management facility site

The Project TMF is located where shown on Figure 18-1 and will eventually cover an area of approximately 20 km^2. This facility has required the construction of retaining walls on the northern and eastern sides to enclose a storage basin, and as such will provide capacity for approximately 18 years of operation.

Figure 18-1 shows the proposed expansion footprint for the TMF, directly north of the existing facility.

18.4 Mining facilities

After commencing in 2015, the target for completion of the mining pre-strip at the Botija Pit was essentially achieved by the end of 2018, to an extent sufficient to expose the initial Botija orebody for process plant start-up. The starter pit had been developed, as had the excavation and formation of four in-pit primary crusher pockets. Two in-pit primary crushers had been installed, along with their associated overland conveyors to the crushed ore stockpile at the plant site.

The first UC rope shovel was assembled and began operating in March 2018, followed by the first of the UC haul trucks in November 2018. As at the end of 2018, two rope shovels and 16 UC trucks were operating in the Botija Pit. An additional two rope shovels, three UC loaders and 14 UC trucks will be progressively commissioned during 2019. In addition, an ongoing pioneering and intermediate fleet comprising diesel hydraulic excavators plus 100 t capacity and 40 t capacity 6x6 trucks continued to operate in Botija on pioneer works.

As at the end of 2018, the first 750 m length of trolley ramp was delineated and procurement of equipment for the trolley line was underway. Electric power into the mine to the trolley assist locations is now provided by a 34 kV transmission line ring main from the Botija substation. At December 2018, approximately 50% of the complete ring main was under construction and is expected to be complete by the second half of 2019.

In-pit water management sumps had been excavated and equipped, and the pit was progressing towards the formation of mining terraces to suit the deployment of UC primary mining equipment.

Grade control drilling was underway throughout 2018 and this incorporated a geometallurgical sampling and testing programme to validate processing recovery projections for the initial production years.

18.4.1 Inpit crushers and overland conveyors

Mining of the Botija starter pit has required the installation of four semi-mobile, in-pit crushers. Each pair of crushers will feed onto an inclined conveyor exiting the pit and thence transferring to two overland conveyors running along the north side of the Botija Pit to the secondary crushing plant. Future development of the pit requires the crushers to be relocated deeper into the excavation, requiring additional conveyors.

When Colina Pit is developed, a new conveyor system will be installed to allow two crushers to be located to the South of Colina (and thus to the north of the Valle Grande Pit, providing accessibilty from both). The conveyor system will comprise two conveyors linking into the tail end of one of the intial overland conveyors.

Figure 18-4 shows a view of one of the two completed in-pit crushers, as at December 2018. The two installations on the opposite side of the beltway remain under construction. Figure 18-5 is a view of the overland conveyors.

Figure 18-4 View of the Botija Pit IPC4

Figure 18-5 View of overland conveyors

18.4.2 Mine services area

Mine mobile equipment will be maintained by a skilled maintenance labour workforce. Where possible, all mining equipment will be maintained at the mine services area (MSA), or in circumstances where machinery cannot be moved, maintenance will be carried out in the field.

The MSA includes the heavy equipment workshop, light vehicle workshop, mine maintenance personnel offices, tyre shop, wash down bay, water services, refuelling station, go-line and mine control facilities for the mining fleet.

Figure 18-6 shows the locations of these facilities, in addition to:

- the location of the process plant site and overland ore conveying routes
- the proposed locations of future IPC positions in the Botija Pit
- the waste dump associated with the Botija Pit

The MSA facilities may be relocated in time, closer to the Colina, Valle Grande and Balboa Pits.

18.4.3 Explosives storage and preparation facilities

A temporary explosives magazine has been established for the construction and mining pre-strip phase, located about 2 km north of the concentrator. This facility receives blasting agents trucked from the off-loading and storage facilities at the port.

A new permanent magazine and emulsion mixing/preparation facilities are being constructed adjacent to the existing facilties and at adequate capacity for the full scale mining operations. The new faciltiies include a dispatch area for mobile mixing units delivering approximately 5 km into the mine.

18.4.4 Fuel and lubricant delivery and storage

A diesel fuel pipeline extends from the port to the Project site, and specifically to fuel storage and dispensing facilities located within the MSA.

The refuelling station comprises fuel unloading pumps, diesel storage tanks, heavy vehicle fuel dispensing pumps and bowsers, a light vehicle fuel dispensing pump and bowser, and fuel transfer pumps to the plant area. Concrete pads are provided for fuel unloading and loading vehicles, and storage tanks are installed in a fully concrete bunded area, with a sump pump which will discharge any spillage or washdownto the oil/water separator sump.

The access roads to the refuelling station have been arranged such that the travel paths of the heavy and light vehicles do not cross.

18.5 Port site and power station

The port site is located at Punta Rincón on the Caribbean Sea (Figure 18-7) and includes facilities for concentrate storage and load-out to handymax/supramax sized vessels (up to 65,000 dwt), in addition to coal receiving facilities, a barge berth, and inbound/outbound freight handling and storage.

The road distance between the port and the mine site is 25 km.

Figure 18-6 Mining facilities

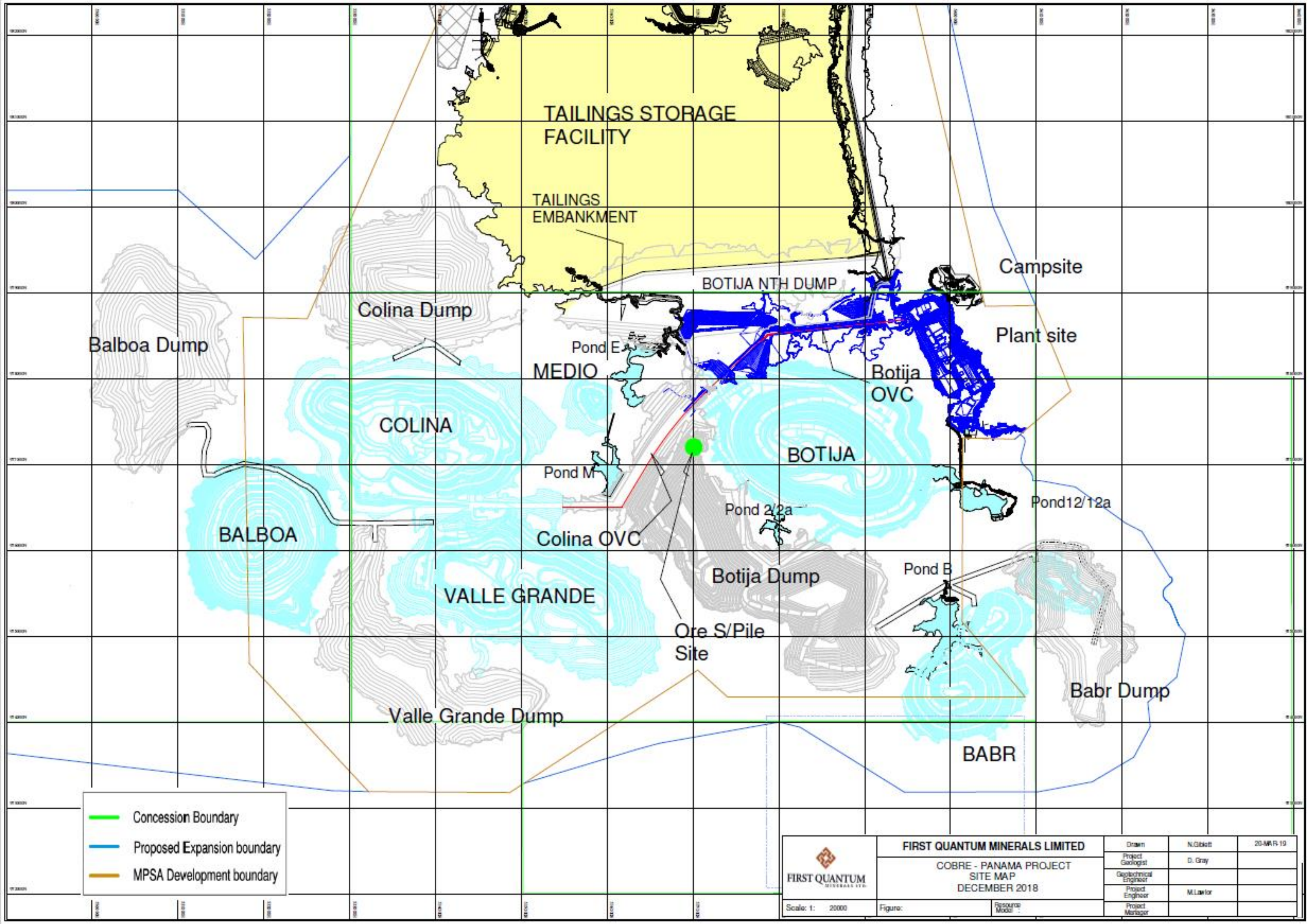


Figure 18-7 View looking over the port site and power station

The port site includes logistics infrastructure for receiving and dispatch of consumables in bulk and in containerised format arriving on regular and charter service vessels. Infrastructure includes:

- office and administration facilities for Government of Panamá customs, immigration and marine authority, and company personnel as part of the International Ship and Port Facility Security Code licence requirements for the Punta Rincón port
- bulk ammonium nitrate facilities for receiving, storage and dispatch by road transport to the emulsion plant located approximately 2 km north of the plant site
- bulk lime facilities for receiving, bag storage, bag breaking, silo storage and dispatch by bulk road tankers to the lime reagent facilities in the plant
- mill ball facilities for receiving, handling, and dispatch by road transport to the plant
- reagent facilities for receiving containers and dispatch by road transport to the plant for unloading and return of empty containers; and
- laydown and other logistics yards for receiving, storage and dispatch of consumables and equipment to the power plant, mine, process plant and other areas of the Project

The power station facilities include two 150 MW coal-fired generators (with provision for future additional trains). This facility operates continuously, 24 hours per day. Coal is being received at the port into a discharge hopper on shore. From there the coal is conveyed (under cover) to storage stockpiles at the power station.

18.6 Power transmission

A dual cicuit 230 kV power transmission line has been erected between the power station at the port and the Project site. There are interconnections to the plant switchyard and onward to the Llano Sanchez substation (ie, the connection point to the national grid; a distance of approximately 120 km).

Electric power into the mine is provided by a 34 kV transmission line ring main from the Botija substation.

18.7 Pipelines

Concentrate, concentrate filtrate return water, and fuel pipelines have been installed on surface between the plant and port site. These are located along the side of the coast access road.

The process plant requires a number of large pipelines for water and tailings management. The tailings lines (ie, six lines currently, with additional required as the Project expands) run alongside the coast access road from the plant area to the north east corner of the TMF. At this point they pass through a cyclone station and split onto the TMF walls.

18.8 Water supply

Potable water for the Project, including the port site, is obtained from local rivers and treated prior to distribution.

During operations, process water will be reclaimed from the TMF and collected from pit dewatering and surface drainage ponds. Water from the TMF will be pumped to the plant site by barge pumps followed by two stages of booster pumps.

A seawater intake and discharge has been constructed at the port site to supply the power plant.

18.9 Accommodation facilities

Accommodation facilities have been built for the construction phase and comprise a number of camps at the plant, TMF and port sites. The various camps are accessible by road and can collectively accommodate over 10,000 people. These facilities will continue to be used during operations in a modified format.

In addition, for the operations phase a number of onsite residential houses are being established for housing Compnay personnel. As at the end of 2018, the first eight houses had been built at the Punta Rincón port and additional houses are to be constructed according to requirements for attraction and retention of staff.

ITEM 19 MARKET STUDIES AND CONTRACTS

19.1 Product marketability

Based on Project metallurgical information, the primary metal product will be a clean standard grade copper concentrate which could be used for blending in all smelter processes. The contained gold and silver contents should not present any processing problems.

19.2 Agreements for sale of concentrate

Memoranda of understanding (MOU) have been negotiated and signed with eleven off-takers for the sale of Project copper concentrate. Counterparties include most major purchasers of copper concentrate from Europe, Asia and South America.

The MOU are adequate for up to 100% of the copper concentrate produced from the Project based on expected ramp up. The MOUs include provisions for sales of a minor proportion of concentrate, or indeed excess production, into the market at spot rates.

19.3 Other contracts and agreements

With production now imminent, all major supply contracts or arrangements are in place. This includes coal, diesel, grinding media, mill liners, blasting services, lubricants, reagents, etc. All of the contracts were competitively tendered and benchmarked against other Company operations, and can thus be considered as being "within industry norms". Raw material rates have been indexed against appropriate independent indicies, and will be adjusted up or down as appropriate.

In August 2012, MPSA entered into a precious metals stream agreement with a subsidiary of Franco-Nevada Corporation for the delivery of by-product precious metals based on copper concentrate production from the Cobre Panamá Project. Under the terms of the agreement, this subsidiary will provide pro-rata funding to MPSA in return for an amount of precious metals (gold and silver bullion) deliverable to them and indexed to the copper concentrate produced from the Project over the life of the operation.

The delivery of gold and silver bullion under the terms of this agreement is an obligation of FQM and is therefore not considered as a deficit to the MPSA gold and silver revenues in the optimisations and cashflow modelling carried out for the Mineral Reserves estimate reported herein.

ITEM 20 ENVIRONMENTAL STUDIES, PERMITTING, LAND, SOCIAL AND COMMUNITY IMPACT

The following information is largely reproduced from the 2015 Technical Report (FQM, July 2015), with updates authored by Michael Lawlor (QP) and reviewed by Anthony Petersen, MPSA Environmental Manager.

20.1 Environmental setting

The Project site is located in a recognised area of high biodiversity, and is subject to heavy tropical rainfall. The region forms part of the MesoAmerican Biological Corridor (MBC), connecting North and South America.

The location is relatively isolated, undeveloped and sparsely populated. The nearest community to the site is the village of Coclecito, located about 12 km southeast, whilst the provincial capital of Penonomé is 49 km further southeast.

Subsistence farming is the primary occupation for the local people. The area has been faced with increasing de-forestation, artisanal (small scale) mining, and changes in land use. Due to the high annual rainfall, increased water erosion and sedimentation have resulted.

20.2 Status of environmental approvals

An Environmental and Social Impact Assessment (ESIA) was completed by Golder Associates (Golder, 2010) in September 2010. The 14,000 page (plus appendices) ESIA was approved by the *Autoridad Nacional del Ambiente* (National Authority of the Environment, ANAM; now referred to as *MiAmbiente*) on 28th December 2011 for the Project as then envisaged.

In 2017, the Company successfully obtained change approvals relating to the siting of the process plant, the decant tower and tunnel at the TSF, and a number of above ground pipelines. The Company also lodged the necessary environmental permits to begin commissioning and operation of the power plant at the port.

In 2018, the Company successfully obtained permits for the power station water concession and the process plant water concession. During 2018, the Company also lodged the necessary environmental permits for water discharge from the power station and the TSF, the permit for the permanent emulsion plant, and the occupation permits to operate the power plant and the processing plant.

The 2011 Project approval was for a mining operation comprising three open pits, at Botija, Colina and Valle Grande. Since then the Project definition and development scope has changed to include aspects that will need to be addressed in a new ESIA, as follows:

- site clearing and mining of additional open pits, ie at Balboa and Botijo Abajo/Brazo
- site clearing and formation of additional waste rock storage facilities
- expansion of the tailings management facility

The expected timeframe for submitting the new ESIA for approval is mid-2014. The proposed expansion of the Project will be subject to a high level of environmental and social impact assessment by MPSA, in anticipation that the proposal will be similarly classified (Category III) by the Panamánian

Government. Environmental management and controls will be applied to the development of the expansion areas that are equivalent to the controls for the existing Project.

Provided that the same process and assessment is conducted as has been for the existing Project, and with similar commitments made, there are therefore no known reasons to indicate that there will be environmental and permitting issues that could materially impact on expanded mining and tailings disposal activities.

20.3 Environmental studies

Extensive environmental studies and social impact assessments were completed between 2007 and 2012 for the 2010 ESIA process. This work involved a number of independent experts and included baseline studies on such as climatic conditions, fauna and flora, hydrology, hydrogeology, air and water quality, cultural and socio-economic circumstances.

