<u>NOTE</u>

The enclosed technical report, titled "Cobre Panamá Project -- Colón Province, Republic of Panamá -- NI 43 - 101 Technical Report" and dated as of June 30th, 2015 amends and restates in its entirety the technical report of the same name and date filed by First Quantum Minerals Ltd. on July 16, 2015 (the "**Superseded Technical Report**"). The enclosed technical report is being filed in order to clarify certain inconsistencies in the Superseded Technical Report regarding the capital cost of the Cobre Panamá Project, which among other things may have implied that the pre-acquisition costs and pre-mining costs of the Cobre Panamá Project were in addition to the total capital spend of $6.4 billion. Since that is <u>not</u> the case, the following changes have made in the enclosed technical report in order to clarify the inconsistencies in the Superseded Technical Report:

1. Table 1-5 on page 21 and Table 21-1 on page 196 have been altered for reasons of clarification. The narrative associated with each Table has been amended accordingly; and

2. The cash-flow tables, being Table 1-6 on page 23 and Table 22-1 on page 207, have been altered to delete reference to mining expenditures prior to 2017. A sentence corresponding to these changes has been included in the accompanying narrative on pages 22 and page 206. The sensitivity analyses in Table 22-2 have been adjusted accordingly.

Cobre Panamá Project

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

30th June 2015

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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). This is the first Technical Report prepared by FQM as an issuer, and in relation to the subject property.

1.1 Project location and ownership

The Cobre Panamá Project is located in the Donoso District 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 13,600[1] ha, and is the first large scale mining project undergoing construction in Panamá.

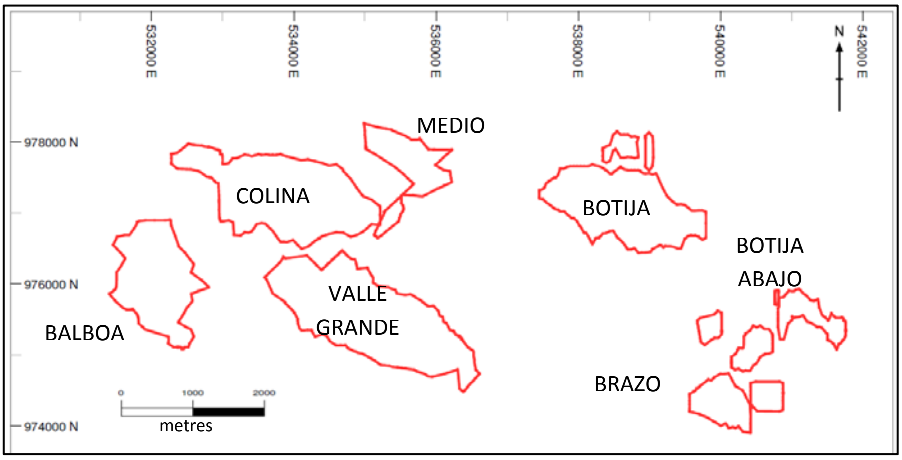
The Cobre Panamá Project is operated by Minera Panamá S.A. (MPSA or the Company), of which 80% is owned by a subsidiary company of FQM (FQM Akubra) and the remaining 20% by Korean Panamá Mining Company (KPMC), a consortium of Korea Resources Corporation and LS-Nikko Copper Inc. FQM acquired the property after a takeover of the previous owners, Inmet Mining Corporation (Inmet), in April 2013.

1.2 Project background

The Cobre Panamá Project, currently under construction, will involve large scale and conventional open pit mining at up to approximately 75 Mbcm of ore and waste mined per annum. The multiple pits will be mined in an optimised sequence and in phases, 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, with differential flotation to produce separate copper and molybdenum concentrate products. Plant tailings will be directed into areas of valley fill and into the depleted open pits. Figure 1-1 shows the location of each of the Cobre Panamá deposits as surface projections of the main ore zones.

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



[1] Of which approximately 700 ha relate to the Molejón gold deposit of Petaquilla Minerals Ltd (PML).

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 bulka bags.

Project power will be generated by a coal-fired power station at the port site and transmitted to the mine site along a new access corridor, which also incorporates the concentrate pipeline.

The current Project construction timeframe extends to Q3 2017, with initial production ramping-up to 74 Mtpa by Q1 2019, and thence to 90 to 100 Mtpa capacity by 2022. The expected Project life is 40 years to 2055.

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) in December 2011. The approved Project was for a mining operation comprising three open pits, at Botija, Colina and Valle Grande.

Since then, and prior to and following the acquisition of Inmet by FQM, 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, i.e. 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
- a production rate increase
- changes in material routing and scheduling in the mine plan
- increased site clearing for development of additional open pits, waste rock storage facilities, tailings expansion and for overland conveyors connecting to in-pit crushers
- 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 the end of 2016. Under the new Ministry of Environment Category III an ESIA has to be reviewed within six months of submittance.

MPSA has identified an additional 150 permits required for development of the Project, including permits for vegetation clearing, water usage and construction activities. Permitting for the three powerline segments from the port site to the mine has been approved by ANAM, although additional permits are required for vegetation clearing and construction. Both permits have been granted for one of the three segments, whilst clearing permits are pending on the other two. Municipal permits will be required in certain areas of the line within each of the three segments.

The 2010 ESIA (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.

1.4 Project development status

Since the FQM acquisition of Inmet, the prime technical focus has been on the key elements of Project site development and infrastructure construction.

Aspects of the Project have changed since the acquisition, including an increase in the processing capacity from 60 Mtpa to 74 Mtpa over the first five years of Project life. Thereafter, a further expansion to up to 90 to 100 Mtpa capacity is now planned, reverting to 74 Mtpa from around year 30.

The development approach for the Project has also changed from an outsourced approach to an in-house, self-perform arrangement, whereby third party engineers and contractors are utilised only for identified specific tasks and work, as per the Company's preferred project execution model.

Site construction earthworks have been the subject of critical review, as has the methodology for initial excavations and construction. Significant quantities of on-site equipment had been purchased from suppliers prior to the acquisition, and these contracts have now been either cancelled or modified. The Company has taken control of all site development activities, thereby reducing the risk of delivery and cost overruns.

The locations of key site infrastructure, including the processing plant, have been reviewed and more practical sites selected to allow for more cost effective construction and better access to the now proposed in-pit crushing and conveying facilities.

The Project is scheduled for construction completion and commissioning in the second half of 2017. Target development timeframes are as follows:

- Q2 2016 – completion of 230kV overland power line
- Q1 2017 – completion of 300MW power station
- Q2 2017 – completion of tailings management facility
- Q3 2017 – completion of process plant construction
- Q4 2017 – process commissioning and first concentrate production

1.5 Geology and mineralisation

The Cobre Panamá Project consists of numerous 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 has led to the 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 1,805 diamond drillholes totalling 346,294 m have been drilled from discovery to August 2013, with many of the deposits drilled to a spacing of 50 m by 50 m, to 200 m by 200 m or greater.

The porphyry deposits occur at the southern margin of a large granodiorite batholith of mid-Oligocene age (36.4 Ma). Mineralisation is hosted in a variety of lithologies, including granodiorite, feldspar-quartz-hornblende porphyry and adjacent andesite volcanics. At many of the deposits the host lithologies and mineralisation have been cross-cut by later dykes of either andesitic or felsic composition. Hydrothermal alteration is primarily silica-chlorite, which is interpreted to be a form of propylitic alteration. Local potassic alteration, consisting of potassium feldspar and secondary biotite, is to be found at Botija. Phyllic and argillic alteration is patchy throughout the four main deposits. High grade mineralisation is associated with intense quartz stockworks.

The most dominant copper sulphide associated with mineralisation is chalcopyrite, with lesser bornite. Sulphides typically occur as dissemination, micro-veinlets, fracture fillings and quartz-sulphide stockworks. Traces of molybdenite are commonly found in quartz veinlets. There is no significant supergene enrichment of copper at Botija, Colina, Valle Grande or Balboa. 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 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 programs 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%

1.7 Mineral Resource summary

Block model resource estimates for Valle Grande, Colina, Medio and Balboa were completed in January 2014 by consultants from Optiro Pty Ltd (Optiro). The Botija, Botija Abajo and Brazo estimates were completed by FQM geologists in January 2014. Supporting the Mineral Resource estimation, a review of the sample preparation methodology, sample analyses and security was completed by Optiro. All of this work was completed under the supervision of David Gray (QP) of FQM.

The Mineral Resource estimates have been generated from drillhole sample assay results and the interpretation of geologic 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 a panel size of 50 mE by 50 mN on 15 m benches, which is 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; this is considered appropriate to the expected 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, and 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 below. The Mineral Resources have been reported inclusive of Mineral Reserves.

Table 1—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**

1.8 Mineral Reserves summary

The detailed mine planning for the Project, including conventional optimisation processes, phased and ultimate pit 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. All mined sulphide ore (initially above an elevated cut-off grade) 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. 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.

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 phased pit designs were developed using the ultimate and selected intermediate pit shells. The ultimate and phased 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.

Detailed life of mine production scheduling was then completed to demonstrate an achievable mine plan and hence allow the reporting of a Mineral Reserve as stated in Table 1-3, 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).

Table 1—3 Cobre Panamá Project Mineral Reserve statement, at June 2015

MINERAL RESERVE AT 30th JUNE 2015 (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	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 &	Proved									
MEDIO	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	603.5	0.35	70.79	0.07	1.10	2,124.5	42.7	1,289.8	21,377.4
	Probable	2,579.0	0.38	56.95	0.07	1.41	9,876.4	146.9	5,651.1	117,024.7
	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**

A cut-off optimisation strategy was adopted for the Mineral Reserves estimation process, whereby an elevated 0.2%Cu cut-off grade was adopted for the period up to 2040, then followed by a period of marginal cut-off grade plant feed for the remainder of the Project life. Whilst the actual marginal cut-off grade varies according to varying process recovery relationships, the overall average cut-off grade over the life of mine at the metal prices listed in Table 1-3, is in the order of 0.19%Cu.

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) commences in July 2015 and processing commences in July 2017. The Project life is 40 years to 2055.
- The total material mined from all pits amounts to 6,465.9 Mt (2,516.8 Mbcm), of which 3,182.5 Mt is ore (including saprock ore, high grade ore, low grade ore) and 3,283.4 Mt is waste (including saprolite, saprock waste and mineralised waste).
- The direct feed ore is 2,785.2 Mt at a grade of 0.40%Cu and 397.3 Mt at a grade of 0.19% Cu is stockpile reclaim.
- The total HG and LG ore mined to stockpiles and reclaimed throughout the Project life is 49.9 Mt at a grade of 0.27% Cu.
- In addition, 223.9 Mt of ultra low grade ore at a grade of 0.17% Cu is mined to stockpile and reclaimed in the final years of the Project.
- The crusher feed ramps up from 2017 to 74 Mtpa by 2019, at which level it remains until 2021. The feed rate then ramps up to 90 Mtpa by 2023, at which level it stays until 2046.
- The rate drops to 74 Mtpa between 2047 and 2054.
- In terms of total plant feed, the average copper grade is 0.42%Cu for the first twenty years, and then 0.32%Cu for the remaining Project life.
- Ignoring the 2017 commissioning year, the average annual copper metal production in the first twenty years is 328,000 tonnes. Thereafter, the annual average is 228,000 tonnes.
- The annual average by-product production is approximately 2,570 tonnes of molybdenum, 97 thousand ounces of gold and 1,570 thousand ounces of silver.
- The total waste mined includes 313.1 Mt of mineralised waste at a grade of 0.12% Cu.
- The overall life of mine strip ratio (tonnes) is 1 : 1.

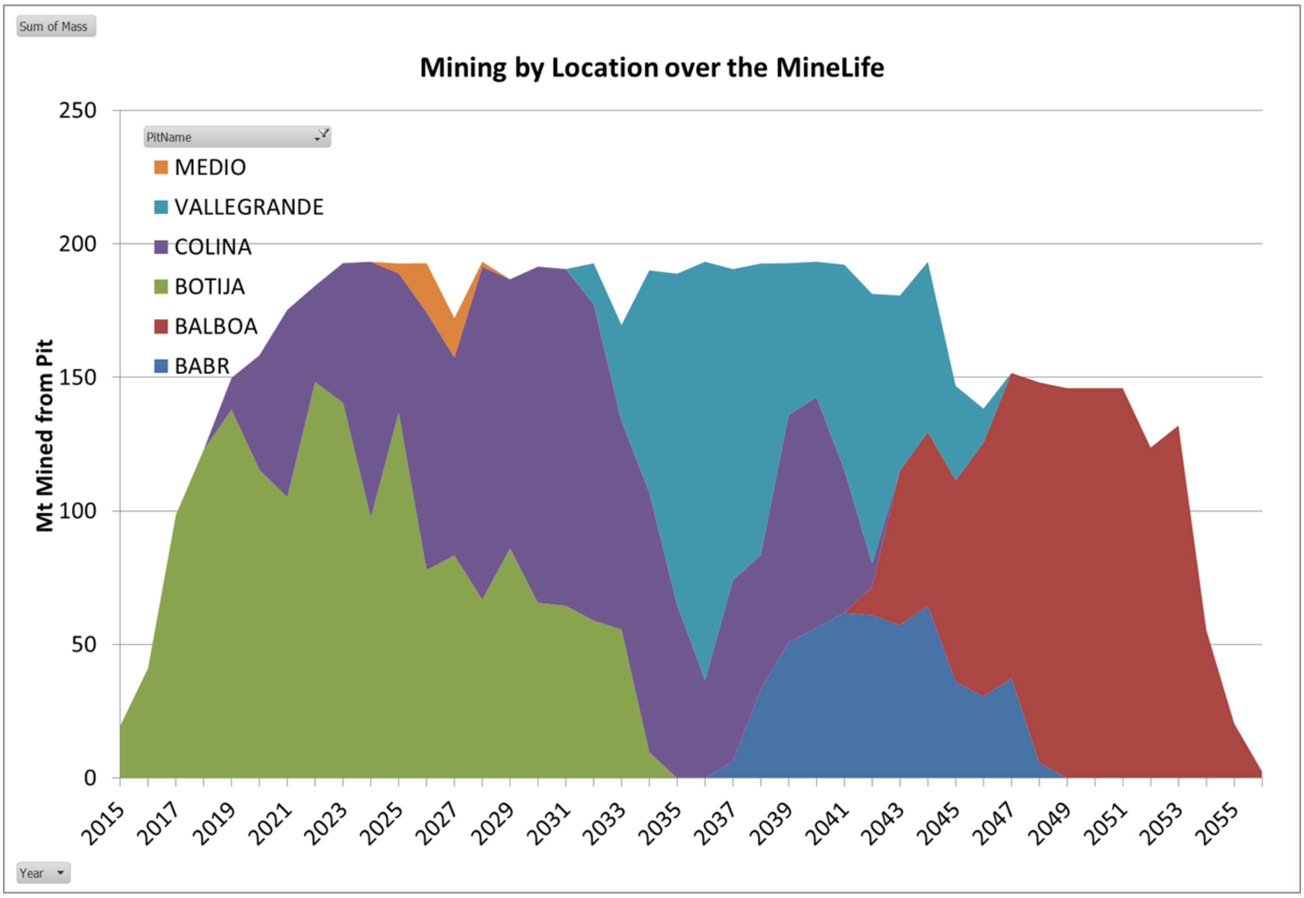
Table 1-4 summarises the life of mine production schedule, whilst Figure 1-2 shows the life of mine mining sequence. Table 1-4 shows that the maximum process rate is around 90 Mtpa. In the event that the plant capacity can process up to 100 Mtpa, there is the potential for currently Inferred Mineral Resource to be included as future plant feed. Alternatively, and on the proviso that this resource can be upgraded to at least an Indicated classification, the Project life could be extended.

A hypothetical inventory inclusive of Inferred Mineral Resource does not form part of the Mineral Reserve estimation process described in this Technical Report.

1.10 Environmental and social summary

The Project site is located in 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. 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.

Figure 1-2 Graph showing mining sequence



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. Through the construction and operational phases, these EMS plans will be updated to reflect changes in site and operational conditions.

As part of the social impact programme, two indigenous communities (300 people) and dispersed Latino people (190) have been subject to physical and economic resettlement in relation with the acquisition of the land polygon of 7,477.54 ha of the mine site.

Table 1—4 Cobre Panamá Project life of mine production schedule

Year	Mined Ore	Grade	Metal	Crusher Feed	Grade	Metal	Cu Metal Recovery		Stockpile Movements												Waste		Ore+Waste	
	(t)	(%TCu)	(t Cu)	(t)	(%TCu)	(t Cu)	(%)	(t)	HG/LG On (t)	HG/LG Off (t)	Balance (t)	ULG On (t)	ULG Off (t)	Balance (t)	Saprock On (t)	Saprock Off (t)	Balance (t)	MW On (t)	MW Off (t)	MW Balance (t)	(t)	Strip Ratio	(bcm)	(t)
2015	19,050	0.10	19												19,050		19,050	128,700		128,700	19,571,030	1,027.4	11,158,053	19,590,080
2016	6,309,300	0.15	9,492						873,840		873,840	636,990		636,990	4,798,471		4,817,521	1,136,520		1,265,220	34,840,827	5.5	19,734,430	41,150,127
2017	41,583,647	0.24	100,939	17,461,029	0.33	57,729	87.9	50,741	2,604,374	1,939,193	1,539,021	8,975,311		9,612,300	14,482,126		19,299,647	9,417,630		10,682,850	56,945,941	1.4	44,643,999	98,529,587
2018	88,877,238	0.36	318,825	70,052,091	0.41	284,658	89.3	254,126	2,209,739	1,539,021	2,209,739	15,356,076		24,968,376	2,798,353		22,098,000	11,139,337		21,822,188	34,092,718	0.4	46,969,152	122,969,956
2019	98,087,634	0.41	399,705	73,995,186	0.47	348,592	90.1	314,210	12,528,701	3,883,053	10,855,388	10,750,975		35,719,351	4,695,825		26,793,825	7,801,088		29,623,276	51,804,102	0.5	63,037,449	149,891,736
2020	80,075,822	0.45	362,927	74,202,840	0.49	364,749	90.4	329,846	2,251,725	11,671,722	1,435,390	6,762,199		42,481,550	8,530,780		35,324,605	5,424,938		35,048,213	78,248,441	1.0	66,999,720	158,324,263
2021	88,234,181	0.43	383,060	74,314,748	0.49	360,592	90.4	325,849		1,435,390		8,026,478		50,508,028	7,328,344		42,652,949	6,954,610		42,002,823	87,118,375	1.0	68,930,187	175,352,556
2022	94,859,219	0.38	355,758	86,999,940	0.39	339,977	89.3	303,607	2,724,384		2,724,384	4,410,995		54,919,023	723,900		43,376,849	8,827,490		50,830,314	89,529,915	0.9	70,541,498	184,389,134
2023	109,021,499	0.36	391,188	89,627,154	0.40	354,418	89.1	315,919		2,724,384		13,574,804		68,493,827	8,543,925		51,920,774	8,487,224		59,317,538	83,791,932	0.8	75,237,179	192,813,431
2024	109,448,816	0.40	433,836	87,956,361	0.44	391,347	89.7	351,179				10,533,103		79,026,930	10,959,353		62,880,127	6,879,529		66,197,066	83,892,871	0.8	75,449,461	193,341,688
2025	96,918,769	0.43	418,962	88,012,816	0.44	387,224	89.8	347,800				81,881		79,108,811	8,824,072		71,704,199	5,508,131		71,705,198	95,778,112	1.0	73,993,347	192,696,881
2026	93,894,143	0.41	381,224	89,999,875	0.41	369,508	89.4	330,345	740,103		740,103	887,215		79,996,026	2,266,950		73,971,149	8,572,411		80,277,609	98,919,288	1.1	74,447,215	192,813,431
2027	92,969,879	0.48	444,966	89,999,875	0.49	439,264	90.4	396,936	1,276,903		2,017,007	902,526		80,898,552	790,575		74,761,724	5,723,455		86,001,064	79,232,260	0.9	67,602,953	172,202,139
2028	97,595,045	0.42	412,442	90,246,450	0.44	397,960	89.8	357,460	3,294,459		5,311,466	1,701,460		82,600,013	2,352,675		77,114,399	8,772,463		94,773,527	95,746,643	1.0	75,503,021	193,341,687
2029	87,126,228	0.39	342,538	88,675,319	0.40	350,402	89.2	312,433		5,311,466				82,600,013	3,762,375		80,876,774	5,189,925		99,963,452	99,664,962	1.1	73,430,441	186,791,190
2030	97,973,705	0.39	378,701	89,999,875	0.40	363,283	89.1	323,596	74,230		74,230	6,423,224		89,023,237	1,476,375		82,353,149	8,505,113		108,468,565	93,545,717	1.0	72,425,643	191,519,422
2031	105,097,338	0.43	448,732	89,999,875	0.46	414,368	89.9	372,509	5,856,166		5,930,396	9,184,147		98,207,384	57,150		82,410,299	9,589,954		118,058,519	85,533,122	0.8	71,842,574	190,630,461
2032	97,609,013	0.38	367,101	90,246,450	0.40	359,728	89.0	320,204		2,956,789	2,973,607	10,319,352		108,526,736			82,410,299	6,598,796		124,657,315	95,130,349	1.0	72,946,498	192,739,362
2033	101,838,220	0.37	372,613	89,999,875	0.40	356,236	89.0	316,961		1,490,532	1,483,075	11,052,402		119,579,138	2,276,475		84,686,774	7,889,400		132,546,715	67,822,007	0.7	64,395,694	169,660,226
2034	102,022,641	0.38	388,631	85,397,402	0.43	368,430	89.5	329,794		1,483,075				119,579,138	18,108,314		102,795,088	3,604,343		136,151,059	88,122,239	0.9	72,456,248	190,144,880
2035	102,085,272	0.37	375,821	88,838,824	0.39	349,889	88.9	311,176						119,579,138	13,246,448		116,041,536	4,830,908		140,981,966	86,787,189	0.9	71,617,284	188,872,461
2036	93,898,233	0.39	370,191	90,246,450	0.40	361,976	89.2	322,845	1,362,478		1,362,478	611,368		120,190,506	1,677,937		117,719,473	5,982,167		146,964,133	99,443,458	1.1	72,990,184	193,341,691
2037	96,800,878	0.38	365,295	89,999,875	0.39	349,856	89.1	311,767	4,511,700		5,874,178	1,941,641		122,132,147	347,662		118,067,136	7,162,594		154,126,728	93,784,009	1.0	72,577,652	190,584,887
2038	90,193,244	0.39	347,466	86,431,517	0.40	344,577	88.7	305,513		5,874,178		8,854,856		130,987,002	781,050		118,848,186	5,154,171		159,280,898	102,487,616	1.1	74,185,403	192,680,861
2039	111,518,207	0.35	385,034	89,999,875	0.38	341,087	88.0	300,200	4,601,580		4,601,580	14,878,402		145,865,404	2,038,350		120,886,536	9,846,721		169,127,619	81,295,225	0.7	73,542,974	192,813,432
2040	103,764,362	0.35	361,341	90,246,450	0.38	339,295	87.6	297,204		4,362,897	238,683	16,705,770		162,571,175	1,175,039		122,061,574	9,896,746		179,024,365	89,577,326	0.9	74,036,274	193,341,688
2041	91,427,895	0.36	332,994	89,999,875	0.37	330,185	87.7	289,736	566,059		804,742	513,000		163,084,175	348,961		122,410,536	6,576,245		185,600,611	98,042,842	1.1	74,622,109	189,470,738
2042	86,467,158	0.34	297,662	87,114,738	0.34	299,637	87.8	263,043		804,742				163,084,175	157,162		122,567,698	10,347,130		195,947,740	92,371,771	1.1	71,690,430	178,838,929
2043	108,818,666	0.32	344,432	89,369,362	0.35	310,645	88.1	273,565				19,273,092		182,357,266	176,212		122,743,911	10,088,333		206,036,073	83,858,011	0.8	74,908,367	192,676,677
2044	81,422,701	0.32	259,671	90,246,450	0.30	275,063	87.3	240,181				15,998,034	25,329,738	173,025,563	507,954		123,251,864	10,635,308		216,671,381	106,059,049	1.3	77,294,068	187,481,750
2045	69,296,456	0.33	227,004	89,999,875	0.29	262,913	86.8	228,282				13,248,326	34,048,629	152,225,260	96,884		123,348,748	8,792,778		225,464,159	123,516,973	1.8	79,725,231	192,813,428
2046	66,422,986	0.36	236,597	89,999,875	0.31	277,831	86.7	240,886				100,610	23,691,786	128,634,084	14,288		123,363,036	12,229,443		237,693,603	112,501,931	1.7	67,396,955	178,924,917
2047	65,864,370	0.36	239,887	74,000,100	0.34	254,114	87.1	221,292					8,135,730	120,498,354			123,363,036	8,927,789		246,621,392	116,386,338	1.8	68,318,448	182,250,708
2048	39,904,568	0.42	169,020	74,202,840	0.31	228,998	86.2	197,461					34,298,272	86,200,082			123,363,036	3,757,261		250,378,653	135,029,889	3.4	65,366,810	174,934,456
2049	34,908,075	0.27	94,046	74,000,100	0.22	162,484	85.6	139,129	19,575		19,575	559,575	39,671,175	47,088,481			123,363,036	14,329,912		264,708,565	118,265,735	3.4	57,400,874	153,173,810
2050	40,556,599	0.28	114,718	74,000,100	0.23	173,244	86.7	150,194	10,125	19,575	10,125	380,363	33,814,413	13,654,430			123,363,036	14,489,437		279,198,002	105,443,395	2.6	54,845,505	145,999,994
2051	69,207,848	0.32	224,647	74,000,100	0.31	232,988	88.3	205,745		10,125			4,782,127	8,872,303			123,363,036	17,102,587		296,300,589	76,209,680	1.1	54,822,161	145,417,528
2052	82,287,223	0.35	288,469	74,202,840	0.36	270,574	88.8	240,165	4,438,367		4,438,367	3,646,015		12,518,319			123,363,036	12,565,800		308,866,389	36,814,691	0.4	45,111,696	119,101,913
2053	58,143,485	0.44	254,949	74,000,100	0.39	286,375	88.8	254,215		4,438,367	0	7,708,500	19,126,747	1,100,071			123,363,036	4,216,387		313,082,776	6,229,237	0.1	24,603,750	64,372,723
2054				74,000,100	0.19	143,023	69.1	98,898					1,100,071	0		72,900,029	50,463,007			313,082,776				
2055				50,463,007	0.19	97,685	68.9	67,333								50,463,007	0			313,082,776				
Undiluted	**3,182,549,611**	**0.38**	**12,000,903**	**3,182,549,611**	**0.38**	**12,000,903**	**88.4**	**10,612,345**	**49,944,509**	**49,944,509**		**223,998,689**	**223,998,689**		**123,363,036**	**123,363,036**		**313,082,776**	**0**		**3,283,435,217**	**1.03**	**2,516,800,941**	**6,465,984,829**
Diluted	**3,181,276,591**	**0.37**	**11,760,885**	**3,181,276,591**	**0.37**	**11,760,885**	**90.2**	**10,612,345**	**49,924,531**	**49,924,531**		**223,909,090**	**223,909,090**		**123,313,690**	**123,313,690**		**312,957,543**	**0**		**3,284,708,237**	**1.03**	**2,516,800,941**	**6,465,984,829**

1.11 Capital and operating cost estimates

Table 1-5 lists the development capital costs included in the Project cashflow model. Included in the total $6,425 million estimated capex is $913 million that was incurred prior to FQM's acquisition of the Project.

Table 1—5 Development capital cost provisions

Capital Item	Incurred Pre-Acquisition (US$M)	Incurred at 30/12/2014 (US$M)	Capex Spend 2015 to 2018 (US$M)	Total Capex Spend (US$M)	Expansion Capex Spend (US$M)
Mine, Port & Infrastructure	480.0	78.2	1,465.0	2,023.2	
Process Plant	62.0	175.3	1,061.6	1,298.9	500.0
Power Plant	209.0	115.6	339.5	664.1	415.0
Indirect Cost	162.0	912.6	1,364.2	2,438.8	
Contingency	-	-	-	-	
Total Project	**913.0**	**1,281.7**	**4,230.3**	**6,425.0**	**915.0**

A detailed derivation of operating costs was completed by MPSA, yielding summary figures as follows:

- overall average ore mining cost = $1.92/t
- overall average waste mining cost = $2.20/t
- mining equipment sustaining capital allowance = $0.26/t mined
- processing operating cost (inclusive of sustaining capital allowance) = $4.20/t
- fixed cost (equivalent G&A cost in variable terms, inclusive of sustaining capital allowance) = $1.25/t
- total processing cost = $5.17/t
- copper metal cost:
  = metal price x royalty + Cu metal cost
  = $3.00 x 5% + $0.277 = $0.43/lb Cu
- molybdenum metal cost:
  = metal price x royalty + Mo metal cost
  = $13.50 x 5% + $0.091 = $0.77/lb Mo
- gold metal cost:
  = 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)
- silver metal cost:
  = 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)

The above cost information was adopted for the optimisation and mine planning work in 2014. The estimates were reviewed in 2015 and updated as follows for sensitivity analyses and cashflow modelling:

- overall average ore mining cost = $1.88/t (now taking account of potential trolley-assist)
- overall average waste mining cost = $1.79/t (now taking account of potential trolley-assist)
- processing operating cost (inclusive of sustaining capital allowance) = $3.92/t

- mining equipment sustaining capital allowance = \$0.26/t mined
- fixed cost (equivalent G&A cost in variable terms, inclusive of sustaining capital allowance) = \$0.85/t
- total processing cost = \$4.77/t
- copper metal cost:
  = net metal price x royalty + Cu metal cost
  = (\$3.00 x 96.43% - \$0.323) x 5% + \$0.323 = \$0.45/lb Cu
- molybdenum metal cost:
  = net metal price x royalty + Mo metal cost
  = (\$13.50 x 86.2% - \$1.334) x 5% + \$1.334 = \$1.85/lb Mo
- gold metal cost:
  = net metal price x royalty + Au refining cost x Au payable
  = ((\$1,200 x 86%) – (\$5.50 x 86%)) x 4% + (\$5.50 x 86%) = \$45.82/oz Au
- The silver metal cost:
  = net metal price adjusted x royalty + Ag refining cost x Ag payable
  = ((\$16.00 x 80%) – (\$0.40 x 80%)) x 4% + (\$0.40 x 80%) = \$0.82/oz Ag

The mining costs listed above take account of ore haulage to designed in-pit crusher locations, and also trolley-assisted waste and ore haulage (ie, potential for future implementation).

1.12 Economic analysis

An economic analysis in the form of an undiscounted cashflow model to support the Mineral Reserve estimate is listed in Table 1-6. This model shows the indicative cashflow and does not replace a more comprehensive financial model that exists for the Project, from which an accurate NPV and IRR can be calculated.

The annual revenues are calculated from the same metal prices as used in the pit optimisation process (Item 15):

- Copper = US\$3.00/lb (US\$6,615/t)
- Molybdenum = US\$13.50/lb (US\$29,762/t)
- Gold = US\$1,200/oz
- Silver = US\$16.00/oz

The payable metal factors are:

- Copper = 96.43%
- Molybdenum = 86.20%
- Gold = 86.00%
- Silver = 80.00%

The Project is cashflow positive from 2018 and payback on the \$6,425M capital spend occurs in 2024.

All mining expenditure incurred prior to commercial production is included in the \$6,425M capital estimate.

Table 1—6 Mineral Reserve cashflow model summary

		TOTAL	<2016 <Year -1	2017 to 2026 Year 1 to 10	2027 to 2036 Year 11 to 20	2037 to 2046 Year 21 to 30	2047 to end Year 31 to end
MINING							
Total ore (inluding direct feed and s/pile reclaim)	Mt	**2,958.6**	5.7	821.6	938.0	814.6	378.6
Total waste (incl. MW not reclaimed)	Mt	**3,507.4**	55.0	839.5	931.2	1,075.0	606.7
Strip ratio		**1.2**	9.7	1.0	1.0	1.3	1.6
TOTAL FEED TO PLANT (before mining dil'n & recovery)							
TOTAL	t	**3,182.5**	0.0	752.6	893.7	893.4	642.9
Cu	%	**0.38**	0.00	0.43	0.42	0.35	0.29
Mo	ppm	**59.57**	0.00	72.24	73.65	54.99	31.54
Au	ppm	**0.07**	0.00	0.08	0.07	0.05	0.07
Ag	ppm	**1.35**	0.00	1.36	1.47	1.34	1.20
TOTAL FEED TO PLANT (after mining dil'n & recovery)							
TOTAL	t	**3,181.3**	0.0	752.3	893.3	893.1	642.6
Cu	%	**0.37**	0.00	0.42	0.41	0.34	0.28
Mo	ppm	**58.40**	0.00	70.82	72.21	53.91	30.92
Au	ppm	**0.07**	0.00	0.08	0.07	0.05	0.06
Ag	ppm	**1.33**	0.00	1.33	1.45	1.31	1.18
AVERAGE RECOVERIES							
Cu	%	**90.2**	0.0	91.6	91.3	89.6	86.9
Mo	%	**54.0**	0.0	55.0	54.6	53.4	51.0
Au	%	**55.4**	0.0	56.6	54.5	50.3	60.2
Ag	%	**45.2**	0.0	47.3	47.3	44.6	39.9
METAL RECOVERED							
Cu	kt	**10,612.3**	0.0	2,923.6	3,363.9	2,750.4	1,574.4
Mo	kt	**100.4**	0.0	29.3	35.2	25.7	10.1
Au	koz	**3,767.6**	0.0	1,123.9	1,091.5	746.0	806.3
Ag	koz	**61,333.4**	0.0	15,208.4	19,633.4	16,788.8	9,702.8
METAL PAYABLE							
Cu	kt	**10,233.3**	0.0	2,819.2	3,243.8	2,652.1	1,518.2
Mo	kt	**86.5**	0.0	25.3	30.4	22.2	8.7
Au	koz	**3,240.2**	0.0	966.5	938.7	641.6	693.4
Ag	koz	**49,066.7**	0.0	12,166.7	15,706.7	13,431.1	7,762.2
GROSS REVENUE							
Cu	$M	**67,681.8**	0.0	18,645.8	21,453.9	17,540.9	10,041.2
Mo	$M	**2,575.4**	0.0	751.8	903.6	659.9	260.0
Au	$M	**3,888.2**	0.0	1,159.8	1,126.4	769.9	832.1
Ag	$M	**785.1**	0.0	194.7	251.3	214.9	124.2
subtotal	**$M**	**74,930.5**	**0.0**	**20,752.2**	**23,735.2**	**19,185.6**	**11,257.5**
CAPITAL COSTS							
Development capex	$M	**6,425.0**	4,995.9	1,429.1	0.0	0.0	0.0
Expansion capex	$M	**915.0**	0.0	915.0	0.0	0.0	0.0
Sustaining capex	$M	**0.0**	0.0	0.0	0.0	0.0	0.0
Closure and reclamation	$M	**78.6**	0.0	0.0	0.0	0.0	78.6
subtotal	**$M**	**7,418.6**	**4,995.9**	**2,344.1**	**0.0**	**0.0**	**78.6**
OPERATING COSTS							
Mining waste	$M	**6,183.2**	0.0	1,503.5	1,667.8	1,925.3	1,086.6
Mining ore	$M	**5,554.3**	0.0	1,545.5	1,764.4	1,532.3	712.1
Mining sustaining	$M	**1,665.4**	0.0	431.9	486.0	491.3	256.2
Processing (incl sustaining)	$M	**12,471.1**	0.0	2,949.2	3,501.8	3,500.9	2,519.1
G&A (incl sustaining)	$M	**2,709.2**	0.0	640.7	760.7	760.5	547.3
Stockpile reclaim	$M	**516.5**	0.0	30.2	14.6	122.3	349.4
subtotal	**$M**	**29,099.7**	**0.0**	**7,101.0**	**8,195.4**	**8,332.7**	**5,470.6**
METAL COSTS (INCLUDING ROYALTIES)							
Cu	$M	**10,568.9**	0.0	2,911.7	3,350.2	2,739.1	1,568.0
Mo	$M	**409.2**	0.0	119.5	143.6	104.9	41.3
Au	$M	**172.6**	0.0	51.5	50.0	34.2	36.9
Ag	$M	**50.2**	0.0	12.5	16.1	13.8	7.9
subtotal	**$M**	**11,201.1**	**0.0**	**3,095.1**	**3,559.8**	**2,891.9**	**1,654.2**
CASHFLOW	**$M**	**27,211.1**	**-4,995.9**	**8,212.0**	**11,980.0**	**7,961.0**	**4,054.1**

1.13 Conclusions and recommendations

In respect of the Mineral Resource estimate, and 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. There are a number of uncertainties with the geological and structural model forming the basis of the Mineral Resource estimate, however, 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 mining plan and production sequence, and one which has taken account of phased mining and processing capacity, “smoothed” equipment usage profiles, longer term in-pit crusher relocations, optimised waste haulage profiles and a practical stockpile building and reclaim 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.