Project development components considered in these baseline studies included:

- three open pit mines (Botija, Colina and Valle Grande)
- mining facilities such as equipment maintenance workshops, fuel storage and dispensing, explosives preparation and storage
- a processing plant (ore concentrator) with an annual capacity of 74 Mt (expanding to 100 Mt) processed
- ore crushing, conveying and ore stockpile facilities in the vicinity of the processing plant
- other site facilities such as a sewage treatment plant, waste incinerator and landfill areas
- a tailings management facility (ie, tailings dam or TMF), in addition to tailings backfill into depleted open pits
- saprolite and waste rock storage facilities (ie, waste dumps or WRSF)
- longer term ore stockpile(s)
- potable and process water reservoirs
- water diversion and sediment control structures
- quarries for construction materials
- camp, security and administration facilities
- assay laboratory
- site roads
- port facilities, including a filtration plant and a coal-fired power station
- power transmission lines
- 25 km of road between the mine and the port, plus site access roads

An environmental management system (EMS) has been developed to include environmental and social management plans. The components of the EMS are:

- Environmental Management System Framework
- Environmental Monitoring Plan
- Biodiversity Action Plan
- Water Management Plan
- Spill Prevention and Control Plan
- Hazardous Materials Management Plan
- Environmental Education and Training Plan

- Erosion and Sediment Control Plan
- Waste Management Plan
- Air Quality and Noise Control
- Port Management Plan
- Environmental Recovery and Abandonment Plan
- Archaeological Resources Management Plan

Specific plans developed for the construction phase include:

- Construction Site Environmental Mitigation and Control Procedures
- Construction Water Management Plan

The ESIA document (Golder, 2010) states that the Project will comply with Panamánian regulations and also meet international standards, in respect of International Finance Corporation (IFC) Performance Standards on social and environmental sustainability.

20.3.1 Biodiversity Action Plan

The Company has implemented its Biodiversity Action Plan in line with IFC Performance Standard 6, to protect and conserve the sensitive biodiversity of the Project area. To this end and continuing through the Project construction phase, collaboration has been maintained with the Kew Botanical Gardens, the Missouri Botanical Gardens, the Smithsonian Tropical Research Institute, the Sea Turtle Conservancy, the Peregrine Fund, and other specialists.

20.3.2 Site water management

The high rainfall intensity in the Project area has led to construction phase challenges due to erosion and sediment runoff. These challenges have been managed by the construction of sedimentation ponds and hydro-seeding of disturbed areas. During operations, all drainage from the open pit, waste dumps and ore stockpiles will be collected and redirected for use as processing water. Surface water flowing from river diversions will be monitored as required, before release to the environment.

20.4 Summary of environmental impacts and management requirements

Over 600 impacts and management commitments are addressed in the 2010 ESIA, during the construction, operations and closure phases.

The main impacts and commitments listed in the ESIA conclusions (Golder, 2010) include (*verbatim*):

- The Project will affect surface water and groundwater quality and quantity. All high magnitude predicted effects are expected to be local in geographic extent.
- The Project will affect air quality, including emissions of dust, SO_2, NOx and greenhouse gases. Available technologies to reduce these emissions will be employed.
- The Project will result an in increase in noise in the immediate vicinity of Project activities. Activities that have a high potential to create noise and vibration impacts will be limited to daytime activities.
- The Project will result in the temporary loss of approximately 5,900 ha of forest. Whilst the majority of this area will be reclaimed to forest, the loss of the forest land base will last for several generations. The loss will be counteracted by reforestation of lands off of the mine site,

economic development to slow existing deforestation rates from a baseline rate of 0.5% per annum, and support for predicted areas management.

- The Project has been designed to minimise the extent of spatial effects, including phased development so that a minimal area is disturbed at any one time, progressive reclamation, and backfilling of mine pits.
- In addition to habitat loss, fauna will be affected by mortality during the clearing of natural habitats, sensory disturbance, and by indirect effects due to changes in hunting and collecting. Mitigation programmes have been designed to reduce these effects.
- Flora and fauna species of concern identified in the assessment are to be found offsite in candidate areas for conservation. Some species will also be relocated from the Project site to other locations. Most importantly, the Project will not result in any species extinctions.
- Freshwater species are also affected by habitat loss, primarily in the Rio del Medio River Basin, and changes to water quality downstream of the Project. Water quality will be monitored and water will be treated if necessary.
- Marine species may be affected by habitat loss from wharf and port/power plant infrastructure construction, or by direct mortality, sensory disturbance and changes in the amount of fishing in the area. The Project will seek to protect hard bottom habitats when possible and compensate with a new 0.5 ha offset hard bottom as required by the ESIA, in addition to controling vessel traffic and underwater noise so as to minimise direct effects.
- A plan has been established to create a conservation area within the concession, to provide support for creation of conservation areas offsite and to provide management or funding support as appropriate to existing and future designated protected areas.
- The Project will create a biodiversity chair at a Panamánian institution of higher learning, and already funds local and international non-governmental organisations (NGOs) to conduct biodiversity related research.
- The Project's target is no net loss of biodiversity and MPSA will work cooperatively with biodiversity experts to design and implement offset programmes to achieve this objective.
- The Project is within the MBC. There will be some temporary loss of habitat and obstruction of fauna moving along the corridor, but reforestation will promote fauna movements, and the development of sustainable communities will help to slow future impacts to the MBC.

In terms of positive outcomes from the ESIA process and the Project development, in general, the ESIA (Golder, 2010) lists the following (*verbatim*):

- The extensive baseline characterisation programme undertaken over three years has added important new information about the socio-environmental context of the Atlantic slope of Panamá.
- The Project will have considerable positive effects to local, regional and national economies, through direct, indirect and induced job creation, procurement from local and other Panamánian businesses, and through royalty and tax revenues. Income taxes, royalties and other fees paid by MPSA to the national treasury are predicted to amount to US$1.6 billion over the life of the Project.
- Direct Project-related employment will occur locally, regionally and nationally. MPSA will encourage the participation of local people. Total construction labour is estimated to annually average over 3,000 workers during the construction phase. An annual average of about 2,000 workers will be directly employed during operations, with about 6,000 direct, indirect and

induced jobs sustained on an annual average over the life-of-mine. Training of workers will also be a positive benefit to the region, and impart important life skills to the local population that will contribute to the development of sustainable communities.

- MPSA will establish a Community Development Foundation for the Project, through which most of MPSA's community development activities in the region will be conducted. MPSA will initially develop and implement specific community development programmes during the construction phase, with $3.6 million in 2010, rising to an estimated annual average of $5 million for the remainder of the construction phase.

The ESIA document (Golder, 2010) goes on to state that (*verbatim*):

- A comprehensive stakeholder engagement programme was undertaken for the Project; stakeholders were consulted beginning at an early stage and throughout the ESIA process. Stakeholders expressed their interest and concerns about job opportunities, effects on water, air, land and forests, livelihoods, in-migration and associated social change, along with their hopes that the Project would generate community benefits such as improved health, education, electricity and water supply.

A condition of the ANAM approval has been a Project commitment for:

- a biodiversity protection programme over a 150,000 ha area
- reforestation of over 7,375 ha outside of the Project area
- reforestation of more than 3,100 ha within the Project area

A modification to the ESIA was prepared in late 2013 to condense the number of management commitments into 370 items. This condensation represents only a change of form, rather than substance, for the approved mitigation measures. The modification was approved by ANAM on 5th December 2013.

During 2017 *MiAmbiente* announced the establishment of the Donoso Multiple-Use Protected Area in Donoso District and a Management Plan was being developed for the area by the Government of Panamá.

20.4.1 Environmental audits

The Project has been audited quarterly against ESIA commitments by a third party specialist and the result provided to the Government of Panamá environmental regulator. As part of a due diligence effort when the Company was contemplating project finance, the Project underwent an environmental (and social) due diligence review by specialist consultants acting for the potential lenders. Although some action items were identified, the Project was considered to be largely in compliance with ESIA commitments.

20.5 Social and community related requirements

The significance of the Project to Panamá is reflected in that it will be the largest private sector investment in the country's history and will provide a substantial proportion of the country's export income. Commensurate with this level of national prominence, the Project is expected to provide considerable social and community benefits, particularly in respect of local employment, provision of schooling, technical and operator training, university funding, and professional development.

As part of the ESIA process, extensive community and stakeholder consultation has taken place. In general, the Project has developed positive relationships with the provincial and national government, as well as with the local community. This outcome has been achieved in circumstances of a relatively impoverished community with essentially no history of mining. For example, a positive outcome of the new mine access road development, with a direct benefit to over fourteen local communities, has been the reduction in travel time from 24 hours to 4-6 hours. The road also allowed the state electrical utility company to install power lines thus bringing 24 hour electricity supply for the first time to these communities.

Arising from the consultation process, a consultant was appointed to produce a Social Development Plan (SDP) for the twenty two impacted communities in the Donoso, Omar Torrijos Herrera and La Pintada Provinces. Since adoption of the SDP in 2010, MPSA has continued to develop and review the plan as community needs change over time. Since 2014, MPSA has worked to establish Community Participatory Committees, with agenda items including water, sanitation, health, education (schools) and commercial development programmes like the agro-extensionism whereas over 400 families now provide fresh produce from their own lands to the mine camps resulting in sales of over $1.75 M per year.

20.6 Resettlement

To 2018, there has been three community resettlement projects involving two indigenous communities at Petaquilla and Chicheme, comprising 86 households, plus 46 farming families. For all but nine of the 46 farming families, the resettlement plan provided compensation for the land claimed under possessory rights. For the nine families, compensation provided for land, dwellings and crops.

Resettlement of the indigenous communities followed the International Finance Corporation Performance Standards. The resettlement plan provided for the construction of entirely new communities, including new houses (house for house was replaced), new schools, communal houses, land title for the new lands (land for land was replaced) and compensation for the farms and crops belonging to each family. Resettlement of the indigenous communities at Petaquilla and Chicheme comprising 86 households was completed in January 2017, and now is in the ongoing phase of livelihood and continued sustainable support.

The Company has performed a mid-term audit of the resettlement programme and will perform the resettlement closure audit report in mid 2019. Both audits are part of the IFC compliance requirements and any recommendations are reviewed and evaluated for compliance purposes. The expanded mine footprint for Balboa, Botija Abajo and Brazo pits and associated TMF expansion will require probable new resettlement processes with impacted communities, and this will be addressed in the new ESIA process.

20.7 Mine closure provisions

The ESIA (Golder, 2010) refers to progressive rehabilitation, where possible, over the course of operations. As buildings and other infrastructure are no longer required, they are to be decommissioned, demolished and/or removed. Diverted water courses are to be reinstated where possible and roads that are no longer required are to be scarified and revegetated.

The ESIA (Golder, 2010) mentions that effluent treatment and monitoring will be performed for a period of three years, post-closure. At closure, the Colina and Valle Grande Pits will be flooded. The

Botija Pit, which will have been backfilled with tailings, will also be flooded. The WRSFs will be covered with compacted saprolite and revegetated. The existing water management structures will be modified to minimise the volumes of water required treatment or management. Detailed concepts are described in the ESIA for the TMF, plant/mine site, port site and power plant at closure.

Golder (December 2017) subsequently produced an update to the mine closure provisions initially estimated by MPSA in 2013. Details on these provisions are listed in Item 21.1.2.

The basis of the Golder update was:

- a review of the anticipated surface water situation at the time of closure, drawing on water balance and modelling information from a Piteau (2017) report
- identification of closure related opportunities, threats and uncertainties
- use of a database of unit costs and rates for such as infrastructure dismantling and demolition, rehabilitation earthworks, water treatment, monitoring and inspections
- itemisation of costs for specific areas of the operation, including the mine, waste dumps, processing plant, TMF, port and power station

Golder (December 2017) provided closure cost estimates for differing circumstances, ie at an unscheduled closure time, after a five year period of operations, and at the scheduled end of Project life.

As at the end of 2018, *MiAmbiente* was reviewing the MPSA Environmental Liability Guarantee for an updated registered Closure Plan with the Governemnt authority. The Environmental Liability Guarantee was approved by *MiAmbiente* in January 2019.

The provisions and closure cost estimates will be periodically reviewed and updated by a specialist consultant.

ITEM 21 CAPITAL AND OPERATING COSTS

21.1 Capital costs

21.1.1 Project development capital

Table 21-1 lists the updated development capital costs included in the Project cashflow model. This table also lists the estimated capital costs for ultimate capacity expansion to 100 Mtpa in 2023, and for associated mine infrastructure and development costs.

Table 21-1 Development capital cost provisions

Capital Item		< 2019	2019	2020	2021	2022	2023	2024	2025	2026	TOTAL
Development capital											
Project capital for 85 Mtpa plant	USD$M	$6,070.0	$230.0								**$6,300.0**
100 Mtpa plant upgrades	USD$M		$90.0	$87.0	$110.0	$40.0					**$327.0**
Colina pioneering, access, engineering, IPC and OVC	USD$M		$45.0	$6.0		$45.0	$16.0				**$112.0**
River diversion works for Colina	USD$M							$33.0	$33.0	$34.0	**$100.0**
Site and other capital	USD$M		$105.0	$7.0	$12.0	$4.0	$2.0				**$130.0**
Total development capital	**USD$M**	**$6,070.0**	**$470.0**	**$100.0**	**$122.0**	**$89.0**	**$18.0**	**$33.0**	**$33.0**	**$34.0**	**$6,969.0**
Total development capital to end 2019, and from 2020	**USD$M**	**$6,540.0**		**$429.0**							**$6,969.0**

$6,300 M is the updated estimate for completion of the Project to 85 Mtpa capacity. Table 21-1 shows estimated capital amounting to $240 M in 2019, required to enable commencement of the expansion to 100 Mtpa capacity, along with initial development and engineering work allowing mining to proceed to the Colina Pit[12].

$105 M of the $240 M in 2019 relates to site and other capital expenses, inclusive of allowances for:

- completion of the Botija Pit pre-strip
- a brought forward amount for completion of the Botija Pit diversion channel and an associated water control embankment
- pioneering mining equipment for the Colina pre-strip
- Colina infill and geotechnical drilling
- early deployment of trolley assist haulage infrastructure
- mine dewatering expenses
- additional spares

The total capital estimate to the end of 2019 is therefore shown in Table 21-1 as $6,540 M.

Table 21-1 shows $404 M of capital allowances from 2020 for continuing plant expansion works and for the development of the Colina Pit such that a starter pit will be ready for inpit crushing and overland conveying of crushed ore to support the plant capacity expansion. Site and other capital items from 2020, amounting to $25 M, include allowances for miscellaneous infrastructure at the port, power plant and accommodation facilities.

[12] Initial Colina expenditure has been brought forward due to the sooner requirement of plant feed from Colina arising from the capacity expansion (ie, not envisaged in the 2015 Technical Report).

Estimate revision

The total Project capital of $6,425 M that was reported in the 2015 Technical Report (FQM, July 2015) has reduced to a new estimate of $6,300 M, despite the increase of the initial production rate from 74 Mtpa to 85 Mtpa processed. This new amount includes the $6,070 M shown in Table 21-1, which has been spent.

The reasons for the reduction are explained as follows:

1. The original capital cost estimate for the Project of $6,425 M, as listed in the 2015 Technical Report (FQM, July 2015), was developed during 2013 and into 2014. A number of assumptions were made in this particular estimate, including assumptions for unit rates that would be applicable to the large volume of earthworks required, and levels of productivity expected for the Project. Also at the time of the estimate preparation, commodity prices were at elevated levels.
2. As the Project progressed from 2015 onwards, real construction data was able to be collected. Significant effort was applied to mastering the earthworks in the difficult conditions, and this focus and effort allowed some capital cost savings to be made. Additionally for the four years of construction, there was a continued focus on productivity for all construction disciplines, and on productivity improvements.
3. During 2016 and 2017 there was a significant downturn in commodity prices, and this presented an opportunity to continue the procurement of various items required for the Project at lower unit rates than had been included in the original capital cost estimate. Lower pricing was obtained for a range of items over a substantial period of time including for example, reinforcing steel for concrete work, structural steel, electrical cabling, piping and mechanical equipment.
4. The scope of work required to expand the Project to 85 Mtpa provided synergies with existing construction. For example: adding an eighth mill in addition to the seven mills already being installed; and adding more flotation cells to a flotation circuit that had been designed to accommodate easy expansion, etc.