Uncertainty in operating costs, 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.

Mine geotechnical risks are considered to be manageable, largely through the adoption of phased pit designs and sequencing, and the ability to map and analyse available exposures before committing to final design slopes. The risk of surface water inflows to the pits has been mitigated by the design of surface diversions, whilst the risk of groundwater and rainfall inflows has been mitigated by the design of staged sump pumps extending up the overall pit slopes.

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 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 an updated commentary on the revised project 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 30th June, 2015.

2.3 Qualified Persons and authors

Optiro was commissioned to prepare Mineral Resource estimates for the Colina, Valle Grande, Balboa and Medio deposits at the Cobre Panamá Project. The Botija, Botija Abajo and Brazo deposit estimates were completed by FQM geologists in January 2014. Ian Glacken of Optiro has authored those items of this Technical Report relating to geology and Mineral Resource estimation.

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.

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.

The following table identifies which items of the Technical Report have been the responsibility of each QP.

Name	Position	NI 43-101 Responsibility
David Gray *BSc (Geology), MAusIMM, PrSciNat(SACNASP)*	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 and Director, Optiro Pty Ltd	Contributing Author Items 3 – 12, 14

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).

2.5 Site visits

The following independent Qualified Persons (QPs) have visited the site:

- David Gray visited the Project in June 2013 and 2014. Mr Gray inspected drill core and drilling sites, reviewed geological, data collection and sample preparation procedures, and carried out independent data verification.
- Michael Lawlor visited the Project in June 2014. Mr Lawlor inspected drill core and visited all accessible areas of the development site.
- Robert Stone visited the Project in April 2013, October 2014 and May 2015. Mr Stone visited all areas of the project including the plant, port and TMF locations, and whilst on site reviewed environmental control infrastructure.

ITEM 3 RELIANCE ON OTHER EXPERTS

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

In relation to Item 20 (Environmental Studies, Permitting, Social and Community Impact), and whilst taking QP responsibility for authoring the information reported therein, Michael Lawlor has submitted the information to and has relied on a technical review provided by Alberto Casas, Environmental Director, MPSA.

ITEM 4 PROPERTY LOCATION, DESCRIPTION AND TENURE

4.1 Project ownership

The Cobre Panamá Project is owned by Minera Panamá S.A. (MPSA) of which 80% is owned by FQM (Akubra) Inc., a subsidiary of First Quantum Minerals Ltd, having taken acquisition of Inmet Mining Corporation in April 2013. The remaining 20% share of MPSA is held by Korean Panamá Mining Company (KPMC), a consortium of Korea Resources Corporation and LS-Nikko Copper Inc.

MPSA (previously Minera Petaquilla S.A.) was granted the project concession rights to the Cobre Panamá (previously Petaquilla) Project under Panamánian Ley Petaquilla, or Law No. 9, in January 1997. This project-specific law gives MPSA rights over the gold, copper and other mineral deposits for the purposes of exploring, extracting, processing, transporting and marketing of all base or precious minerals located in the concession area.

4.2 Project location

The Cobre Panamá Project is located in the Colón Province of north central Panamá, approximately 120 km west of Panamá City (Figure 4-1). Colón is in the north central part 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.

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'.

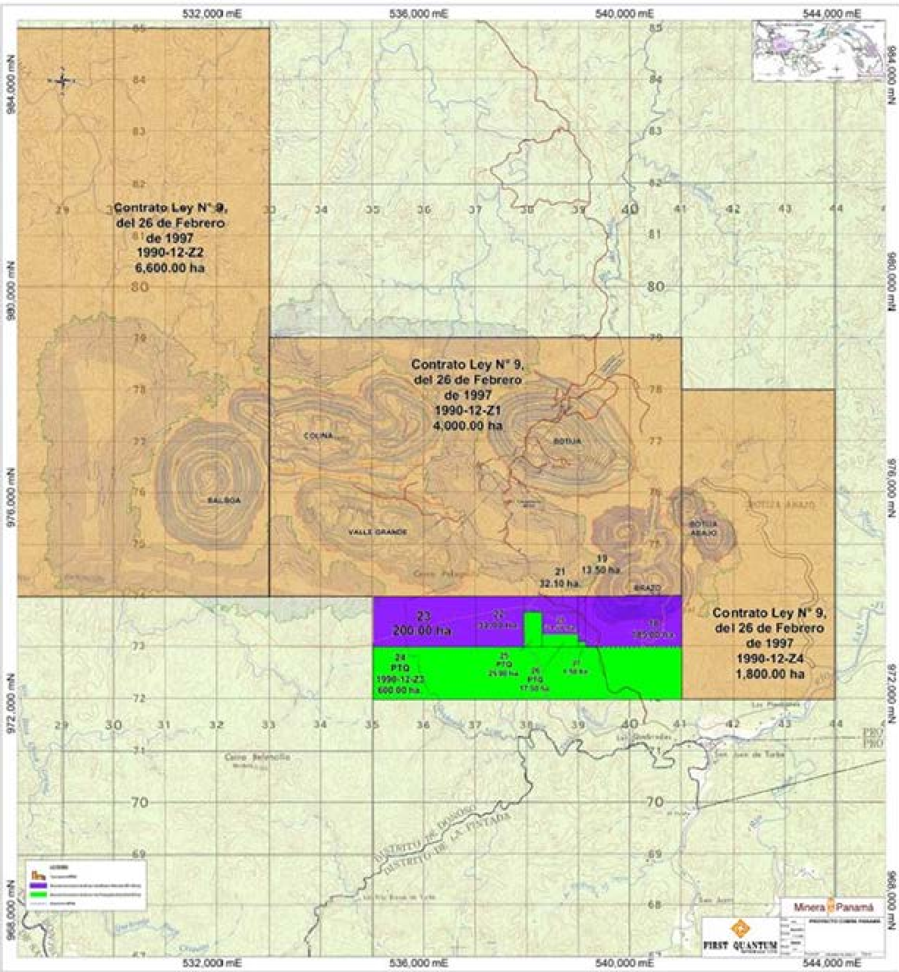
Figure 4-1 Cobre Project location map (source: Rose *et al*, 2012)

4.3 The Project tenement area

The property consists of four concessions totalling 13,600 hectares (Figure 4-2). The geographic co-ordinates for each zone are provided in Table 4—1.

- 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, 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. Although not shown on Figure 4-2, the Balboa deposit is located within Zona No. 2.
- Zona No. 3: Has a total surface area of 1,200 hectares and lies within the Jurisdiction of San José del General, Donoso District, 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 Jurisdiction of San José del General, Donoso District, 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 forms part of the Zona No. 3 concession and relates 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 now covers 644.9 hectares of the total 13,600 hectare concession area (it was previously 600 hectares)[2].

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



[2] 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 Panama Project, and as a result the Zona 3 sub-concession boundaries, have changed to that as shown in Figure 4-2.

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	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	

4.4 Obligations of the concession

The concession was awarded in February 1997 for a twenty year term, with two renewal periods of the same duration. Under the terms of Law No. 9, the annual fee is $3.00/ha, and the last payment was made in April 2015 in respect of the modified area shown in Figure 4-2.

4.5 Legal status

The concession rights to the Cobre Panamá Project are outlined by Law No. 9, from January 1997. Under this law, MPSA has rights to acquire or lease state lands located within the concession area.

The Molejón Gold Project Agreement, formed in June 2005, gave PML the property and mineral rights over a portion of the concession area, known as the Molejón sub-concession, to permit it to develop the Molejón gold deposit. PML was also given the right to explore and mine gold deposits (where greater than 50 percent of the present value being derived was from the gold or other precious metals content) in the larger concession area provided that it did not interfere with MPSA's ability or interest to exploit the total concession area. The copper rights to the sub-concession are still retained by MPSA. 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, and as a result the Zona 3 sub-concession boundaries have changed to that as shown in Figure 4-2.

Under Law No. 9, 1997, MPSA is exercising its rights to acquire or lease state lands located within the proposed tailings basin area (Figure 4-3). Negotiation by MPSA with existing private holders with surface rights has been completed and MPSA now has the land rights on the entire land polygon of 7,477.5 ha (shown as the orange line in Figure 4-3).

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 Reserve Area, MPSA will conform to the requirements of IFC PS 6.

Most of the land required for the proposed port facilities at Punta Rincón, including land required for construction and permanent use of the site, has been acquired by MPSA during 1998 and 2000. Outstanding parcels of the additional land required (6,800 ha) for the mine expansion not already acquired will be done so in accordance with the standards mentioned above.

4.6 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) and was approved on December 28th, 2011. This ESIA covers 7,586 hectares of the total concession, plus the port site.

MPSA intends to expand this to cover all the major deposits by obtaining permits before the end of 2016.

MPSA has identified an additional 150 permits required for development of the Project, which include permits for vegetation clearing, water usage and construction activities.

Permitting for the powerline from the port to the mine site has been approved. The power line easement is split into three segments, i.e. Tramo I, Tramo II and Tramo III. All of the Tramos are included in the approved ESIA for the whole mine Project. In addition to the approval given by ANAM, there are easement permits to be applied for, such as for vegetation clearing and construction permits. Tramo I has both permits already. Tramo I and Tramo II still require tree clearing permits and municipal permits in certain areas of the line within these tramos, which are pending.

Establishment of the port facilities will also engage the Panamánian maritime authority and general construction will engage with the municipality of Donoso.

4.7 Royalties and taxation

As governed by Law No. 9, 1997, MPSA is required 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". The royalty as set out in the Mining Code was increased to 5% for base metals and 4% for precious metal concentrates on October 1st, 2011.

Changes to Law 9 will be made to align with the new Code upon renewal of MPSA's mining concession in 2017.

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

Under Law No. 9, development of the project is subject to certain fiscal incentives. These include:

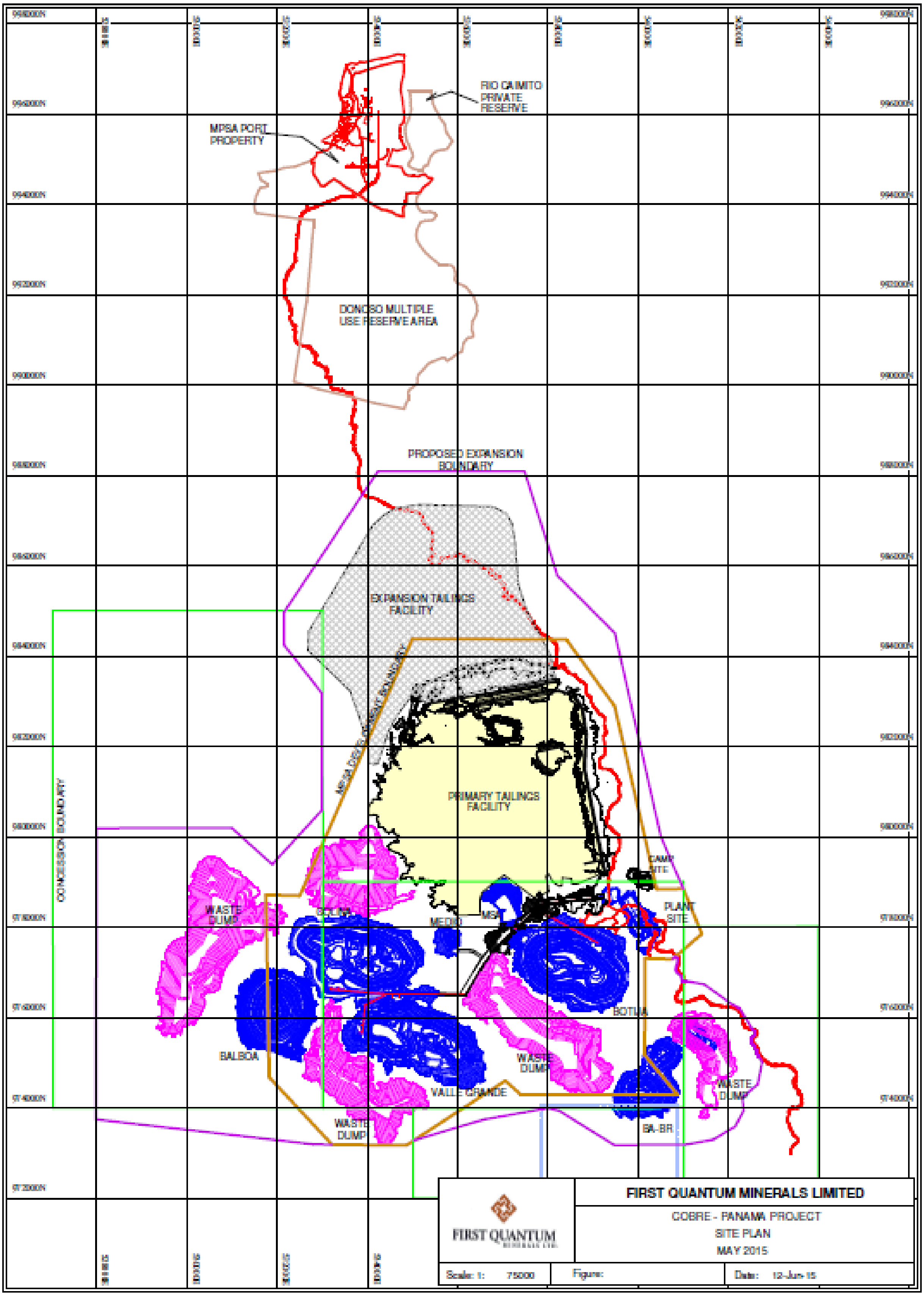
1. "Exoneration for the Company, its Affiliates, contractors and subcontractors of any import tax or duty, contribution, charge, consular fee, lien, duty or another tax or contribution, or of any name or class that fall [are levied] on the introduction and import of equipment, machinery, materials, parts, diesel and Bunker C and other petroleum derivatives"

2. "Income tax exoneration applicable to remittances or transfers abroad, made to pay commissions, loans, royalties, returns, charges for professional advice or administration incurred outside the national territory"

3. "Excepting only the respective mining royalties and royalties, as long as the Company has not finished repaying the debt which the Company or its Affiliates acquire for construction and development of [the Project], the Company and its Affiliates shall be totally exempt from payment of any type of tax, fee, duty, charge, lien, contribution or tribute that may be levied due to any reason in relation to the development of THE PROJECT, except municipal taxes."

Quotations above are from an English translation of Law 9.

Figure 4-3 Location of the planned facilities for the Cobre Panamá Project



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 but due to the dense vegetation and topography, movement of drilling equipment and staff is still frequently completed by helicopter and, as such, a helicopter pad has been constructed at the Colina camp. There is an existing airplane runway at Coclecito; however frequent thick cloud cover in the region 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 area is characterised by rugged topography with dense rainforest cover. The topography is low elevation (less than 300 metres) with elevations ranging from 70 masl to 300 masl. Relatively 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 large rainfall.

5.3 Local resources

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 will be made to hire a workforce from the local communities; however, as this region of Panamá is relatively sparsely populated, additional personnel will be required from other areas of the country. Skilled expatriate personnel will be 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 the national workforce.

5.4 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 main camps, including the Colina, Teton

Kiwara, TMF and Cusa (Dorado) camps, as well as several smaller camps in more remote areas of the concession. The main site camps are all accessible by road and can collectively accommodate over 2,600 people. Additional facilities at the Colina and Teton camps include drill core storage sheds and sample preparation areas. A 1,800 man camp was completed at the Port site in 2013. Figure 5-1 depicts the current project infrastructure.

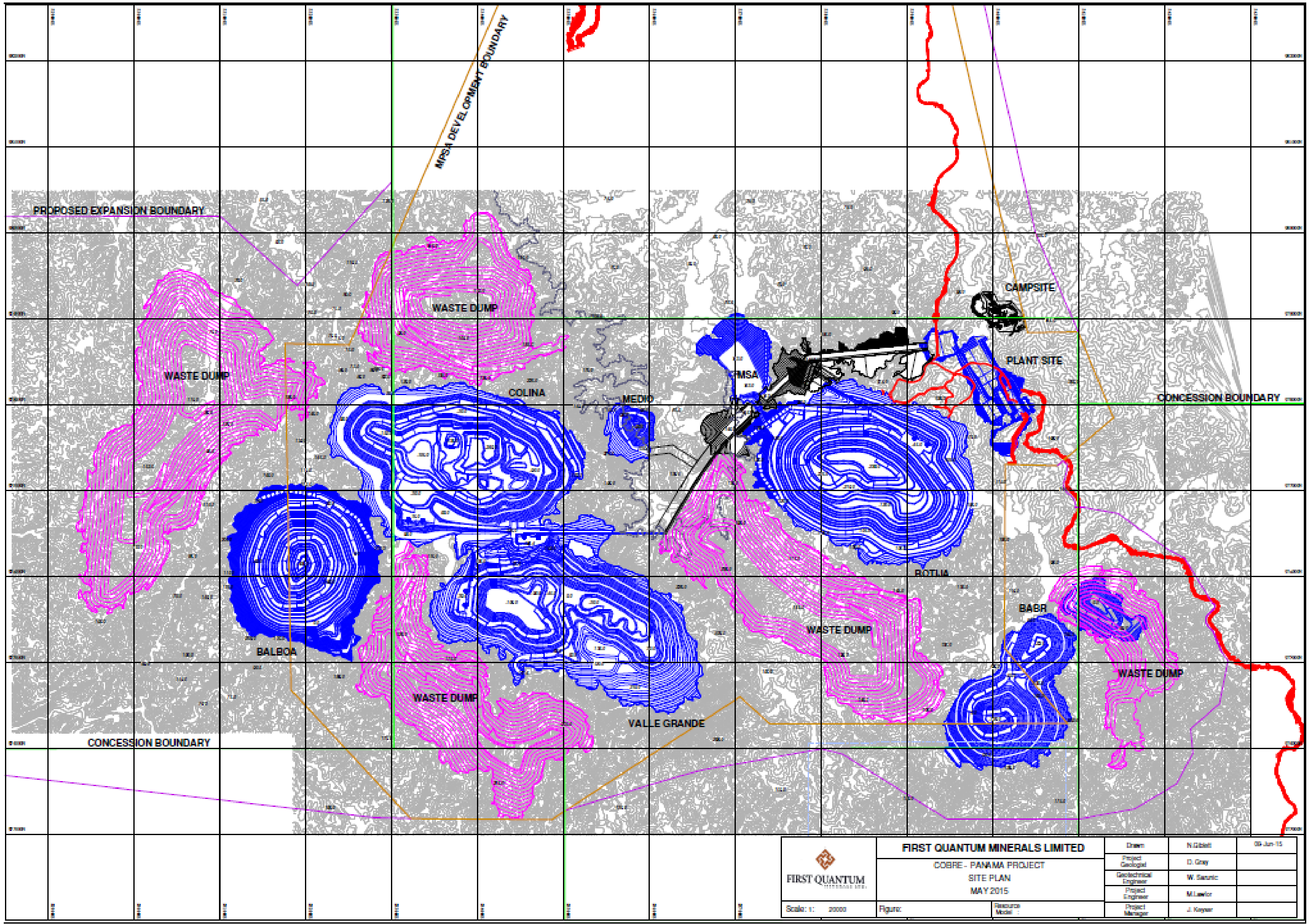
MPSA is in the process of building an extensive road network to improve access across the property and reduce the reliance on helicopters as the main method of transport. This includes upgrades to existing roads, including the main access road from Llano Grande (now completed), as well as building of the Eastern Access road. The upgrade of the Moléjon and La Pintada by-pass roads was recently completed whilst sealing of the main access road has also been completed.

Further development of the Project requires the construction of a port facility and an adjacent 300 MW power plant at Punta Rincón, approximately 25 km north of the mine and proposed plant area, which is in progress. Other required development includes power transmission lines, pipeline infrastructure for both water and metallurgical concentrates, worker accommodation and other support facilities (maintenance and storage areas, etc.). A road linking the port facility and the mine/plant site has been constructed and is currently being upgraded.

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.

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.

Figure 5-1 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 1,805 diamond drillholes (346,294 m) have been completed (up to October 2013).

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

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 September 2008 Inmet acquired Petaquilla Copper Ltd, and in November 2008 Inmet acquired Teck Cominco's remaining share in MPSA, taking Inmet to a 100 % interest in MPSA. In October 2009, Inmet announced an option agreement with the Korea Panamá Mining Corporation (KPMC; a consortium of LS-Nikko Copper Inc. and Korea Resources Corporation) under which KPMC could acquire a 20% interest in the Cobre Panamá copper project. On January 10th 2012 KPMC exercised its option, acquiring the full 20% interest.

In April 2013, FQM successfully completed the acquisition of Inmet, retaining the 20% interest held by KPMC.

6.2 Previous Mineral Resource estimates

Mineral Resource estimates have previously been generated for a number of porphyry copper-type deposits on the Cobre Panamá property including the Botija, Colina, Valle Grande, Balboa, Brazo and Botija-Abajo mineralised zones. Prior to the update described in this Technical Report, the most recent 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.

6.3 Previous Mineral Reserve estimates

A Mineral Reserve estimate for the Project was last estimated in March 2013, at an effective date of December 2013. The estimate was based on metal prices of \$2.25/lb Cu, \$13.50/lb Mo, \$1000/oz Au and \$16.00/oz Ag. The design and production schedule upon which the Mineral Reserves was determined, accounted for the Botija, Colina, Valle Grande, Balboa, Brazo and Botija-Abajo mineralised zones.

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.

6.4 Production from the property

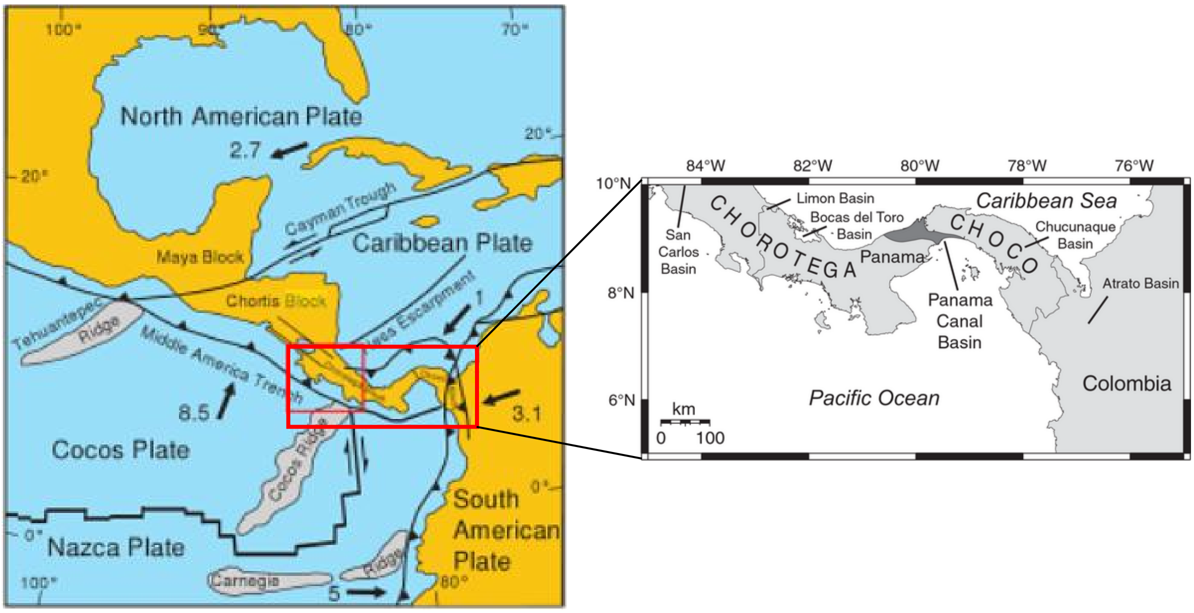
To date there has been no production from the property.

ITEM 7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional geological setting

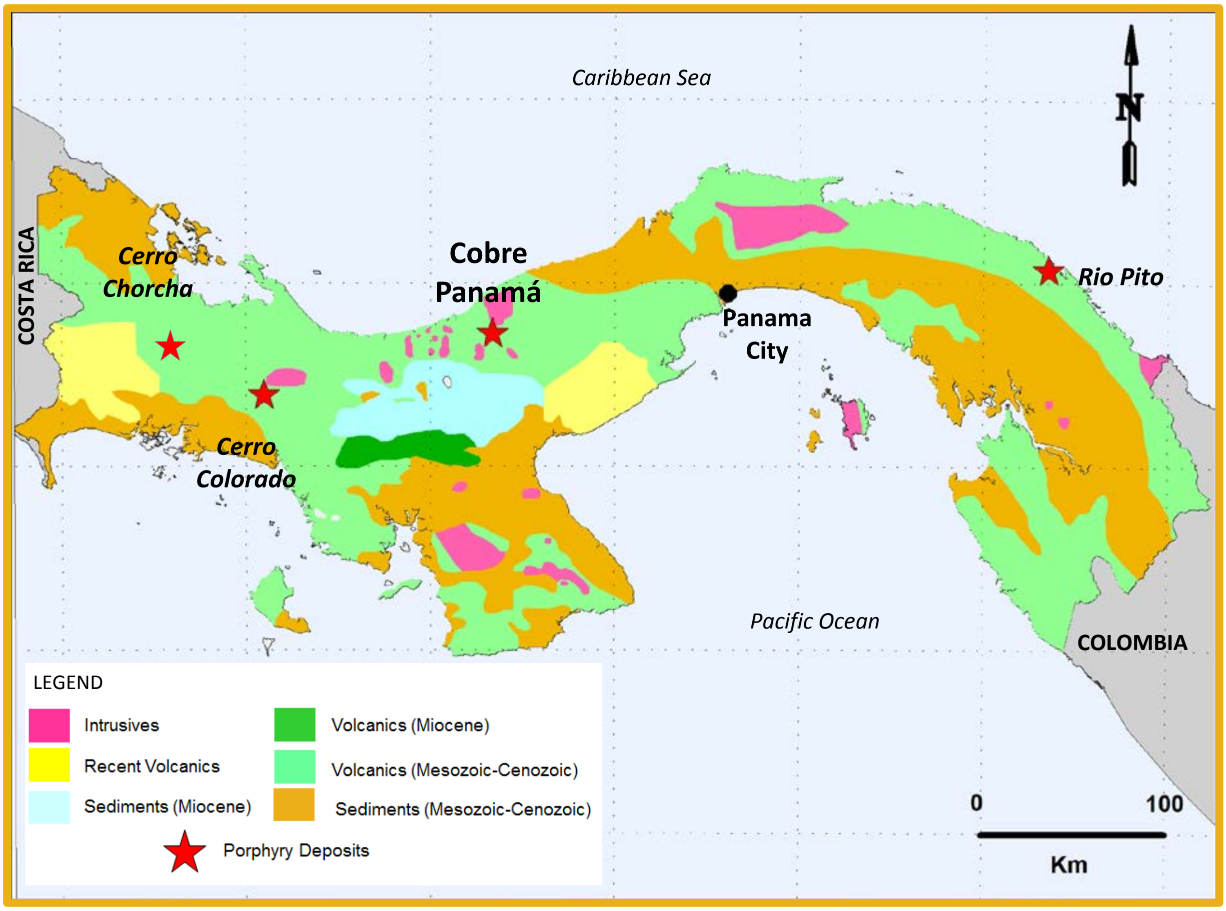
The Panamá Isthmus is a narrow strip of land that connects the North and South Americas at the junction of several tectonic plates, including the Caribbean, Nazca, Cocos and continental North and South American Plates. South vergent subduction and related arc volcanism, high-angle strike-slip and block 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.

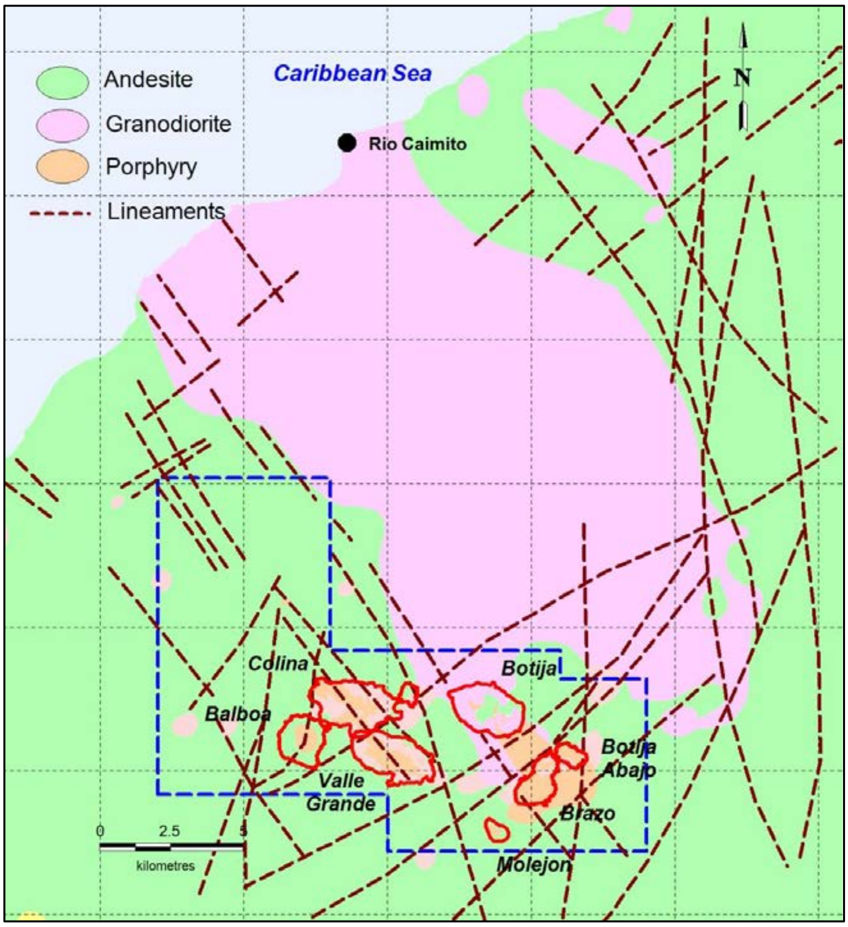
In the area of the Cobre Panamá deposits, the oldest rocks are submarine andesites, basalt flows and tuffs, intercalated with clastic sedimentary rocks and reef limestones, of probably Eocene to early Oligocene age. This suggests that the volcanic arc became emergent during the mid-Oligocene period, 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 7-4.

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

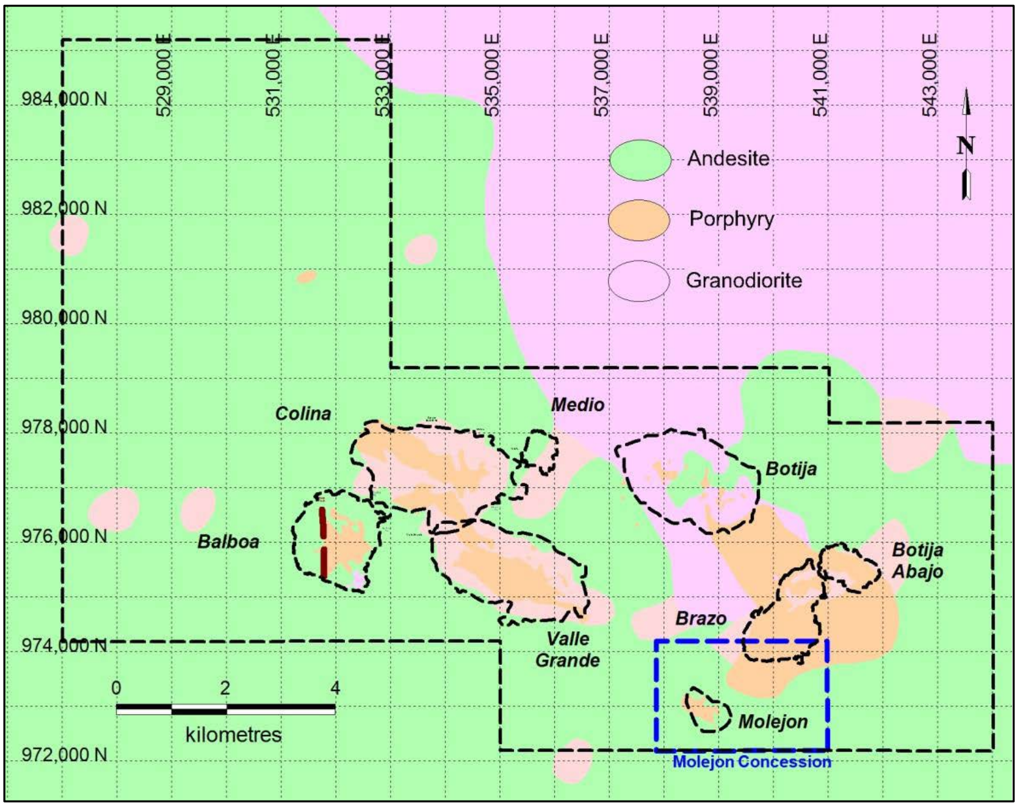


Seven main lithological units have been recognised at the Cobre Panamá Project. From oldest to youngest these units are:

- a sequence of undifferentiated andesite volcanics and volcaniclastics
- granodiorite
- a suite of a feldspar-quartz-hornblende porphyries that are likely to be 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)[3]



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

[3] The Molejon Concession boundary has recently changed – refer to Item 4.3.

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

There have been three generations of structures identified in mapping at the Cobre Panamá Project (Model Earth, 2013). The earliest features are roughly east-west to northeast-southwest trending, 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 and often bound 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 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

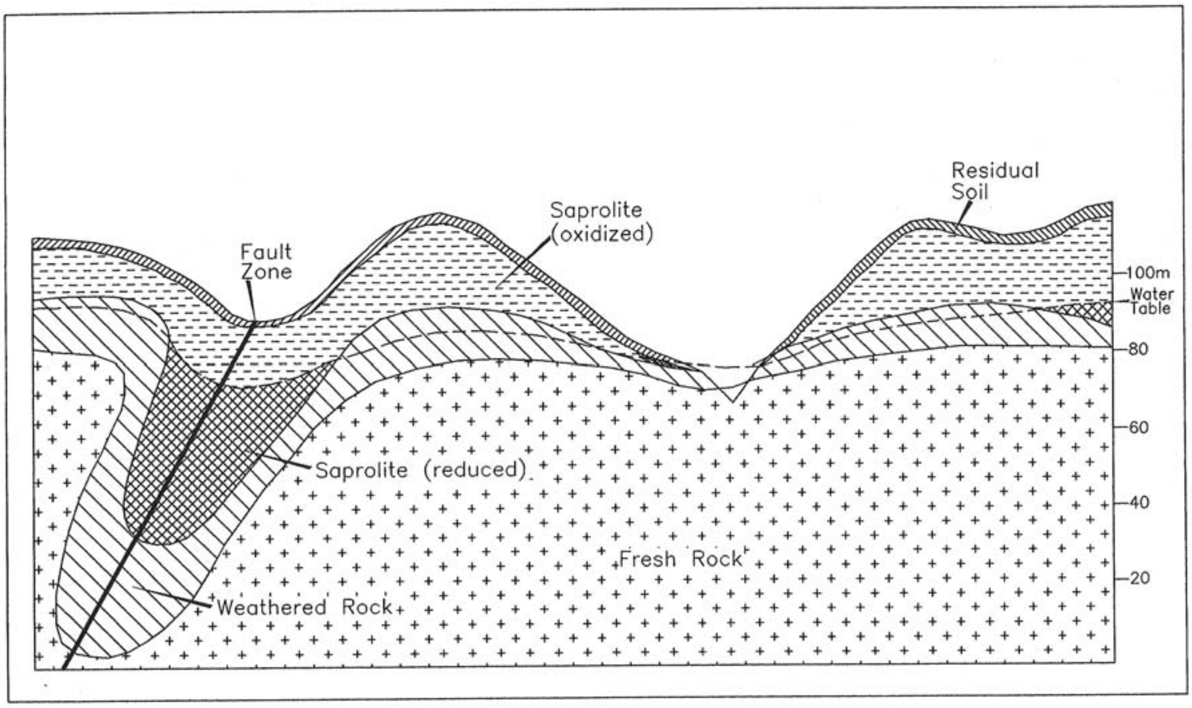
Most of the property area is covered by a thin layer (1cm to 10 cm) of black organic material which overlies a thin layer (<1 m) of residual soil. This in turn overlies a 1 m to 20 m thick layer of orange to white (oxidised) or green (reduced) saprolite (Figure 7-5). The depth of the saprolite layer 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

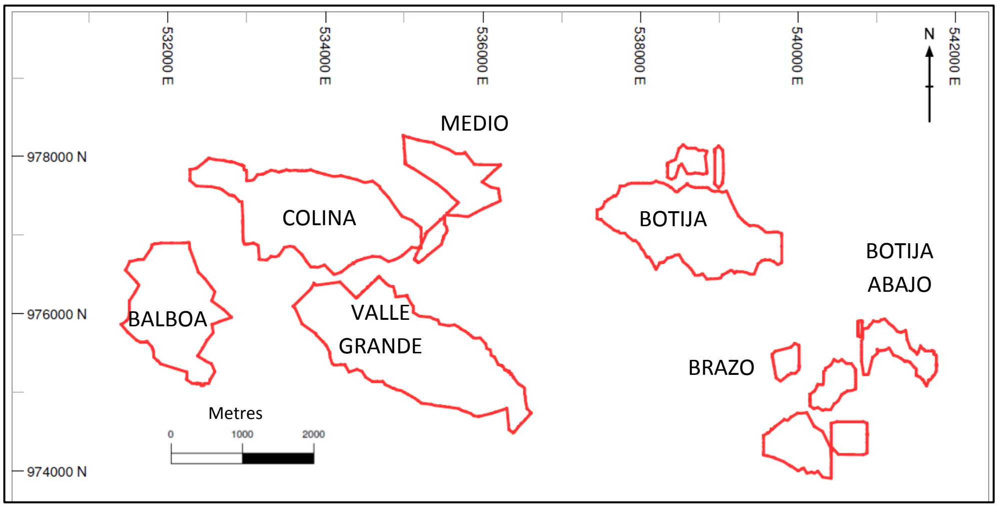
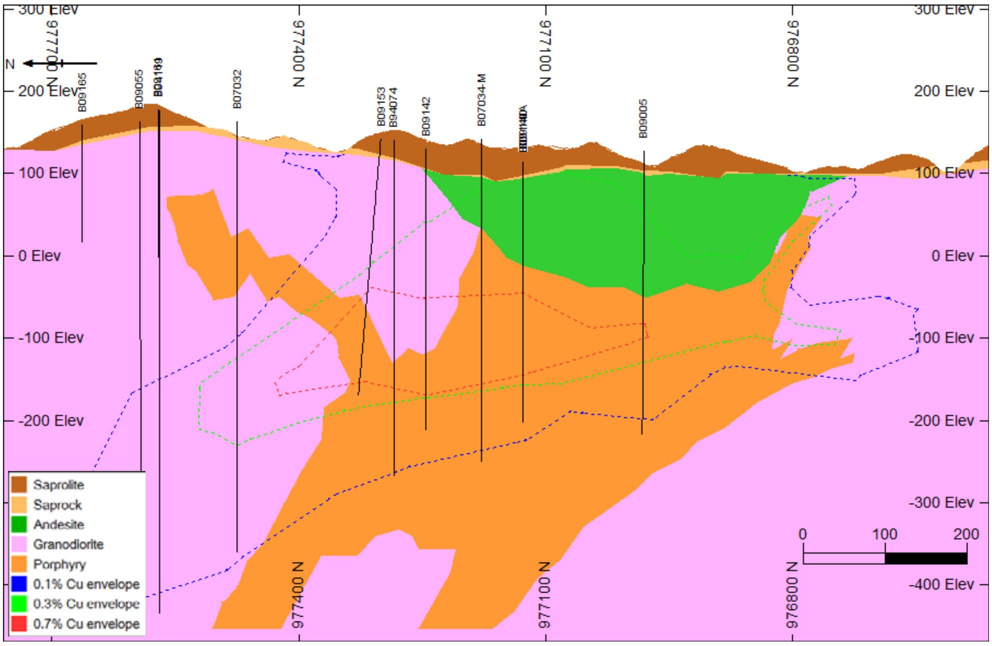


Figure 7-7 Botija south-north geology cross-section at 538140 mN



Extensive work has recently been completed at Botija to better understand the geochemistry, mineralogy and possible structural controls on the mineralisation. This included studies by Halley (2013) and Model Earth (2013).

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.

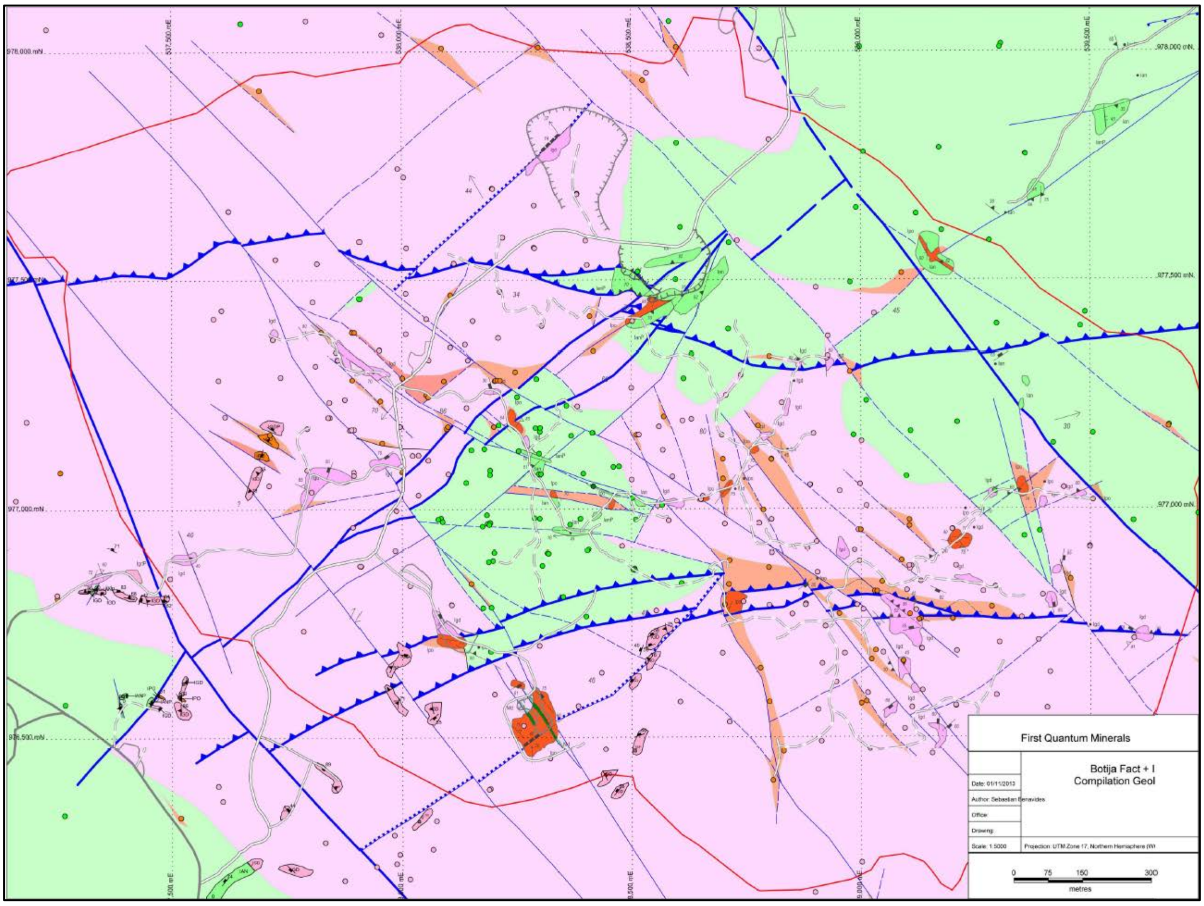
Detailed field mapping and interpretation of structures at Cobre Panamá was completed by Model Earth Pty Ltd in September 2013. The main focus at Botija was to complete detailed mapping of the deposit and its surroundings to better understand the geology, assist in focussing any exploration and improve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



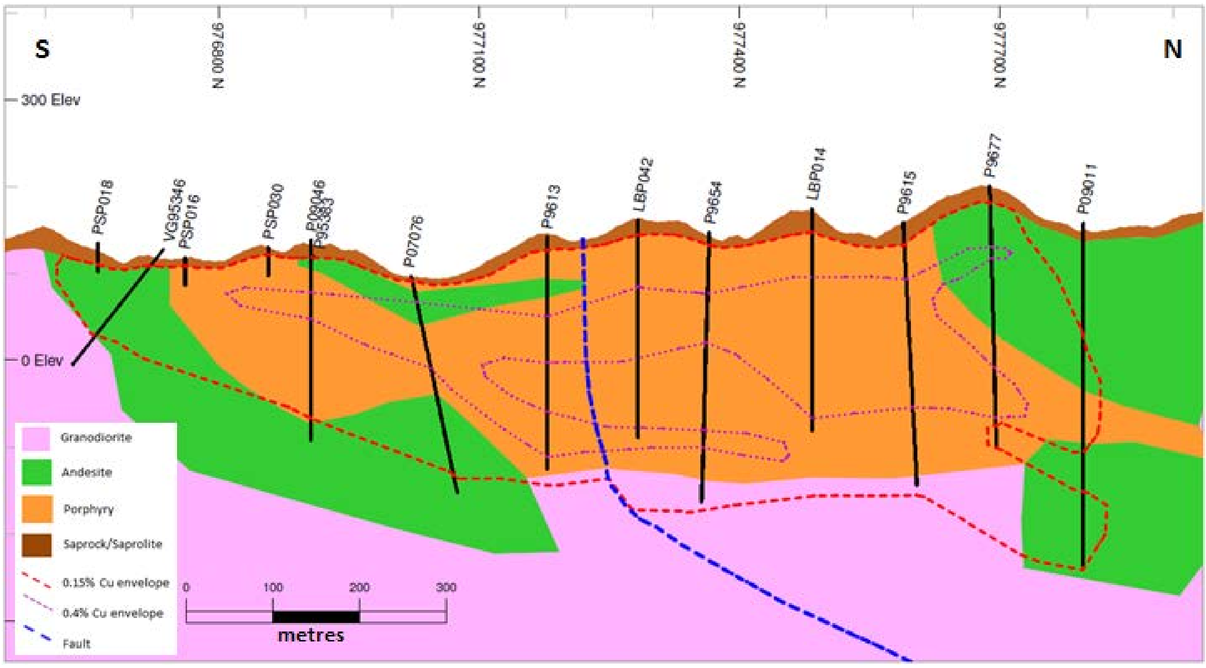
Copper mineralisation at Botija is characterised by a quartz stockwork of “A” and “B” veinlets within the central porphyry dyke complex and throughout the contact zones. This stockwork forms a locus for high grade copper and molybdenum mineralisation (Gustafson and Hunt, 1975) and hosts most of the chalcopyrite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

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).

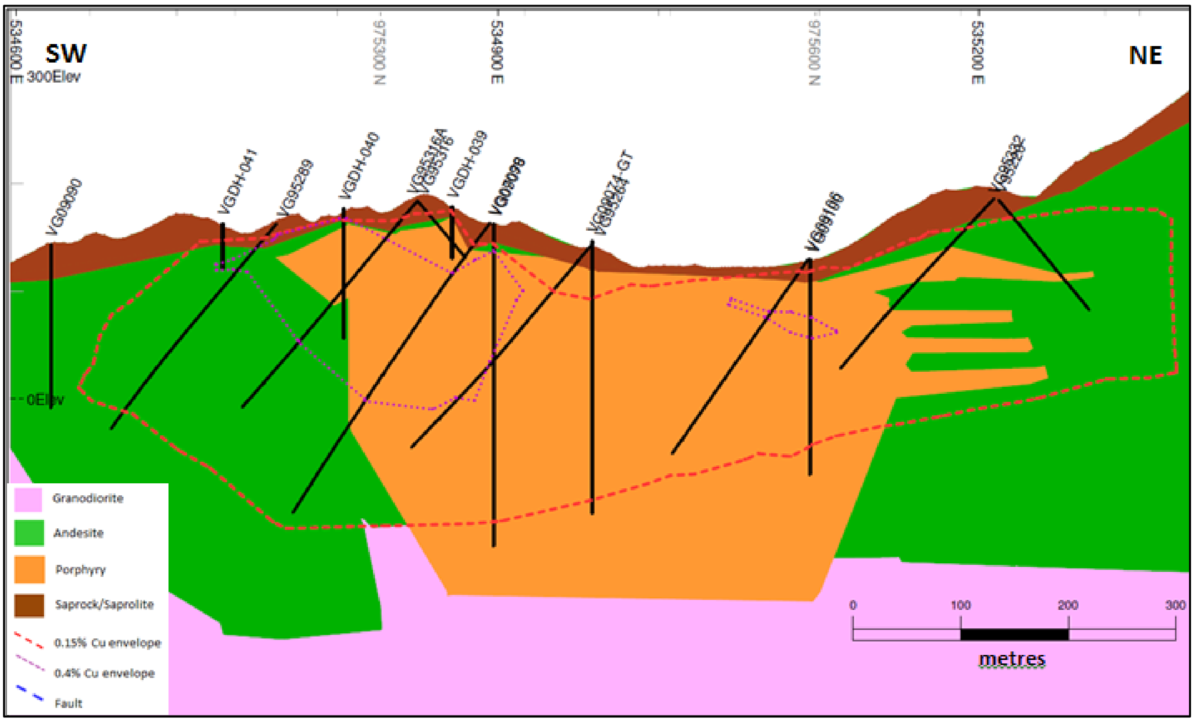
Secondary copper minerals, including chalcocite and covellite, have been observed at Colina, occurring mainly as sooty coatings on chalcopyrite at the base of the saprolite or adjacent to structures penetrating the underlying sulphide domains from surface. Locally, malachite has been observed within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



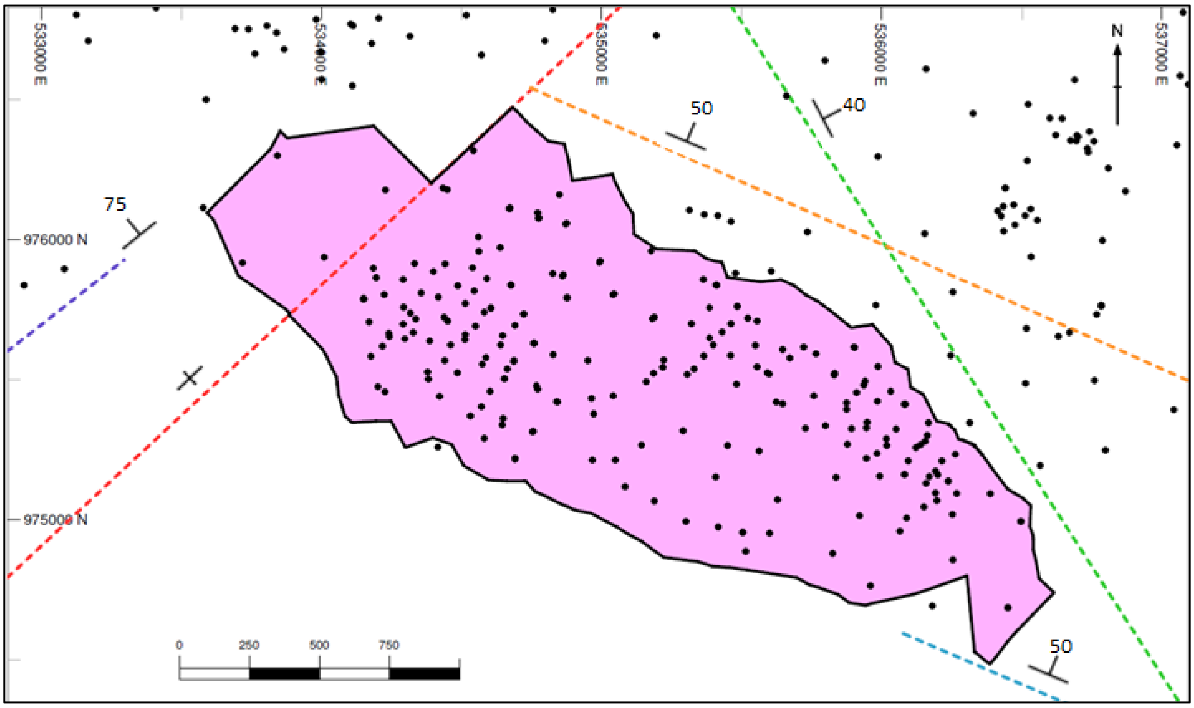
The most commonly logged alteration type logged at Valle Grande is propylitic. This is likely due to the dominance of andesite as the host rock. Silica-chlorite alteration is irregular, often occurring deep within the feldspar-quartz-hornblende porphyry dyke. Potassic alteration distribution is erratic, 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)

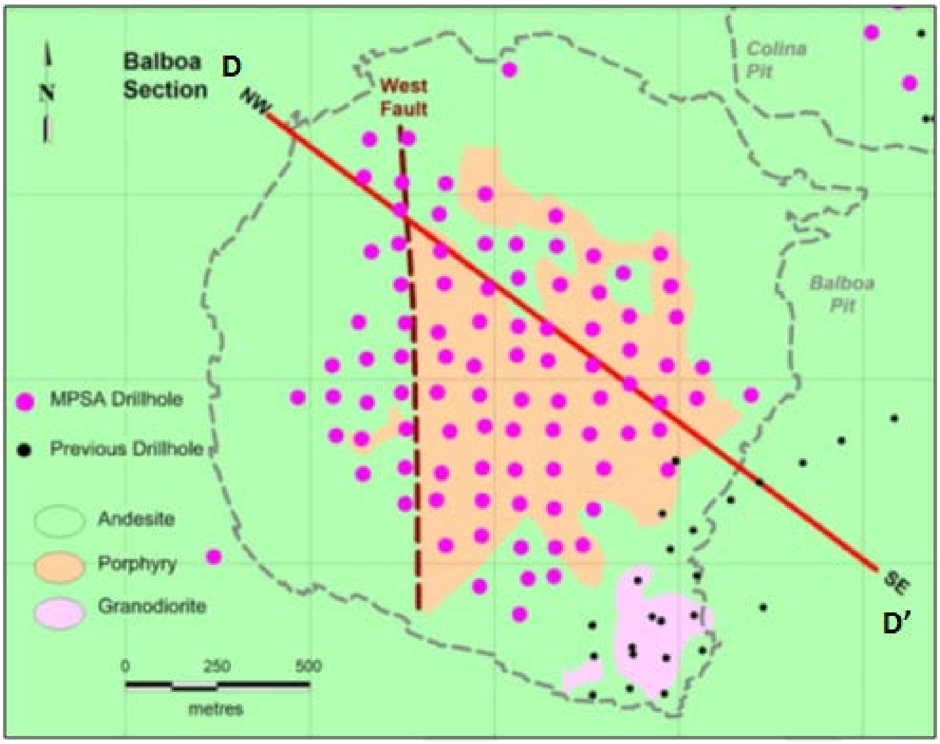
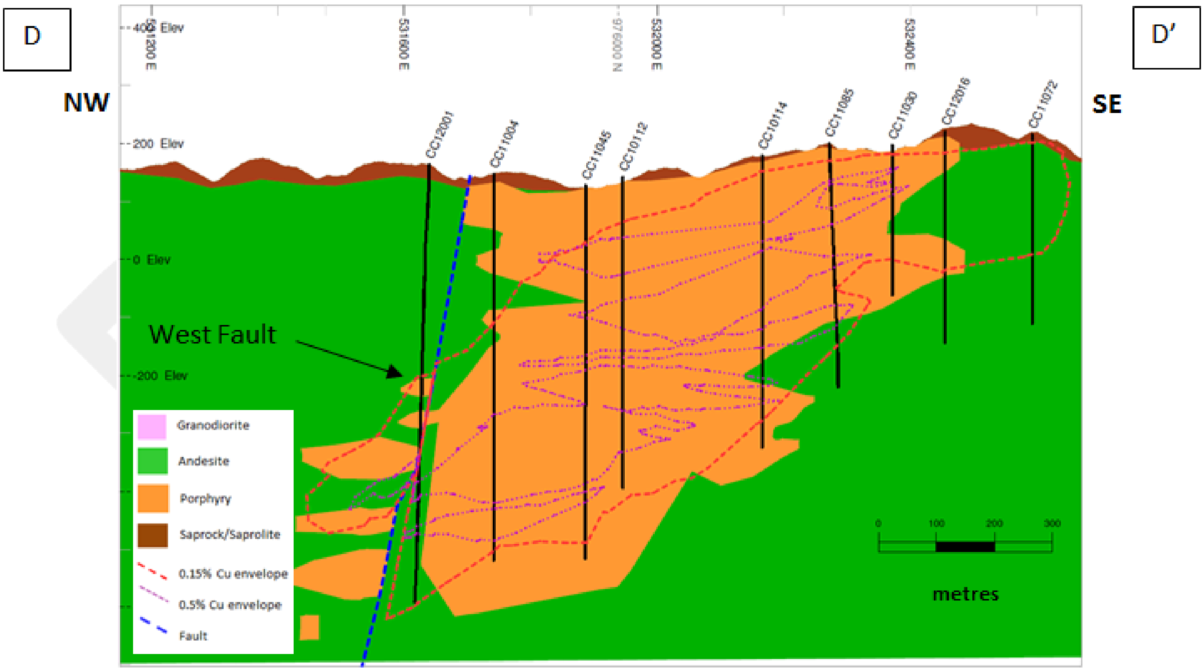


Figure 7-14 Oblique section looking northeast through the Balboa deposit



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.

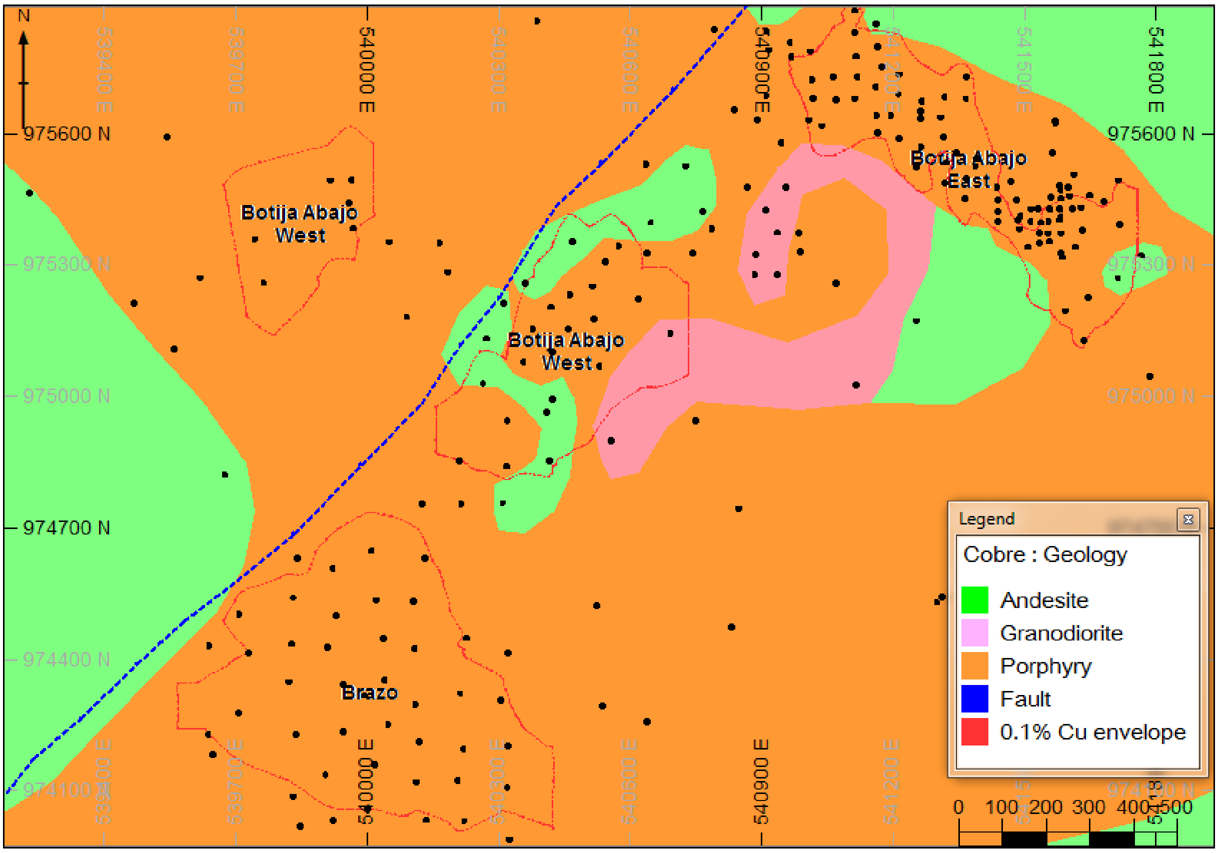
7.8.2 Botija Abajo

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 strucures.
- 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)

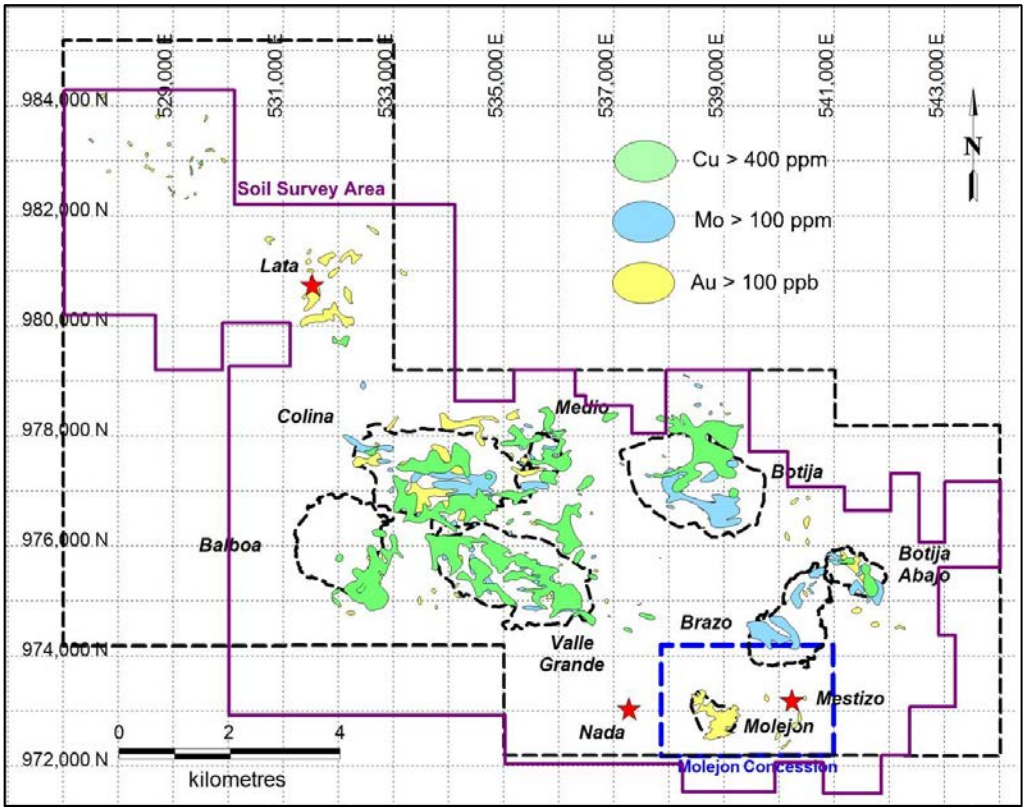
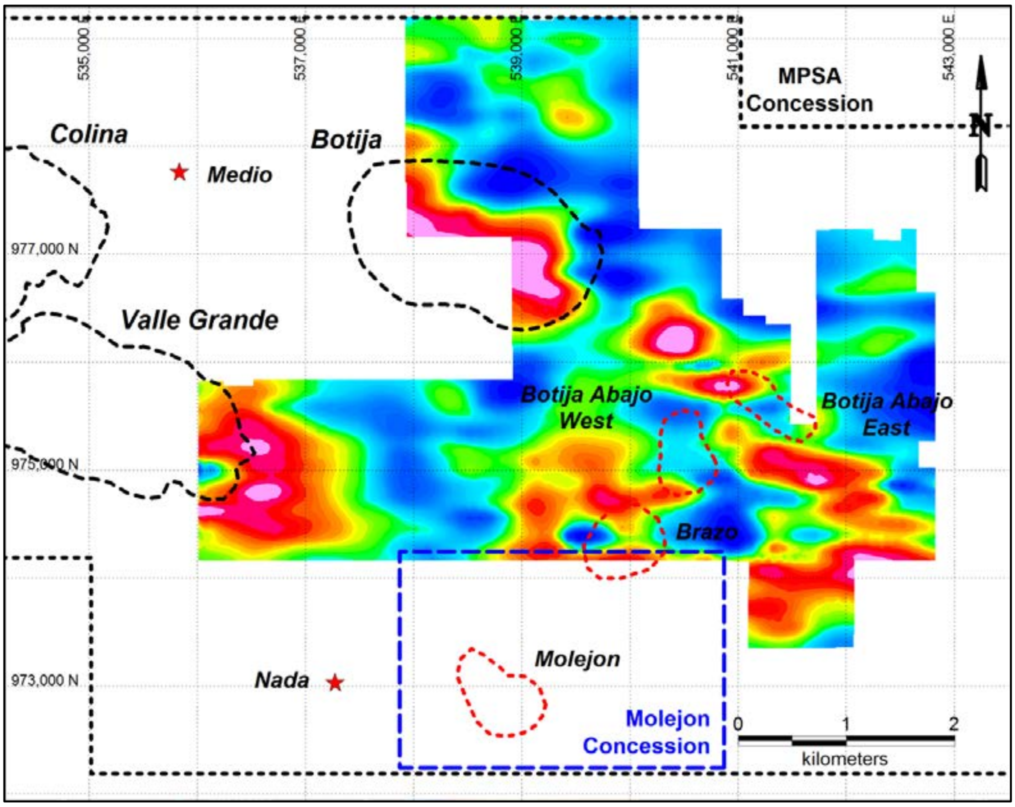


Figure 9-2 Plan of IP Chargeability at -150 m level (source: Rose *et al*, 2012)



ITEM 10 DRILLING

The following information describes the drilling activities carried out prior to Project acquisition. Whilst no further drilling has been carried out 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	2007-2013	251	55,038.7	95	27,630.6	79	15,854.7
Total		**446**	**93,000**	**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	825	188,996
Total	**121**	**53,487**	**295**	**50,015**	**401**	**39,368**	**1,805**	**346,294**

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

As at June 2014, 1877 holes had been drilled for 359,494 m. September 2013 was the cut-off date for drilling data used for the Mineral Resource estimate described in Item 14.

10.2 Historical drilling

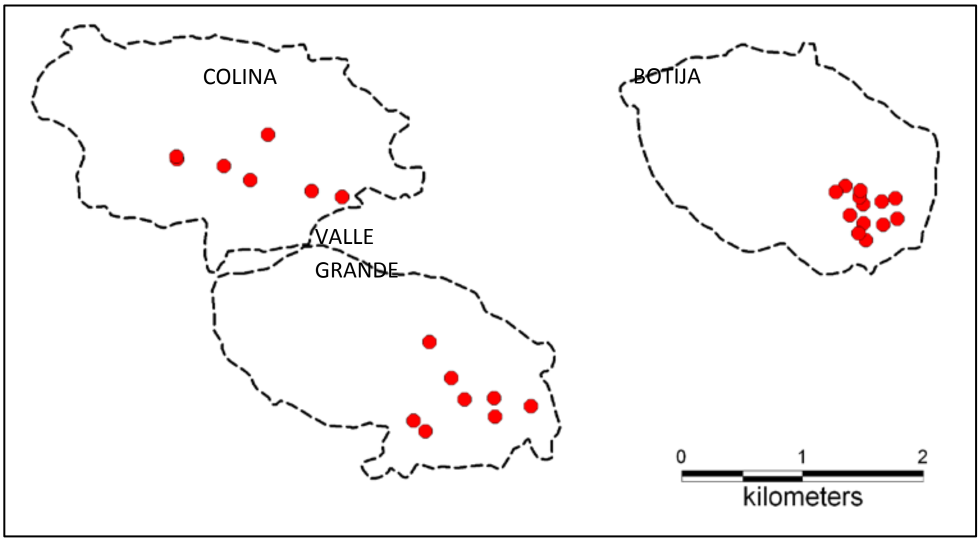
10.2.1 United Nations Development Programme (1967-1969)

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	446	93,000
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	**1,805**	**346,294**

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.

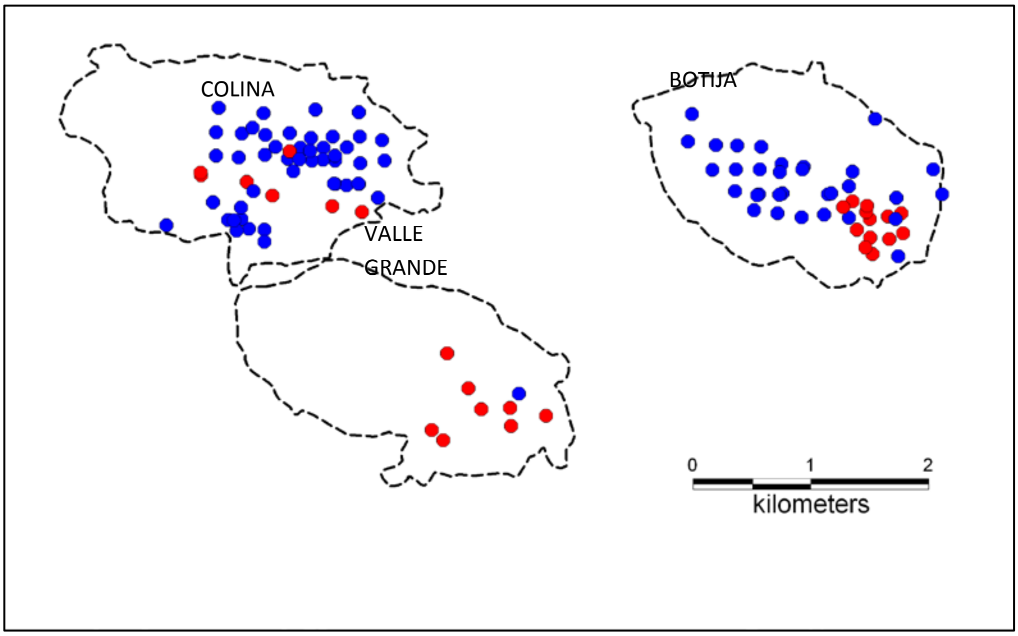
Core recovery averaged 95.5% (PMRD, 1977). Core recovery was poor in weathered intervals but excellent in fresh rock. No drill core, sample rejects or pulps from this drilling campaign remains.