The above combination of capital cost savings and synergies has allowed an expanded 85 Mtpa plant to be constructed for less than the originally estimated cost of a 74 Mtpa plant.

21.1.2 Sustaining capital costs

Consistent with the 2015 Technical Report (FQM, July 2015), an allowance of $0.26/t mined has been adopted as a sustaining capital cost allowance for mining. Allowances for process and G&A sustaining costs have been adopted as 5% of the respective operating costs.

The allowance for mining equipment sustaining capital was determined by MPSA in January 2013. This estimate has been reviewed and considered to be an appropriate continuing allowance, based on:

- specified service life hours for each mining equipment item
- calculated annual equipment requirements and annual operating hours
- a detailed annual equipment replacement schedule linked to capital cost estimates

An additional $60 M has been included to account for budgeted tailings dam capital expenditure, and an amount of $148.6 M has been included for planned component replacements and for the sustaining of haul road trolley assist infrastructure.

21.1.3 Mine closure provisions

Table 21-2 lists the updated closure cost provisions (Golder, December 2017) included in the Project cashflow model, adopting the estimated costs that would be incurred at the end of the Project life. These costs provide for rehabilitation of the entire Project site.

Table 21-2 Closure cost provisions

Description	Mine and Plant ($M)	Port and Power Station ($M)	TMF ($M)	Total ($M)	2049 (year 31) ($M)	2050 (year 32) ($M)	2051 (year 33) ($M)	2052 (year 34) ($M)	2053 (year 35) ($M)	2054 (year 36) ($M)
Rehabilitation Measures										
Infrastructure aspects	$41.4	$2.5	$2.2	$46.1	$5.7	$5.0	$6.6	$9.3	$7.6	$11.9
Mining aspects	$10.7	$0.3	$23.5	$34.5	$4.2	$3.7	$4.9	$7.0	$5.7	$8.9
General surface rehabilitation	$6.1	$0.4	$0.0	$6.6	$0.8	$0.7	$0.9	$1.3	$1.1	$1.7
Water management	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal	**$58.2**	**$3.2**	**$25.8**	**$87.2**	**$10.7**	**$9.4**	**$12.4**	**$17.6**	**$14.4**	**$22.6**
Post-closure Measures	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
On-going water treatment	$4.5	$0.0	$0.0	$4.5	$0.6	$0.5	$0.6	$0.9	$0.8	$1.2
Surface water treatment	$0.0	$0.0	$0.0	$0.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Groundwater modelling	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Rehabilitation measures	$0.2	$0.0	$0.1	$0.3	$0.0	$0.0	$0.0	$0.1	$0.0	$0.1
Care and maintenance	$1.1	$0.0	$0.3	$1.4	$0.2	$0.1	$0.2	$0.3	$0.2	$0.4
Subtotal	**$5.9**	**$0.1**	**$0.4**	**$6.3**	**$0.8**	**$0.7**	**$0.9**	**$1.3**	**$1.0**	**$1.6**
Additional Allowances	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Preliminary and general	$14.7	$0.8	$6.4	$21.9	$2.7	$2.4	$3.1	$4.4	$3.6	$5.7
Contingencies	$5.9	$0.3	$2.6	$8.8	$1.1	$1.0	$1.3	$1.8	$1.4	$2.3
Additional studies	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal	**$20.6**	**$1.1**	**$9.0**	**$30.7**	**$3.8**	**$3.3**	**$4.4**	**$6.2**	**$5.1**	**$8.0**
Total	**$84.7**	**$4.3**	**$35.2**	**$124.2**	**$15.3**	**$13.4**	**$17.7**	**$25.1**	**$20.5**	**$32.2**

21.2 Operating costs

21.2.1 Mining costs

Updated mining costs for this Technical Report were re-estimated from first principles using haul profiles and productivity estimates related to current mine designs, a preliminary long term plan and production schedule, the actual equipment fleet, and updated ore/waste haulage destinations.

The mining costs took account of short ore hauls to notional IPC installations; they did not account for faster cycle times arising from trolley-assisted haulage. There were two cost components:

- a base mining cost, including drill, blast, load and haul costs up to a reference bench elevation (Table 21-3), and
- an incremental mining cost, comprising of:
  - drill and blast for ore and waste (Table 21-4) and;
  - haulage beyond the reference bench level, and to respective ore tipping and waste dumping destinations (Table 21-5).

Table 21-3 Fixed mining costs

Cost Item	Fixed Mining Costs - with IPCC	
	Waste ($/t)	Ore ($/t)
Management	$0.008	$0.008
Operations O/H	$0.030	$0.030
Engineering	$0.011	$0.011
Geology	$0.010	$0.010
Technical Services	$0.010	$0.010
Drilling	$0.006	$0.006
Blasting	$0.012	$0.012
Loading	$0.003	$0.003
Hauling	$0.074	$0.074
Stockpile Rehandle	$0.010	$0.010
Services	$0.066	$0.066
Dewatering	$0.061	$0.061
Contract Clearing & Pre-stripping	$0.000	$0.000
Reclamation	$0.002	$0.002
Subtotal Base Mining Cost	**$0.304**	**$0.304**

Table 21-4 Drill and Blast Variable Costs

Drill & Blast Variable Cost	Waste ($/t)	Ore ($/t)
Drilling	$0.096	$0.101
Blasting	$0.271	$0.286
Subtotal Drill & Blast Cost	**$0.367**	**$0.387**

Table 21-5 shows the incremental haulage costs applied for each 15 m vertical change in elevation from the reference RL, for uphill and downhill hauls. These costs were developed from the unit costs of truck haulage for a 15 m change in elevation, plus an allowance for ancillary plant support.

Table 21-5 Incremental haul costs

Incremental Haulage Cost	
Waste ($/t)	Ore ($/t)
$0.0385	$0.0394

A total mining cost for a nominal RL in each pit was calculated using the following inputs:

- fixed mining cost (Table 21-3)
- variable drill and blast cost (Table 21-4)
- load and haul cost (including ancillary support) based on haul profiles for ore and waste to nominated destinations

The total mining cost estimate at each mining bench level within each pit was then determined based on a reference bench cost in each pit (Table 21-6), and then incrementing the cost upwards and downwards using the incremental haulage costs shown in Table 21-5.

Table 21-6 Reference bench RL mining costs

Pit	Base RL	Waste ($/t)	Ore ($/t)
Botija	-120 m	$2.64	$2.58
Colina	120 m	$1.77	$1.59
Medio	150 m	$1.58	$1.83
Valle Grande	225 m	$1.56	$1.38
BABR	195 m	$1.69	$1.75
Balboa	-30 m	$2.13	$2.18

The equivalent weighted average mining costs based on the above calculations are shown in Table 21-7. This table shows the average costs without and with adjustments to account for trolley-assisted haulage. Experience at the Company's Kansanshi operation in Zambia suggests a potential 25% reduction in haulage costs with the implementation of trolley-assist. Table 21-7 shows the unadjusted overall weighted average mining costs derived from the new incremental relationships, in addition to adjusted overall weighted average mining costs assuming the following:

- approximately 40% of the total mining costs are attributable to haulage
- of this 40%, a saving of 25% could be expected for all waste hauls
- of this 40%, a saving of 25% could be expected on around half of the ore hauls

Table 21-7 Overall average mining costs

Pit	Units	Unadjusted total cost	Adjusted for trolley-assist		
			Haulage cost	TA saving	Total cost
BOTIJA	$/t waste	$2.30	$0.92	-$0.23	$2.07
	$/t ore	$2.33	$0.93	-$0.12	$2.22
	$/t total	$2.32	$0.93	-$0.17	$2.15
COLINA	$/t waste	$1.88	$0.75	-$0.19	$1.70
	$/t ore	$1.91	$0.76	-$0.10	$1.81
	$/t total	$1.90	$0.76	-$0.14	$1.76
MEDIO	$/t waste	$1.60	$0.64	-$0.16	$1.44
	$/t ore	$1.98	$0.79	-$0.10	$1.88
	$/t total	$1.75	$0.70	-$0.14	$1.61
VALLE GRANDE	$/t waste	$1.75	$0.70	-$0.17	$1.57
	$/t ore	$1.79	$0.71	-$0.09	$1.70
	$/t total	$1.77	$0.71	-$0.13	$1.64
BABR	$/t waste	$1.92	$0.77	-$0.19	$1.73
	$/t ore	$2.18	$0.87	-$0.11	$2.07
	$/t total	$2.05	$0.82	-$0.16	$1.89
BALBOA	$/t waste	$1.97	$0.79	-$0.20	$1.78
	$/t ore	$2.27	$0.91	-$0.11	$2.16
	$/t total	$2.09	$0.84	-$0.17	$1.92
ALL PITS	**$/t waste**	**$1.97**	**$0.79**	-$0.20	**$1.78**
	$/t ore	**$2.11**	**$0.79**	-$0.10	**$2.01**
	$/t total	**$2.05**	**$0.82**	-$0.15	**$1.90**

The savings due to the application of trolley assist that are summarised in the above table have been applied in each pit on an individual bench basis and then combined into a simple relationship to estimate mining costs by depth in each pit as shown in Table 21-8.

Table 21-8 Applied mining costs ($/t) relationship, varying by depth

	Waste	Ore
Botija	2.0984 -(Zelev * (0.0353/15))	2.0794 - (Zelev * (0.0346/15))
Colina	1.8476- (Zelev * (0.0353/15))	1.8331 - (Zelev * (0.0370/15))
Medio	1.7824 - (Zelev * (0.0363/15))	2.2971 - (Zelev * (0.0757/15))
Valle Grande	1.9327 - (Zelev * (0.0362/15))	1.8410 - (Zelev * (0.0370/15))
BABR	1.9933- (Zelev * (0.0362/15))	2.2327 - (Zelev * (0.0458/15))
Balboa	1.8588 - (Zelev * (0.02722/15))	2.0106 - (Zelev * (0.0373/15))

Although Table 21-3 shows a nominal allowance for ore rehandling, an estimated cost was included in the cashflow modelling to account for active and longer term ore stockpile rehandling to suitably located IPC positions. The unit cost of $1.19/t ore reclaimed was developed from the equipment costs used in the mining cost model, development of haul cycle times for each potential stockpile source and IPC destination, and material rehandled in the mining schedule.

21.2.2 Processing and G&A costs

Project operating costs were originally derived from total dollar, life of mine cost estimates produced by MPSA in January 2013. The costs were subsequently adjusted to account for the proposed plant throughput increase, and as shown in Tables 21-9 and 21-10 for processing and G&A costs, respectively. The estimates are shown in unit cost terms, for the purposes of the original mine optimisation/planning and for subsequent cashflow modelling to support the Mineral Reserve estimate.

In the original MPSA estimates, a power cost was included for provision of excess generated electricity back into the national grid. In the updated cost estimates, in circumstances of increased plant throughput and power consumption, this cost has been removed.

Revised process operating and G&A cost estimates reflect updated estimates of:

- consumption rates for grinding media, liners and reagents
- electrical power consumption rates
- unit costs for grinding media, liners and reagents, electrical power and maintenance parts
- plant personnel numbers
- annual assay costs
- an allowance for tailings operation costs for such as piping and spiggotting

The original sustaining allowances were removed from the cashflow model operating cost estimates and included as sustaining capital allowances.

The process operating costs shown in Table 21-9 are the estimated longer term average annual costs when operating at 100 Mtpa capacity. For the purposes of the cashflow model, and to reflect the shorter term budget produced by the site operating team, the average unit processing cost for 2019 to 2025 is $4.27/t processed (yielding an overall Project life average of $3.84/t processed). Similarly, the G&A costs shown in Table 21-10 are also the estimated longer term average annual costs when operating at 100 Mtpa capacity. The shorter term average unit G&A cost for 2019 to 2025 is $1.10/t processed (yielding an overall Project life average of $0.62/t processed).

There is an additional fixed annual operating cost to be included in the cashflow model and that is $10.3 Mpa for the port operations.

Table 21-9 Process operating costs, pit optimisation vs cashflow model

Processing costs	Pit Optimisation					Cashflow Model				
	Consumption rates		Unit costs		$/t	Consumption rates		Unit costs		$/t
Grinding media					**0.81**					**0.55**
SAG mill	0.401	kg/t milled	0.94	$/kg	0.38	0.316	kg/t milled	1.05	$/kg	0.31
Ball mill	0.381	kg/t milled	0.94	$/kg	0.36	0.230	kg/t milled	1.06	$/kg	0.23
Regrind mill	0.079	kg/t milled	0.94	$/kg	0.07	0.019	kg/t milled	0.52	$/kg	0.01
Liners					**0.25**					**0.38**
Gyratory crusher mantle liners	0.001	kg/t milled	4.43	$/kg	0.004	0.004	kg/t milled	5.00	$/kg	0.02
Gyratory crusher concave liners	0.001	kg/t milled	3.38	$/kg	0.004	0.004	kg/t milled	4.00	$/kg	0.02
Secondary crusher mantle					0.00			2,150	op hours life	0.03
Secondary crusher liners					0.00			2,150	op hours life	0.03
Pebble crusher mantle liners	0.001	kg/t milled	4.25	$/kg	0.005			2,500	op hours life	0.01
Pebble crusher bowl liners	0.001	kg/t milled	4.31	$/kg	0.005			2,500	op hours life	0.01
SAG Mill liners	0.052	kg/t milled	3.38	$/kg	0.171	0.055	kg/t milled	4.00	$/kg	0.21
Ball Mill Liners	0.021	kg/t milled	2.87	$/kg	0.059	0.022	kg/t milled	3.50	$/kg	0.07
Reagents					**0.59**					**0.52**
Sodium Hydrosulphide (NaHS)	10.165	kg/t dry concentrate	1.16	$/kg (0.0137 t conc / t ore)	0.16	7.624	kg/t dry concentrate	1.08	$/kg (0.0137 t conc / t ore)	0.18
Lime	0.987	kg/t ore	0.22	$/kg	0.22	0.987	kg/t ore	0.14	$/kg	0.13
Flotation Frother (MIBC)	0.028	kg/t ore	3.23	$/kg	0.09	0.028	kg/t ore	3.10	$/kg	0.08
Moly Frother					0.00	0.0008	kg/t ore	3.91	$/kg	0.00
Collector (SIPX)	0.031	kg/t ore	1.88	$/kg	0.06	0.035	kg/t ore	1.93	$/kg	0.07
Promoter (A3302)	0.010	kg/t ore	4.93	$/kg	0.05	0.026	kg/t ore	1.15	$/kg	0.03
Dispersant					0.00	0.090	kg/t ore	1.74	$/kg	0.02
Fuel Oil	0.028	kg/t dry concentrate	0.74	$/kg (0.0137 t conc / t ore)	0.00	0.028	kg/t dry concentrate	0.73	$/kg (0.0137 t conc / t ore)	0.00
Flocculant Plant and Port (AF303)	0.068	kg/t dry concentrate	3.03	$/kg (0.0137 t conc / t ore)	0.00	0.068	kg/t dry concentrate	3.21	$/kg (0.0137 t conc / t ore)	0.00
NaOH	0.000	kg/t ore	1.91	$/kg	0.00	0.008	kg/t ore	1.83	$/kg	0.01
Carbon Dioxide (CO2)	0.038	kg/t dry concentrate	0.48	$/kg (0.0137 t conc / t ore)	0.00	0.001	kg/t dry concentrate	0.40	$/kg (0.0137 t conc / t ore)	0.00
Anti-Scalant	0.008	kg/t ore	1.74	$/kg	0.01					0.00
Plant general operating supplies			0.05	$/t av. assumption	**0.05**			0.05	$/t av. assumption	**0.05**
Electricity					**1.20**					**0.93**
Mill	20.600	kWh/t milled	0.055	$/kWh	1.14	25.346	kWh/t milled	0.035	$/kWh	0.89
Site & Services	0.450	kWh/t milled	0.055	$/kWh	0.02	0.579	kWh/t milled	0.035	$/kWh	0.02
Port	0.700	kWh/t milled	0.055	$/kWh	0.04	0.646	kWh/t milled	0.035	$/kWh	0.02
Maintenance parts			0.68	$/t av. assumption	**0.68**			0.68	$/t av. assumption	**0.50**
Mobile equipment			0.10	$/t av. assumption	**0.10**			0.10	$/t av. assumption	**0.10**
Labour	34.00	$M/annum	100.00	Mtpa	**0.34**	34.00	$M/annum	100.00	Mtpa	**0.32**
Assay laboratory	1.08	$M/annum	100.00	Mtpa	**0.01**	1.08	$M/annum	100.00	Mtpa	**0.02**
Subtotal, plus					**4.02**					**3.37**
allowance for tailings operation					**0.00**					**0.25**
sustaining capex allowance	606.70	$M	3,352.12	Mt	**0.18**					
power cost credit										
			TOTAL PROCESS OPERATING COST		**4.20**			**TOTAL PROCESS OPERATING COST**		**3.62**