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.

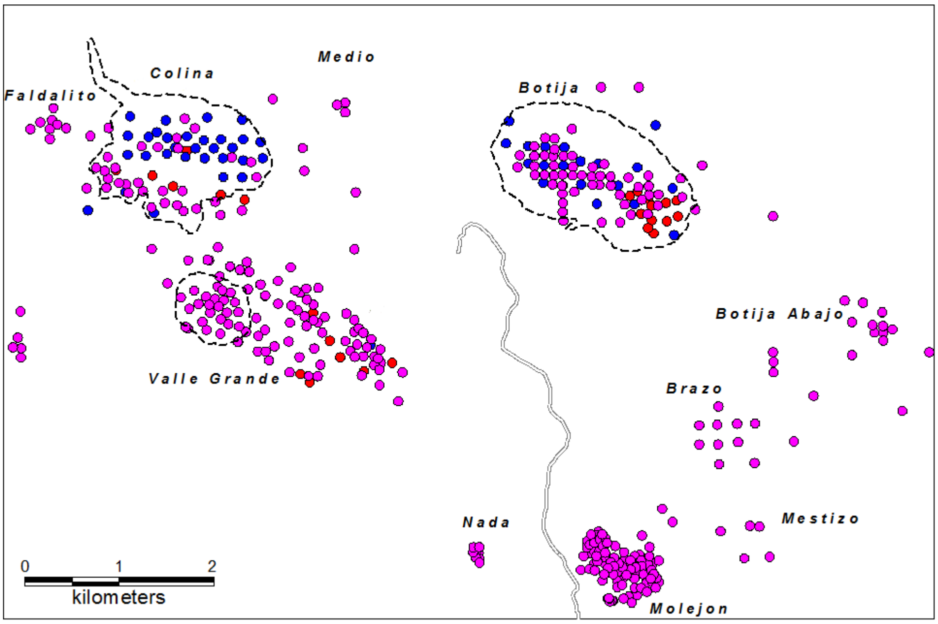
Figure 10-2 Plan view of PMRD (blue) and UNDP (red) collar locations (Source: MPSA, 2014)



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)

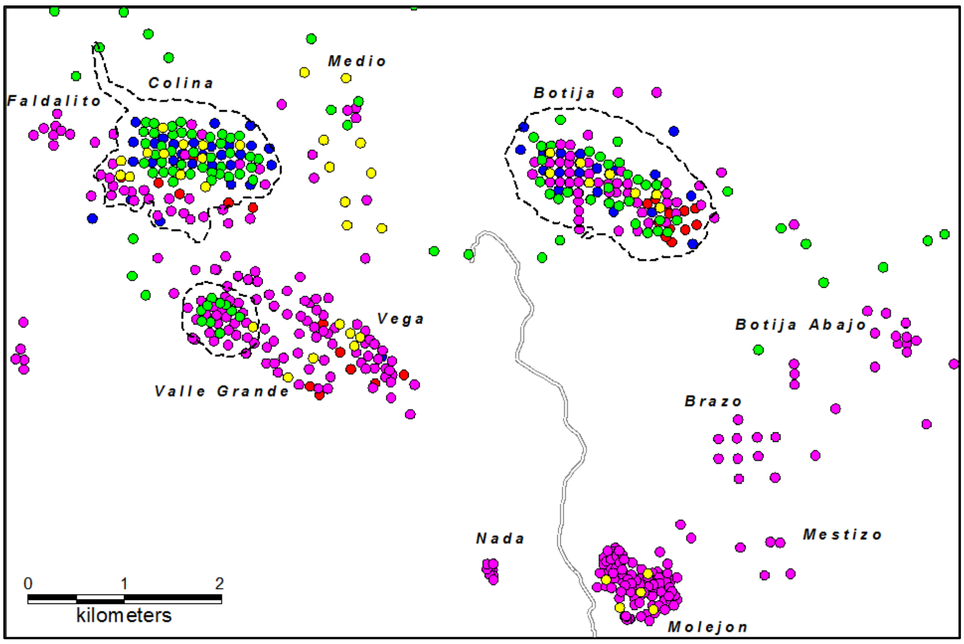


10.2.4 Teck (1996-1997)

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)



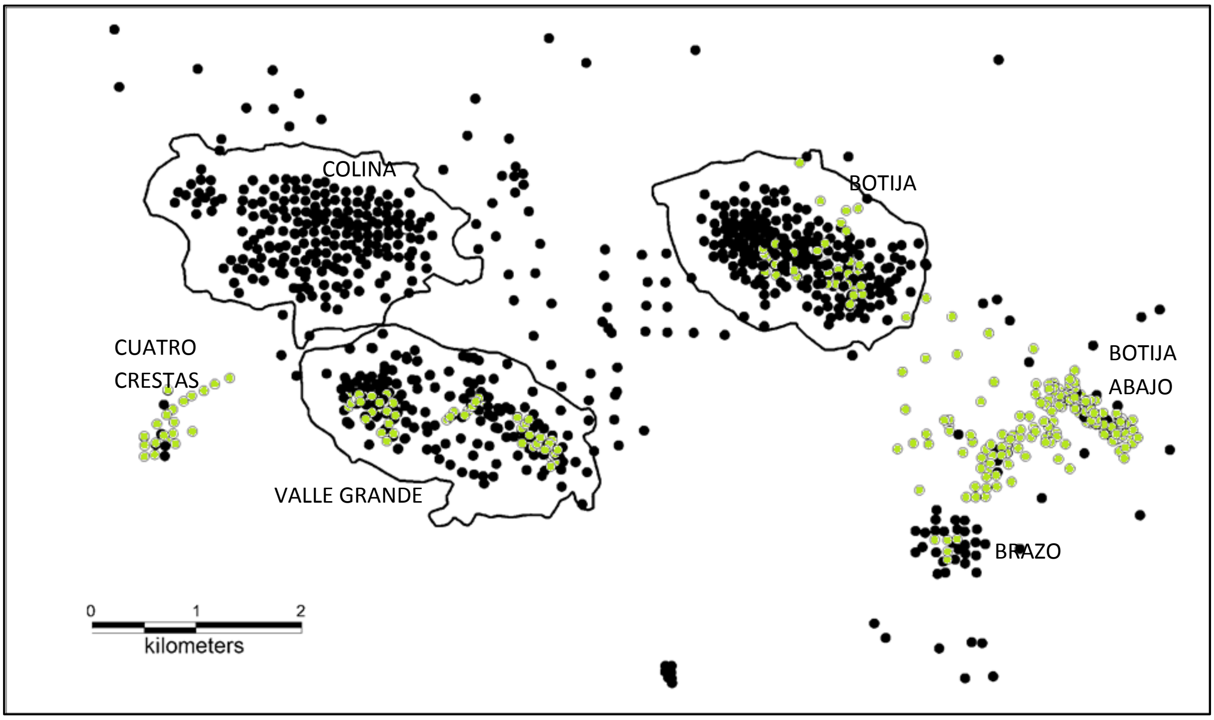
10.2.5 Petaquilla Copper (2006-2008)

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).

10.3 MPSA (2007-2013)

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.

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



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 damarea.

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

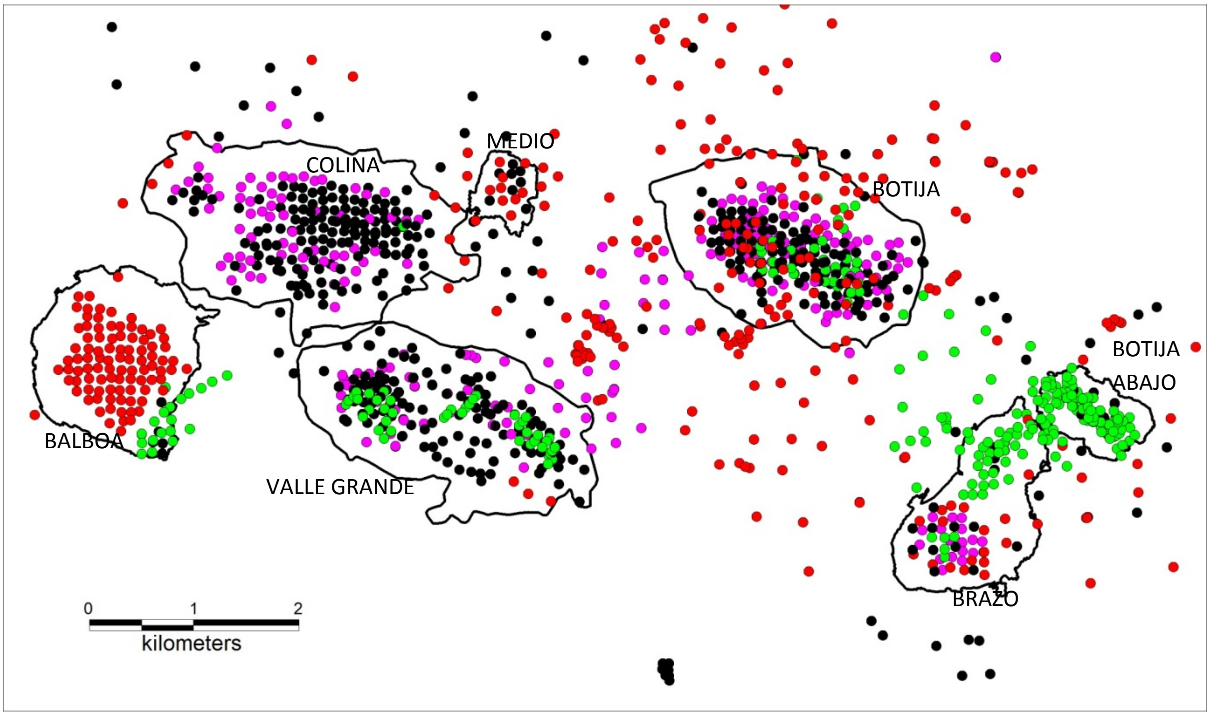
Core recovery data from MPSA, Teck and Adrian range from 1% to 100%, with an overall average of 93%. Less than 5% of the sample intervals have recoveries below 50%, whilst over 90% of the data have recoveries above 80%.

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

10.4 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.

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)



10.4.1 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.

10.4.2 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.

10.4.3 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.

10.5 Downhole surveying

10.5.1 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.

10.5.2 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 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.

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. Optiro has undertaken 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.

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.

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 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

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 recent 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%

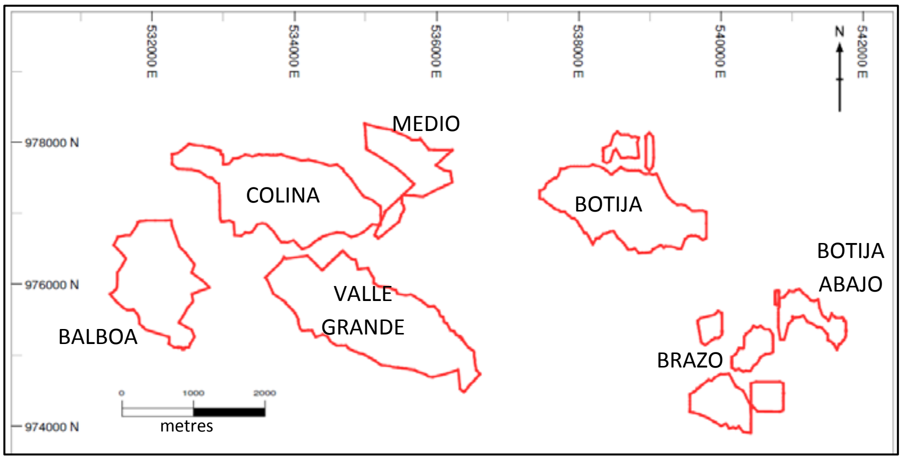
ITEM 14 MINERAL RESOURCE ESTIMATES

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.

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.

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.

Since finalising these estimates in January 2014 and the current date of this Technical Report, there has been no additional mineral resource drilling or improvements to the geological understanding of the Cobre Panamá deposits. Accordingly, the data used for these Mineral Resource estimates are currently valid.

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.

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 parent blocks sized at 50 mE by 50 mN. 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	446	93,000
Botija Abajo and Brazo	295	50,015
Valle Grande	265	46,229
Balboa	121	53,487
Total	**1,404**	**306,684**

Geological logging data imported into the database includes lithological codes, alteration codes, visually estimated pyrite, chalcopyrite, bornite and chalcocite percentages and vein density.

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

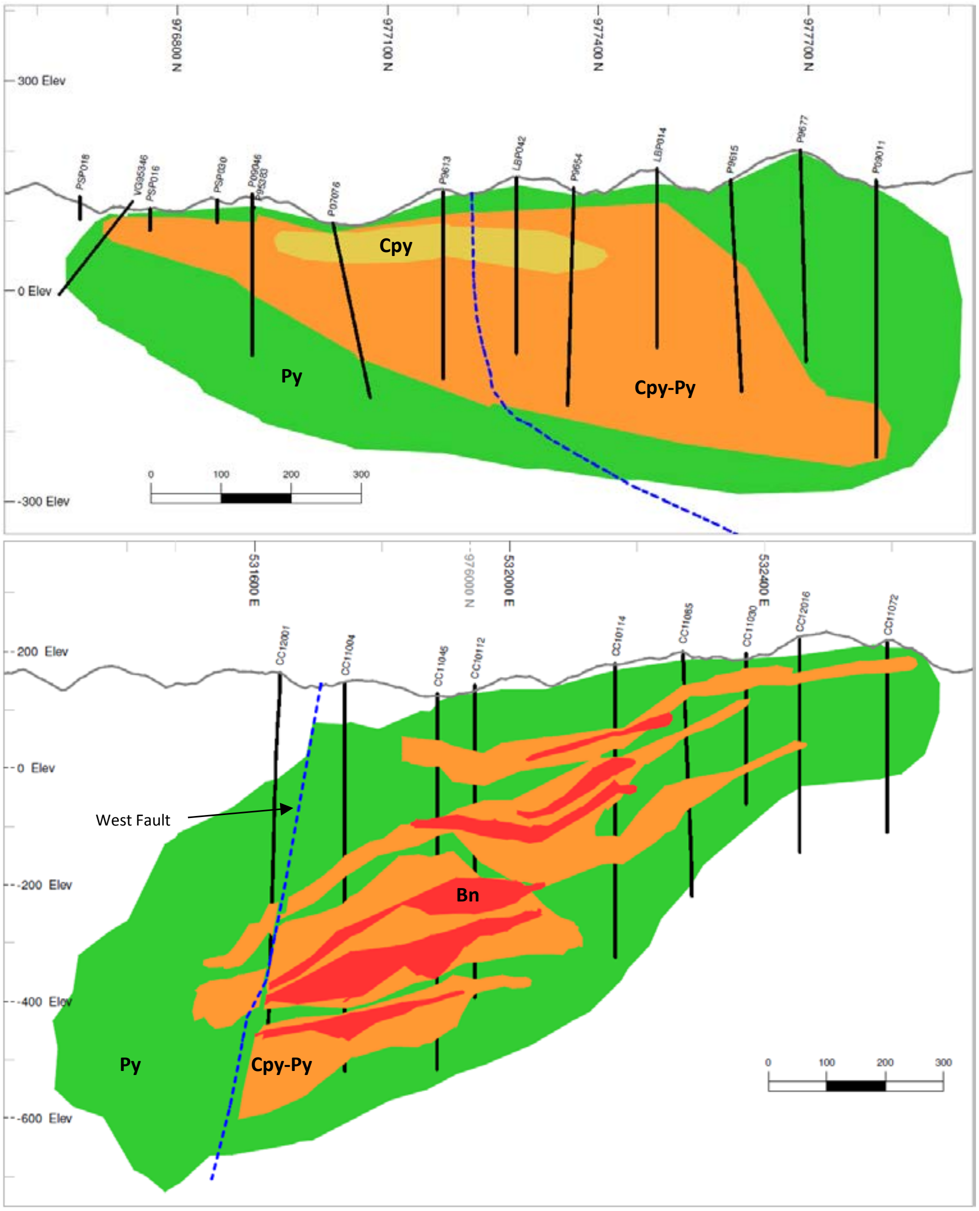
As discussed in Item 7, the database includes geological logging of alteration types. The alteration intersections were reviewed to determine if a 3D interpretation of alteration types could be generated. The distribution of the logged alteration types is spatially erratic and it was therefore not possible 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

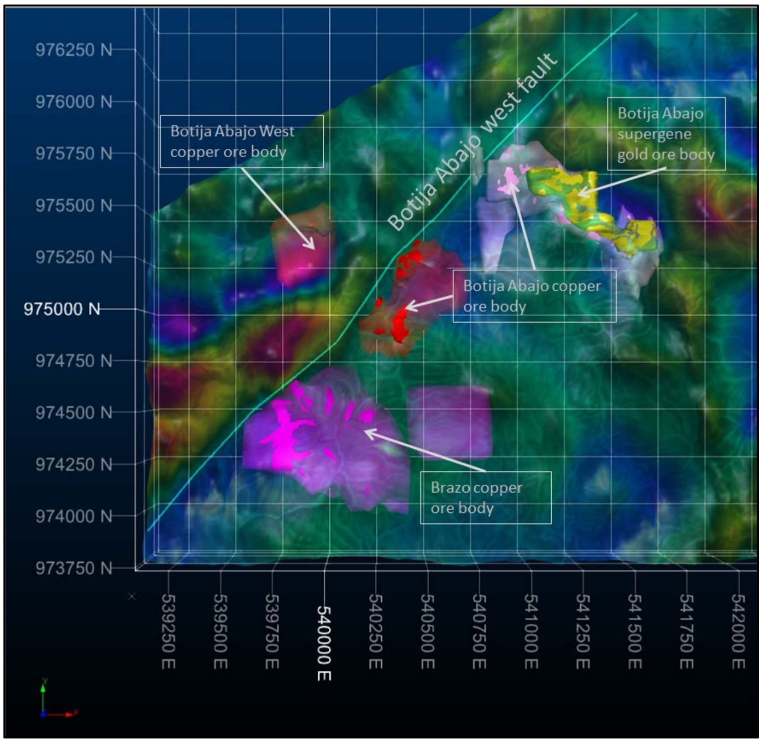
As discussed in Item 7, faults have been modelled at Colina, Medio, Valle Grande, Balboa, Botija Abajo and Brazo based on offsets observed in the geological and mineralisation modelling. While field mapping and logging data at Botija suggests the presence of faulting, no major fault offsets controlling mineralisation were observed. Continued FQM study through detailed logging and mapping, 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

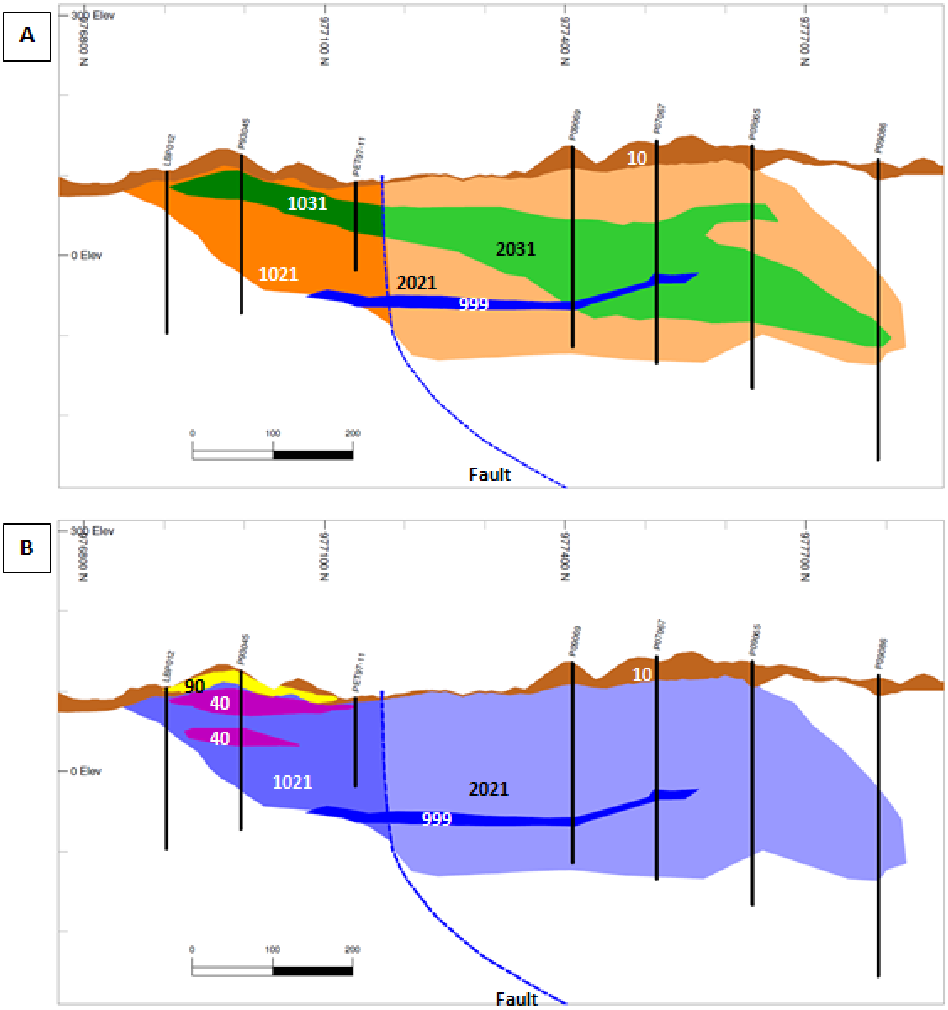
The mineralisation at the Cobre Project tends to occur within the porphyritic rocks and extends into the contact area of the adjacent plutonic/volcanic rocks. For each deposit, examination of the drillhole intersections and grade inflections in log-probability plots was used to select nominal cut-off 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.

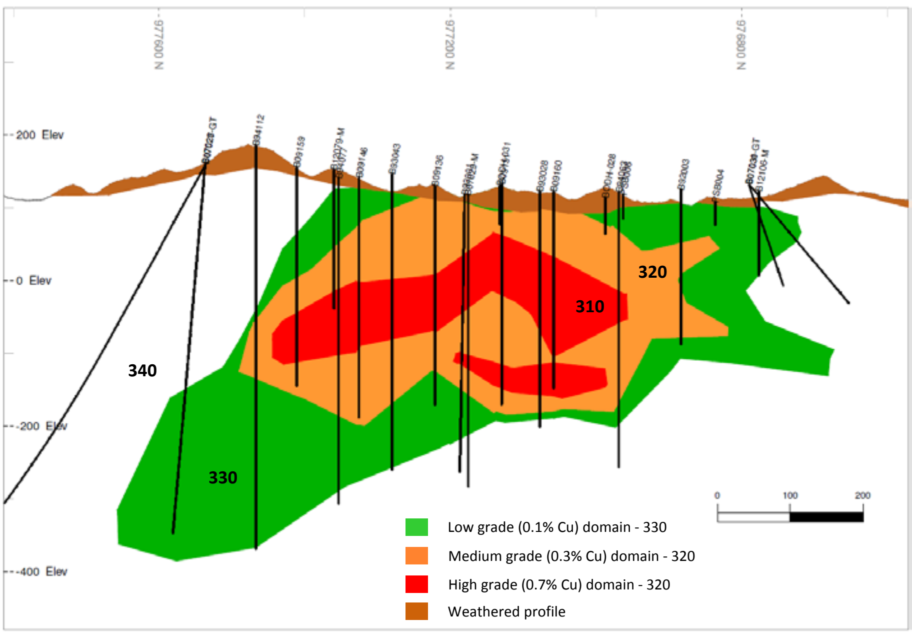
Copper mineralisation at Botija was based on nominal copper cut-off grades of 0.15% for low grade, 0.3% for medium grade and 0.7% for high grade mineralisation (Figure 14-5). Gold, molybdenum and silver were each estimated within the same copper domains due to reasonable to good correlations with copper 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

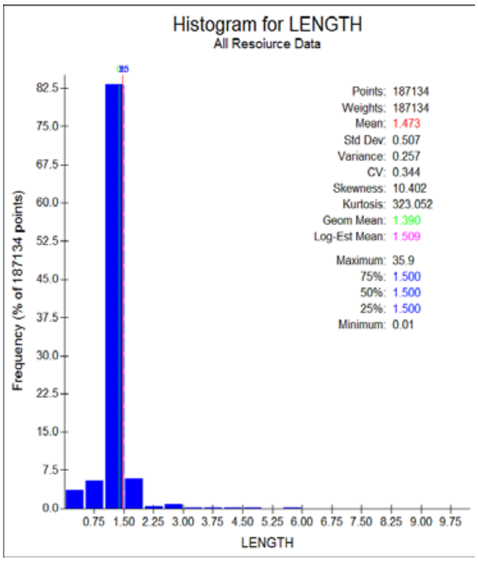
Statistical analysis of the data, including spatial statistics, was carried out using Snowden Supervisor software. Data distributions for copper, gold, molybdenum and silver were investigated by domain with histograms and probability plots and statistical parameters for the composited copper, gold, molybdenum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

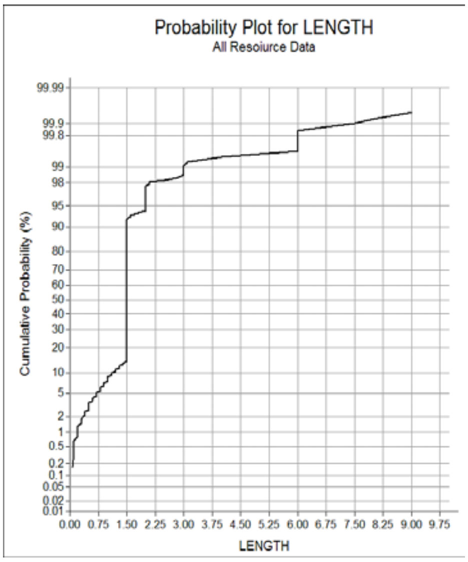
The objective of the mineralisation interpretations was to separate mixed populations and reduce internal variability, thereby assisting with spatial analysis and providing a more robust estimate. Molybdenum and silver histograms showed some evidence of possible detection limit artefacts associated with 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	340
		Low grade	330
		Medium grade	320
		High grade	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

	Copper			Gold			Molybdenum			Silver		
Domain	**Number**	**Mean**	**CV**	**Number**	**Mean**	**CV**	**Number**	**Mean**	**CV**	**Number**	**Mean**	**CV**
Botija												
310	54,212	0.89	2.77	50,643	0.21	1.30	54,162	133	0.71	50,966	2.25	1.45
320	54,212	0.56	2.19	50,643	0.11	1.29	54,162	109	0.70	50,966	1.56	1.18
330	54,212	0.26	1.55	50,643	0.05	1.01	54,162	47	0.51	50,966	0.92	0.89
340	54,212	0.07	0.90	50,643	0.02	0.53	54,162	15	0.27	50,966	0.48	0.52
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	1524	0.032	1.14	1524	39.33	1.93	1524	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	3.2	1.8	2000	20.0
	320	3.2	1.2	2000	18.0
	330	2.0	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

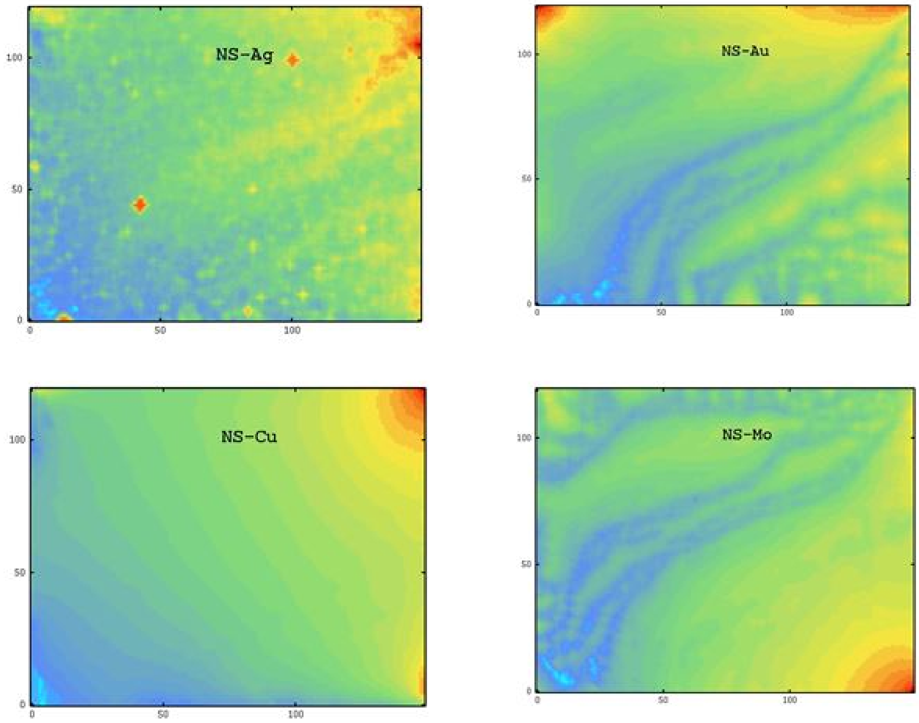


Figure 14-9 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Balboa; hot colours signify high values and vice versa

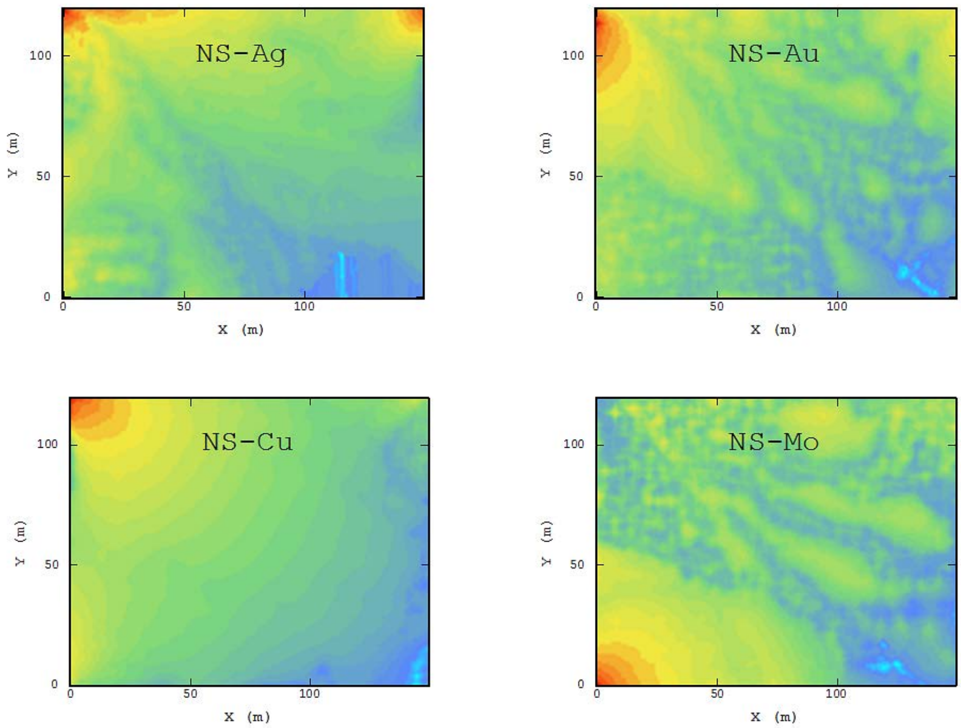


Figure 14-10 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Colina; hot colours signify high values and vice versa

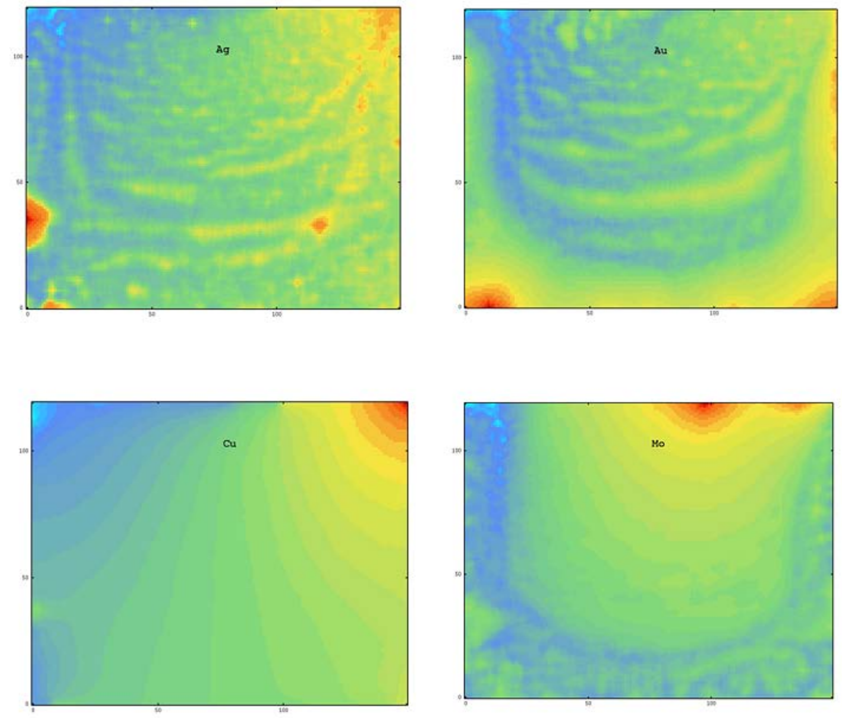


Figure 14-11 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Valle Grande; hot colours signify high values and vice versa

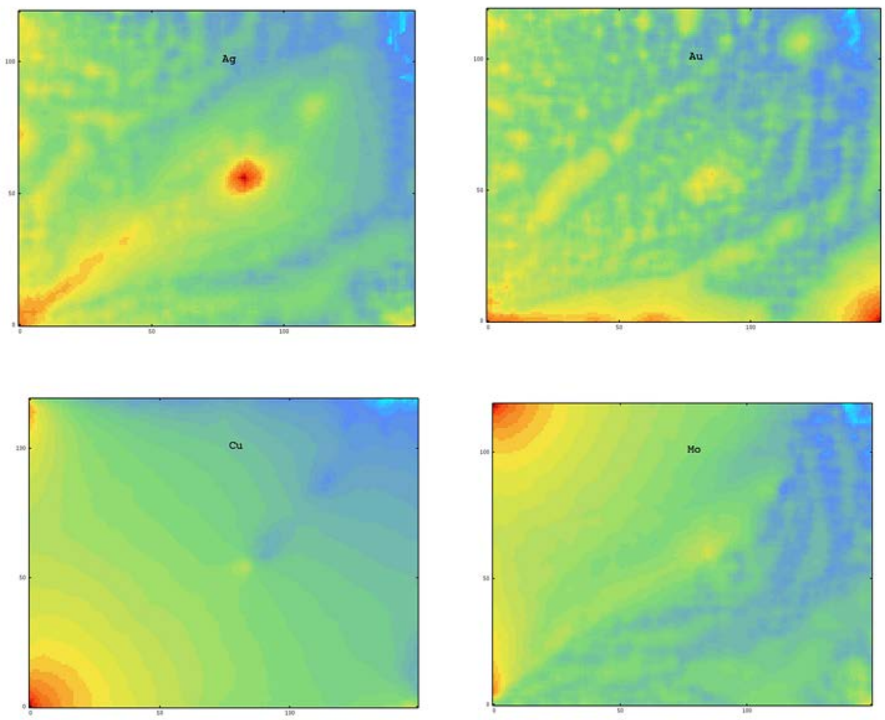
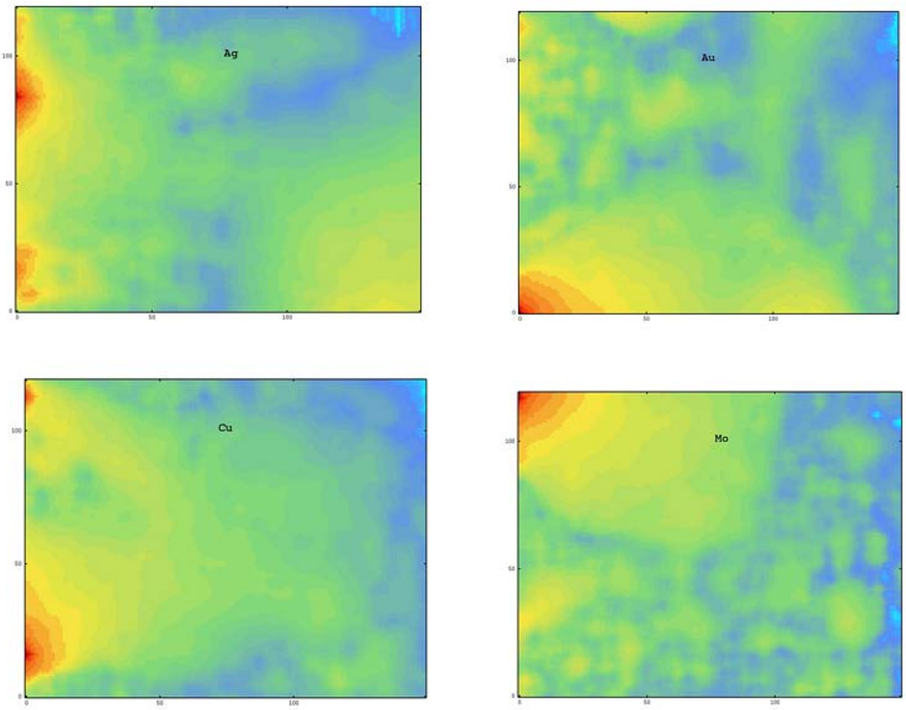


Figure 14-12 Self organizing feature maps for silver (Ag), gold (Au), copper (Cu) and molybdenum (Mo) at Medio; hot colours signify high values and vice versa



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.