Table 21-10 G&A operating costs, pit optimisation vs cashflow model

General & Admin Costs	Pit Optimisation					Cashflow Model				
	Consumption rates		Unit costs		$/t	Consumption rates		Unit costs		$/t
Finance					**0.37**					**0.15**
Labour					0.05					0.02
Services					0.04					0.02
Insurance					0.22					0.09
Other Operating Costs					0.06					0.02
External relations					**0.04**					**0.01**
Labour					0.01					0.01
Services					0.01					0.00
Other Operating Costs					0.01					0.00
Environment					**0.11**					**0.05**
Labour					0.02					0.01
Services					0.09					0.04
Other Operating Costs					0.01					0.00
Security					**0.09**					**0.04**
Labour					0.01					0.00
Services					0.08					0.03
Human Resources					**0.16**					**0.07**
Labour					0.04					0.02
Services					0.11					0.04
Other Operating Costs					0.02					0.01
Executive					**0.09**					**0.03**
Labour					0.04					0.02
Mobile Equipment					0.01					0.00
Services					0.01					0.00
Other Operating Costs					0.01					0.01
Site Services					**0.30**					**0.12**
Labour					0.14					0.06
Mobile Equipment					0.03					0.01
Fixed Equipment					0.04					0.02
Mine Allocation					0.05					0.02
Marine Services					0.04					0.02
Power					**0.05**					**0.04**
G&A	0.826	kWh/t milled	0.055	$/kWh	0.03	0.826	kWh/t milled	0.035	$/kWh	0.03
Site services	0.251	kWh/t milled	0.055	$/kWh	0.01	0.251	kWh/t milled	0.035	$/kWh	0.01
Subtotal, plus					**1.20**					**0.51**
sustaining capex allowance	181.02	$M	3,352.12	Mt	**0.05**					
power cost credit										
				TOTAL G&A OPERATING COST	**1.25**				**TOTAL G&A OPERATING COST**	**0.51**

21.2.3 Metal costs

In addition to royalties, metal costs for each of the copper and molybdenum concentrates comprise:

- concentrate transport charges (ocean freight)
- concentrate refining charges
- payable rates for each metal recovered into concentrate

Table 21-11 lists the metal costs adopted for the pit optimisation described in Item 15. Table 21-11 also lists updated information that was available for subsequent cashflow modelling following a more recent cost review by Metal Corp Trading AG.

Table 21-11 Metal costs, pit optimisation vs cashflow modelling

	Opt'n	Cashflow
Copper concentrate charges		
Cu con grade, %	25.0%	26.8%
Cu con overland freight, $/dmt	$0.00	$2.75
Cu con ocean freight, $/dmt	$40.00	$48.50
Cu con treatment, $/dmt	$70.00	$90.00
Cu refining, $/lb payable	$0.07	$0.09
Cu payable, %	96.43%	96.15%
TCRCs, $/lb Cu payable	**$0.277**	**$0.338**
Total metal cost		
Metal price, $/lb	$3.00	$3.07
royalty rate, %	5.0%	2.0%
Cu metal cost, $/lb Cu payable	**$0.43**	**$0.40**
Molybdenum concentrate charges		
Mo con grade, %	52.0%	52.0%
Mo con overland freight, $/dmt	$0.00	$2.75
Mo con ocean freight, $/dmt	$90.00	$90.00
Mo con treatment, $/dmt	$0.00	$0.00
Cu removal charge, $/lb	$0.00	$0.69
Mo refining, $/lb payable	$0.00	$0.55
Mo payable, %	86.20%	86.20%
TCRCs, $/lb Mo payable	**$0.091**	**$1.334**
Total metal cost		
Metal price, $/lb	$13.50	$8.83
royalty rate, %	5.0%	2.0%
Mo metal cost, $/lb Mo payable	**$0.77**	**$1.46**
Gold in concentrate charges		
Au refining, $/oz payable	$5.50	$5.10
Au payable, %	92.00%	90.00%
TCRCs, $/lb Au payable	**$5.060**	**$4.590**
Total metal cost		
Metal price, $/oz	$1,200	$1,310
royalty rate, %	5.0%	2.0%
Au metal cost, $/oz Au payable	**$65.060**	**$28.078**
Au metal cost, $/lb Au payable	**$948.77**	**$409.46**
Silver in concentrate charges		
Ag refining, $/oz payable	$0.40	$0.44
Ag payable, %	90.00%	90.00%
TCRCs, $/lb Ag payable	**$0.360**	**$0.398**
Total metal cost		
Metal price, $/oz	$16.00	$18.87
royalty rate, %	5.0%	2.0%
Au metal cost, $/oz Ag payable	**$1.160**	**$0.729**
Ag metal cost, $/lb Ag payable	**$16.92**	**$10.63**

The updated estimates accounted for:

- inclusion of a concentrate overland freight rate (for the distance between the plant site and the port; previously omitted)
- updated ocean freight rate for copper concentrate
- updated copper treatment and refining charges
- inclusion of a copper removal charge from the molybdenum concentrate (previously omitted)
- inclusion of a molybdenum refining charge (previously omitted)
- adjustment of the gold and silver payable percentages to reflect possible concentrate sales to a range of customers in Europe, Japan, China, India and South Korea
- correction of the royalty payment to be made on a net return basis (in conjunction with revised long term metal price projections)

ITEM 22 ECONOMIC ANALYSIS

22.1 Principal assumptions

In accordance with Part 2.3 (1) (c) of the Rules and Policies of Canadian National Instrument (NI) 43-101, the economic analysis set out below does not include Inferred Mineral Resources.

The economic analysis in the form of a simple cashflow model is intended to support the Mineral Reserve estimate, and in order to demonstrate a positive cashflow for each year of mining and processing. The development capital costs, sustaining capital costs and longer term rehabilitation costs are included in the model for completeness. The cashflow model forms part of a more comprehensive Project financial model which extends to depreciation, tax, financing and inter-company cashflows.

22.1.1 Production schedule

The production schedule forming the basis of the cashflow model is the same as that listed in Table 16-5 of Item 16.

22.1.2 Concensus metal pricing

The annual revenues in the cashflow model are calculated from late 2018 consensus pricing information from a number of banks and financial service companies, as listed in Tables 22-1 to 22-4.

Table 22-1 Consensus copper pricing information for cashflow modelling

Pricing Date	2019E ($/lb)	2020E ($/lb)	2021E ($/lb)	2022E ($/lb)	2023E ($/lb)	LT ($/lb)
11 Dec '18	2.84	2.95	3.28			3.10
18 Dec '18	3.25	3.51	3.13	3.18	3.25	3.25
18 Oct '18	2.86	2.86	2.86	2.86	n/a	2.86
17 Dec '18	3.08	3.25	3.40	3.50	3.70	
17 Dec '18	2.80	2.60				3.00
17 Dec '18	3.08	3.29				2.99
27 Nov '18	3.25	3.40				3.25
03 Dec '18	2.95	3.05				3.25
12 Dec '18	2.75	3.00				3.00
13 Dec '18	2.80	2.79				3.20
17 Dec '18	2.95	3.02	3.31	3.76	3.86	2.90
14 Dec '18	3.09	2.96	2.90	3.04	3.10	3.19
24 Dec '18	2.80	2.85	2.90			3.10
17 Dec '18	2.63	3.00	3.25	3.50		3.00
10 Oct '18	3.00	3.20	3.50	3.75		3.00
22 Oct '18	3.20	3.30	3.50	3.50		3.10
06 Dec '18	2.89	3.23	3.30	3.30		2.95
Average	**2.95**	**3.07**	**3.21**	**3.38**	**3.48**	**3.07**

Table 22-2 Consensus molybdenum pricing information for cashflow modelling

Pricing Date	2019E ($/lb)	2020E ($/lb)	2021E ($/lb)	2022E ($/lb)	2023E ($/lb)	LT ($/lb)
18 Dec '18	11.88	10.00	9.00	8.50	8.50	8.50
17 Dec '18	11.25	11.00	10.50	10.00	9.50	
11 Dec '18	11.25	11.51	11.78			
17 Dec '18	11.91	12.13	12.00	12.50	12.75	11.00
24 Dec '18	11.25	10.00	8.00			7.00
17 Dec '18	10.00	9.50	9.50	9.50		8.50
10 Oct '18	10.00	10.00	10.00	10.00		10.00
06 Dec '18	10.40	10.30	9.00	9.00		8.00
Average	**10.99**	**10.56**	**9.97**	**9.92**	**10.25**	**8.83**

Table 22-3 Consensus gold pricing information for cashflow modelling

Pricing Date	2019E ($/oz)	2020E ($/oz)	2021E ($/oz)	2022E ($/oz)	2023E ($/oz)	LT ($/oz)
11 Dec '18	1,269	1,300	1,300			1,350
18 Dec '18	1,283	1,250	1,530	1,200	1,200	1,200
18 Oct '18	1,222	1,259	1,305	1,344		1,385
17 Dec '18	1,300	1,300	1,300	1,300	1,300	
17 Dec '18	1,252	1,300				1,300
17 Dec '18	1,210	1,280				1,300
27 Nov '18	1,250	1,300				1,300
12 Dec '18	1,300	1,300				1,300
13 Dec '18	1,241	1,278				1,400
17 Dec '18	1,219	1,313	1,319	1,363	1,425	1,250
14 Dec '18	1,295	1,295	1,290	1,300	1,300	1,301
24 Dec '18	1,200	1,250	1,300			1,350
17 Dec '18	1,300	1,300	1,300	1,300		1,300
10 Oct '18	1,300	1,300	1,300	1,300		1,300
06 Dec '18	1,300	1,325	1,350	1,375		1,300
Average	**1,263**	**1,290**	**1,329**	**1,310**	**1,306**	**1,310**

Table 22-4 Consensus silver pricing information for cashflow modelling

Pricing Date	2019E ($/oz)	2020E ($/oz)	2021E ($/oz)	2022E ($/oz)	2023E ($/oz)	LT ($/oz)
11 Dec '18	16.00	18.00	18.00			18.00
18 Dec '18	15.75	17.00	17.50	18.00	18.25	18.25
18 Oct '18	14.83	15.23	15.93	16.03		16.03
17 Dec '18	17.50	17.50	17.50	17.50	17.50	
17 Dec '18	16.00	17.50				20.00
27 Nov '18	16.00	17.50				18.00
13 Dec '18	14.40	14.85	14.85			24.00
17 Dec '18	15.69	16.50	17.50	18.50	19.63	18.00
14 Dec '18	16.75	17.20	18.00	19.00	20.00	22.25
24 Dec '18	14.65	16.50	18.00			19.50
17 Dec '18	16.50	16.50	16.50	16.50		16.50
10 Oct '18	17.00	17.00	17.00	17.00		17.00
Average	**15.92**	**16.77**	**17.08**	**17.50**	**18.85**	**18.87**

22.2 Cashflow model inputs

22.2.1 Revenue inputs

In addition to the metal pricing information in Tables 22-1 to 22-4, other revenue related inputs to the model are listed below.

The overall average recoveries (after mining dilution, and where metal recoveries for each individual block are calculated according to a recovery algorithm, where a copper recovery downrate is applied to initial processing years, and where stockpiled saprock ore has a further 20% downrate applied) are:

- copper = 90.2%
- molybdenum = 53.4%
- gold = 55.9%
- silver = 45.0%

The payable metal factors are:

- copper = 96.15%
- molybdenum = 86.2%
- gold = 90.0%
- silver = 90.0%

22.2.2 Capital costs

According to the itemisation shown in Table 21-1, the updated Project capital cost of $6,300 M was included in the cashflow model, along with the $240 M estimated cost for 100 Mtpa expansion works in 2019 (and inclusive of mine development and engineering work allowances). Additional capital expenditure items as summarised in Table 21-1 and amounting to $429 M, were included in the model for the Project expansion period from 2020.

Mining, processing and G&A sustaining capital costs according to the allowances described in Item 21.1 were also included in the model, except for 2019 (the first year of production). As per the commentary in Item 21.1.3, a total sustaining allowance of $60 M was included for tailings dam works, split evenly between 2019 and 2020. A $148.6 M allowance for planned component replacements and for the sustaining of haul road trolley assist infrastructure was included for the period 2019 to 2026.

22.2.3 Operating and metal costs

With some differences for the initial schedule years, the modelled unit operating costs are essentially the same costs as summarised in Item 21[13]:

- mining waste = $1.76/t waste (overall average for all pits)
- mining ore = $1.94/t ore (overall average for all pits)
- processing = $3.84/t ore (overall average)
- G&A = $0.62/t ore (overall average)

[13] Relative to the mining cost information in Table 21-7, the modelled overall average ore and waste mining costs differ due to the inclusion of site budgeted costs for the period 2019 to 2023. The modelled average processing and G&A costs differ for the same reason.

- fixed cost for the port operations = \$10.3 M per year
- stockpile reclaim costs = \$1.19/t reclaimed

The modelled metal costs (including TCRC's and royalties) equate to the same costs as summarised in Item 21:

- copper = \$0.40/lb
- molybdenum = \$1.46/lb
- gold = \$28.08/oz
- silver = \$0.73/oz

Mining, process and G&A operating costs for 2019, in addition to metal costs for that year, have been expensed.

22.3 Cashflow model outcomes

The basic cashflow model to support the Mineral Reserve estimate is listed in Table 22-5. Table 22-6 summarises the average annual physicals and costs that relate to this model.

The Project is cashflow positive from 2020 and payback on the \$6,540 M capital spend occurs in 2024 (ie, payback on the capital inclusive of \$6,070 M spent, \$230 M of remaining capital for the 85 Mtpa expansion, and \$240 M for initial 100 Mtpa expansion and associated development and engineering costs) .

The total undiscounted cashflow for the Project, from the outset, is \$31,686.5 M. As at 31st December 2018, the NPV of the projected gross operating income (ie, excluding the precious metal stream) and capital expenditure of the project is \$14,307 M, calculated assuming mid-period cash flows.

The NPV is stated on the basis that the \$6,070 M of historic capital spend for the development and construction of the Project has been spent. In looking forward from the end of 2018, the adoption of an 8.5% discount rate is considered appropriate and aligned with rates recommended by the Company's independent valuation experts. With the exclusion of the historic capital spend from the discounted cashflow, the presentation of an IRR value is considered to be not applicable.

22.4 Project value and sensitivity analysis

A sensitivity analysis was completed as part of the pit optimisation work described in Item 15.4.5. The most sensitive optimisation variable is copper metal price (and recovery, since the magnitude of impact is the same). Continuing this analysis, and based on the cashflow model, Table 22-7 lists a number of Project variables and the base, minimum and maximum values of these adopted for further sensitivity analysis. The base development cost is the forward capital spend from 2019 (ie, excluding capital already spent). Table 22-8 summarises the impact on undiscounted cashflow of varying these parameters by +/- 10%. Table 22-9 summarises the impact on Project $NPV_{8.5}$.

As expected, copper price is the most sensitive of these variables; a 10% increase in average copper price over the life of the Project would increase the cashflow by 18%, and increase the $NPV_{8.5}$ by the same percentage. A 10% reduction in the copper price over the life of the Project would reduce the cashflow by 18% and decrease the $NPV_{8.5}$ by the same percentage.