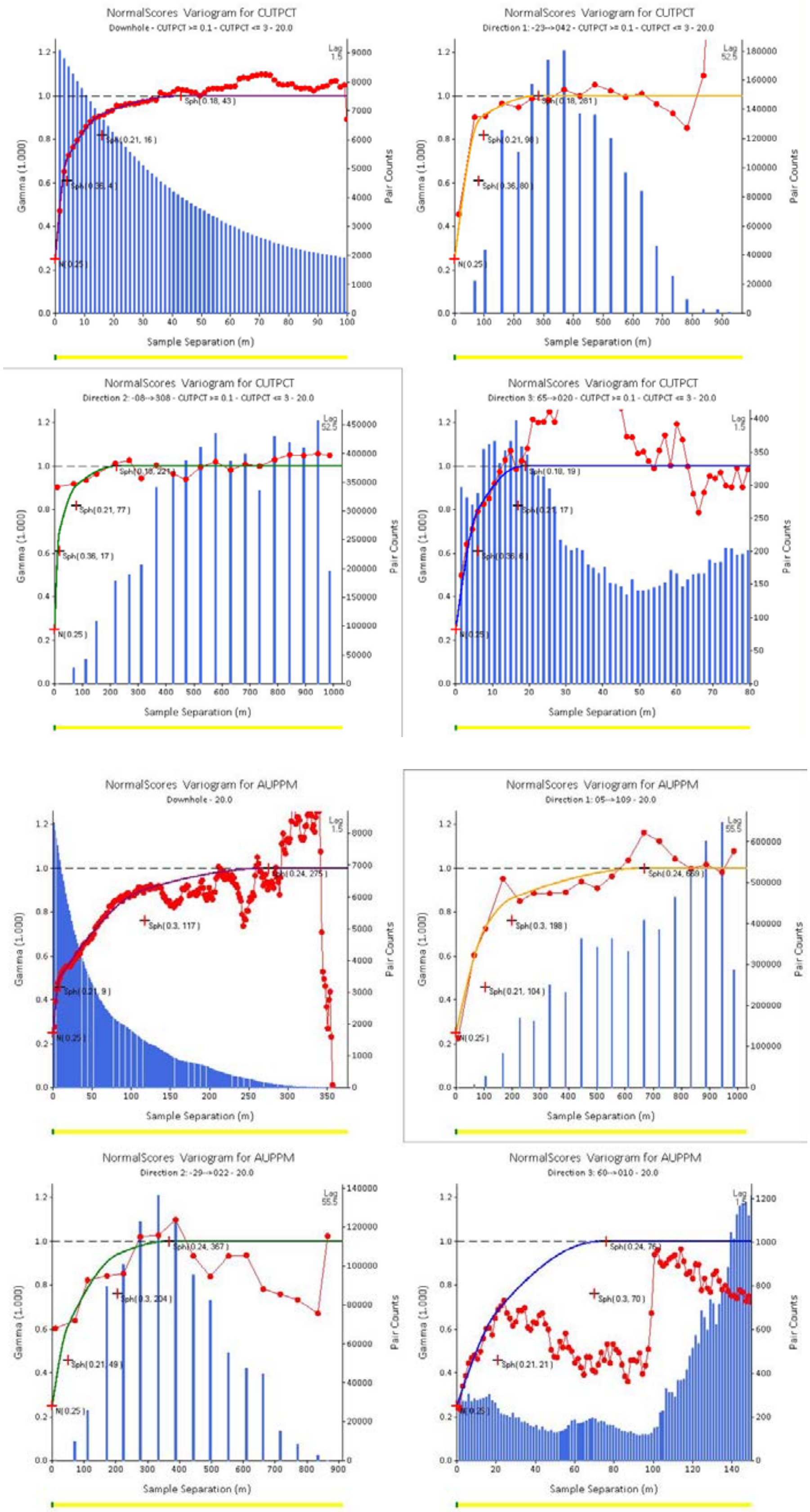
At Valle Grande there is no correlation between copper, gold, silver or molybdenum within the low grade domains. Within the medium grade domain there is a moderate to poor correlation between copper and gold (0.53) and copper and silver (0.69).Variogram analysis

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)

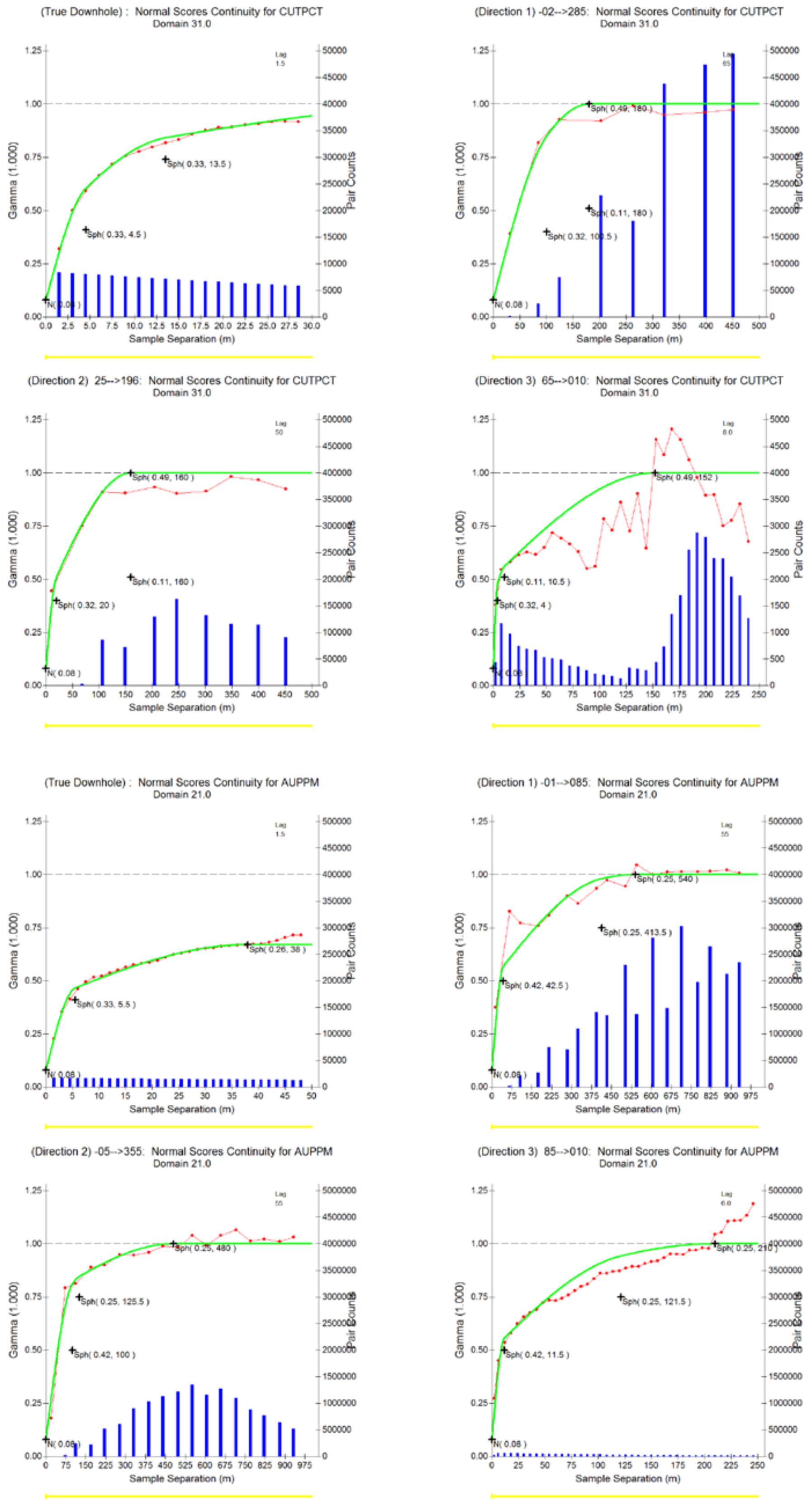


At Colina, nugget values for each element were similar, ranging from 7% to 28% of the total variability. For both copper and gold, the longest ranges of continuity within the main low grade domains were oriented at between 1° and -7° to the east with a range of 130 m (copper) and 540 m (gold). For the 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 -5° 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 -3° to -6° 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 -6° 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 -25° 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. Variograms 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 -10° 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. Block estimates were controlled by the parent block dimension. 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

	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	1200	10
Northing	973,700	979,200	110	50	220	25
Elevation	-795	470	85	15	85	15

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

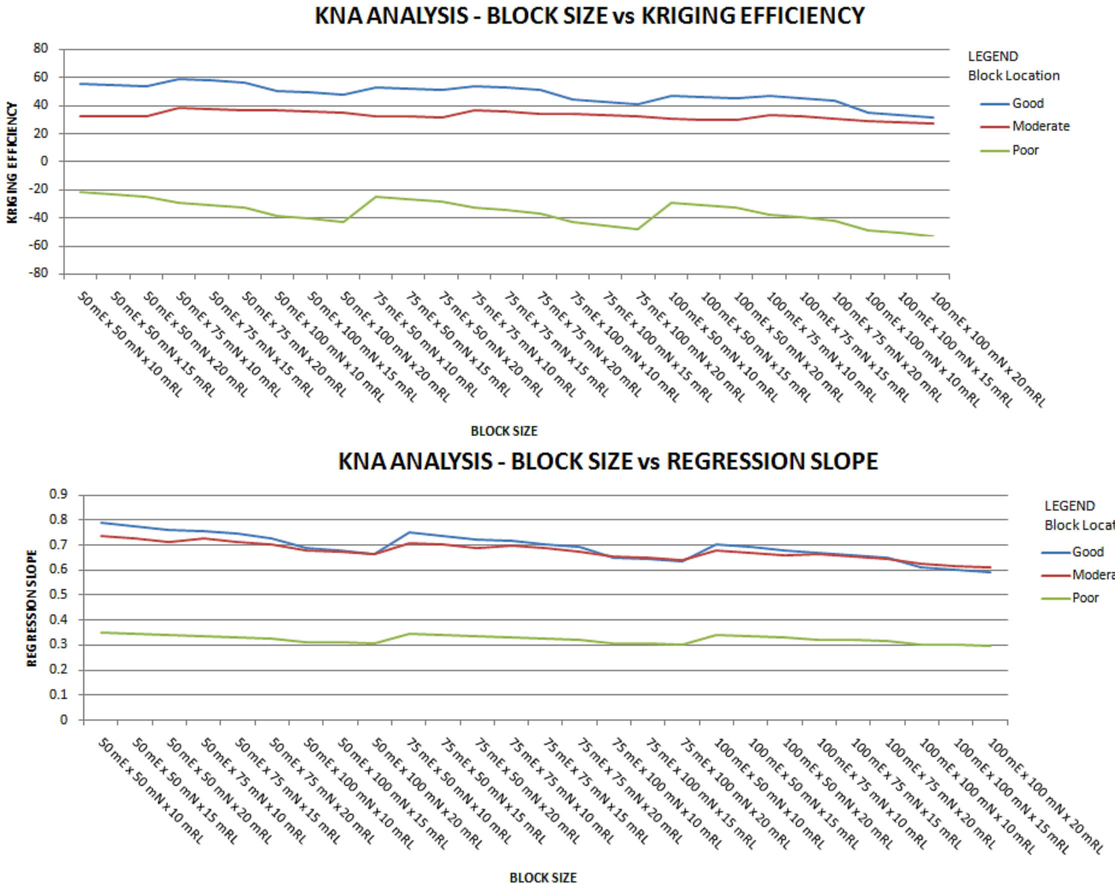
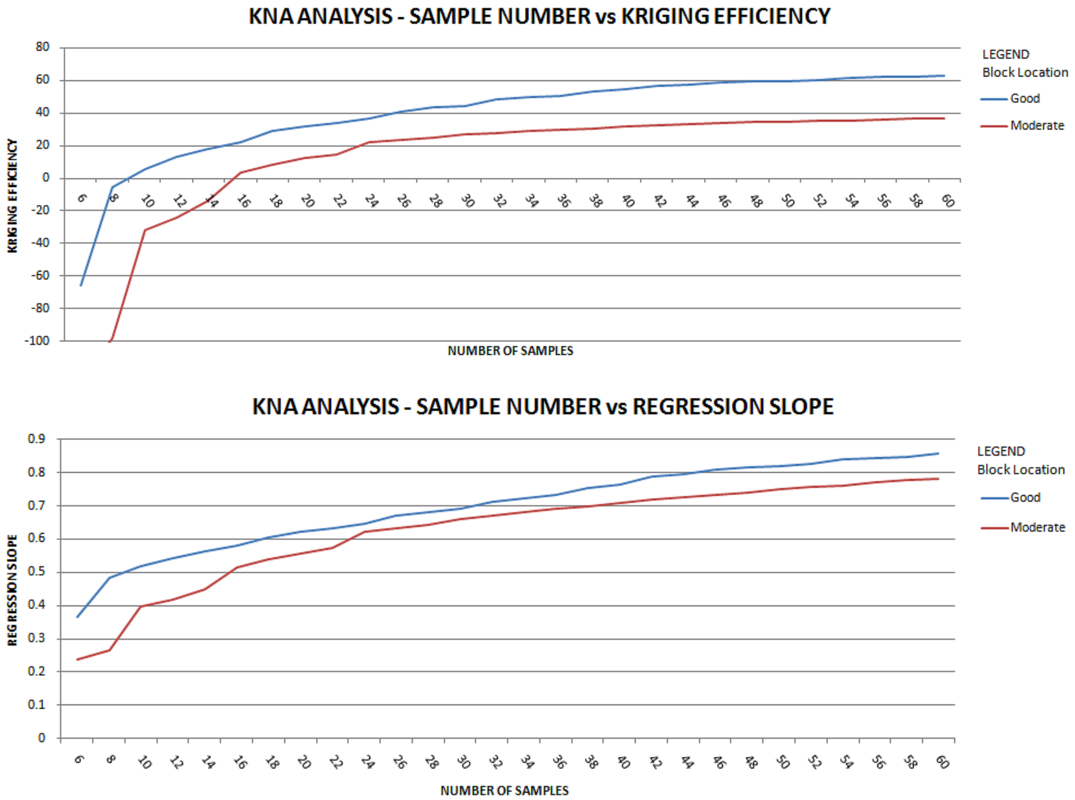


Figure 14-16 Kriging Neighbourhood Analysis to optimise sample numbers



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

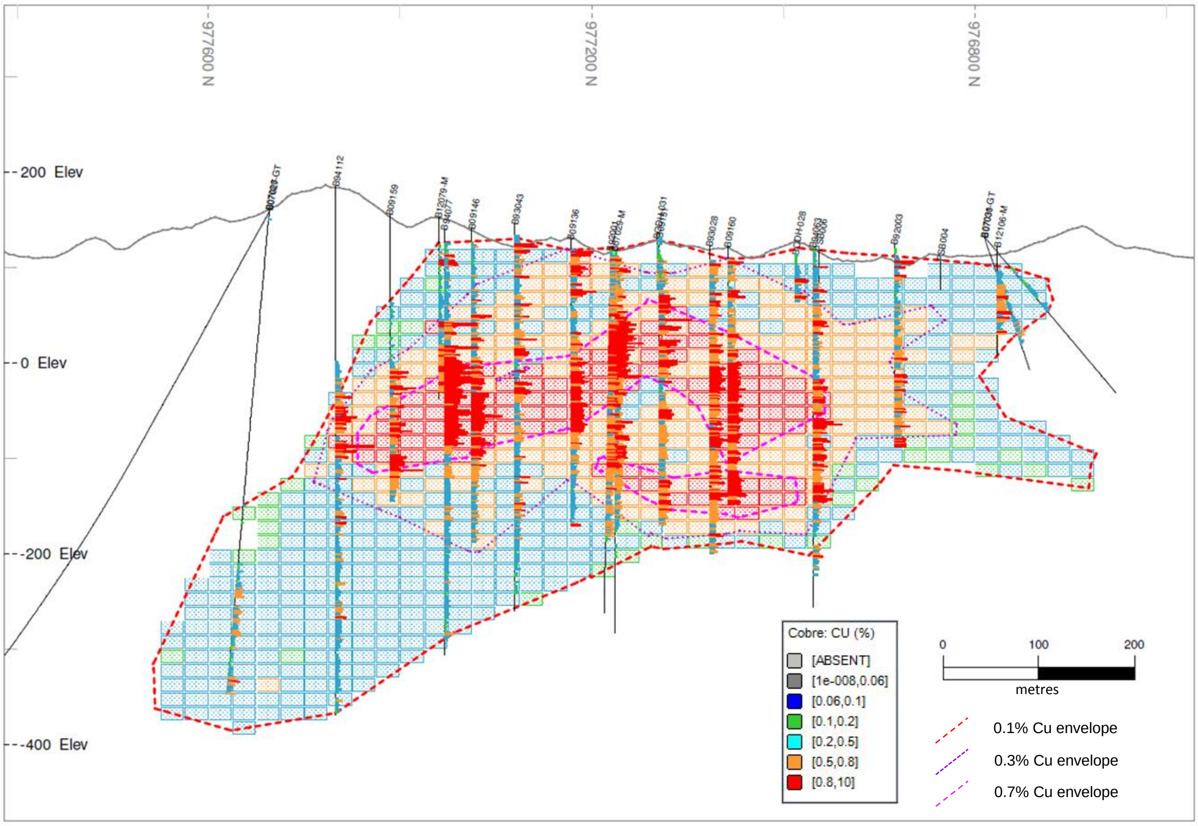


Figure 14-18 South-North cross section of Colina along 533,800 mE demonstrating the correlation between the drillhole data and the ordinary kriged (OK) model

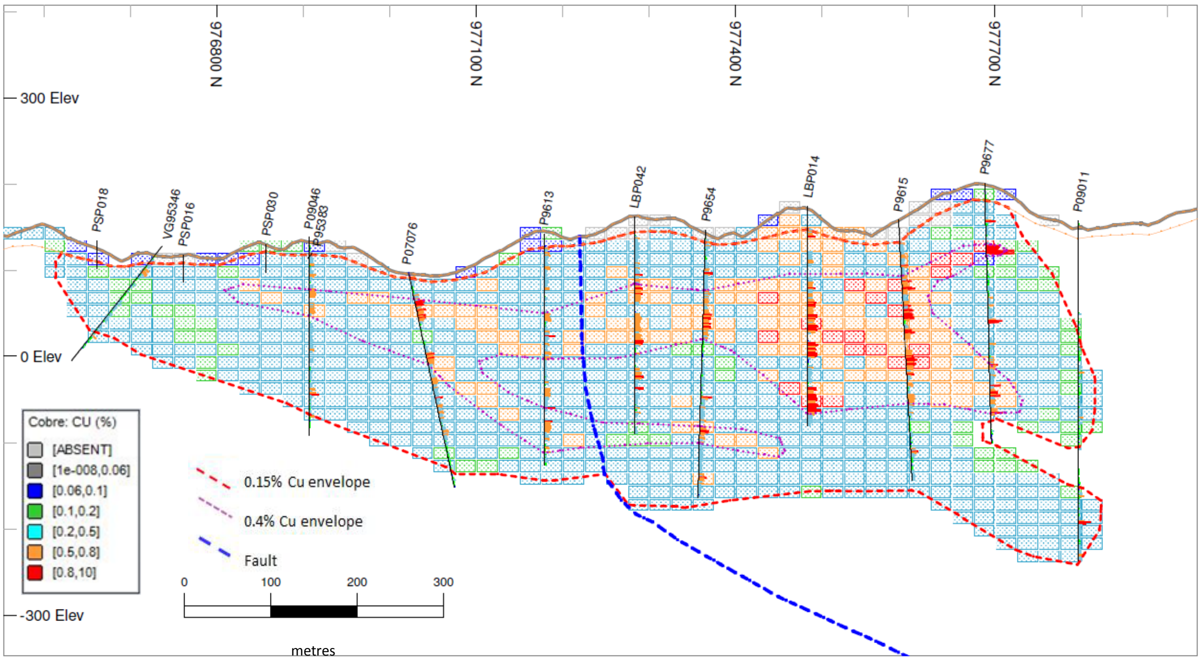


Figure 14-19 Oblique cross section looking northwest of Valle Grande demonstrating the correlation between the drillhole data and the ordinary kriged (OK) model

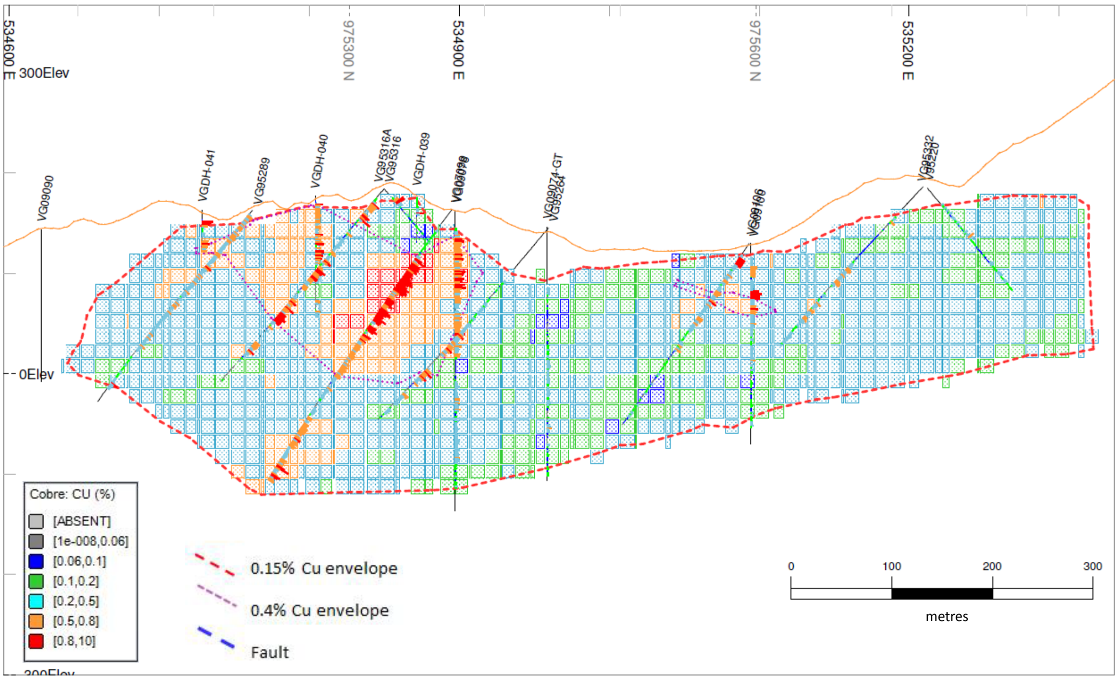


Figure 14-20 Oblique cross section looking northeast through the Balboa deposit demonstrating the correlation between the drillhole data and the ordinary kriged (OK) model

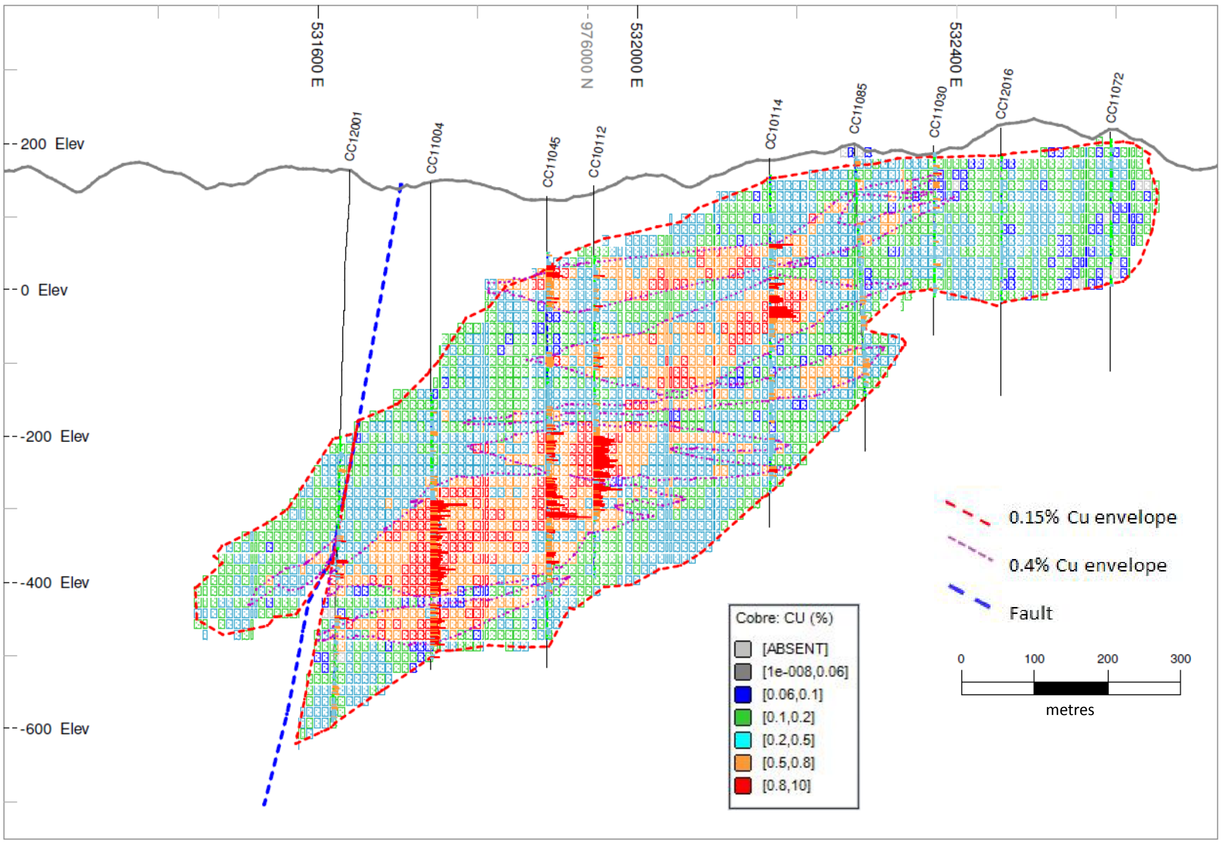
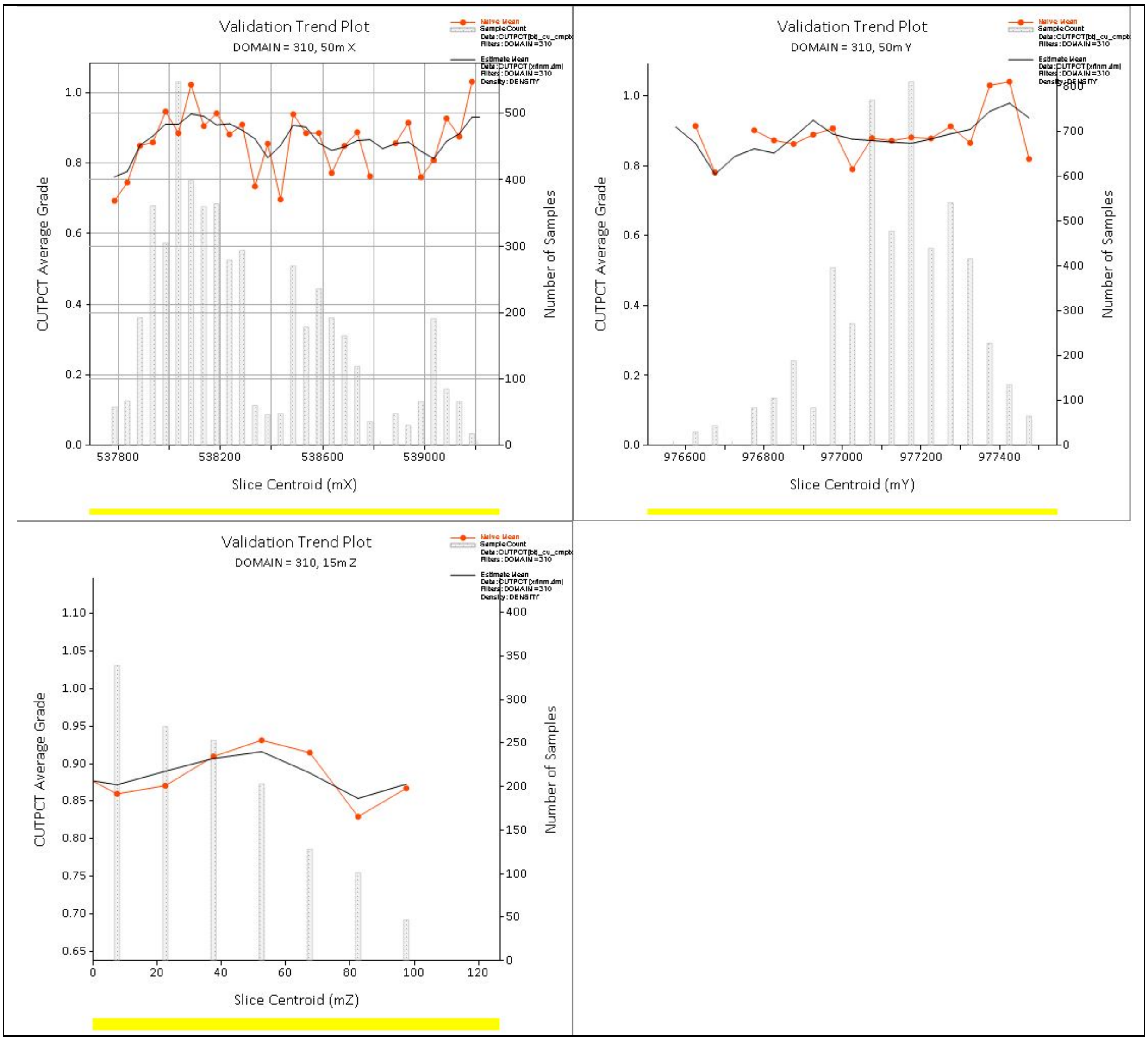


Table 14—7 Validation statistics for the main domains of each deposit

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**	**1.50**	**0.91**
Difference	**-0.62%**	**-0.21%**	**0.30%**	**6.17%**	**-0.59%**	**-1.85%**	**-16.1%**	**-1.9%**	**18.70%**	**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	**5.60%**	**0.80%**		**7.70%**	**2.60%**		**6.9%**	**13.0%**		**0.30%**	**1.10%**	
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	**-1.10%**	**-3.50%**		**-2.20%**	**-5.90%**		**1.4%**	**-1.6%**		**-1.30%**	**-2.20%**	
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	**-3.50%**	**-4.00%**		**-10.60%**	**-10.80%**		**-6.3%**	**-4.5%**		**-3.00%**	**-1.60%**	
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	**0.10%**	**-4.80%**		**-0.20%**	**16.10%**		**0.1%**	**-0.1%**		**1.90%**	**16.10%**	
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**	**0.74**	**2.63**	**0.34**
OK estimate	**0.58**	**0.70**	**0.25**	**0.08**		**0.03**	**48.6**	**16.0**	**54.8**	**0.79**	**2.80**	**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**	**0.30**	**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%**		**-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**	**0.20**			**0.18**		**45.7**	**38.6**		**0.91**	**0.68**	
OK estimate	**0.58**	**0.20**			**0.18**		**43.3**	**32.3**		**0.98**	**0.70**	
Difference	**3.20%**	**0.00%**			**2.23%**		**-5.23%**	**-16.31%**		**8.13%**	**2.94%**	

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.

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



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.

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 reserve generation.

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.

At Cobre, 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 is 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.

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

Bulk density sample data is not available for most MPSA drillhole samples and as a result does 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

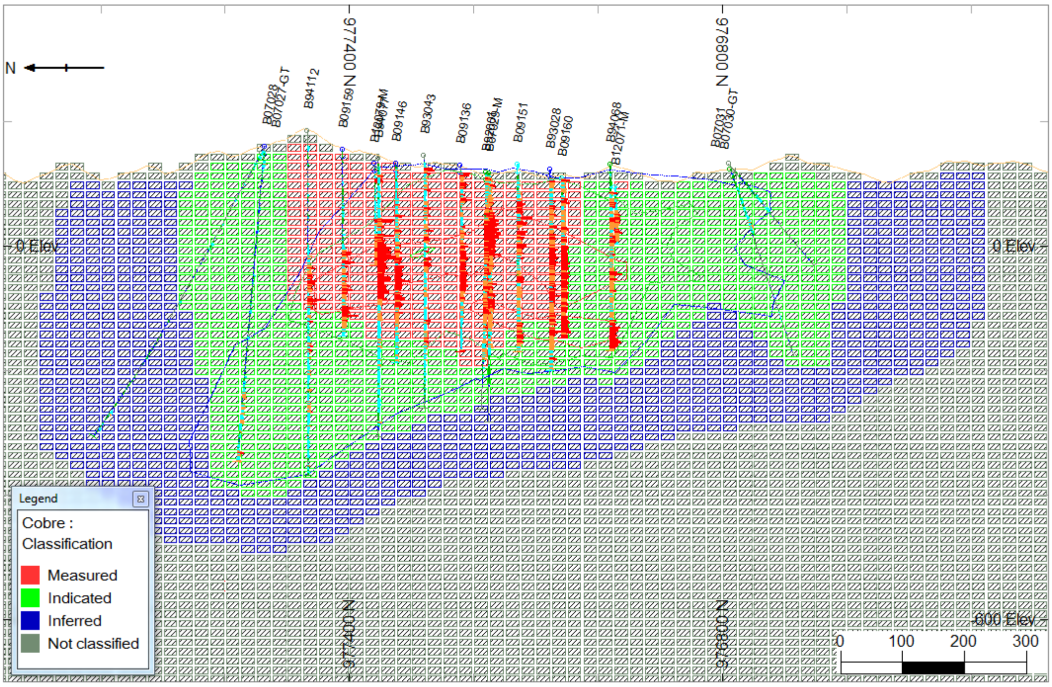


Table 14—9 Cobre Panamá Mineral Resource statement, at June 2015 above a 0.15% Copper Cut-off Grade. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

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**

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

This estimate of Mineral Resources for Cobre Panamá took into consideration the following changes since the previous (May 2010) estimate:

1. drillhole additions from extensional and infill drilling at Botija (~30 holes), Valle Grande (~3 holes), Colina (~10 holes) and Botija Abajo (~7 holes)
2. continued geological mapping and 3D geology modelling
3. improved understanding of controls on mineralisation, including faulting, phases of porphyry emplacement and the effects of alteration
4. increased confidence in historical Adrian and Teck sampling QAQC results through additional field duplicate sampling and analysis at a recognised umpire laboratory
5. estimation into a smaller block size as per the selective mining unit (SMU)
6. the use of improved constrained estimation routines for more accurate local block estimates with reduced grade smoothing and smearing
7. inclusion of the Balboa and Medio deposit Mineral Resource estimates, and
8. modification of mining input parameters for deriving an improved total copper (Cu) cut-off grade which has considered key mining input parameters including revised metal prices, mining costs and metal recovery figures that are better aligned with FQM practices and strategies.