Table 22-5 Mineral Reserve undiscounted cashflow model

Year		TOTAL	<2019	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053	2054	2055
MINING																																								
Total ore	Mt	**3,143.7**		76.0	94.2	101.0	109.8	124.8	109.1	110.9	123.8	123.4	108.2	119.8	116.3	111.1	102.8	103.7	102.8	104.8	110.6	114.5	101.6	106.5	105.6	103.6	81.9	75.5	63.1	62.0	50.7	66.4	66.3	61.0	56.7	37.4	27.2	10.7		
Total waste (incl. MW not reclaimed)	Mt	**3,157.8**		62.7	74.7	76.1	94.2	79.3	100.4	98.3	85.3	85.1	100.9	89.3	88.0	92.2	101.4	100.1	101.4	99.4	94.2	89.6	102.6	97.7	99.1	100.5	122.2	129.2	130.2	131.2	131.7	136.3	109.4	65.9	37.0	36.8	14.6	0.8		
Total Mining	Mt	**6,301.6**		138.7	169.0	177.1	204.0	204.1	209.5	209.3	209.1	208.4	209.1	209.1	204.3	203.2	204.2	203.8	204.2	204.2	204.7	204.2	204.2	204.2	204.7	204.2	204.1	204.7	193.3	193.1	182.5	202.7	175.7	127.0	93.7	74.2	41.8	11.4		
Strip ratio	W:O	**1.00**		0.83	0.79	0.75	0.86	0.64	0.92	0.89	0.69	0.69	0.93	0.75	0.76	0.83	0.99	0.97	0.99	0.95	0.85	0.78	1.01	0.92	0.94	0.97	1.49	1.71	2.06	2.12	2.60	2.05	1.65	1.08	0.65	0.98	0.54	0.07		
Stockpile reclaim	Mt	**334.0**		14.9	10.0	3.7	1.2	3.9	0.0	1.1	0.0	1.2	0.0	0.0	3.3	0.6	0.0	0.0	3.9	0.0	0.0	0.0	2.0	0.0	1.2	0.0	3.4	0.0	11.9	13.4	25.0	8.7	8.7	14.0	18.3	37.6	47.8	64.3	33.8	
PIT TO MILL DIRECT																																								
TOTAL	Mt	**2,813.1**		32.3	71.6	81.3	88.9	96.1	100.0	98.9	100.0	98.8	100.0	100.0	96.7	99.0	100.0	100.0	95.8	100.0	100.0	100.0	98.0	100.0	98.8	100.0	81.6	75.0	63.1	61.6	50.0	66.3	66.3	61.0	56.7	37.4	27.2	10.7		
Cu	%	**0.39**		0.40	0.41	0.41	0.45	0.46	0.44	0.42	0.41	0.37	0.42	0.45	0.42	0.43	0.44	0.38	0.39	0.35	0.37	0.39	0.38	0.39	0.34	0.35	0.33	0.33	0.34	0.35	0.36	0.35	0.39	0.36	0.40	0.39	0.31	0.55		
Mo	ppm	**60.19**		79.38	73.65	71.27	78.04	75.08	69.45	73.55	61.93	62.78	76.11	88.40	76.73	78.48	72.25	67.27	64.82	61.04	68.16	76.03	80.39	69.15	55.53	35.63	47.13	27.51	29.58	33.74	20.41	22.28	25.69	21.89	25.69	28.69	23.24	17.87		
Au	ppm	**0.07**		0.09	0.09	0.08	0.09	0.09	0.10	0.10	0.09	0.07	0.08	0.08	0.08	0.07	0.07	0.05	0.05	0.04	0.05	0.04	0.05	0.05	0.05	0.06	0.05	0.09	0.06	0.06	0.09	0.07	0.08	0.07	0.08	0.07	0.07	0.16		
Ag	ppm	**1.38**		1.38	1.33	1.31	1.39	1.41	1.39	1.34	1.36	1.32	1.38	1.48	1.46	1.43	1.52	1.65	1.60	1.59	1.54	1.52	1.38	1.46	1.32	1.32	1.40	1.16	1.01	0.93	1.34	1.07	1.24	1.14	1.27	1.31	1.21	2.16		
PIT TO STOCKPILE																																								
TOTAL	Mt	**330.6**		43.7	22.7	19.6	20.8	28.6	9.1	12.0	23.8	24.6	8.2	19.8	19.6	12.1	2.8	3.7	7.0	4.8	10.6	14.5	3.6	6.5	6.8	3.6	0.3	0.5	0.0	0.4	0.7	0.1	0.0	0.0	0.0	0.0	0.0	0.0		
Cu	%	**0.21**		0.24	0.20	0.16	0.17	0.22	0.20	0.26	0.23	0.21	0.26	0.16	0.16	0.15	0.23	0.23	0.18	0.29	0.22	0.20	0.27	0.31	0.20	0.21	0.30	0.33	0.44	0.23	0.16	0.52	0.00	0.00	0.00	0.15	0.00	0.00		
Mo	ppm	**41.31**		52.27	31.50	26.29	27.01	38.92	51.15	39.90	50.06	45.74	54.73	34.70	31.48	32.70	26.12	40.69	46.42	65.75	55.22	66.32	63.13	34.04	13.45	11.30	54.51	11.87	39.29	55.73	23.08	44.64	0.00	0.00	0.00	43.90	0.00	0.00		
Au	ppm	**0.05**		0.06	0.05	0.04	0.04	0.05	0.07	0.10	0.07	0.05	0.05	0.05	0.04	0.06	0.04	0.03	0.02	0.07	0.03	0.04	0.07	0.06	0.05	0.04	0.11	0.29	0.00	0.00	0.05	0.00	0.00	0.00	0.00	0.03	0.00	0.00		
Ag	ppm	**1.02**		1.04	0.92	0.74	0.80	1.00	1.07	1.16	1.00	0.94	1.45	1.01	0.88	0.68	1.42	1.34	1.28	1.27	1.23	1.30	1.49	1.37	1.00	0.82	0.66	1.66	0.97	0.93	1.17	0.79	0.00	0.00	0.00	0.59	0.00	0.00		
STOCKPILE TO MILL																																								
TOTAL	Mt	**334.0**		14.9	10.0	3.7	1.2	3.9	0.0	1.1	0.0	1.2	0.0	0.0	3.3	0.6	0.0	0.0	3.9	0.0	0.0	0.0	2.0	0.0	1.2	0.0	3.4	0.0	11.9	13.4	25.0	8.7	8.7	14.0	18.3	37.6	47.8	64.3	33.8	
Cu	%	**0.21**		0.32	0.27	0.28	0.35	0.47	0.00	0.43	0.00	0.34	0.00	0.00	0.29	0.29	0.00	0.00	0.26	0.00	0.00	0.00	0.26	0.00	0.51	0.00	0.35	0.00	0.16	0.17	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.20	0.33	
Mo	ppm	**41.38**		65.35	50.66	50.80	60.81	94.96	0.00	38.73	0.00	86.16	0.00	0.00	62.73	62.73	0.00	0.00	35.06	0.00	0.00	0.00	73.64	0.00	51.08	0.00	29.74	0.00	34.17	33.62	34.17	34.06	34.13	34.13	34.13	34.13	34.13	39.76	58.83	
Au	ppm	**0.05**		0.07	0.07	0.07	0.06	0.09	0.00	0.06	0.00	0.05	0.00	0.00	0.03	0.03	0.00	0.00	0.03	0.00	0.00	0.00	0.02	0.00	0.05	0.00	0.05	0.00	0.04	0.05	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.05	0.09	
Ag	ppm	**1.07**		1.20	1.53	1.54	1.18	1.37	0.00	1.76	0.00	1.29	0.00	0.00	1.51	1.51	0.00	0.00	1.48	0.00	0.00	0.00	1.42	0.00	1.68	0.00	1.19	0.00	0.93	0.96	0.93	0.93	0.93	0.93	0.93	0.93	0.93	1.04	1.39	
TOTAL FEED TO PLANT (after mining dilution & recovery)																																								
TOTAL	Mt	**3,147.1**		47.19	81.60	85.05	90.16	100.01	100.00	100.00	100.00	100.00	100.00	100.00	100.00	99.59	100.00	100.00	99.71	100.00	100.00	100.00	100.00	100.00	100.00	100.00	85.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	33.79	
Cu	%	**0.373**		0.38	0.40	0.40	0.45	0.46	0.44	0.42	0.41	0.37	0.42	0.45	0.42	0.43	0.44	0.38	0.38	0.35	0.37	0.39	0.38	0.39	0.34	0.35	0.33	0.33	0.31	0.31	0.29	0.33	0.36	0.32	0.34	0.28	0.21	0.25	0.33	
Mo	ppm	**58.19**		74.94	70.84	70.37	77.81	75.85	69.45	73.17	61.93	63.05	76.11	88.40	76.27	78.38	72.25	67.27	63.64	61.04	68.16	76.03	80.25	69.15	55.48	35.63	46.44	27.51	30.31	33.72	25.00	23.65	26.67	24.17	27.75	31.42	30.18	36.65	58.83	
Au	ppm	**0.07**		0.08	0.09	0.08	0.09	0.09	0.10	0.10	0.09	0.07	0.08	0.08	0.07	0.07	0.07	0.05	0.05	0.04	0.05	0.04	0.05	0.05	0.05	0.06	0.05	0.09	0.06	0.06	0.07	0.07	0.08	0.07	0.07	0.06	0.05	0.07	0.09	
Ag	ppm	**1.35**		1.32	1.35	1.32	1.39	1.41	1.39	1.35	1.36	1.32	1.38	1.48	1.46	1.43	1.52	1.65	1.60	1.59	1.54	1.52	1.38	1.46	1.32	1.32	1.39	1.16	1.00	0.93	1.20	1.06	1.20	1.10	1.19	1.12	1.03	1.20	1.39	
AVERAGE RECOVERIES																																								
Cu	%	**90.2%**		81.5%	91.0%	91.2%	91.9%	92.1%	91.9%	91.6%	91.2%	90.5%	91.5%	91.8%	91.3%	91.5%	91.6%	90.7%	90.8%	90.2%	90.6%	90.8%	90.8%	91.1%	90.2%	90.4%	89.7%	88.7%	88.1%	88.5%	89.2%	89.1%	89.4%	89.1%	89.5%	88.6%	88.1%	84.0%	71.8%	
Mo	%	**53.4%**		14.9%	54.9%	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%	54.8%	53.8%	53.3%	52.3%	52.0%	52.1%	52.4%	52.4%	54.3%	53.7%	58.5%	66.4%	59.2%	54.0%	60.7%	47.1%	51.3%	50.0%	48.3%	38.2%	
Au	%	**55.9%**		52.5%	56.3%	55.9%	58.9%	59.9%	61.4%	59.9%	59.7%	53.9%	56.0%	55.1%	54.8%	53.7%	54.2%	50.2%	50.9%	46.5%	50.6%	46.3%	46.5%	48.7%	50.5%	58.6%	49.8%	54.4%	57.8%	56.3%	61.4%	62.8%	62.7%	60.9%	63.3%	58.5%	57.5%	55.9%	47.0%	
Ag	%	**45.0%**		22.0%	47.2%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.3%	47.2%	46.9%	46.5%	44.4%	45.7%	36.8%	36.0%	39.2%	45.7%	40.5%	37.5%	42.2%	38.7%	38.3%	39.4%	38.9%	38.7%	
METAL RECOVERED																																								
Cu	kt	**10,586.4**		144.8	293.9	312.6	373.7	424.9	401.5	385.1	376.0	334.9	387.5	412.3	380.8	390.7	402.6	341.9	346.3	313.2	337.1	349.6	342.4	356.0	307.1	320.4	250.9	220.0	205.8	208.4	195.5	220.9	241.9	215.7	231.3	183.4	141.8	156.1	79.2	
Mo	kt	**97.8**		0.5	3.2	3.3	3.9	4.2	3.8	4.0	3.4	3.5	4.2	4.9	4.2	4.3	4.0	3.7	3.5	3.3	3.7	4.1	4.2	3.6	2.9	1.9	2.1	1.1	1.2	1.5	1.2	1.1	1.1	1.1	1.0	1.2	1.1	1.3	0.8	
Au	koz	**3,876.2**		65.1	125.8	125.1	157.2	181.0	201.4	184.3	180.5	116.1	143.3	133.8	130.7	125.1	125.9	84.0	86.2	63.8	87.7	66.3	67.5	73.1	73.3	111.9	67.5	114.8	78.6	76.8	107.0	104.5	118.1	95.8	110.4	81.7	73.3	92.8	45.9	
Ag	koz	**61,226.2**		440.9	1,674.3	1,707.2	1,906.2	2,141.6	2,112.6	2,049.0	2,062.3	2,009.1	2,101.7	2,247.6	2,227.1	2,171.6	2,309.5	2,502.3	2,424.6	2,414.5	2,343.9	2,307.2	2,103.4	2,206.6	1,972.8	1,890.3	1,738.5	1,030.3	867.0	880.4	1,326.2	1,032.4	1,087.1	1,117.0	1,108.8	1,030.9	979.5	1,121.7	579.9	
CONCENTRATE GRADE																																								
Cu	%	**26.8%**		26.8%	27.3%	27.5%	28.6%	28.8%	28.2%	27.9%	27.7%	26.7%	27.9%	28.5%	27.8%	28.0%	28.3%	26.9%	27.0%	26.2%	26.7%	27.0%	26.9%	27.2%	26.0%	26.3%	25.7%	25.8%	25.3%	25.4%	24.9%	25.8%	26.5%	25.6%	26.1%	24.5%	23.1%	23.9%	25.7%	
Mo	%	**52.0%**		52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	52.0%	
METAL PAYABLE																																								
Cu	kt	**10,193.0**		139.4	283.1	301.2	360.7	410.1	387.3	371.2	362.5	322.3	373.7	397.8	367.1	376.8	388.4	329.2	333.5	301.3	324.5	336.7	329.7	342.9	295.2	308.2	241.1	211.5	197.7	200.2	187.7	212.4	232.8	207.3	222.5	176.0	135.7	149.5	76.1	
Mo	kt	**84.3**		0.5	2.7	2.8	3.3	3.6	3.3	3.5	2.9	3.0	3.6	4.2	3.6	3.7	3.4	3.2	3.0	2.9	3.2	3.5	3.6	3.1	2.5	1.6	1.8	1.0	1.1	1.3	1.1	0.9	0.9	0.9	0.8	1.0	1.0	1.1	0.7	
Au	koz	**3,488.5**		58.6	113.2	112.6	141.4	162.9	181.2	165.9	162.5	104.5	129.0	120.4	117.7	112.6	113.3	75.6	77.6	57.4	79.0	59.6	60.7	65.8	65.9	100.7	60.8	103.3	70.8	69.1	96.3	94.0	106.3	86.3	99.4	73.5	66.0	83.5	41.3	
Ag	koz	**55,103.6**		396.8	1,506.9	1,536.5	1,715.5	1,927.4	1,901.3	1,844.1	1,856.1	1,808.2	1,891.5	2,022.9	2,004.4	1,954.4	2,078.6	2,252.1	2,182.1	2,173.1	2,109.5	2,076.5	1,893.1	1,986.0	1,775.5	1,701.3	1,564.6	927.3	780.3	792.3	1,193.6	929.2	978.4	1,005.3	997.9	927.8	881.6	1,009.5	521.9	
GROSS REVENUE (based on varying concentrate grade																																								
Cu	$M	$69,693.1		$907.9	$1,918.3	$2,133.0	$2,685.3	$3,143.1	$2,622.6	$2,514.0	$2,454.5	$2,182.9	$2,530.3	$2,694.0	$2,486.1	$2,551.3	$2,630.0	$2,229.0	$2,258.3	$2,040.2	$2,197.5	$2,280.2	$2,232.4	$2,322.4	$1,999.4	$2,087.4	$1,632.8	$1,432.2	$1,338.6	$1,355.7	$1,270.9	$1,438.1	$1,576.3	$1,403.7	$1,506.7	$1,191.6	$918.6	$1,012.6	$515.3	
Mo	$M	$1,680.9		$11.0	$63.7	$62.4	$72.7	$81.3	$64.1	$67.6	$57.2	$58.2	$70.3	$81.6	$70.4	$72.1	$66.7	$62.1	$58.6	$56.2	$61.6	$68.0	$70.4	$60.4	$48.5	$31.3	$34.7	$18.8	$20.5	$24.8	$20.9	$17.6	$18.1	$18.5	$16.4	$20.3	$19.0	$22.3	$12.8	
Au (assay deduction ignored)	$M	$4,565.7		$74.0	$146.1	$149.6	$185.3	$212.7	$237.4	$217.3	$212.8	$136.8	$168.9	$157.7	$154.1	$147.5	$148.4	$99.0	$101.6	$75.2	$103.4	$78.1	$79.6	$86.2	$86.4	$131.9	$79.6	$135.3	$92.7	$90.5	$126.2	$123.1	$139.2	$113.0	$130.2	$96.2	$86.4	$109.3	$54.1	
Ag (assay deduction ignored)	$M	$1,030.2		$6.3	$25.3	$26.2	$30.0	$36.3	$35.9	$34.8	$35.0	$34.1	$35.7	$38.2	$37.8	$36.9	$39.2	$42.5	$41.2	$41.0	$39.8	$39.2	$35.7	$37.5	$33.5	$32.1	$29.5	$17.5	$14.7	$14.9	$22.5	$17.5	$18.5	$19.0	$18.8	$17.5	$16.6	$19.0	$9.8	
Subtotal - Gross Revenue	**$M**	**$76,969.9**		**$999.2**	**$2,153.3**	**$2,371.2**	**$2,973.4**	**$3,473.4**	**$2,960.0**	**$2,833.6**	**$2,759.5**	**$2,412.0**	**$2,805.2**	**$2,971.5**	**$2,748.4**	**$2,807.7**	**$2,884.3**	**$2,432.5**	**$2,459.7**	**$2,212.5**	**$2,402.3**	**$2,465.5**	**$2,418.1**	**$2,506.5**	**$2,167.7**	**$2,282.7**	**$1,776.7**	**$1,603.8**	**$1,466.5**	**$1,486.0**	**$1,440.5**	**$1,596.4**	**$1,752.1**	**$1,554.1**	**$1,672.1**	**$1,325.6**	**$1,040.7**	**$1,163.3**	**$592.0**	
CAPITAL COSTS																																								
Development capital	$M	$6,300.0	$6,070.0	$230.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0																													
Expansion capital	$M	$669.0		$240.0	$100.0	$122.0	$89.0	$18.0	$33.0	$33.0	$34.0																													
Subtotal - Development Capital Costs	**$M**	**$6,969.0**	**$6,070.0**	**$470.0**	**$100.0**	**$122.0**	**$89.0**	**$18.0**	**$33.0**	**$33.0**	**$34.0**																													
SUSTAINING CAPITAL COSTS																																								
Sustaining (Mining, Processing, G&A)	$M	$2,493.9		$80.0	$90.0	$90.0	$112.0	$90.0	$90.0	$90.0	$90.0	$75.4	$75.5	$75.5	$74.3	$73.9	$74.3	$74.2	$74.2	$74.2	$74.4	$74.2	$74.2	$74.2	$74.4	$74.2	$71.1	$69.2	$66.3	$66.2	$63.4	$68.7	$61.7	$49.0	$40.4	$35.3	$26.9	$19.0	$7.5	
Closure and reclamation	$M	$124.2																																$15.3	$13.4	$17.7	$25.1	$20.5	$32.2	
Subtotal - Sustaining Capital Costs	**$M**	**$2,618.1**		**$80.0**	**$90.0**	**$90.0**	**$112.0**	**$90.0**	**$90.0**	**$90.0**	**$90.0**	**$75.4**	**$75.5**	**$75.5**	**$74.3**	**$73.9**	**$74.3**	**$74.2**	**$74.2**	**$74.2**	**$74.4**	**$74.2**	**$74.2**	**$74.2**	**$74.4**	**$74.2**	**$71.1**	**$69.2**	**$66.3**	**$66.2**	**$63.4**	**$68.7**	**$61.7**	**$64.3**	**$53.8**	**$53.0**	**$52.0**	**$39.5**	**$39.6**	
MINE OPERATING COSTS																																								
Mining ore	$M	$6,136.8		$136.0	$162.2	$171.6	$182.3	$234.2	$232.3	$227.6	$239.5	$238.2	$211.4	$240.6	$240.5	$235.4	$209.8	$197.9	$201.9	$209.5	$207.9	$214.5	$173.6	$182.5	$184.7	$178.6	$150.6	$147.6	$127.6	$123.3	$104.7	$140.1	$147.9	$143.3	$140.2	$97.5	$71.2	$30.1		
Mining waste	$M	$5,539.6		$99.0	$128.6	$129.6	$155.5	$148.5	$186.2	$178.9	$148.4	$148.8	$196.4	$170.8	$154.8	$155.9	$177.4	$171.5	$180.7	$180.9	$162.8	$148.4	$157.5	$148.1	$159.6	$167.2	$204.6	$226.5	$223.4	$234.0	$231.8	$247.1	$204.8	$127.5	$73.6	$76.2	$32.1	$1.9		
PROCESS AND G&A OPERATING COSTS																																								
Processing	**$M**	$12,088.2		$264.7	$375.5	$369.3	$380.2	$402.6	$402.5	$382.5	$372.5	$372.5	$372.5	$372.5	$372.5	$371.0	$372.5	$372.5	$371.5	$372.5	$372.5	$372.5	$372.5	$372.5	$372.5	$372.5	$318.2	$282.0	$282.0	$282.0	$282.0	$282.0	$282.0	$282.0	$282.0	$282.0	$282.0	$282.0	$132.7	
G&A	**$M**	$1,957.9		$122.0	$119.9	$115.2	$102.8	$92.0	$64.4	$50.8	$50.8	$50.8	$50.8	$50.8	$50.8	$50.6	$50.8	$50.8	$50.6	$50.8	$50.8	$50.8	$50.8	$50.8	$50.8	$50.8	$43.2	$38.1	$38.1	$38.1	$38.1	$38.1	$38.1	$38.1	$38.1	$38.1	$38.1	$38.1	$17.2	
Stockpile reclaim	**$M**	$397.3		$17.8	$11.9	$4.4	$1.4	$4.6	$0.0	$1.3	$0.0	$1.4	$0.0	$0.0	$4.0	$0.8	$0.0	$0.0	$4.7	$0.0	$0.0	$0.0	$2.4	$0.0	$1.4	$0.0	$4.0	$0.0	$14.2	$15.9	$29.8	$10.4	$10.3	$16.6	$21.8	$44.7	$56.8	$76.5	$40.2	
Subtotal - Operating Costs	**$M**	**$26,119.9**		**$639.5**	**$798.2**	**$790.1**	**$822.2**	**$881.9**	**$885.4**	**$841.1**	**$811.2**	**$811.8**	**$831.1**	**$834.7**	**$822.6**	**$813.7**	**$810.6**	**$792.8**	**$809.4**	**$813.8**	**$794.0**	**$786.3**	**$756.8**	**$753.9**	**$769.0**	**$769.1**	**$720.6**	**$694.1**	**$685.3**	**$693.3**	**$686.3**	**$717.6**	**$683.1**	**$607.4**	**$555.7**	**$538.5**	**$480.3**	**$428.6**	**$190.0**	
METAL COSTS (INCLUDING ROYALTIES)																																								
Cu	$M	$9,107.6		$123.9	$250.2	$267.0	$314.1	$357.0	$334.6	$323.7	$317.4	$289.2	$325.2	$341.4	$320.6	$326.9	$335.1	$294.1	$297.0	$274.0	$290.8	$299.5	$294.5	$304.0	$269.6	$279.1	$221.8	$194.3	$183.8	$185.9	$176.7	$195.1	$210.0	$191.4	$202.6	$167.4	$134.7	$145.0	$70.1	
Mo	$M	$315.5		$1.7	$10.4	$10.7	$12.6	$13.7	$12.3	$13.0	$11.0	$11.2	$13.5	$15.6	$13.5	$13.8	$12.8	$11.9	$11.2	$10.8	$11.8	$13.0	$13.5	$11.6	$9.3	$6.0	$6.7	$3.6	$3.9	$4.8	$4.0	$3.4	$3.5	$3.5	$3.2	$3.9	$3.6	$4.3	$2.4	
Au	$M	$108.8		$1.8	$3.5	$3.6	$4.4	$5.1	$5.7	$5.2	$5.1	$3.3	$4.0	$3.8	$3.7	$3.5	$3.5	$2.4	$2.4	$1.8	$2.5	$1.9	$1.9	$2.1	$2.1	$3.1	$1.9	$3.2	$2.2	$2.2	$3.0	$2.9	$3.3	$2.7	$3.1	$2.3	$2.1	$2.6	$1.3	
Ag	$M	$44.5		$0.3	$1.2	$1.2	$1.3	$1.6	$1.5	$1.5	$1.5	$1.5	$1.5	$1.6	$1.6	$1.6	$1.7	$1.8	$1.8	$1.8	$1.7	$1.7	$1.5	$1.6	$1.4	$1.4	$1.3	$0.8	$0.6	$0.6	$1.0	$0.8	$0.8	$0.8	$0.8	$0.8	$0.7	$0.8	$0.4	
Subtotal - Metal Costs	**$M**	**$9,576.4**		**$127.7**	**$265.3**	**$282.5**	**$332.4**	**$377.3**	**$354.1**	**$343.3**	**$334.9**	**$305.1**	**$344.2**	**$362.5**	**$339.4**	**$345.8**	**$353.1**	**$310.2**	**$312.4**	**$288.3**	**$306.7**	**$316.0**	**$311.5**	**$319.2**	**$282.4**	**$289.7**	**$231.6**	**$201.9**	**$190.6**	**$193.4**	**$184.7**	**$202.2**	**$217.6**	**$198.4**	**$209.7**	**$174.4**	**$141.1**	**$152.7**	**$74.3**	
CASHFLOW	**$M**	**$31,686.5**	**$6,070.0**	**$317.9**	**$899.8**	**$1,086.6**	**$1,617.8**	**$2,106.2**	**$1,597.4**	**$1,526.3**	**$1,489.3**	**$1,219.8**	**$1,554.4**	**$1,698.8**	**$1,512.2**	**$1,574.2**	**$1,646.4**	**$1,255.4**	**$1,263.7**	**$1,036.2**	**$1,227.2**	**$1,288.9**	**$1,275.6**	**$1,359.1**	**$1,042.0**	**$1,149.7**	**$753.3**	**$638.6**	**$524.3**	**$533.0**	**$506.2**	**$607.9**	**$789.7**	**$684.0**	**$852.9**	**$559.7**	**$367.3**	**$542.5**	**$288.1**	