Individual deposit changes, as compared to the May 2010 estimates (Rose et al, 2010), generally relate to reductions in ore tonnages associated with the change from an unconstrained estimation method to a constrained method.

The Inferred Mineral Resource statement for 2015 has significantly lower contained metal than the previous 2010 statement. The lower contained metal is primarily due to reduced volumes of above cut-off mineralisation located in the peripheral zones of each deposit. The reduced Inferred tonnages are due to the modelling of shorter ranges of grade continuity for the 2015 estimate. In addition, sharp grade changes along boundaries between low grade and ultra-low grade (waste) volumes do not support the extrapolation of low and medium grades into the surrounding ultra-low grade (waste) volumes as per the 2010 estimate.

A detailed comparison of the 2015 and 2010 estimates is provided in Table 14—10.

Table 14—10 Comparison of Mineral Resource estimate results as at June 2015 and the previous results as at May 2010; copper cut-off grade was 0.15%

First Quantum Minerals Limited - Cobre Panama Comparison of Mineral Resource estimate results for FQML as at January 2014 and Inmet as at December 2012 and May 2010 Cu cut-off grade was 0.15%																			
Deposit	**Resource classification**	**Tonnage Million tonnes**			**Cu%**			**Mo%**			**Au g/t**			**Ag g/t**			**Contained Cu Kilo tonnes**		
		Jun-15	**May-10**	**% Var**	**Jun-15**	**May-10**	**% Var**	**Jun-15**	**May-10**	**% Var**	**Jun-15**	**May-10**	**% Var**	**Jun-15**	**May-10**	**% Var**	**Jun-15**	**May-10**	**% Var**
Botija	Measured + Indicated	1,008	1,168	-14%	0.39	0.38	2%	0.007	0.007	0%	0.08	0.07	6%	1.17	1.13	3%	3,997	4,416	-10%
Colina	Indicated	1,032	1,178	-12%	0.39	0.35	9%	0.007	0.007	0%	0.06	0.05	13%	1.58	1.45	9%	4,076	4,157	-2%
Medio	Indicated	63	na	+	0.28	na	+	0.004	na	+	0.03	na	+	0.96	na	+	194		+
Valle Grande	Indicated	602	671	-10%	0.36	0.34	5%	0.006	0.006	1%	0.04	0.04	6%	1.37	1.34	3%	2,228	2,307	-3%
Balboa	Indicated	647	na	+	0.35	na	+	0.002	na	+	0.08	na	+	1.37	na	+	2,427	+	+
Botija Abajo	Indicated	114	184	-38%	0.31	0.28	11%	0.004	0.004	4%	0.06	0.09	-32%	0.93	0.91	3%	368	507	-27%
Brazo	Indicated	228	71	224%	0.36	0.43	-17%	0.004	0.004	-6%	0.05	0.12	-61%	0.81	0.73	10%	833	302	176%
Total Measured and Indicated		**3,695**	**3,271**	**13%**	**0.37**	**0.36**	**3%**	**0.006**	**0.007**	**-14%**	**0.07**	**0.06**	**5%**	**1.32**	**1.27**	**4%**	**14,122**	**11,688**	**21%**
Botija	Inferred	152	407	-63%	0.23	0.21	12%	0.004	0.004	0%	0.03	0.03	10%	0.78	0.72	9%	354	847	-58%
Colina	Inferred	125	1,090	-89%	0.26	0.24	10%	0.006	0.005	16%	0.05	0.03	49%	1.20	1.23	-2%	329	2,595	-87%
Medio	Inferred	189	na	+	0.25	na	+	0.005	na	+	0.03	na	+	1.25	na	+	482		+
Valle Grande	Inferred	363	1,141	-68%	0.29	0.24	21%	0.005	0.005	17%	0.03	0.03	9%	1.14	1.03	11%	1,048	2,727	-62%
Balboa	Inferred	79	na	+	0.23	na	+	0.003	na	+	0.04	na	+	0.96	na	+	180	+	+
Botija Abajo	Inferred	67	287	-77%	0.27	0.22	27%	0.005	0.005	8%	0.06	0.07	-20%	1.25	0.87	43%	182	616	-70%
Brazo	Inferred	76	269	-72%	0.21	0.27	-21%	0.003	0.004	-28%	0.01	0.07	-92%	0.73	0.55	32%	162	724	-78%
Total Inferred		**1,051**	**3,194**	**-67%**	**0.26**	**0.24**	**11%**	**0.005**	**0.005**	**5%**	**0.04**	**0.04**	**-7%**	**1.08**	**1.00**	**8%**	**2,737**	**7,509**	**-64%**

ITEM 15 MINERAL RESERVE ESTIMATES

15.1 Introduction

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

As part of the estimation process, pit optimisation aspects and detailed pit designs were completed by FQM personnel overseen and supervised by Michael Lawlor (QP) of FQM. All operating cost, recovery and revenue information for the optimisations, in addition to operational parameters for the open pit designs, were reviewed by Michael Lawlor (QP).

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 Reserves.

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) and phased pit outlines (shells) were used to create practical and detailed open pit designs accounting for the siting of in-pit crushers, in-wall drainage sumps, 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 was regularised to an agreed smallest mining unit (SMU) block size, suitable for the scale of proposed primary mining equipment, ie down to dimensions of 5 m vertical x 100 m x 75 m for saprolite and 15 m vertical x 200 m x 125 m for non-saprolite.

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.

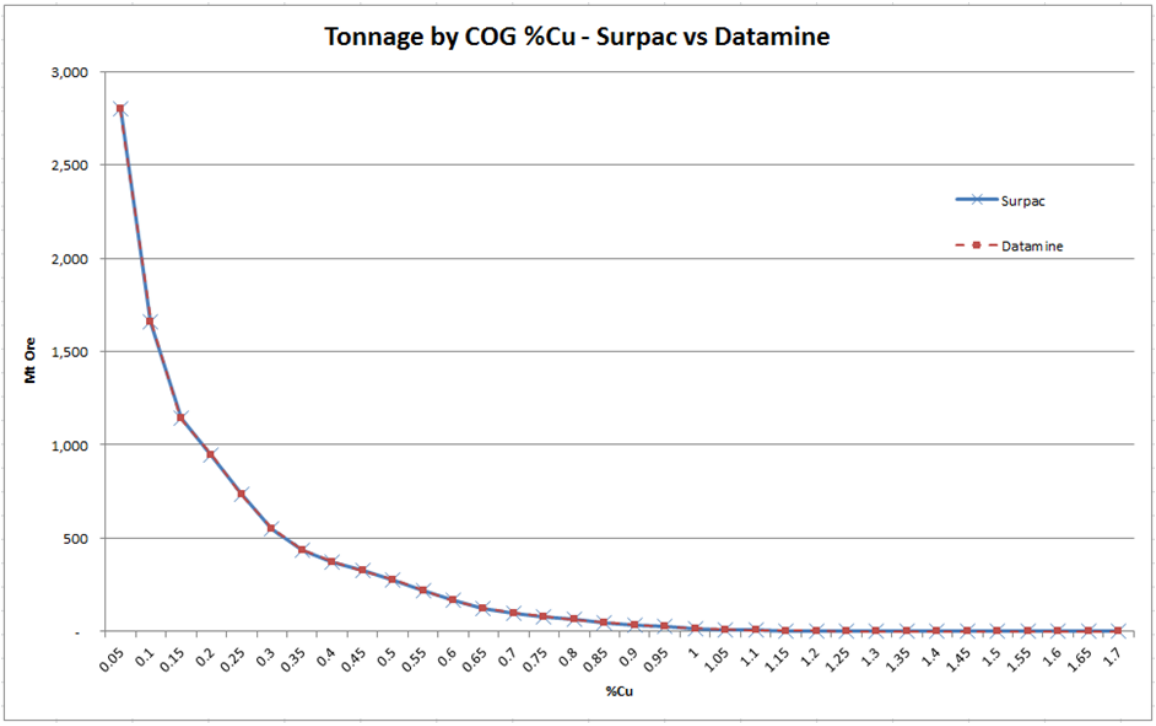
15.4 Pit Optimisation

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 sulphide 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.

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



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 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

Pit slope design criteria

Pit optimisation input included overall slope design angles as listed in Table 15-1. The geotechnical engineering basis for these design angles is outlined in Item 16.

Metal prices

The optimisation inputs for metal prices were as follows:

- Copper = US$3.00/lb (US$6,615/t)
- Molybdenum = US$13.50/lb (US$29,762/t)
- Gold = US$1,200/oz
- Silver = US$16.00/oz

Table 15—1 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
Saprolite	Varies by wall height (m) Non- River Diversion Areas	65	32.5
		60	33.0
		55	34.0
		50	35.0
		45	36.0
		40	37.5
		35	39.5
		30	40.2
		25	40.2
		20	40.2
	Varies by wall height (m) River Diversion Areas	65	29.5
		60	30.0
		55	31.0
		50	32.0
		45	33.0
		40	34.0
		35	36.0
		30	38.5
		25	40.2
		20	40.2

Metal recoveries

The input processing recovery projections were the same as those listed in Item 13 (Table 15-2).

Table 15—2 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%

For the saprock ore within each deposit, the recovery from stockpiles was reduced by 20%.

Operating costs

Since the Project will be mill constrained, the process operating costs are the sum of the fixed and variable costs. These costs are the same for each deposit:

- operating cost = $4.20/t
- fixed cost (equivalent G&A cost in variable terms) = $1.25/t
- total processing cost = $5.75/t

Details of these costs are outlined in Item 21.

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 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 Whittle input was essentially:

- 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 weighted average variable mining costs are $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.

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 (ocean freight)
- concentrate refining charges
- payable rates for each metal recovered into concentrate

Table 15-3 lists the concentrate charges and payable percentages.

Table 15—3 Cobre Panamá concentrate charges

	2014
Copper Concentrate	
Cu con - Cu grade, %	25.00%
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
Au refining, $/oz payable	$5.50
Ag refining, $/oz payable	$0.40
Cu payable, %	96.43%
Au payable, %	92.00%
Ag payable, %	90.00%
Cu metal cost, $/lb Cu payable	**$0.277**
Molybdenum Concentrate	
Mo con - Mo grade, %	52.00%
Mo con overland freight, $/dmt	$0.00
Mo con ocean freight, $/dmt	$90.00
Mo con treatment, $/dmt	$0.00
Cu removal charge, $/lb Cu	$0.00
Mo refining, $/lb payable	$0.00
Mo payable, %	86.20%
Mo metal cost, $/lb Mo payable	**$0.091**

The Panamánian government levies a royalty on each metal recovered[4]. From the above table, therefore:

[4] 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". The royalty as set out in the Mining Code, was increased to 5% for base metals and 4% for precious metal concentrates on October 1st, 2011. Changes to Law 9 will be made to align with the new Code upon renewal of MPSA's mining concession in 2017.

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

- The copper metal cost is:
  = metal price x royalty + 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[5] + Au refining cost x Au payable
  = \$1,200 x 5% + \$5.50 x 0.92 = \$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 0.90 = \$1.16/oz Ag (\$16.92/lb Ag)

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 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

[5] In the optimisations, for simplicity, a 5% royalty rate was adopted for all four metals.

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

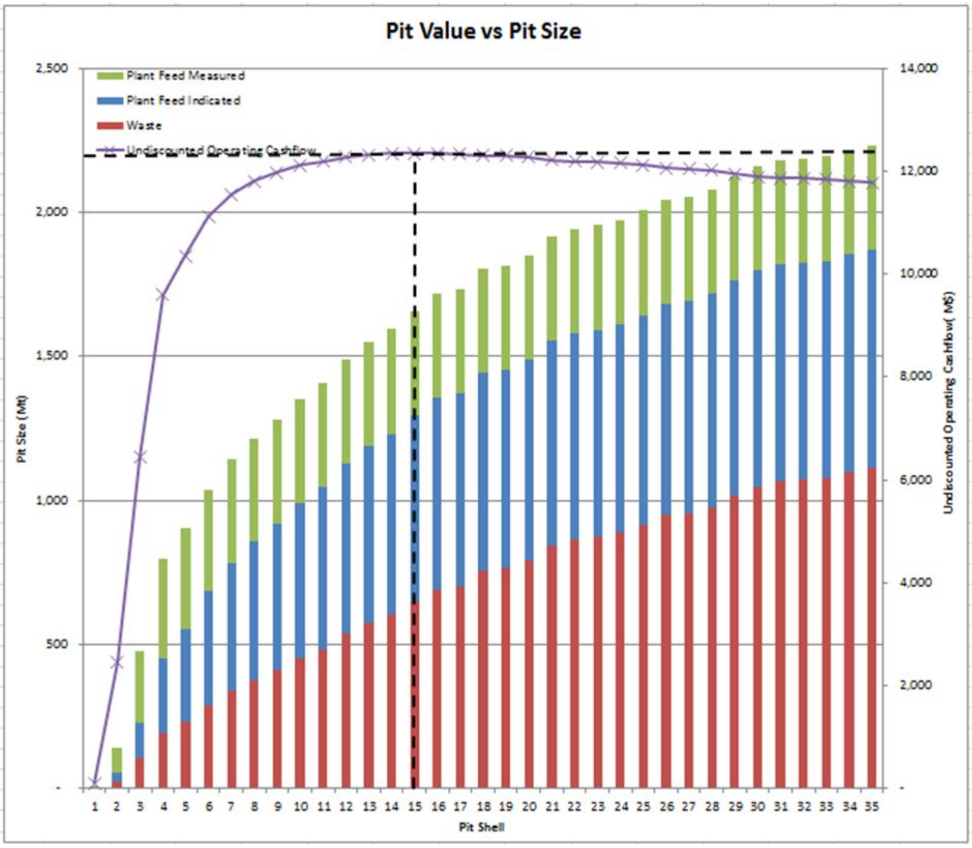


Figure 15-3 Colina/Medio Pit optimisation results

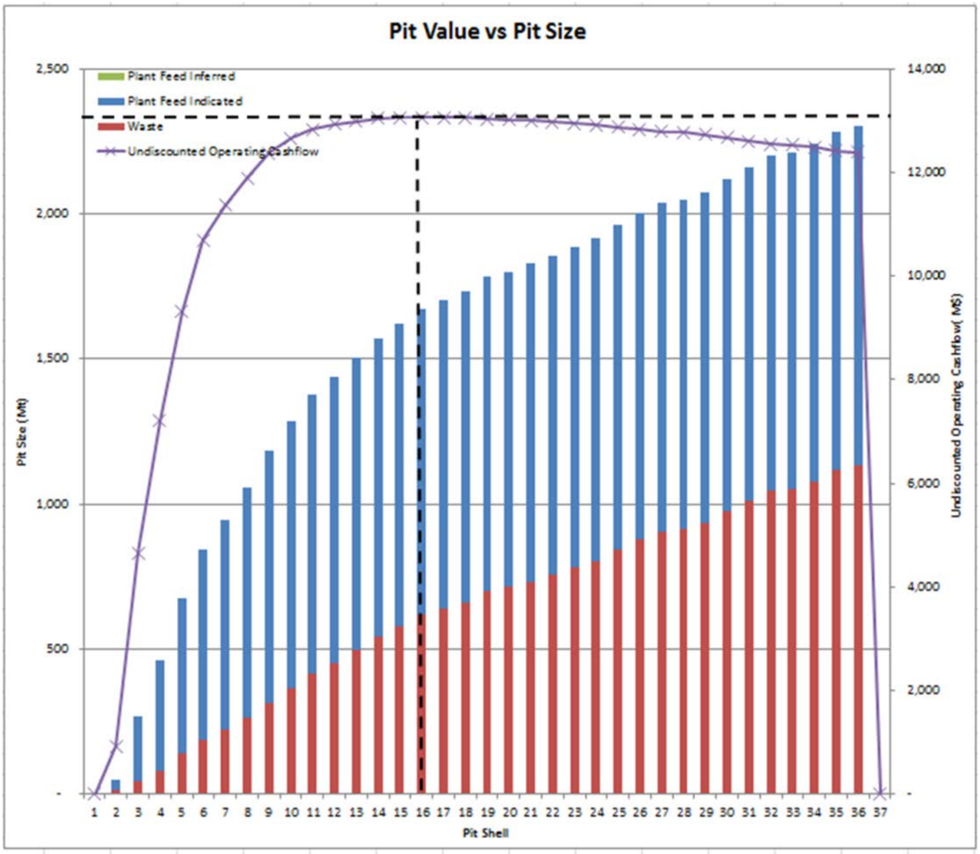


Figure 15-4 Valle Grande Pit optimisation results

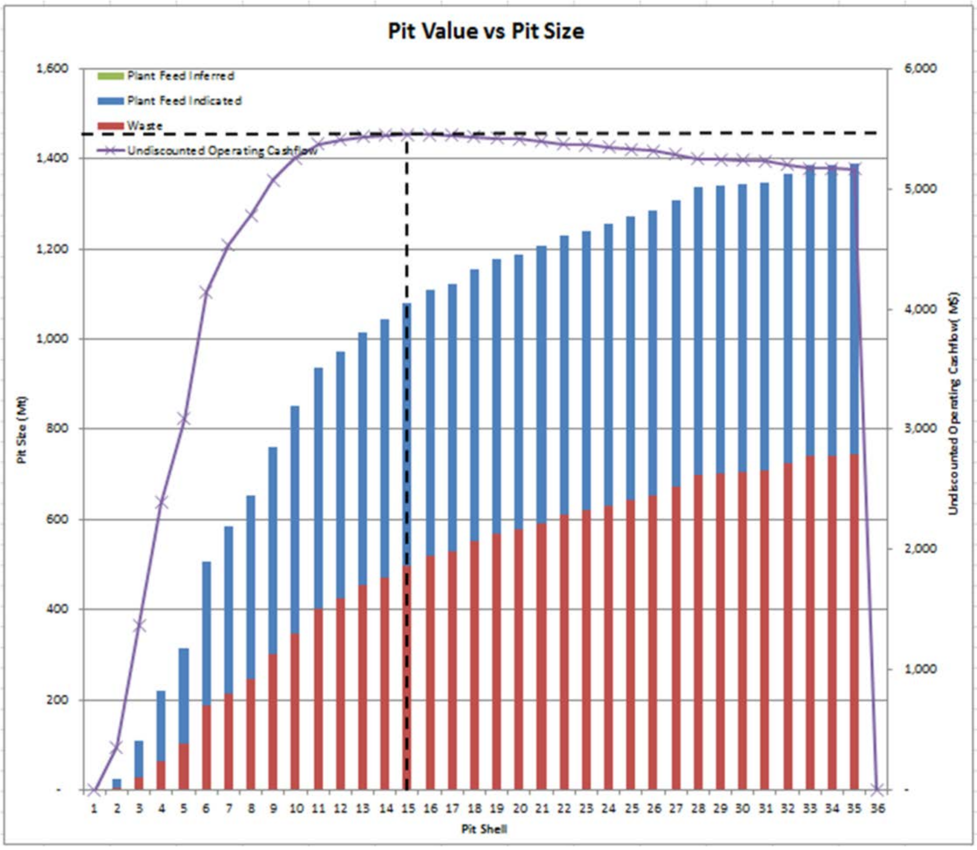


Figure 15-5 Balboa Pit optimisation results

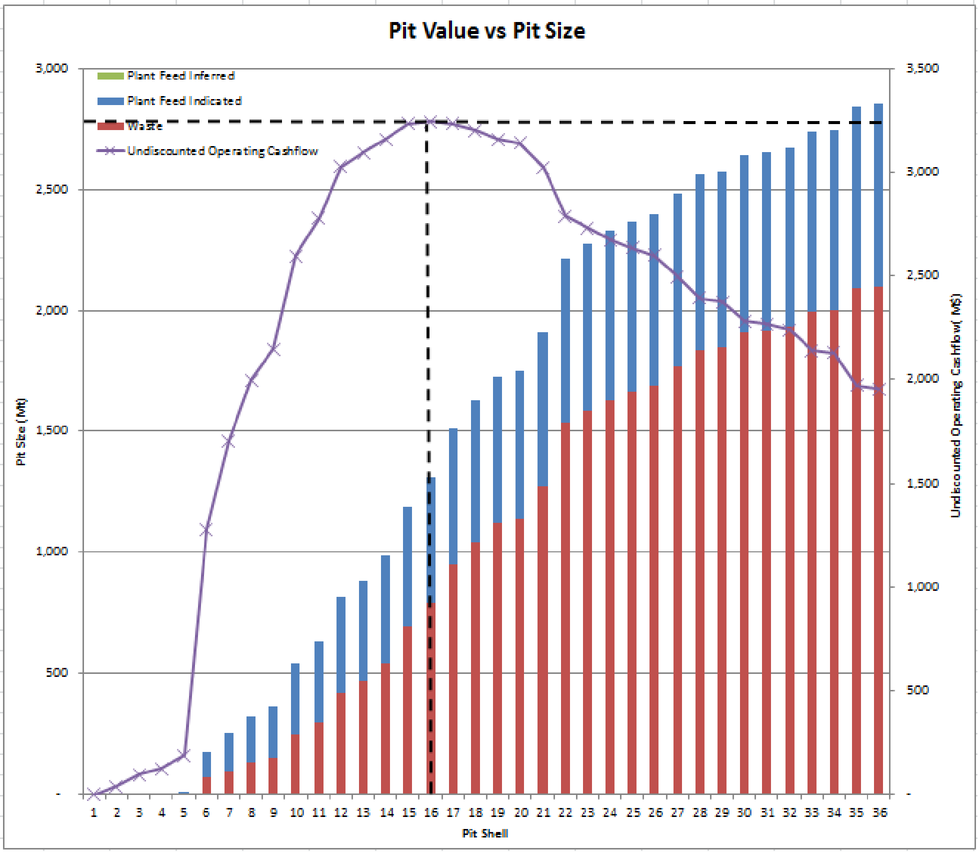


Figure 15-6 Botija Abajo, Brazo (BABR) Pit optimisation results

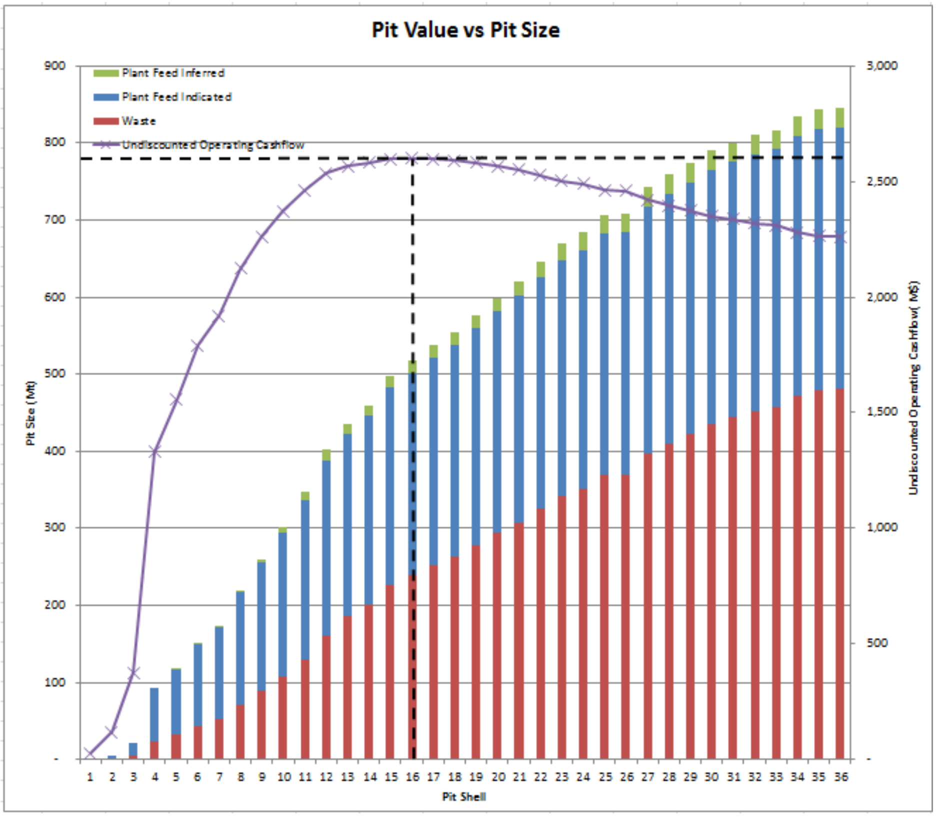


Table 15—7 Cobre Panamá optimal pit shell inventories

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**

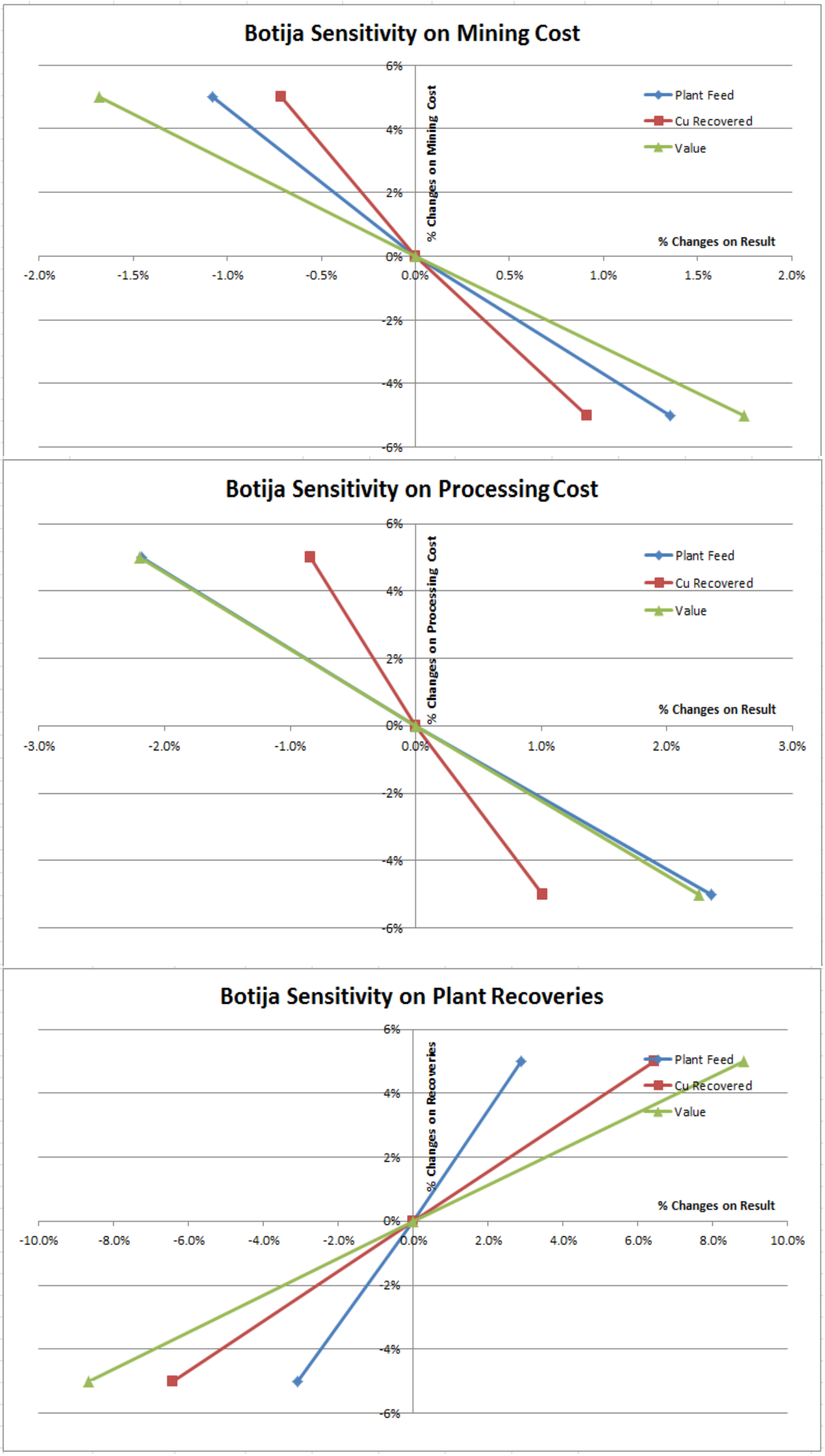
Various intermediate shells were selected for the purposes of phased pit designs. These are listed with their respective inventories in Item 15.5.

15.4.5 Optimisation sensitivity

Optimisation sensitivity analyses were run, for each deposit, and separately for a range of mining costs, processing costs and processing recoveries. The results are shown in Figures 15-7 to 15-11 and confirm that processing recovery is the most sensitive variable (sensibly, since this is related to net return). The magnitude of the impact of varying recovery would be exactly the same for metal price.