Table 22-6 Mineral Reserve cashflow model average annual physicals and costs

MINERAL RESERVES - CASHFLOW SUMMARY	UNIT	TOTAL	< 2019 < Year 1	2019 to 2023 Year 1 to 5	2024 to 2028 Year 6 to 10	2029 to 2033 Year 11 to 15	2034 to 2028 Year 16 to 20	2039 to 2043 Year 21 to 25	2044 to 2048 Year26 to 30	2049 to end Year 31 to end
ANNUAL AVERAGES - PHYSICALS										
Mining tonnage, ore plus waste	Mtpa	**180.0**		178.6	209.1	204.9	204.3	204.4	189.5	69.6
Mill feed	Mtpa	**87.4**		80.8	100.0	99.9	99.9	92.0	75.0	91.7
Cu recovered	kt	**294.1**		310.0	377.0	385.7	337.7	290.9	214.5	167.9
Mo recovered	kt	**2.717**		3.0	3.8	4.2	3.7	2.3	1.2	1.1
Au recovered	koz	**107.7**		130.8	165.1	119.9	74.3	88.1	97.0	83.3
Ag recovered	koz	**1,700.7**		1,574.0	2,066.9	2,291.6	2,318.7	1,767.7	1,038.6	989.6
ANNUAL AVERAGE UNIT COSTS										
Mining ore	$/t mined	**$1.94**		$2.29	$2.44	$2.39	$2.07	$1.54	$1.01	$3.11
Mining waste	$/t mined	**$1.76**		$1.31	$1.49	$1.50	$1.55	$1.92	$3.70	$1.61
Processing (excluding sustaining)	$/t processed	**$3.84**		$4.44	$3.81	$3.73	$3.73	$3.73	$3.76	$3.77
G&A (excluding sustaining)	$/t processed	**$0.62**		$1.37	$0.53	$0.51	$0.51	$0.51	$0.51	$0.51
Stockpile reclaim	$/t rehandled	**$1.19**		$1.19	$1.19	$1.19	$1.19	$1.19	$1.19	$1.19
Metal costs (including royalties)	$/t processed	$3.04		$3.43	$3.36	$3.42	$3.07	$2.88	$2.64	$2.33

Table 22-7 Mineral Reserve cashflow model sensitivity parameters

Sensitivity parameter	Base	Minimum		Maximum	
Cu metal price ($/lb)	$3.09	90%	$2.78	110%	$3.40
Mo metal price ($/lb)	$9.04	90%	$8.14	110%	$9.95
Au metal price ($/oz)	$1,308	90%	$1,177	110%	$1,439
Mine operating costs ($/t mined)	$1.85	90%	$1.67	110%	$2.04
Plant operating costs ($/t processed)	$3.84	90%	$3.46	110%	$4.23
G&A operating costs ($/t processed)	$0.62	90%	$0.56	110%	$0.68
Metal costs (incl royalties) ($M)	$9,576	90%	$8,619	110%	$10,534
Development capital ($M)	$899	90%	$809	110%	$989
Sustaining capital ($M)	$2,494	90%	$2,245	110%	$2,743

Table 22-8 Mineral Reserve cashflow model, undiscounted cashflow sensitivity, from end 2018

	$M at -10%	$M Base	$M at +10%
Copper price	$30,927	$37,757	$44,586
Mo pric e	$37,592	$37,757	$37,921
Au pric e	$37,309	$37,757	$38,204
Mining costs	$38,924	$37,757	$36,589
Processing costs	$38,965	$37,757	$36,548
G & A costs	$37,952	$37,757	$37,561
Metal costs	$38,714	$37,757	$36,799
Development capital	$37,846	$37,757	$37,667
Sustaining capital	$38,018	$37,757	$37,495

Table 22-9 Mineral Reserve cashflow model, $NPV_{8.5}$ sensitivity, from end 2018

	$M at -10%	$M Base	$M at +10%
Copper price	$11,731	$14,307	$16,883
Mo pric e	$14,240	$14,307	$14,374
Au pric e	$14,134	$14,307	$14,480
Mining costs	$14,727	$14,307	$13,887
Processing costs	$14,739	$14,307	$13,875
G & A costs	$14,394	$14,307	$14,220
Metal costs	$14,661	$14,307	$13,953
Development capital	$14,388	$14,307	$14,226
Sustaining capital	$14,405	$14,307	$14,208

There is a view from the Company's geochemistry consultants (Piteau, 2018), that the value of saprock ore reclaimed from a long term stockpile may not be realised due to the degradation that could occur over such a long period. The sensitivity of deleting the saprock ore reclaim from the cashflow production schedule in the final two years of operation was therefore tested. Under these circumstances, the Project value would be reduced by less than 0.5%.

ITEM 23 ADJACENT PROPERTIES

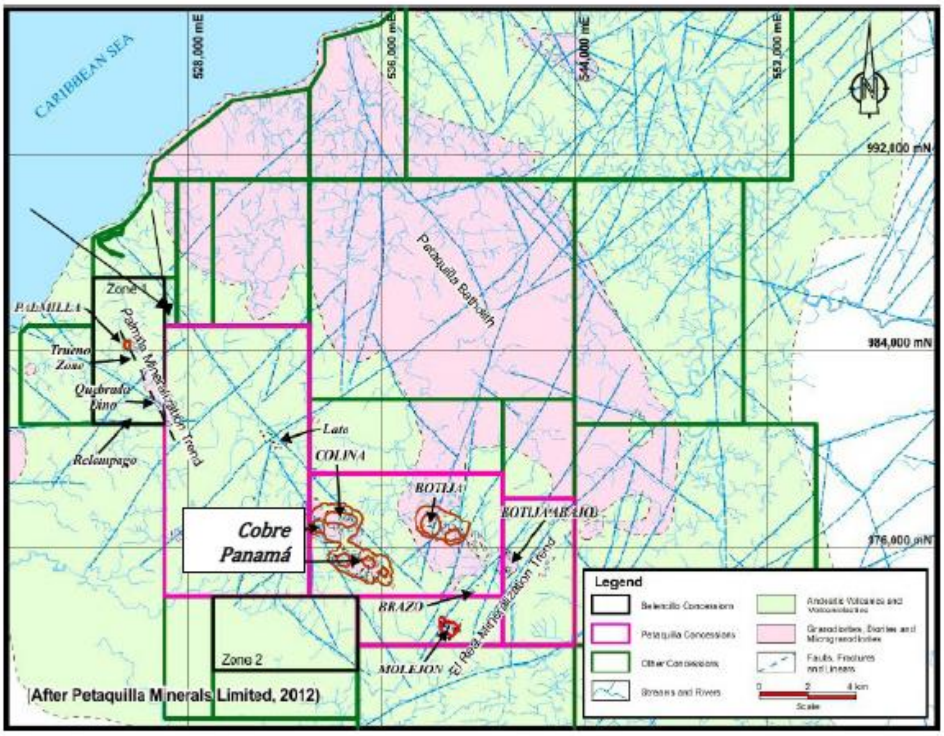
23.1 Introduction

The following information is reproduced in part from the 2015 Technical Report (FQM, July 2015). It has not been verified by the QPs for this Technical Report and the information is not necessarily indicative of the mineralisation on the Cobre Panamá property which is the subject of this Technical Report.