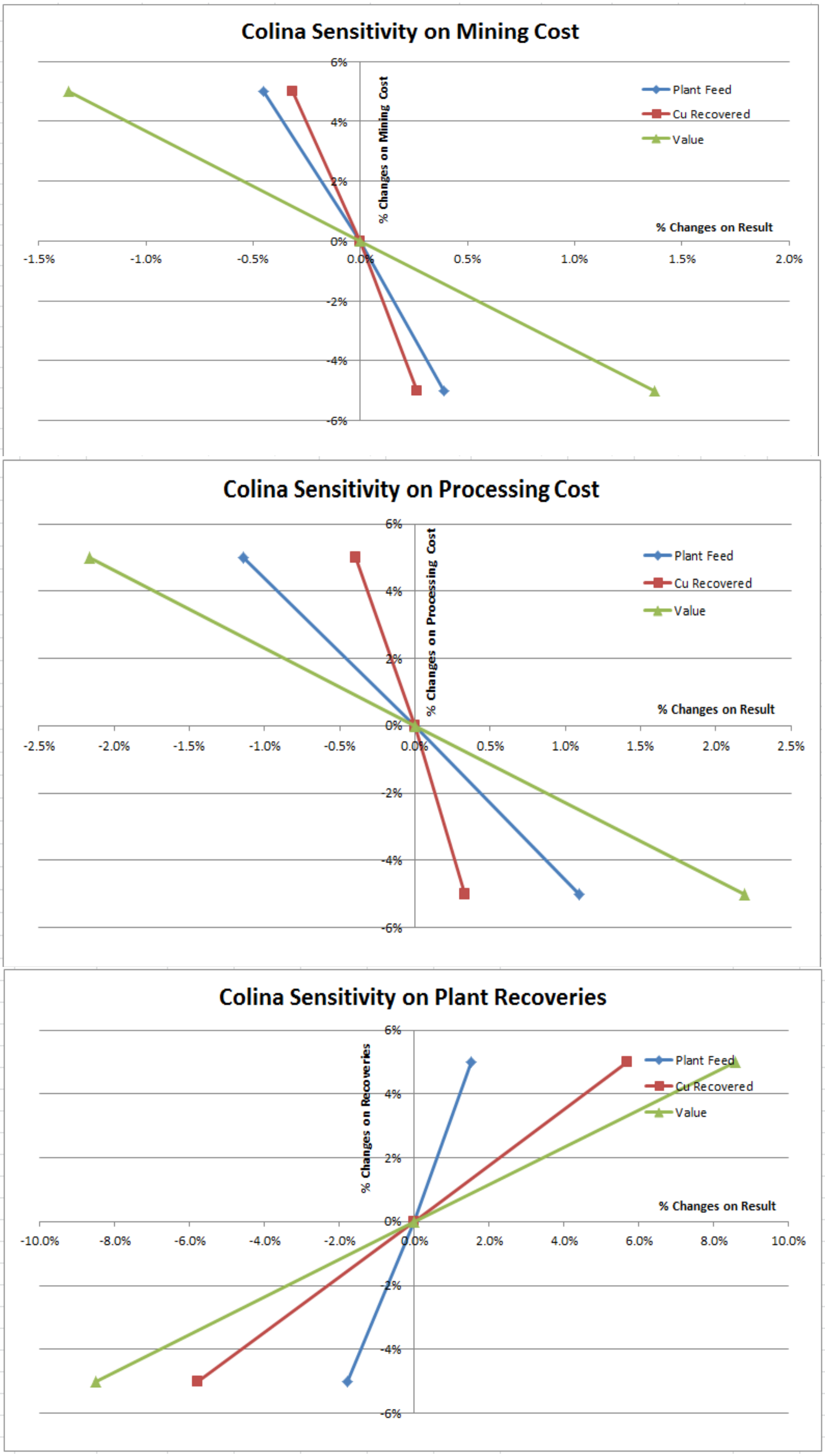
Botija

Figure 15-7 Botija optimisation sensitivity analysis



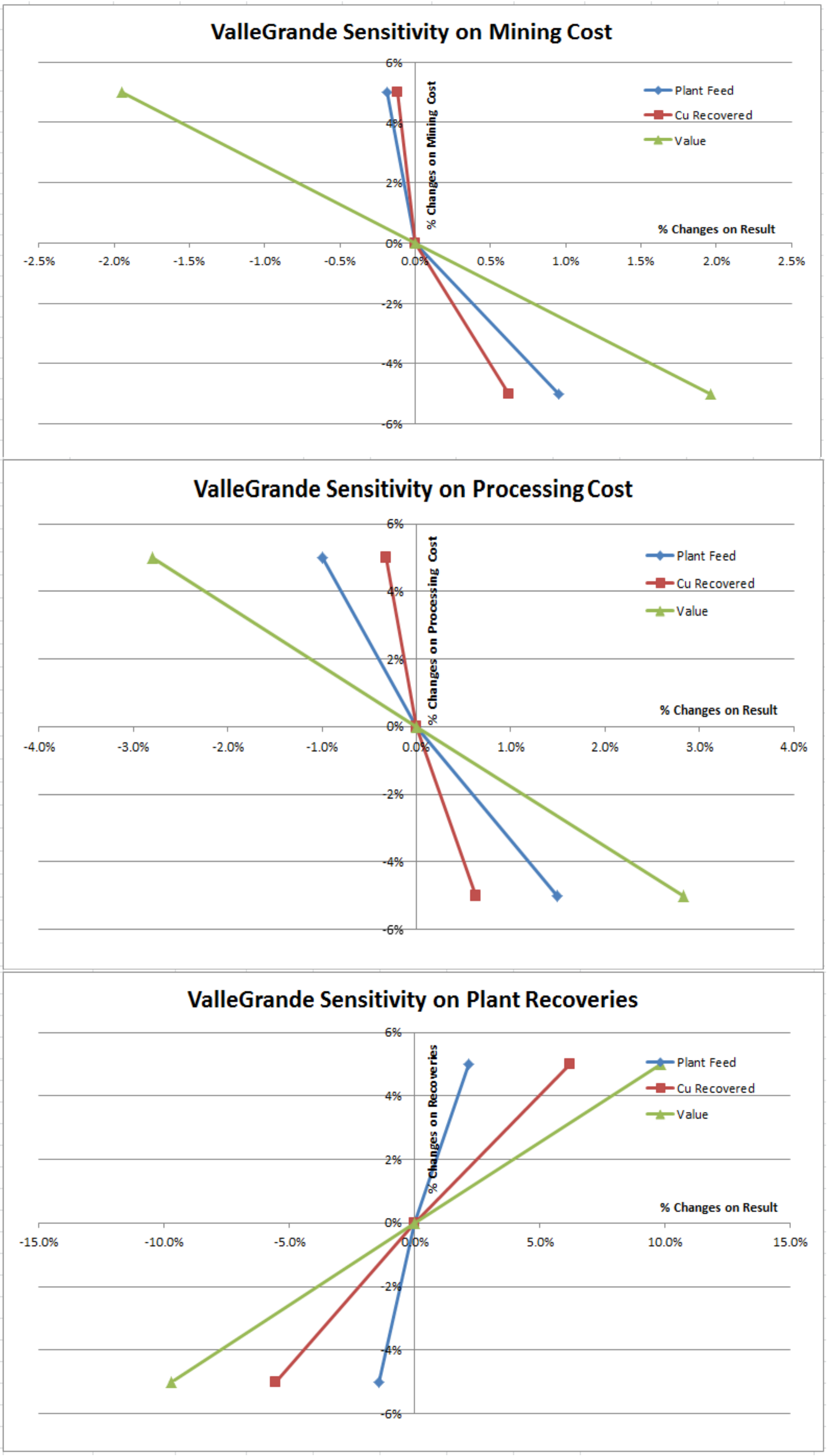
Colina

Figure 15-8 Colina optimisation sensitivity analysis



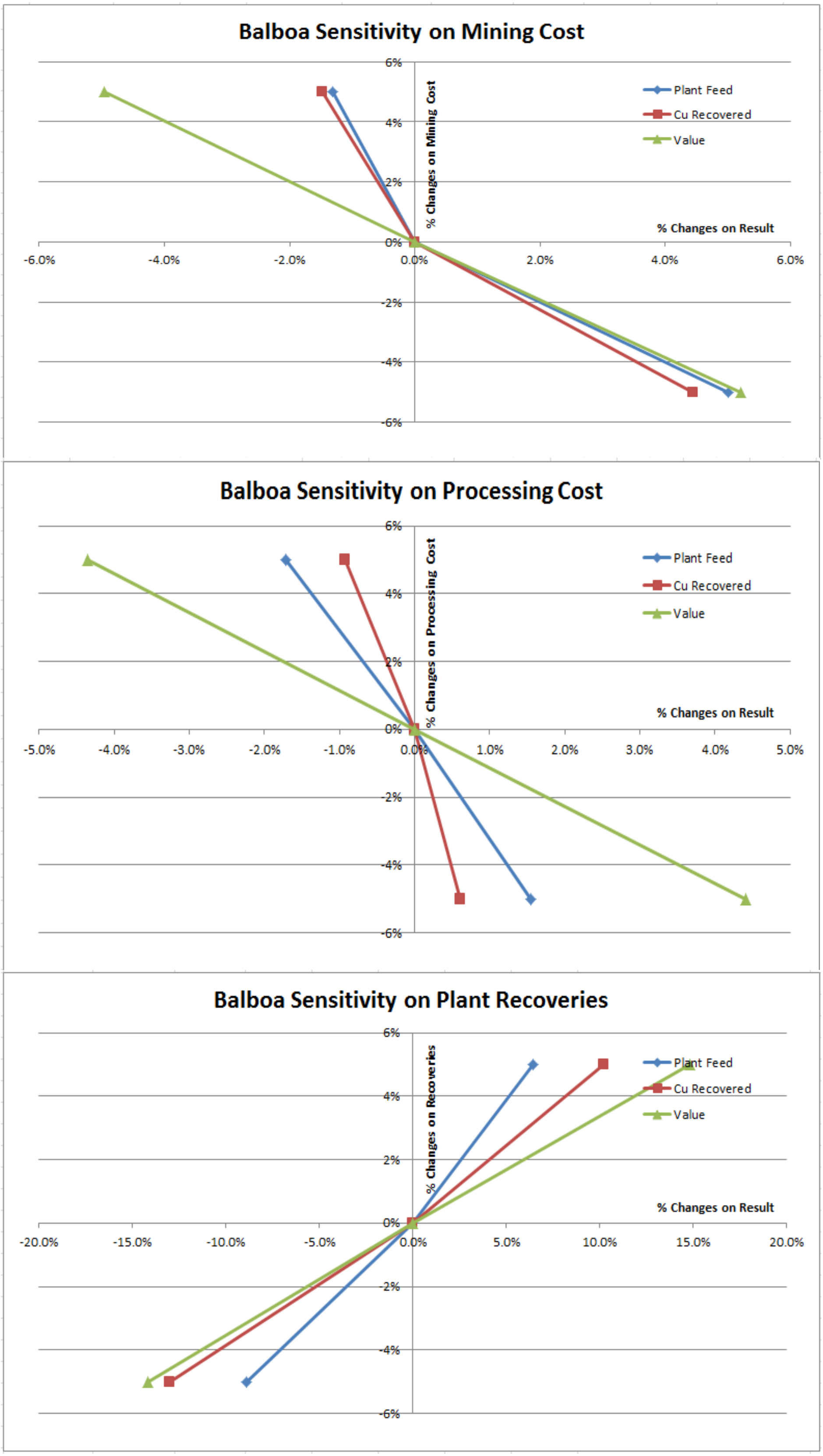
Valle Grande

Figure 15-9 Valle Grande optimisation sensitivity analysis



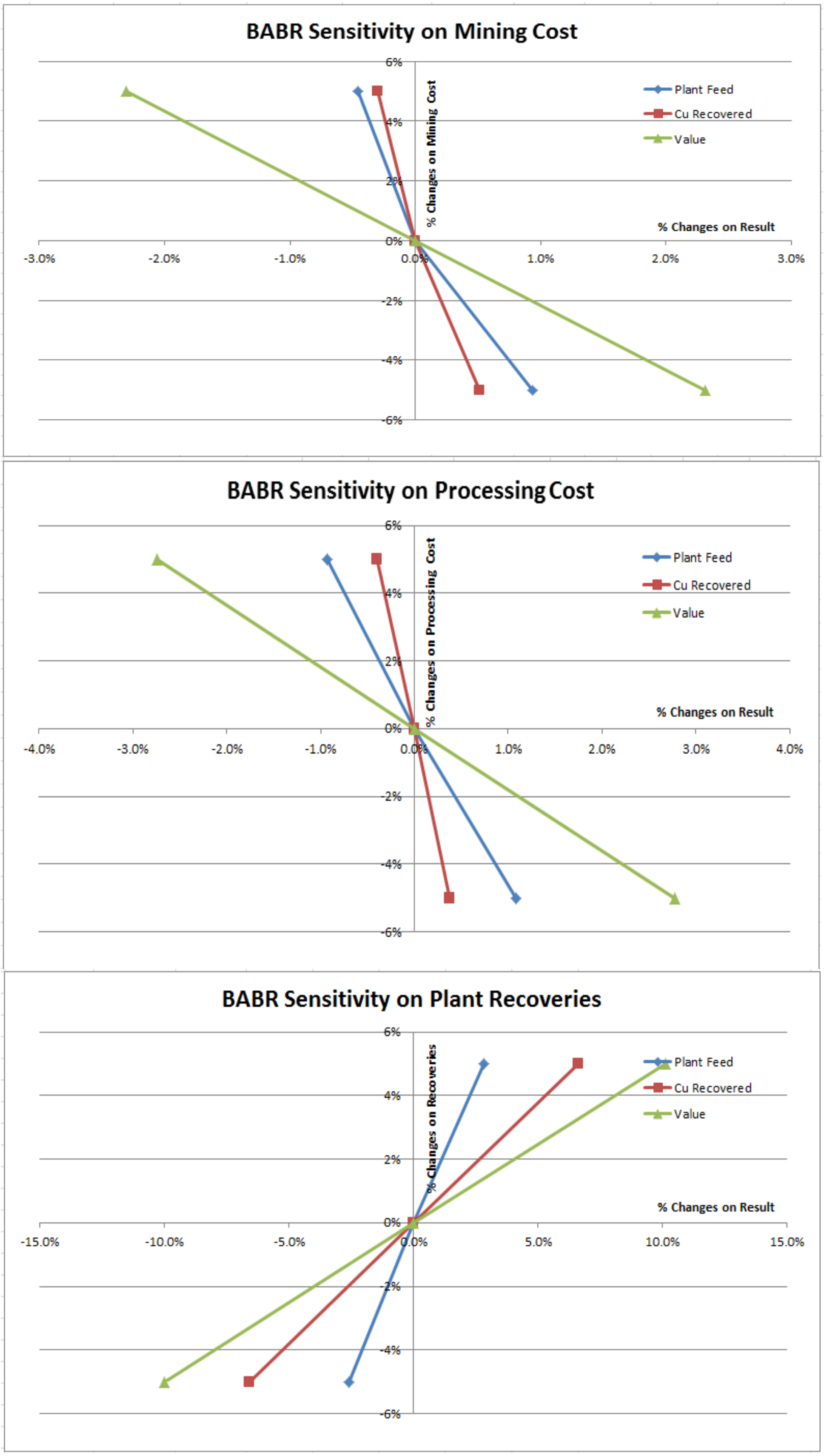
Balboa

Figure 15-10 Balboa optimisation sensitivity analysis



BABR

Figure 15-11 BABR optimisation sensitivity analysis



Sensitivity to revised mining cost estimates

More detailed mining cost estimates were developed subsequent to the optimisations described herein. Rather than being based on generic 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 2027).

Sensitivity to revised processing and metal cost estimates

In the same way that mining costs were revised subsequent to the optimisations, processing costs and metal costs were also reviewed and updated. Item 21 provides a commentary on these (lower) processing and (higher) metal cost updates, whilst Table 15-8 shows the impact on the total NR due to the metal cost updates. The overall impact is that the marginal cut-off grade is essentially unchanged.

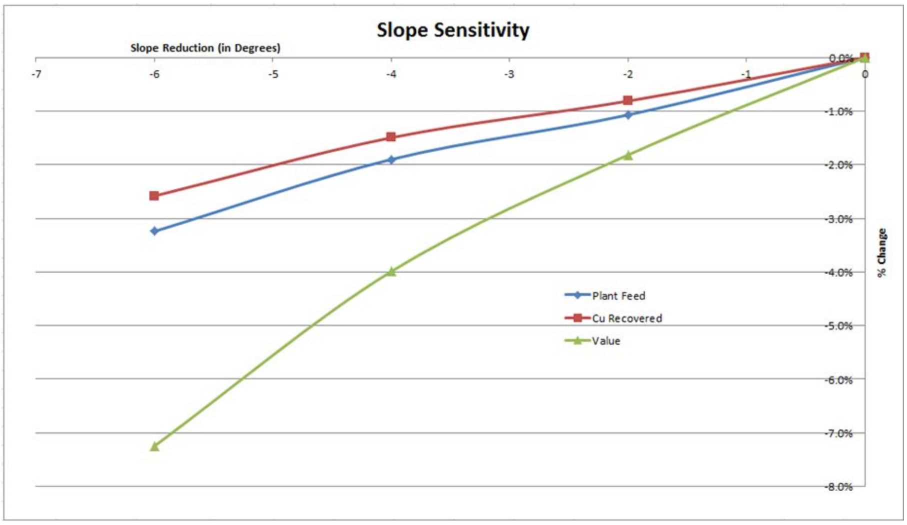
Table 15—8 Revised Cobre Panamá Net Return values

	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	2.01	2.01	2.01	2.01
Mo Net Return, $/lb	11.49	11.49	11.49	11.49
Mo Net Return, $/lb (recovered)	6.32	6.09	6.32	6.32
CuEq Net Return, $/lb (recovered)	0.07	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	64.73	64.73	64.73	64.73
Au Net Return, $/oz	1,135.27	1,135.27	1,135.27	1,135.27
Au Net Return, $/lb	16,555.64	16,555.64	16,555.64	16,555.64
Au Net Return, $/lb (recovered)	8,977.47	7,693.27	11,360.06	8,277.82
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	1.12	1.12	1.12	1.12
Ag Net Return, $/oz	14.88	14.88	14.88	14.88
Ag Net Return, $/lb	217.00	217.00	217.00	217.00
Ag Net Return, $/lb (recovered)	102.64	102.64	108.50	65.10
CuEq Net Return, $/lb (recovered)	0.02	0.01	0.01	0.00
Total Net Return, $/lb (recovered)	2.48	2.25	2.40	2.27
Total Net Return, $/10kg (recovered)	**54.74**	**49.71**	**53.01**	**50.12**

Sensitivity to pit slope design parameters

An analysis was also completed on the sensitivity to changes in overall slope angle (Figure 15-12). The analysis was done for the Botija Pit only and indicates that flattening the overall slope by around 4^{o} reduces the value of that pit by 4%.

Figure 15-12 Botija optimisation sensitivity analysis on overall slope angle



15.5 Cut-off grade strategy

An optimisation sensitivity analysis was also completed to assess the impact of elevated cut-off grades, above the marginal grades listed in Table 15-6. This was done to assess potential improvements to the plant feed grade profile in the first half of the LOM production schedule, coupled with stockpile building and reclaim strategies. The results in Table 15-9 lead into the production scheduling exercise described in Item 16.3.

Table 15—9 Cobre Panamá optimisation results at varying elevated cut-off grades

Ultimate Pits		Marginal COG	0.15% Cu COG	0.20% Cu COG	0.25% Cu COG	0.30% Cu COG
Plant Feed	Mt	3,432.8	3,105.4	2,705.6	2,195.1	1,733.6
Feed Grade	%TCu	0.35	0.38	0.41	0.45	0.50
Cu Metal Insitu	kt	12,098.0	11,715.3	11,074.1	9,949.6	8,714.1
Waste	Mt	2,793.6	2,946.6	3,346.4	3,856.9	4,318.4
Total Mined	Mt	6,226.4	6,052.0	6,052.0	6,052.0	6,052.0
Strip Ratio		0.81	0.95	1.24	1.76	2.49
Recovered Cu	kt	10,850.9	10,539.8	9,995.2	9,023.5	7,941.1
Recovered Mo	kt	105.1	99.1	91.3	78.9	66.0
Recovered Au	koz	3,929.6	3,686.0	3,510.6	3,189.5	2,850.4
Recovered Ag	koz	64,421.3	60,522.7	55,348.1	47,514.1	39,577.0
Undisc. Cashflow	**M$**	36,695.2	36,550.3	35,094.6	31,466.8	26,918.7
Ultimate Pits		**Variance from Marginal COG Optimisation**				
Plant Feed	Mt		-9.5%	-21.2%	-36.1%	-49.5%
Feed Grade	%TCu		7.0%	16.1%	28.6%	42.6%
Cu Metal Insitu	kt		-3.2%	-8.5%	-17.8%	-28.0%
Waste	Mt		5.5%	19.8%	38.1%	54.6%
Total Mined	Mt		-2.8%	-2.8%	-2.8%	-2.8%
Strip Ratio			16.6%	52.0%	115.9%	206.1%
Recovered Cu	kt		-2.9%	-7.9%	-16.8%	-26.8%
Recovered Mo	kt		-5.7%	-13.1%	-24.9%	-37.2%
Recovered Au	koz		-6.2%	-10.7%	-18.8%	-27.5%
Recovered Ag	koz		-6.1%	-14.1%	-26.2%	-38.6%
Undisc. Cashflow	**M$**		-0.4%	-4.4%	-14.2%	-26.6%

15.6 Detailed pit designs

A series of phased pit designs were developed using the ultimate and selected intermediate pit shells (Table 15-10) and according to the design and planning parameters listed below.

Table 15—10 Cobre Panamá intermediate pit shell inventories

		Botija				Colina		ValleGrande	Balboa	BABR
		Box Cut	Pushback1	Pushback2	Pushback3	Pushback1	Pushback2	Pushback1	Pushback1	Pushback1
Shell No		2	3	4	8	3	6	6	7	7
Plant Feed	Mt	118.5	370.1	603.8	838.8	51.0	541.7	243.6	155.4	120.1
Feed Grade	%Tcu	0.48	0.42	0.41	0.38	0.54	0.44	0.42	0.34	0.44
Cu Metal Insitu	kt	562.9	1,569.2	2,445.3	3,204.1	276.3	2,405.1	1,030.2	528.2	531.9
Waste	Mt	23.9	107.2	194.0	376.6	11.1	113.6	97.1	97.9	52.3
Total Mined	Mt	142.5	477.3	797.8	1,215.4	62.1	655.3	340.7	253.2	172.4
Strip Ratio		0.20	0.29	0.32	0.45	0.22	0.21	0.40	0.63	0.44
Recovered Cu	kt	519.4	1,441.7	2,236.4	2,919.6	257.5	2,206.1	902.0	478.1	471.5
Recovered Mo	kt	5.7	14.7	22.7	31.5	3.1	20.8	9.7	1.0	2.7
Recovered Au	koz	244.7	699.0	1,001.7	1,232.8	119.4	784.3	225.8	307.5	185.9
Recovered Ag	loz	2,520.0	7,363.6	11,552.8	15,159.9	1,373.2	13,086.3	5,692.9	2,682.9	1,091.7
Undisc. Cashflow	**M$**	**2,470.2**	**6,443.6**	**9,602.8**	**11,800.3**	**1,287.7**	**9,741.5**	**3,644.8**	**1,705.2**	**1,919.4**

15.6.1 Design and planning parameters

Table 15-11 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 15-1[6].

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

- benches (interval between berms) are mined to a height of 30 metres 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 15-13)
- all waste ramps = 50 m (allowing for future trolley-assisted haulage)
- maximum conveyor incline angle = 1 : 3.7 = 15°
- maximum haul ramp gradient = 1 : 10 = approximately 6°

The future trolley-assist system will require wider up-haul ramps to cater for the catenary wire towers, so have been designed at 50 m width with a 1:10 gradient.

Further, detailed design parameters relating to the dimensions for IPC pockets are provided in Item 16.

[6] Item 16.10.2 provides further information on pit slope design criteria, together with the findings from geotechnical reviews carried out after the pit optimisations and detailed pit designs had been completed.

Figure 15-13 Schematic cross section across conveyor ramp

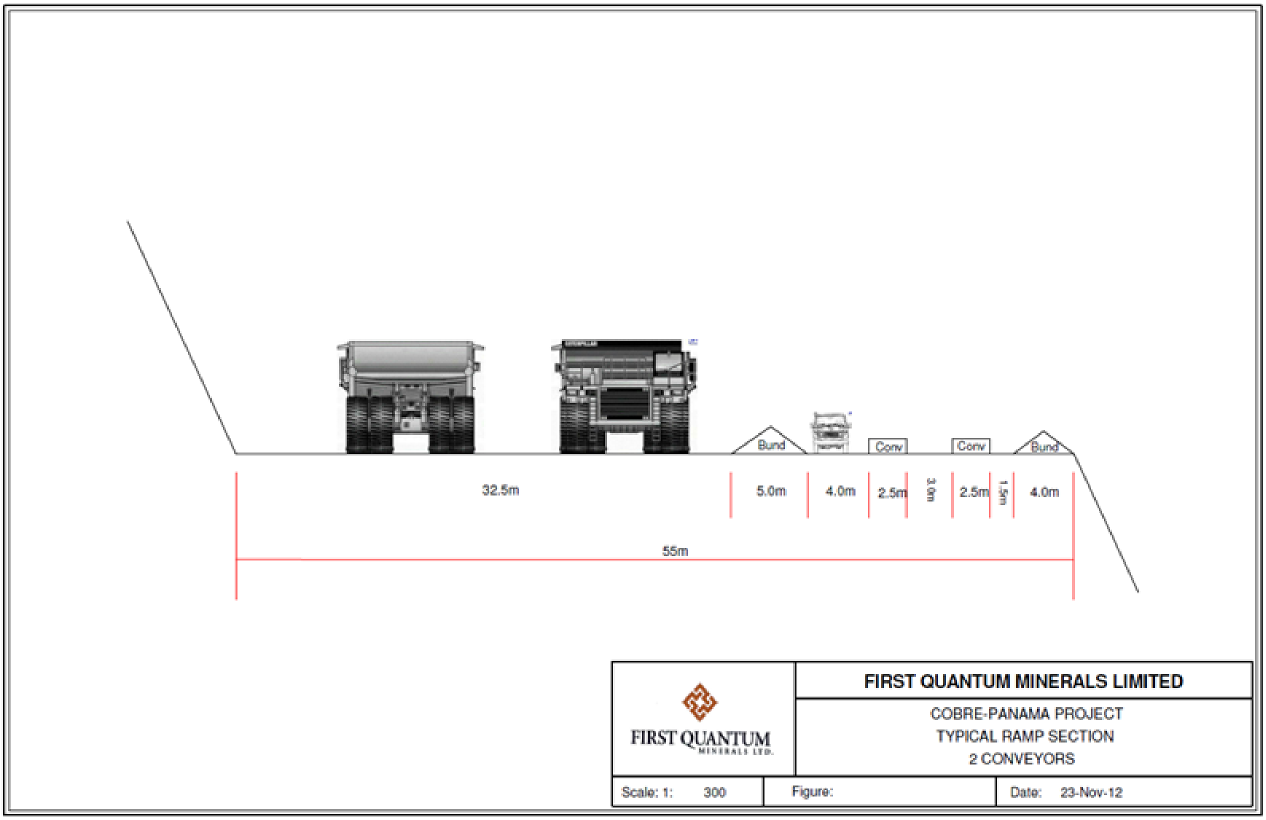


Table 15—11 Cobre Panamá pit slope design criteria

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	41.8	31.6	90	46.0	Double Benching 30 meters	70	18.1
		2	42.0	31.1	90	46.0		70	18.1
	South Wall River Diversion Diversion Sectors	3	40.3	32.5	90	44.0		69	19.5
		4	38.4	34.2	90	42.0		67	20.6
		5	40.3	31.5	90	44.0		67	18.3
		6	41.0	31.9	90	45.0		69	18.5
		7	41.0	31.6	90	45.0		68	17.9
	West Wall	8	41.1	31.7	90	45.0		69	18.5
		9	41.9	31.5	90	46.0		70	18.1
		10	42.0	31.2	90	46.0		70	18.1
		11	41.1	32.8	90	45.0		71	19.7
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	-	-	-	33.0		68	5.7
		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
Working									
37.5 degree	Bot 1, VG		-	-	-	37.5	15	55	9
40.0 degree	Bot 2, Col 1		-	-	-	40.0	15	60	9
43.0 degree	Bot 3, Col 2		-	-	-	43.0	15	65	9
45.0 degree	Bot 3, Col 3-4		-	-	-	45.0	15	65	9
42.0 degree	Balboa		-	-	-	42.0	30	68	21

* 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

15.6.2 Phased and ultimate pit designs

Figures 15-14 to 15-32 show the phased and ultimate pit designs produced from respective pit shell outlines.