23.2 Regional mining and exploration

The Cobre Panamá deposits are located within a cluster of adjacent mineral deposits associated with intrusives and mineralisation related to granodiorite and andesite bodies, and a northwest-southeast structural trend (Figure 23-1). These adjacent properties include the Molejón and Palmilla deposits, being mined and explored respectively, by Petaquilla Minerals Ltd (PML) of Vancouver, Canada[14].

Figure 23-1 Location of adjacent properties (SGS Canada Inc, 2013)



There are no adjacent copper-producing properties. The closest significant copper-producing property is the Cerro Colorado porphyry copper deposit, located 120 km to the west.

23.3 Molejón

The Molejón gold deposit is located approximately 4 km south of the Botija deposit. The Molejón area is underlain by Tertiary volcanics and subvolcanic andesites, which have been intruded by feldspar-quartz porphyry dykes. Significant gold grades are associated with three northeast-trending structurally controlled zones of quartz-carbonate breccia and adjacent altered zones that dip at a shallow angle of 25° to the northwest. The main near-surface mineralised zone is approximately 10

[14] MPSA now owns the PML rights to the Palmilla deposit.

m thick. The deposit is interpreted to be a low-sulphidation, quartz-adularia epithermal gold deposit. Main zone veins locally exhibit typical banded cockade textures, and gold occurs as electrum. Economic mineralisation comprises the oxidized portion of the breccia, feldspar quartz porphyry, and feldspar andesite flows.

On May 8, 2012, PML released an updated NI 43-101 Technical Report for Molejón, in which Proven and Probable reserves are summarized as shown in Table 23-1.

Table 23-1 Molejón Mineral Resource estimate (reference: Berhe Dolbear, 2011)

Class	Tonnes	Au Grade (g/t)	Ounces
Proven	9,072,000	1.549	451,884
Probable	6,259,000	0.951	191,382
Total	15,331,000	1.305	643,266

23.4 Palmilla

The Palmilla copper-gold-silver deposit is located 10 km northwest of the Cobre Panamá deposits and 15 km southwest of the port development at Punta Rincon. The Mineral Resource listed in Table 23-2 has been reproduced from a 2013 NI 43-101 Technical Report (SGS Canada Inc., 2013).

Table 23-2 Palmilla Mineral Resource estimate (SGS Canada Inc, 2013)

	Tonnage	Au g/t	Cu %	Ag g/t	Au Eq. g/t	Ounces of Au	Pounds of Cu	Ounces of Ag	Ounces of Au Eq.
MEASURED	2,500,000	0.81	0.29	0.99	1.26	64,700	16,000,000	79,700	101,000
INDICATED	24,530,000	0.56	0.24	0.86	0.94	444,700	128,000,000	676,100	740,000
MEAS+IND	**27,020,000**	**0.59**	**0.24**	**0.87**	**0.97**	**509,400**	**143,900,000**	**755,800**	**841,000**
INFERRED	11,060,000	0.40	0.22	0.67	0.76	144,000	54,600,000	239,200	269,000
	COG: 0.35 g/t Au Equivalent **BASE CASE**								

- Numbers may differ due to rounding
- Gold equivalent (Au Eq.) is calculated using Au, Cu and Ag prices and recoveries
- Gold $1,400/oz., Copper $ 3.50/lb., Silver $ 30 /oz.
- Recoveries : Au 85%, Ag and Cu 75%
- Costs : $2.1/t mined, processing and G&A $14.00/t processed
- Fixed density of 2.67 t/m^3
- Royalties Au 4% and Cu 5% NSR

ITEM 24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant information or explanation required to make this Technical Report understandable and not misleading.

ITEM 25 INTERPRETATIONS AND CONCLUSIONS

25.1 Mineral Resource modelling and estimation

The Mineral Resource estimates for Botija have been updated prior to the issue of this Technical Report. The Botija estimate has benefited from additional close spaced reverse circulation drilling in the pre-strip areas. The resource estimates for the remaining Cobre Panama deposits remain unchanged and were all preceded by revisions to the geological models which focussed on the roles of structure, the phases of porphyry emplacement and the impacts of alteration in constraining copper and gold grades. In addition to the geological work, the assay quality information generated over the exploration and development history of Cobre Panamá was collated, analysed and documented. A core duplicate sampling study executed since the last resource estimate has supported assumptions and conclusions regarding the quality of the assay data.

Mineral Resource classification criteria remain unchanged from previous estimates and are based upon data quality, assay quality, sample spacing, geological continuity and estimation metrics. Apart from Botija there has been no change in the Mineral Resource classification. For Botija, the 2018 estimated combined Measured and Indicated Mineral Resource tonnes have reduced by 4% due to pre-stripping depletions. The added RC data at Botija has however increased the copper grades by 2.8%. The current Mineral Resource tabulation has been guided by the application of a cut-off grade derived from a series of pit shells based upon all resource categories.

In the opinion of David Gray (QP) the classifications applied to the mineralisation at Cobre Panamá fairly reflect the levels of geological and grade confidence.

25.1.1 Uncertainty and risk

Geological and structural

While the Cobre Panamá total Mineral Resource is large by world standards, it has been split into seven separate mineralisation occurrences, and in many areas the drill spacing remains relatively wide. Notwithstanding this, the broad mineralised envelopes are well-known and have been precisely defined. The influence of faulting on the mineralisation has been recognised since previous estimates, but the precise position of many of the post mineralisation faults still remains unclear. To this extent there are some risks associated with the geological interpretation. The full extents of each of the mineralised systems has not been fully defined, and future extensional drilling may result in small to moderate increases in resources and possibly some changes to the positions of mining infrastructure.

The risk in the overall tonnage and grade at Cobre Panamá is low.

Mineral Resource estimation

Risks associated with the Mineral Resource estimate largely pertain to the definition of the estimation domains from the geological model and interpretation, and as such are low. The estimation approach used reflects common practice among major mining houses and is believed to represent good to best practice in a global context.

25.2 Mine planning and Mineral Reserve estimation

The Mineral Reserve estimate for the Cobre Panamá Project is the product of a thorough and conventional process reflecting ultimate pit designs constrained by appropriate optimal pit shells. The pit optimisation process incorporated the best available information at the time, including variable processing recovery relationships determined from metallurgical testwork and analysis. Both planned and unplanned mining dilution were considered in the Mineral Resource modelling and optimisation process, respectively.

The pit optimisation work completed for the 2015 Technical Report (FQM, July 2015) has not been updated for this Technical Report. Original optimisation sensitivity analyses indicated that processing recovery and copper metal price were the most sensitive optimisation variables; there has been no material change to these particular variables since that time. Notwithstanding this, recent sensitivity analyses have been completed to assess the impact on the optimisations of updated operating costs and metal costs (TCRCs and royalties). These latest analyses confirm the original indications.

The ultimate pit designs take account of the IPCC concept and incorporate detailed crusher pocket layouts, haulage/tipping access and in-pit conveyor routes. Waste dumps and ore stockpiles have been included into the mine site layout plan, and haulage simulations and waste dump planning have been undertaken to optimise the mining fleet requirements and the mining/dumping sequence.

The in-pit development strategy has changed since that described in the 2015 Technical Report (FQM, July 2015). In place of phased pit development, mine planning now focusses on the development of terraced mining benches. From experience in the Company's Zambian operations, broad terraces rather than conical shaped pit phase cutbacks are considered to be more suited to the deployment of ultra class mining equipment. Furthermore, in the prevailing high rainfall environment of Panamá, the terraced bench strategy enables a means of better managing prolonged rainfall events and the potential for production disruptions.

Volume comparisons between the design ultimate pits and the corresponding pit shells indicate acceptable overall differences. There are localised zones however, for Botija Pit specifically, where the current ultimate design pit is outside of the optimal shell limits. This is a primarily related to the inclusion of IPCC infrastructure and the desire to minimise peninsulas within the pit which would otherwise create situations of temporary ore sterilisation. Engineering and mine planning work continues in this regard in order to arrive at a reasonable longer term crusher location within the Botija Pit mine plan as it approaches the ultimate pit limits.

There are related engineering plans for modifying the Botija Pit ultimate design, and these include a revision to the longer term in-pit conveyor route and the possibility of removing a surface run-off diversion channel which has been designed for the north side of the pit. Planning is also in progress for access to and the development of the Colina Pit, initial plant feed from which is currently scheduled for 2024. Part of this continuing planning effort relates to the future siting of IPCC infrastructure for Colina, taking into account similar planning considerations and constraints as those for the Botija ultimate pit.

In the opinion of Michael Lawlor (QP), therefore, the Mineral Reserve estimate reflects an achievable longer term mining plan and production sequence and one which has taken account of the newly devised terraced mining strategy. This revised strategy allows the desired processing capacity and plant feed grade to be achieved, whilst accounting for reasonable equipment usage profiles, longer term in-pit crusher relocations, planned ore and waste haulage profiles, and a practical stockpile building and reclaim strategy. As is conventionally required, short and medium term mine planning effort will be necessary for implementing practical terrace bench and access layouts.

It is recognised that iterations to the mine and related infrastructure plans will continue and that before committing to the ultimate mining limits, pit optimisations and Mineral Reserve estimates will be updated accordingly.

25.2.1 Uncertainty and risk

Mining

There is considered to be minimal risk attributable to the mining method and primary equipment selected for the Cobre Panamá Project. The method and equipment items are conventional and suitable for a large scale bulk mining project. The change to a terraced mining strategy is considered to be an enhancement to the effective utilisation of the ultra class equipment.

Mining, processing and metal costs

The mining costs that were originally estimated for pit optimisation and cashflow modelling as reported for the 2015 Technical Report (FQM, July 2015), were updated for this Technical Report and accounted for revised haulage profile definition and productivity information related to current mine designs and the purchased equipment fleet. The revised mining cost estimates continue to make allowance for potential trolley-assisted haulage savings (ie, faster cycle times and reduced fuel consumption based on experience at the Company's Kansanshi operations in Zambia).

Operating costs and metal cost inputs (ie, accounting for royalties, concentrate transport and refining charges) for optimisation and cashflow modelling were originally determined from MPSA cost estimates for a smaller ultimate scale of operations. The processing costs included general and administration costs (G&A costs). Process and G&A costs have now been updated to account for the changed scale of operations, and revised consumables consumption rates and unit costs.

In terms of operating cost uncertainty, the original optimisation sensitivity analyses indicated an approximate -3% and -4.5% impact on net value due to a 10% increase in mining and process operating costs, respectively. Sensitivity analyses were completed for the current cashflow model, incorporating the operating cost estimate updates. These analyses confirmed that a 10% increase in mining costs would impact on the undiscounted cashflow by -4%. The magnitude of impact would be the same for processing costs, whereas 10% increases in G&A costs and metal costs, would result in a -1% and -3% cashflow impact, respectively.

At these impact levels there is less risk to the selected pit shells, as the basis for all following pit design and production scheduling work, due to reasonable variances in mining, processing and metal costs. Optimal pit shell limits are more influenced by revenue related variables.

Future pit optimisations completed to reflect changes in mining strategy and infrastructure siting will take account of continuing updates to operating and metal costs.

Mine geotechnical engineering

The geotechnical engineering completed by CNI (March and April, 2013) was based on limited data drawn from drilling, mapping and laboratory testing. Additional geological structural information provided to CNI after the pit designs had been completed allowed CNI to review the designs in the context of these interpreted structures and modify their design recommendations accordingly.

As the Botija Pit pre-strip has progressed, pit wall batters have become progressively exposed in various lithological units such as saprolite, saprock and fresh rock. A review consultant (Peter O'Bryan and Associates) completed a reappraisal of the Botija Pit slope design parameters based on geotechnical information gleaned from these exposures and from updated stability assessments and calculations.

An outcome of this reappraisal has been the recommendation of an overall slope configuration, inclusive of lower batter heights, which is marginally steeper than the original CNI recommendation. Geotechnical risk for the Botija Pit overall slopes is managed by the continued recommended inclusion of wide berms at 90 m vertical intervals.

With the exception of Colina, no specific mine geotechnical investigations have been completed for the other Cobre Panamá deposits. The Botija Pit slope design parameters have been extrapolated to these proposed pits and hence there is some uncertainty as to the applicability of this extrapolated information.

Since the Botija and Colina Pits are to be mined first, it follows that actual operating experience will allow for these extrapolated parameters to be refined before mining of the other pits commences.

Hydrogeology

The hydrogeological work that was addressed in the 2015 Technical Report (FQM, July 2015) has been considerably enhanced through the establishment of a site water management team. Under the supervision of this team, a number of new bores have been drilled and equipped with piezometers. Two previously drilled production bores are expected to be pump tested during 2019, and the drawdown response measured by the expanded array of piezometers. These measurements are intended to provide the basis for subsequent hydrogeological modelling and drilling/equipping of new production bores.

From operating experience in the Botija Pit pre-strip, it is apparent that an effective means of pit slope depressurisation will be afforded by horizontal drain holes rather than by numerous vertical bores. The volume of groundwater that is expected to be piped away from horizontal drains and pumped from dewatering bores is not expected to be material in the context of the significant runoff into the pits from rainfall events.

In terms of rainfall and runoff inflows to the Botija Pit, the terraced bench mining approach is considered to be a more effective management approach to the risk of inundation, as opposed to the previous mining strategy of phased pit cutbacks. The proposed arrangement of staged pumping from the base of the deepening pit has also changed since the 2015 Technical Report. Booster pumping is now being considered rather than staged pumping into holding dams/tanks located at intermediate levels of the pit.

A major water course (the Botija River) is in the process of being diverted around the southern crest of the Botija Pit. There is a geotechnical risk associated with the siting of this diversion channel, and more specifically the diversion embankment site, immediately behind the southern crest of the open pit. This risk is being managed by the excavation of the embankment foundation down to competent rock and the engineered construction of the embankment itself.

Groundwater and rainfall inflow management practices at the other pits will likely follow the Botija Pit precedent. A surface stream diversion channel will eventually be required at Colina, and construction of diversion works at that site will likely be similar as for Botija.

25.3 Processing

The predominantly copper/molybdenum sulphide ore is amenable to conventional differential flotation processing, with lesser gold and silver recovered into the copper concentrate.

Various metallurgical test work has been undertaken on the Cobre Panamá Project since 1968, commensurate with the various levels of preliminary feasibility and prefeasibility studies that were completed up until 1998. In 1997 an extensive programme of metallurgical testing was designed to confirm earlier studies on the metallurgical response of the Botija and Colina ores, ie the first ores to be mined and processed.

Confirmatory batch laboratory flotation testwork was conducted during 2014 by ALS Metallurgy in Perth, Western Australia.

By virtue of the adopted conventional processing technology, the amount of test work (including confirmatory testwork) completed, and the adoption of variable process recovery relationships, there is considered to be minimal risk attributable to the processing of Cobre Panamá ores.

Based on results from recent geometallurgical sampling and testwork in the Botija Pit pre-strip, no change to recommended processing recovery relationships is warranted.

25.4 Environmental compliance

Extensive environmental studies and social impact assessments were completed between 2007 and 2012 for the 2010 ESIA process. Over 600 impacts and management commitments are addressed in the ESIA, during the construction, operations and closure phases.

The Project Environmental and Social Impact Assessment (ESIA) was approved by the *Autoridad Nacional del Ambiente* (National Authority of the Environment, ANAM; now referred to as *MiAmbiente*) in December 2011. The approved Project was for a mining operation comprising four open pits, at Botija, Colina, Medio and Valle Grande. Since then the Project definition and development scope has changed and the ESIA studies and documentation will need to be updated. The expected timeframe for the submission of a new ESIA is mid- 2024.

An environmental management system (EMS) has been developed to include the environmental and social management plans and commitments. From the construction into the operational phases, these EMS plans have been updated to reflect changes in site and operational conditions.

25.5 Permitting

Mine expansion, including mining of the Balboa, Botija Abajo and Brazo deposits, and adjacent waste dumping, will require access to additional properties covering up to 6,800 ha. MPSA intends to initiate the acquisition of these properties in accordance with the procedures established by Law No. 9 and other applicable Panamánian laws.

ITEM 26 RECOMMENDATIONS

26.1 Extensional drilling and grade control drilling

It is recommended that extensional drilling is carried out to resolve the mineralisation boundaries of the major orebodies in the Colina-Medio area. This is important to confirm the ultimate pit shells and thus the mining and infrastructure (overland conveyor route, stockpiles, etc.) footprint. At Botija, reverse circulation grade control drilling is recommended to continue at the 15 m grid spacing.

26.2 Mine planning and Mineral Reserves estimation

As an extension of the interpretive commentary in Item 25.2, there are several areas of continuous improvement that are recommended. This work is recommended to be completed in conjunction with the continuing engineering and mine planning studies for future mining infrastructure and layouts, as follows:

- review the ultimate pit designs to improve design efficiency (especially in terms of localised zones requiring waste reduction)
- optimise the IPC locations and conveyor alignments for future installations
- optimise haul road alignments and waste dump development to suit the application of trolley assisted haulage
- optimise the inpit development and external dump development sequence in order to reduce haulage requirements and improve operational efficiency

26.3 Cost modelling

In the 2015 Technical Report (FQM, July 2015) and as an item of continuous improvement, it was recommended that all operating costs be reviewed and analysed in terms of the production scale and profile described in the report. In particular, it was recommended that the mine operating costs should be re-assessed against the now developed IPCC concepts for ore haulage to specific locations over time, in successive iterations of LOM production plans and schedules.

These recommendations have been addressed to a certain extent, although continuing efforts will be required as the Project proceeds into the operational phase. As a case in point, for the revised operating cost estimates reported in this Technical Report, cognisance has been given to the inclusion of budgeted costs for the short to medium term, as estimated by the mine operations team.

26.4 Geotechnical engineering

The review items and recommendations by geotechnical consultants CNI (2014, 2015) have been followed up with confirmatory geotechnical mapping and observations of actual exposures in the Botija Pit. This mapping and observational review work is recommended to continue, along with the drilling and analysis of geotechnical information from proposed bores required to test the Botija Pit south wall infrastructure. Planning and logistics are underway in relation to the recommended geotechnical drilling for the Colina Pit access development and for the proposed initial IPC locations at that pit.

26.5 Hydrogeology

The hydrogeological work programme involving piezometer installations and water level monitoring, to be followed by groundwater modelling, is endorsed as a precursor before committing to expensive production bore siting and drilling. Considering the favourable performance observed from the installation of horizontal drains, a natural recommendation would be for the continued installation of these as a means of depressurising pit slopes.

ITEM 27 REFERENCES

ALS, 2014: *Metallurgy Report A155473, metallurgical testwork conducted upon samples from the Cobre Panamá Project,* May.

AMEC, 2007. *Petaquilla Project, Panamá NI 43-101 Technical Report*; AMEC Report, 174 p.

AMEC Earth & Environmental, 2010. *Open Pit Geomechanics, Mina De Cobre Project, Panamá SA.* Report to Minera Panamá SA, February.

Behre Dolbear, 2012. *A technical report on the Botija Abajo Project – A satellite deposit of the Molejon Mine*. Prepared for Petaqulla Minerals LTD. September.

Call & Nicholas Inc., 2015. *Geotechnical Review of the Botija Revision 14 Open-Pit Design*. Report to Minera Panamá SA, May.

Call & Nicholas Inc., 2014. *Geotechnical Review of the Botija Revision 9 Open-Pit Design*. Report to Minera Panamá SA, June.

Call & Nicholas Inc., 2013. *Geotechnical Slope Angle Study for the Proposed Botija Open Pit Mine.* Report to Minera Panamá SA, March.

Call & Nicholas Inc., 2013. *Saprolite and Saprock Slope Angle Recommendations for the Mina De Cobre Mining Project*. Report to Minera Panamá SA, April.

CIM, 2014. *Definition Standards – For Mineral Resources and Mineral Reserves,* May.

Cruden, A., 1998. *On the emplacement of tabular granites, Journal of the Geological Society*, London. Vol. 155, p. 853-862.

Davis, B., 2010. *Memorandum – Analysis of twin drill holes*, to Colin Burge of MPSA.

de Boer J.Z., M.S. Drummond, M.J. Bordelon, M.J. Defant, H. Bellon, and R.C. Maury, 1995. *Cenozoic Magmatic Phases of the Costa Rican Island Arc (Cordillera de Talamanca); in Geologic and Tectonic Development of the Caribbean Plate Boundary in Southern South America*, Mann, P, ed.; Geological Society of America, Special Paper 295, p. 35-56.

Ecometrix Inc., 2014. *DRAFT – 2014 Update to Predicted Water Quality of the TMF Discharge during Operations at Mina de Cobre, Panama*. Report to Minera Panamá S.A, April.

Escalante G., and Astorga A., 1994. *Geology of Eastern Costa Rica and Northern Panama*. Journal of Geology, Central America, special volume Terremoto de Limon, p. 1-14.

First Quantum Minerals Ltd, 2015. *Cobre Panamá Project. Colon Province, Republic of Panamá, NI 43-101 Technical Report*. July.

Francois-Bongarcon, D., 1995. *Memorandum - Petaquilla check assays*. Issued to Adrian Resources, October.

Golder Associates, 2017. *Closure Planning to inform and advance the Asset Retirement Obligation for the Cobre Panama Project at Project Initiation, as at December 2017*. Report to Minera Panamá S.A, December.

Golder Associates, 2010. *Environmental and Social Impact Assessment, Project Mina de Cobre Panamá*. Report to Minera Panamá S.A, September.

Gray, D., 2013. *Results from check sampling of skeleton Adrian/Teck drill core*. Internal report prepared by FQM. August.

Gustafson, L.B. and Hunt, J.P., 1975. *The porphyry copper deposit at El Salvador, Chile. Economic Geology*, v. 70, p. 857-912.

Halley, S., 2013. *The geochemistry and mineralogy of the Botija Deposit*, a powerpoint presentation to MPSA.

Jaacks, J and Candia, W., 2011. *Memorandum - Results of the Cobre Panama Project check analysis programme*. Issued to Freeport McMoran Exploration, Geochemical Applications International Inc., December.

JORC, 2012. *Australasian Code for Reporting of Mineral Resources and Ore Reserves, Effective December 2012.* Prepared by the Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC).

Kesler, S.E., J.F. Sutter, J.J. Issigonis, L.M. Jones, and R.L. Walker, 1977. *Evolution of Porphyry Copper Mineralization in an Oceanic Island Arc: Panamá*; *Economic Geology*, v. 72, p. 1142-1153.

Kirkby, M.X., Jones, D.S. and MacFadden, B.J., 2008. *Lower Miocene Stratigraphy along the Panamá Canal and Its Bearing on the Central American Peninsula.* PLoS ONE, 3(7): e2791 DOI: 10.1371/journal.pone.0002791.

Lakefiled, 1997. *The Recovery of Copper, Molybdenum and Gold from Petaquilla Project Samples.*

Love, D.A., 2011: *Rock types and geology of the Balboa area*, Internal MPSA report.

Lowell, D.J. and Guilbert, J.M. 1970. *Lateral and Vertical Alteration-Mineralization Zoning in Porphyry Ore Deposits*. *Economic Geology* vol. 65, pp 373-408.

McArthur, G.F., S. Harris, S. Kenwood, and D. Laudrum, 1995. *1994 Summary Report on the Petaquilla Project*; Adrian Resources Limited, Internal Report.

Mann, P. 1995. Preface: *Geologic and Tectonic Development of the Caribbean Plate Boundary in Southern South America*. Geological Society of America, Special Paper 295, p. xi-xxxii.

Model Earth, 2013a. *Mineralogical, metallurgical and physical property mapping in 3D*. Report prepared by Ian E. Neilson.

National Instrument (NI) 43-101, *Standards of Disclosure for Mineral Projects,* Supplement to the OSC Bulletin, April 8.

O'Bryan, Peter and Associates, 2018. *Cobre Panamá, Botija Open Pit Slope Analysis, Derivation of Fresh Rock Design Parameters.* Technical memorandum to First Quantum Minerals Ltd, October.

Panamá Mineral Resources Development Company, 1977. *Preliminary Feasibility Report of the Petaquilla Project*; PMRD Internal Report, 42 p.

Piteau Associates South Africa, 2017. *Preliminary Assessment of Post-Closure Water Balance and Discharge Chemistry, Cobre Panamá Mine, Rev II.* Memorandum report to First Quantum Minerals Ltd, November.

Piteau Associates South Africa, 2018. *Update of Acid Rock Drainage Risks and Management Strategy, Cobre Panamá.* Letter report to to Minera Panamá SA, February.

Reeves, R.A. 2012: *Evaluation of Adding a Ball Mill to an SABC-B Grinding Circuit to increase Copper Production.* Hazen Research October 29, 2012.

Rose, W.L., Davis, B. and Sim, R., 2010. *Mina de Cobre Panamá Project, Panamá, NI 43-101 Technical Report.* Unpublished report prepared by WLR Consulting, Inc, for Inmet Mining Corporation, May.

Rose, W.L., Davis, B. and Sim, R., 2012. *Mina de Cobre Panama Project, Panama, NI 43-101 Technical Report*. Unpublished report prepared by WLR Consulting, Inc, for Inmet Mining Corporation.

Rose, W.L., Davis, B. and Sim, R., *et al*, 2013. *2012 Mineral Resource & Reserve Update Report.* Unpublished report prepared by WLR Consulting, Inc, for Inmet Mining Corporation, March.

Schultz, C.P. 2012a: *Flotation Study on Copper-Gold-Molybdenum Ore Composites from the Balboa Orebody in Panama*. Hazen Research January 13, 2012.

Schultz, C.P. 2012b: *Progress Report of Flotation Study on Copper – Gold-Molybdenum Ore composites from the Brazo Orebody in Panama*. Hazen Research May 22, 2012.

Schultz, C.P. 2012c: *Gold Recovery in Rougher Flotation of Copper-Gold Molybdenum ore composites from the Balboa and Brazo orebodies in Panama*, Revision 1. Hazen Research September 20, 2012.

Schultz, C.P. 2012d: *Flotation Study on Copper-Gold-Molybdenum ore Composites from the Brazo Orebody in Panama*. Hazen Research October 26, 2012.

Schultz, C.P. 2012e: *Flotation Study on Copper-Gold-Molybdenum ore Composites from the Balboa Orebody in Panama*. Hazen Research October 29, 2012 (5).

Sillitoe, R. H., 2013. *Comments on geological models for the Cobre Panama porphyry copper deposits, Panama*. March.

Simons 1998. *Petaquilla Project Feasibility Study*. Issued to Teck Cominco, January.

SGS, 2012, *The Recovery of Copper and Gold from Samples of the Botija and Colina Deposits.*

SGS Canada Inc., 2013. *NI 43-101 Palmilla Deposit Resource Update, Rio Belencillo Zone 1 Concession, Colon Province, Panamá.* Report to Petaquilla Minerals Ltd, October 9.

Universidad de Panamá, Instituto de Geociencias, 2013. *Evaluation of the Study, Probabilistic Seismic Hazard Assessment (PSHA) for Cobre Panamá*. Report to Minera Panamá S.A., September.

ITEM 28 CERTIFICATES

David Gray
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I, David Gray, do hereby certify that:

1. I am the Group Mine and Resource Geologist employed by First Quantum Minerals Ltd.

2. This certificate applies to the technical report entitled "Cobre Panamá Project NI 43-101 Technical Report", dated effective 29th March 2019 (the "Technical Report").

3. I am a professional geologist having graduated with a Bachelor of Science degree with Honours (1988) in Geology from Rhodes University in Grahamstown, South Africa.

4. I am a Member of the Australasian Institute of Mining and Metallurgy and a Fellow Member of the Australian Institute of Geoscientists (FAIG).

5. I have worked as a geologist for a total of twenty five years since my graduation from university. I have gained over 15 years experience in production geology, over 5 years of exploration management of precious, base metal and copper deposits. Over the last ten years I have consulted to and held senior technical mineral resource positions in copper mining companies operating in Central Africa and worldwide.

6. I have read the definition of "qualified person" as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.

7. I most recently personally inspected the Cobre Panamá property described in the Technical Report in August 2018.

8. I am responsible for the preparation of those portions of the Technical Report relating to geology, data collection, data analysis and verification and Mineral Resource estimation (namely Items 3 to 12 and 14).

9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.

10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in the assurance of sampling QAQC, optimisation of estimation methods and the development of geology and mineralisation models.

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

12. 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 for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 29th dayof March 2019 at West Perth, Western Australia, Australia.

David Gray

Michael Lawlor
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I, Michael Lawlor, do hereby certify that:

1. I am a Consultant Mining Engineer employed by First Quantum Minerals Ltd.

2. This certificate applies to the technical report entitled "Cobre Panamá Project NI 43-101 Technical Report", dated effective 29th March 2019 (the "Technical Report").

3. I am a professional mining engineer having graduated with an undergraduate degree of Bachelor of Engineering (Honours) from the Western Australian School of Mines in 1986. In addition, I have obtained a Master of Engineering Science degree from the James Cook University of North Queensland (1993), and subsequent Graduate Certificates in Mineral Economics and Project Management from Curtin University (Western Australia).

4. I am a Fellow of the Australasian Institute of Mining and Metallurgy.

5. I have worked as mining and geotechnical engineer for a period in excess of twenty five years since my graduation from university. Within the last ten years I have held senior technical management positions in copper mining companies operating in Central Africa, and before that, as a consulting mining engineer working on mine planning and evaluations for base metals operations and development projects worldwide.

6. I have read the definition of "qualified person" as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.

7. I most recently personally inspected the Cobre Panamá property described in the Technical Report in September 2018.

8. I am responsible for the preparation of those portions of the Technical Report relating to Mineral Reserve estimation and Mining, namely Items 15 and 16, respectively, and for Items 1, 2, and 18 to 26.

9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.

10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in mine planning and the preparation of scoping studies, commencing in 2014.

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

12. 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 for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 29th day of March 2019 at West Perth, Western Australia, Australia.

M Lawlor

Michael Lawlor

Robert Stone
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I, Robert Stone, do hereby certify that:

1. I am Technical Manager employed by First Quantum Minerals Ltd.

2. This certificate applies to the technical report entitled “Cobre Panamá Project NI 43-101 Technical Report”, dated effective 29th March 2019 (the “Technical Report”).

3. I am a professional process engineer having graduated with an undergraduate degree of Bachelor of Science (Honours) from the Camborne School of Mines in 1984.

4. I am a Member of the Institue of Materials, Minerals and Mining (UK). I have been a Chartered Engineer through the Institute of Materials, Minerals and Mining since 1991.

5. I have worked as process engineer and metallurgist for a period in excess of thirty years since my graduation from university. For the last fifteen years I have been in the employ of First Quantum Minerals Ltd in both technical and managerial roles. Of these, seven years were as a manager of process plants producing copper in concentrate, copper as electrowon cathode, gold concentrate and cobalt metal by RLE. The remaining eight years were in a technical role as Consulting Process Metallurgist responsible for development of First Quantum Minerals Ltd projects worldwide including copper/cobalt in Central Africa, nickel in Australia and copper/molybdenum in Panama.

6. I have read the definition of “qualified person” as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a “qualified person” for the purposes of NI 43-101.

7. I most recently personally inspected the Cobre Panamá property described in the Technical Report in December 2018.

8. I am responsible for the preparation of those portions of the Technical Report relating to mineral processing/metallurgical testing and recovery methods, namely Items 13 and 17, respectively.

9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.

10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in project planning and the preparation of engineering studies, commencing in 2013.

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

12. 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 for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 29th day of March 2019 at West Perth, Western Australia, Australia.

Robert Stone