Botija

Figure 15-14 Botija Pit boxcut design

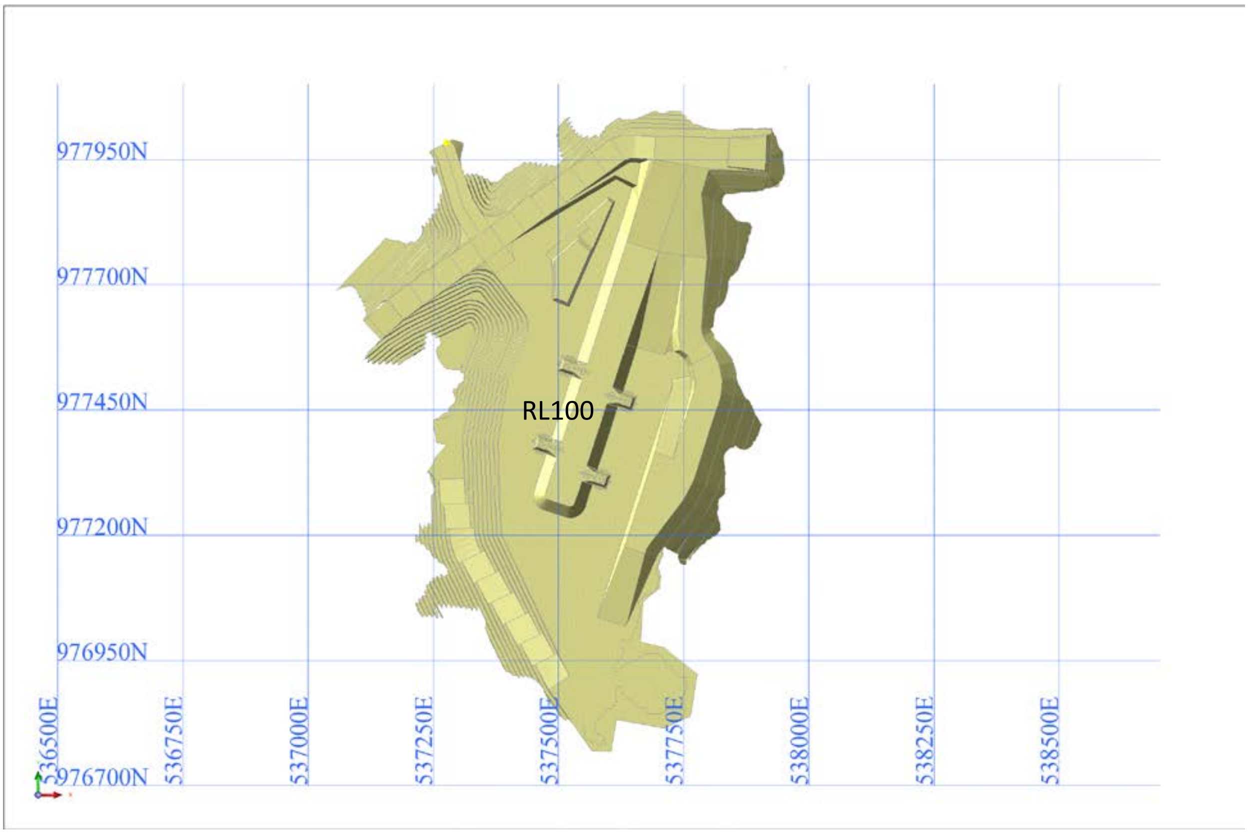


Figure 15-15 Botija Pit phase 1 design

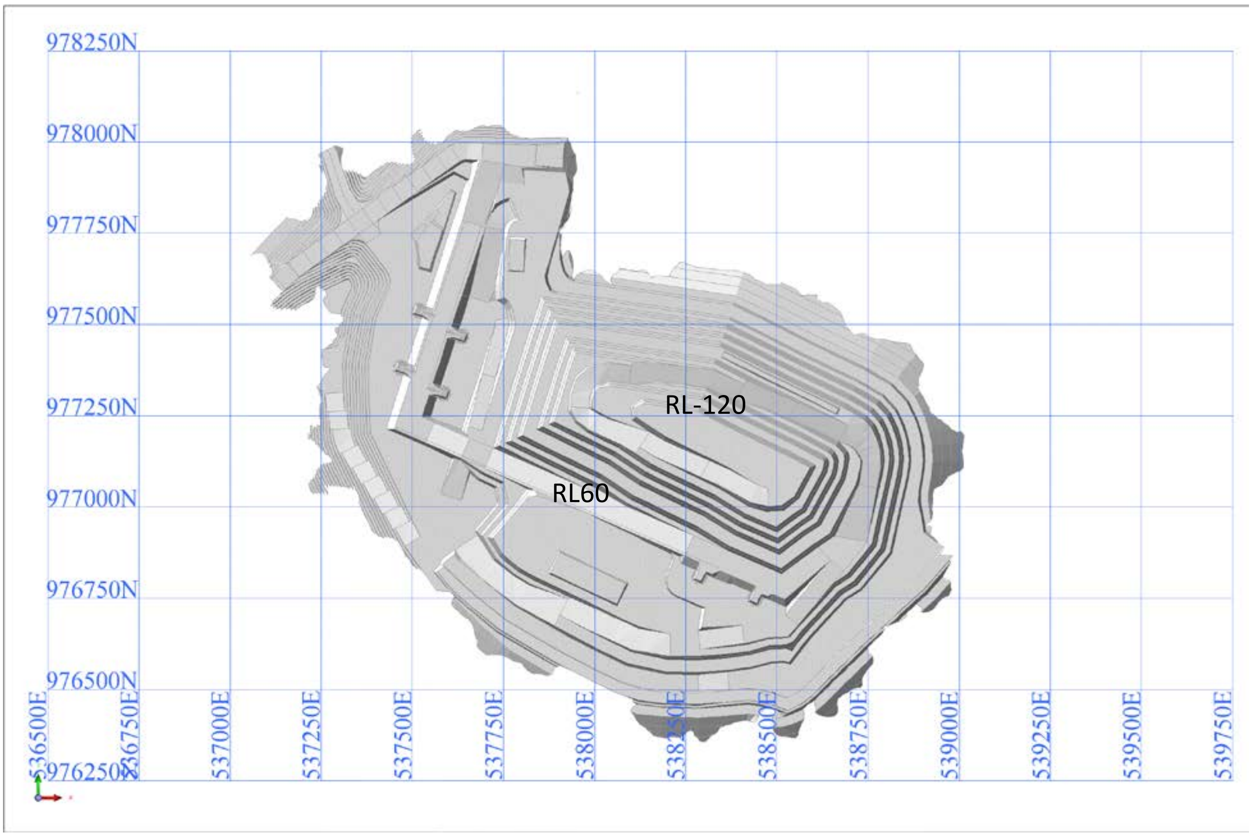


Figure 15-16 Botija Pit phase 2 design

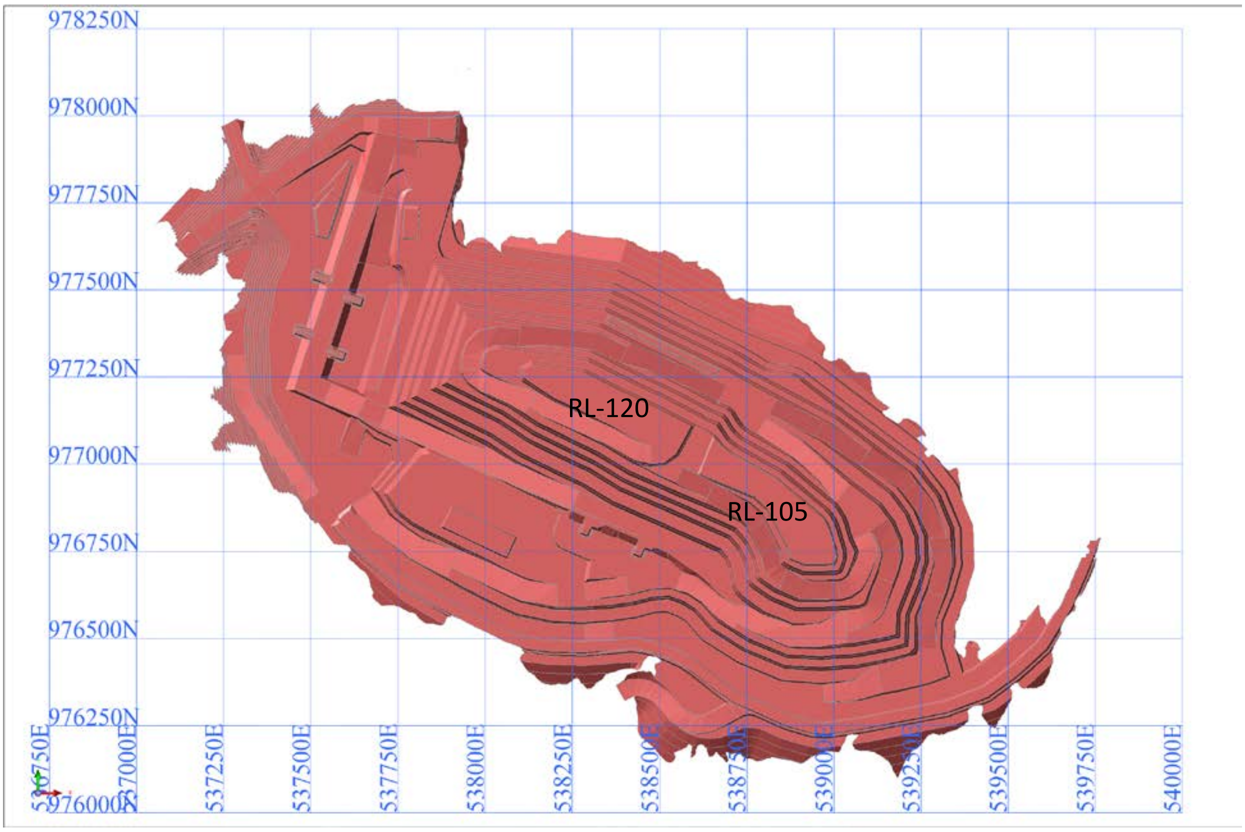


Figure 15-17 Botija Pit phase 3 design

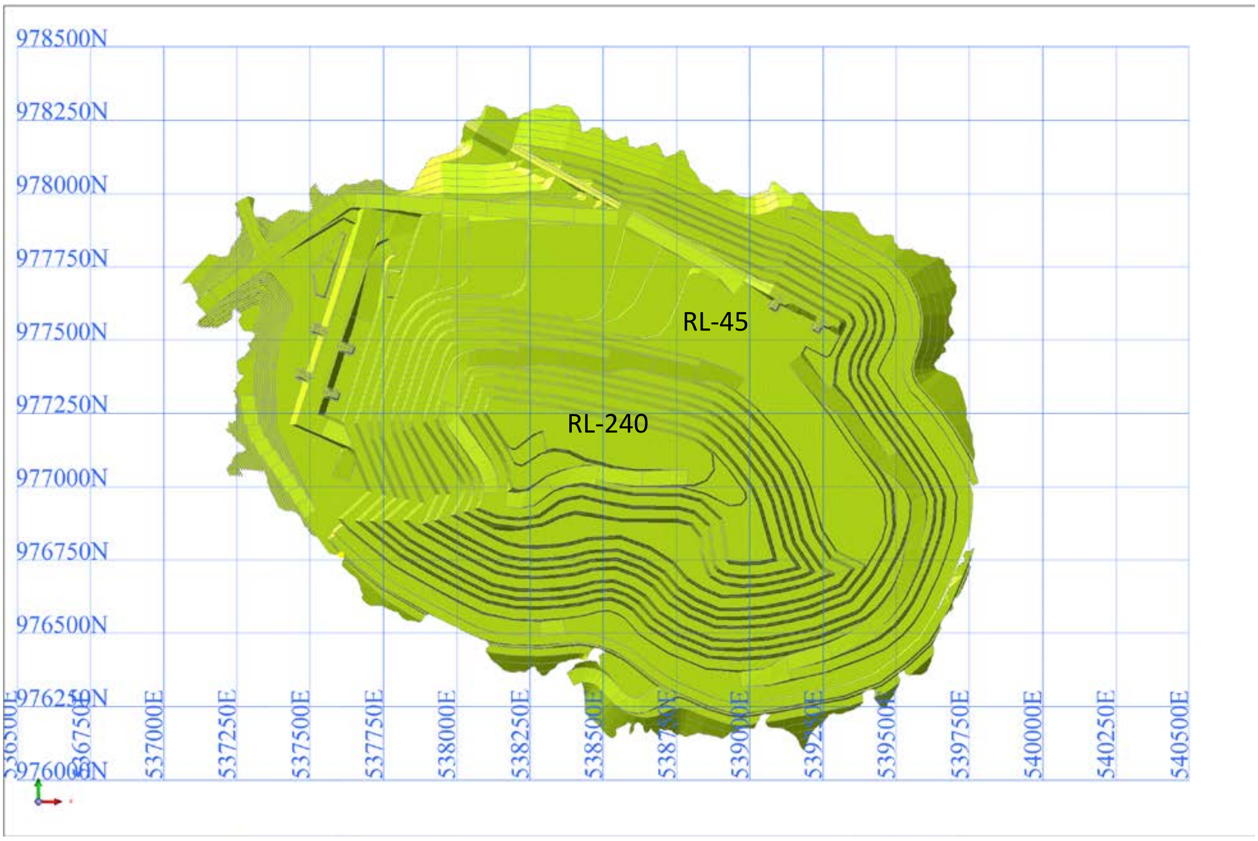


Figure 15-18 Botija ultimate pit design

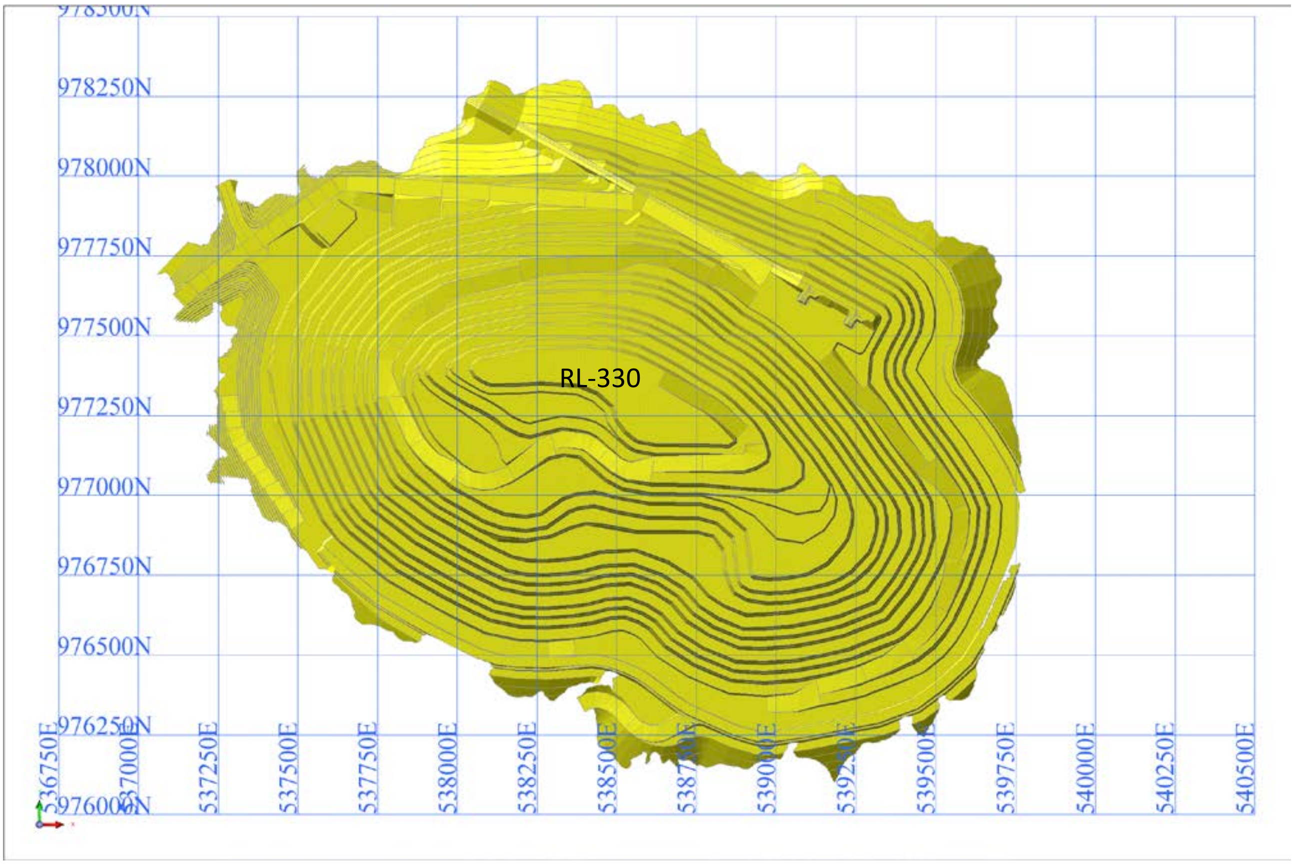
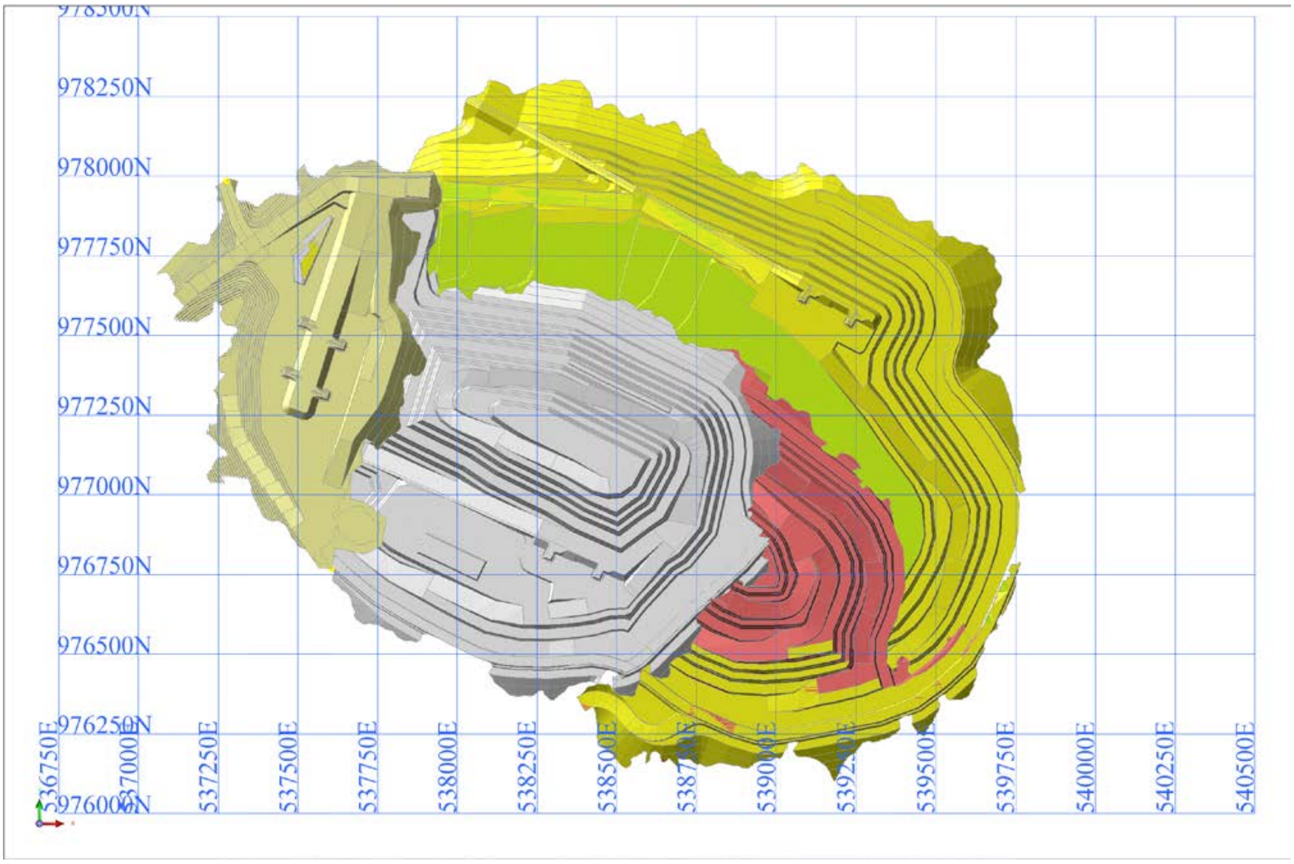


Figure 15-19 Botija Pit design showing all stages



Colina

Figure 15-20 Colina Pit phase 1 design

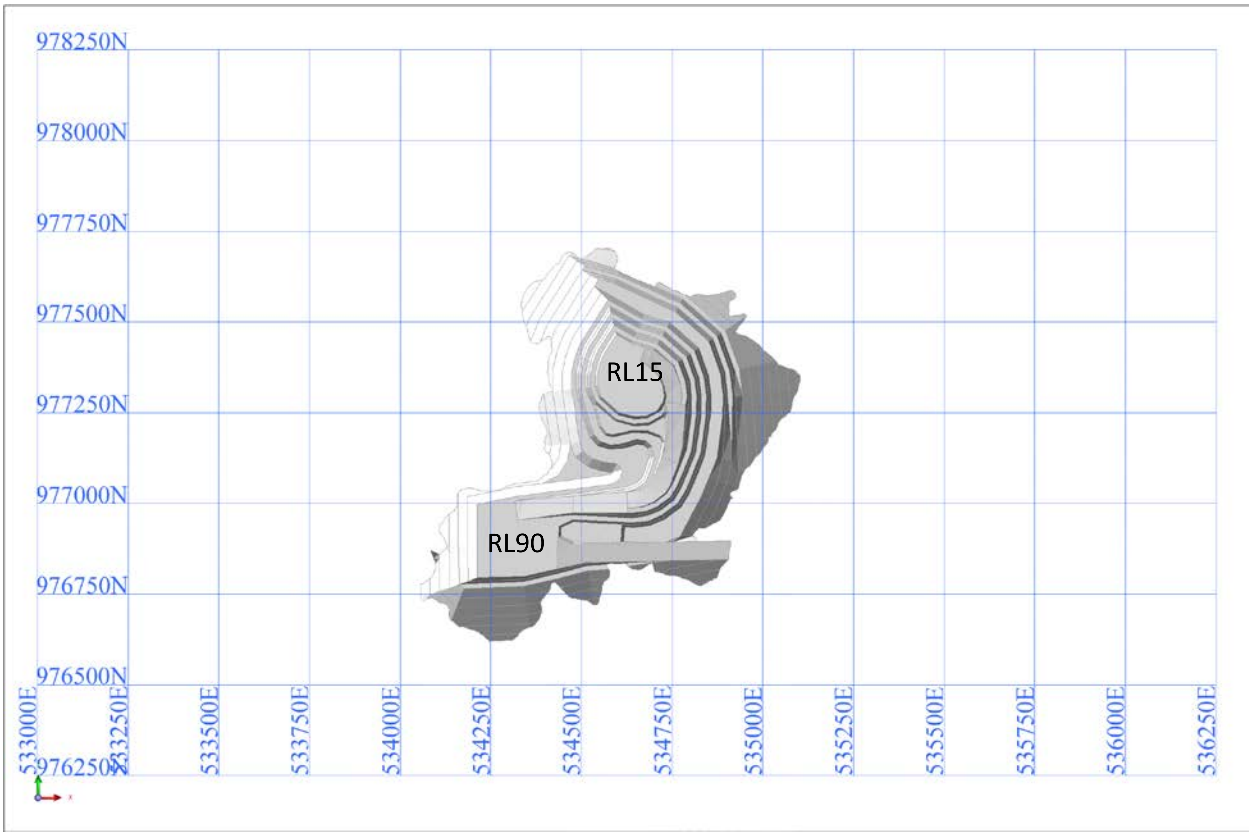


Figure 15-21 Colina Pit phase 2 design

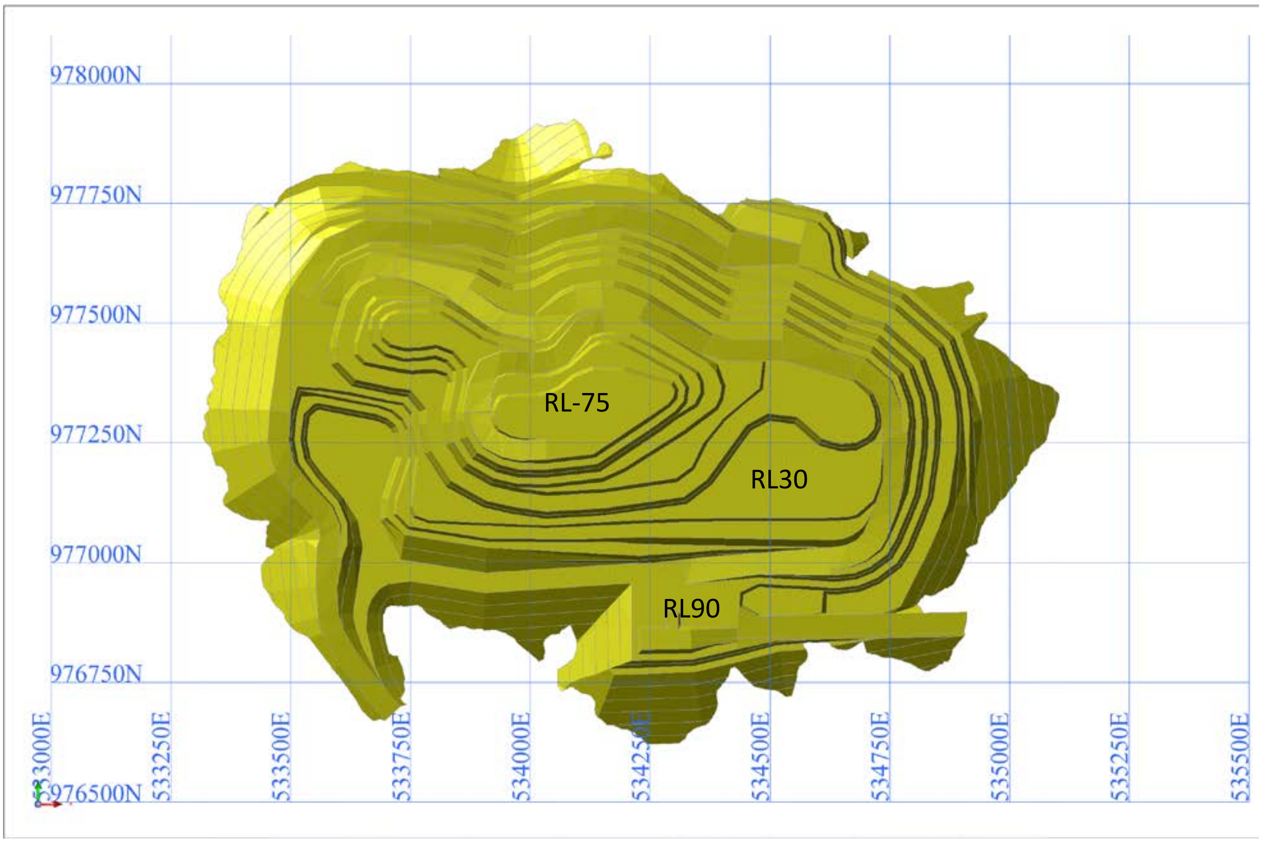


Figure 15-22 Colina ultimate pit design

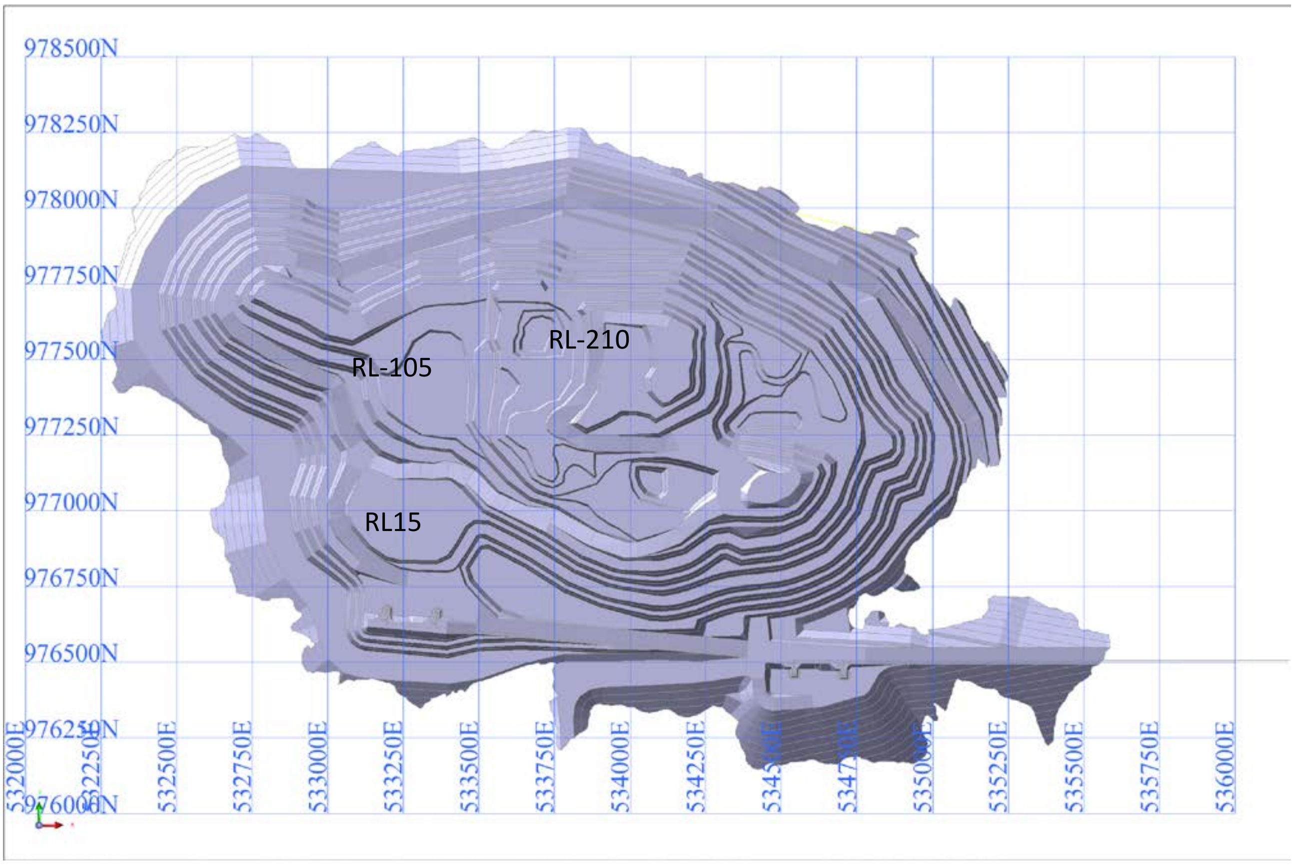
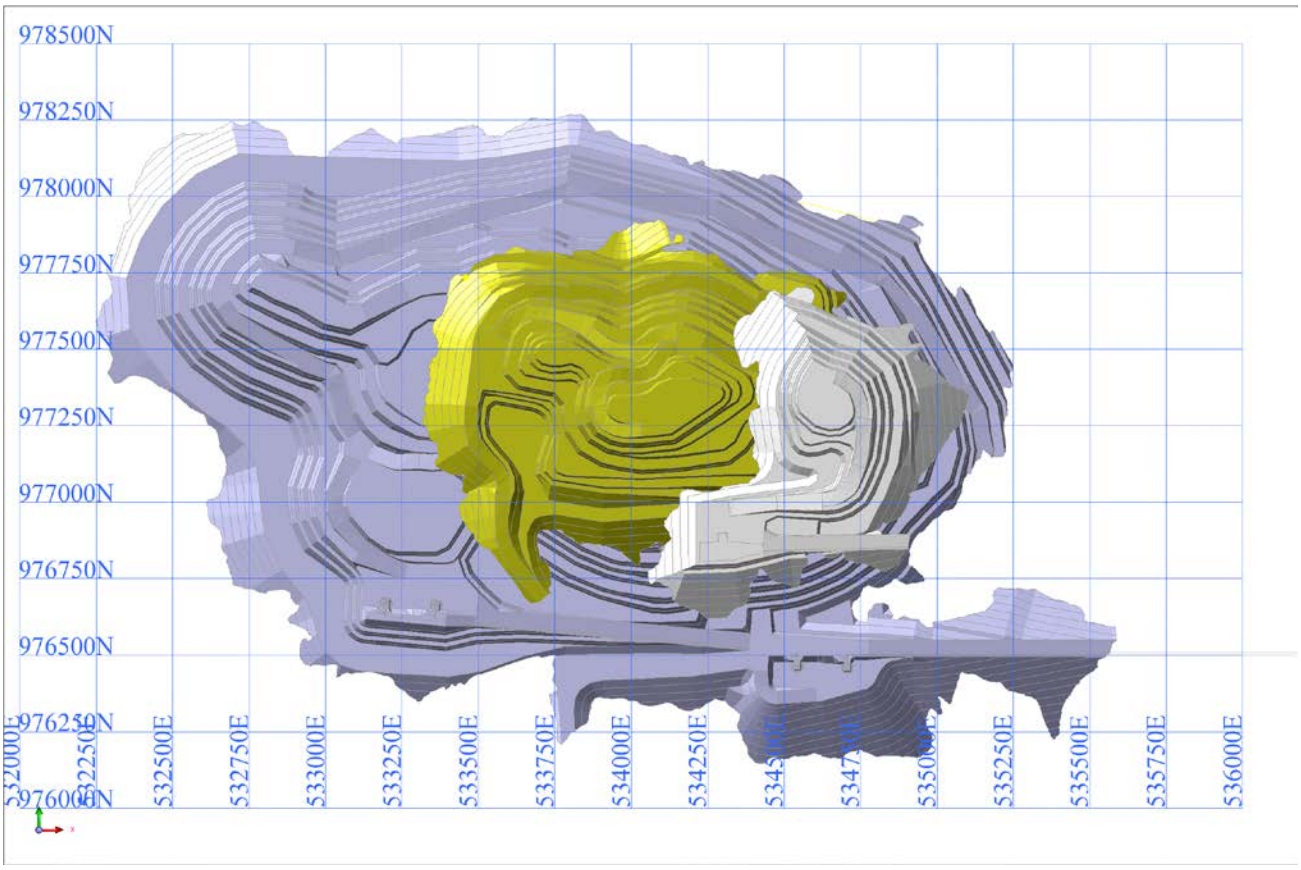


Figure 15-23 Colina Pit design showing all stages



Valle Grande

Figure 15-24 Valle Grande Pit phase 1 design

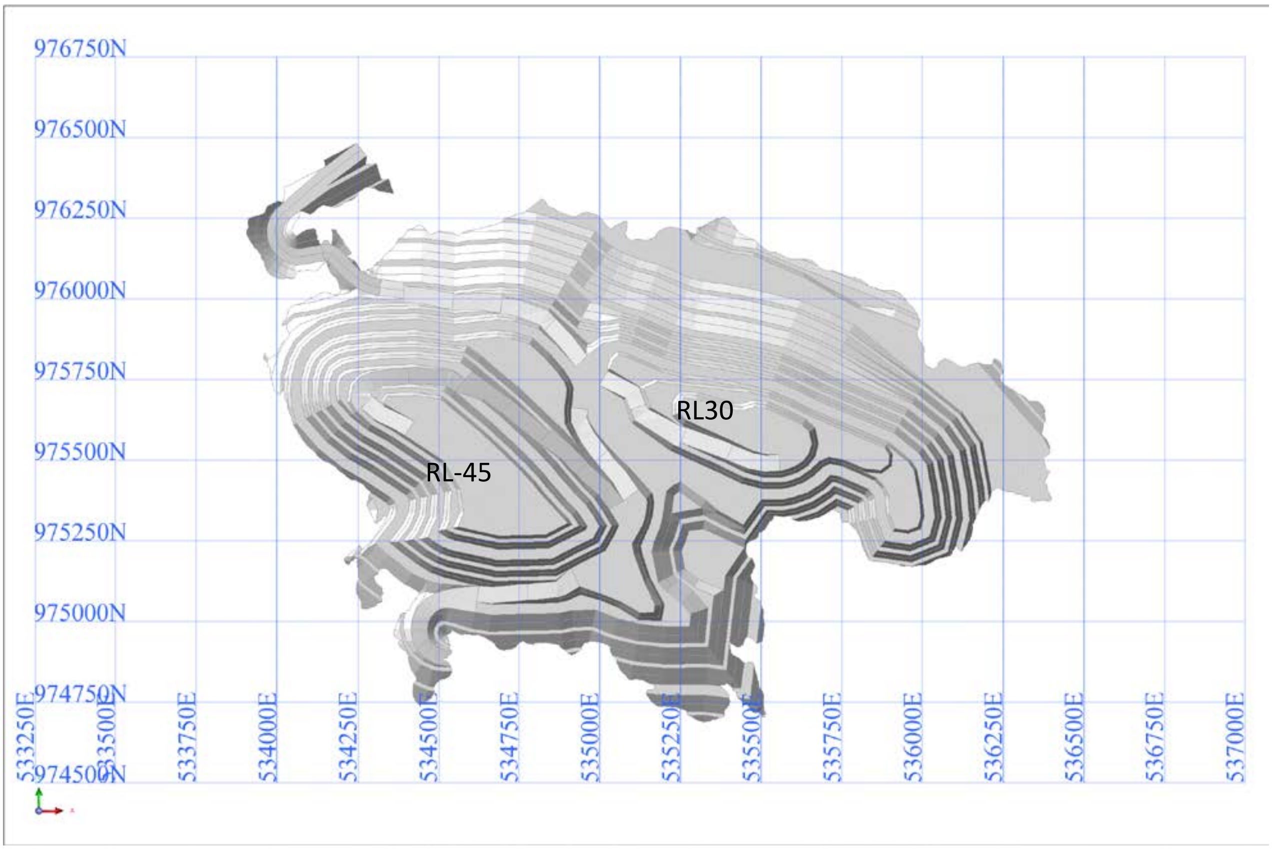


Figure 15-25 Valle Grande ultimate pit design

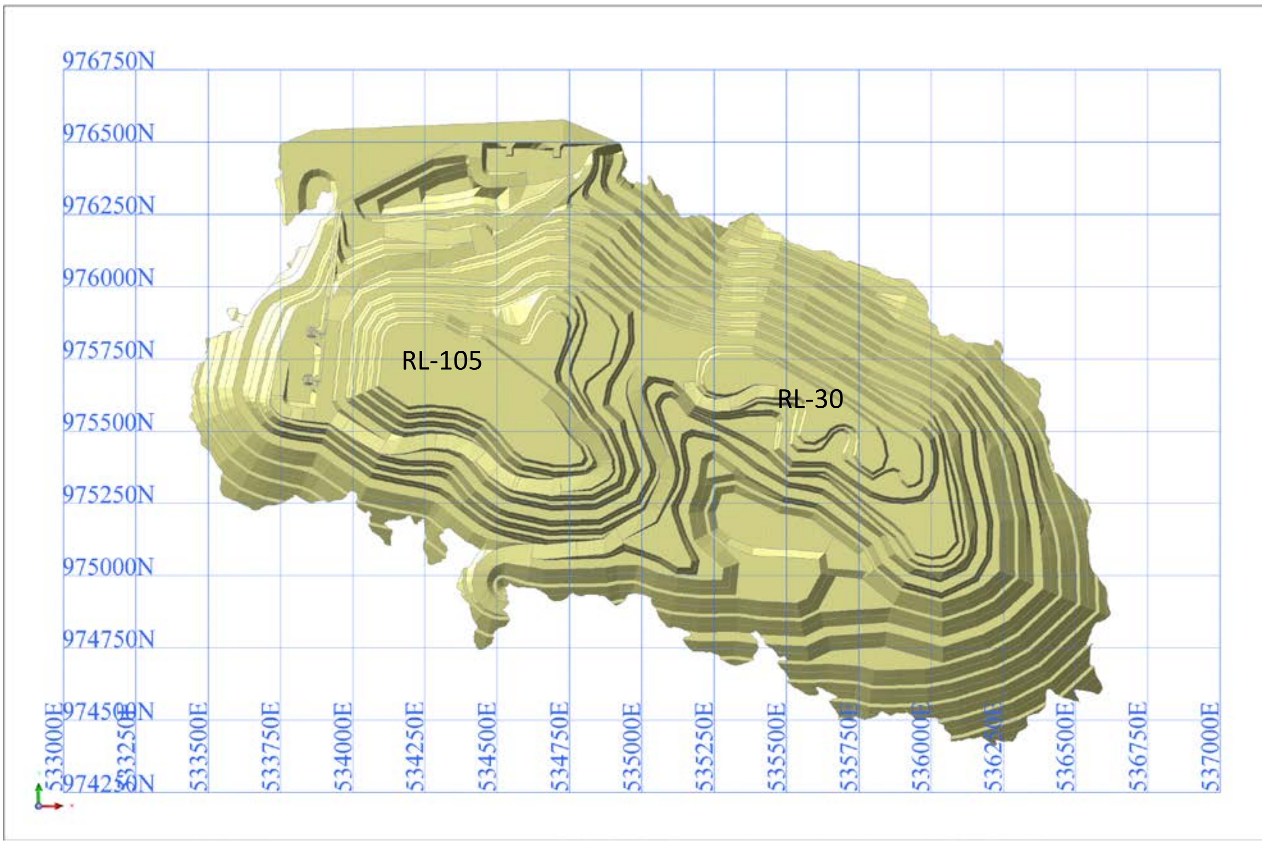
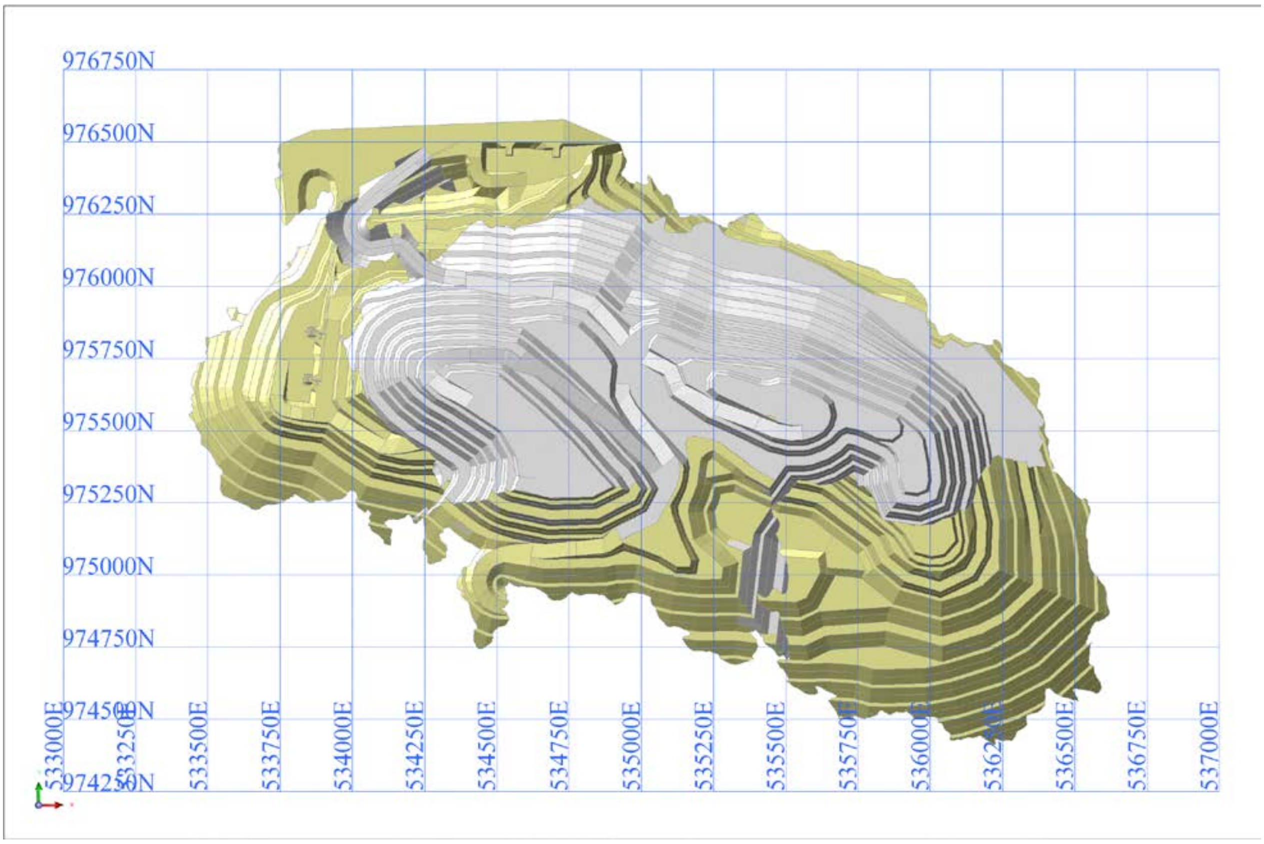


Figure 15-26 Valle Grande Pit design showing all stages



Balboa

Figure 15-27 Balboa Pit phase 1 design

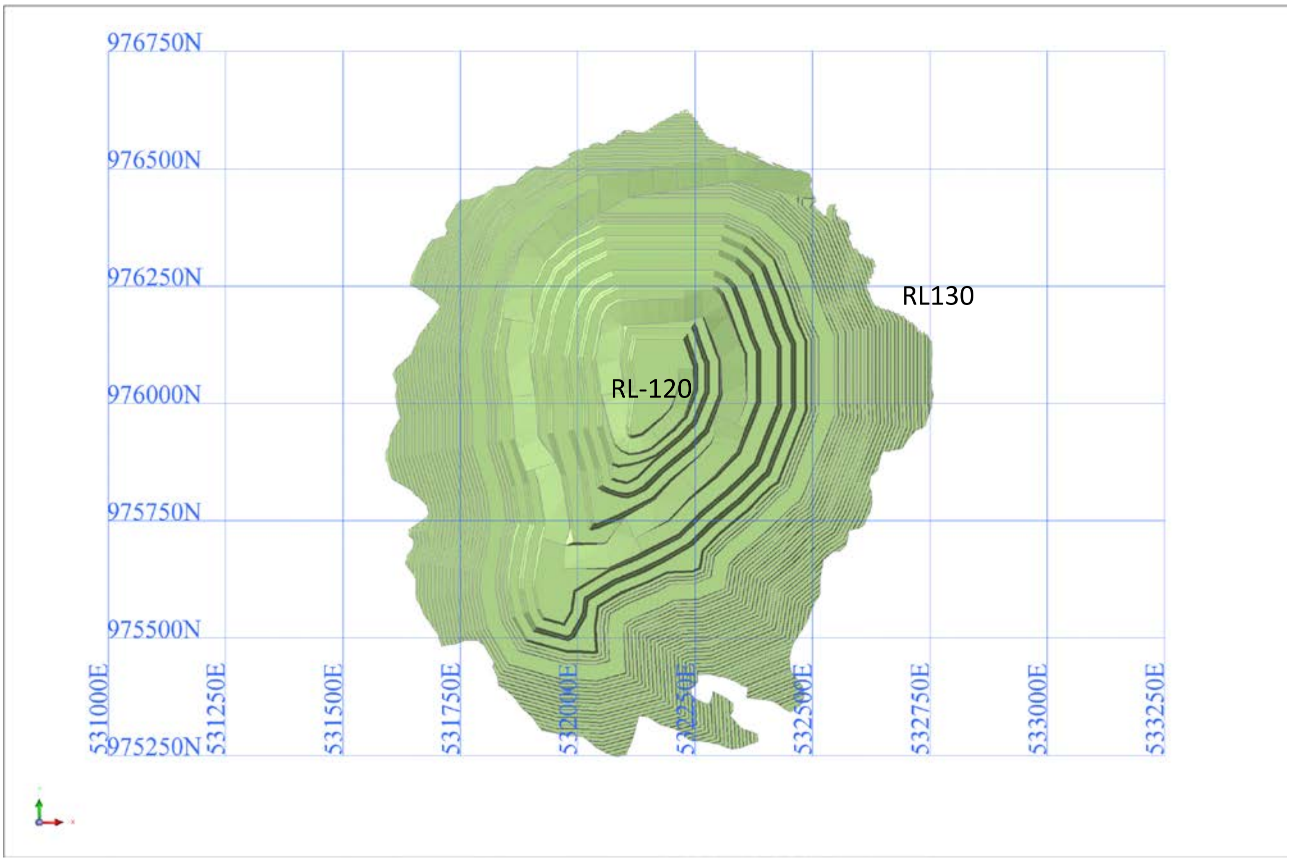


Figure 15-28 Balboa ultimate pit design

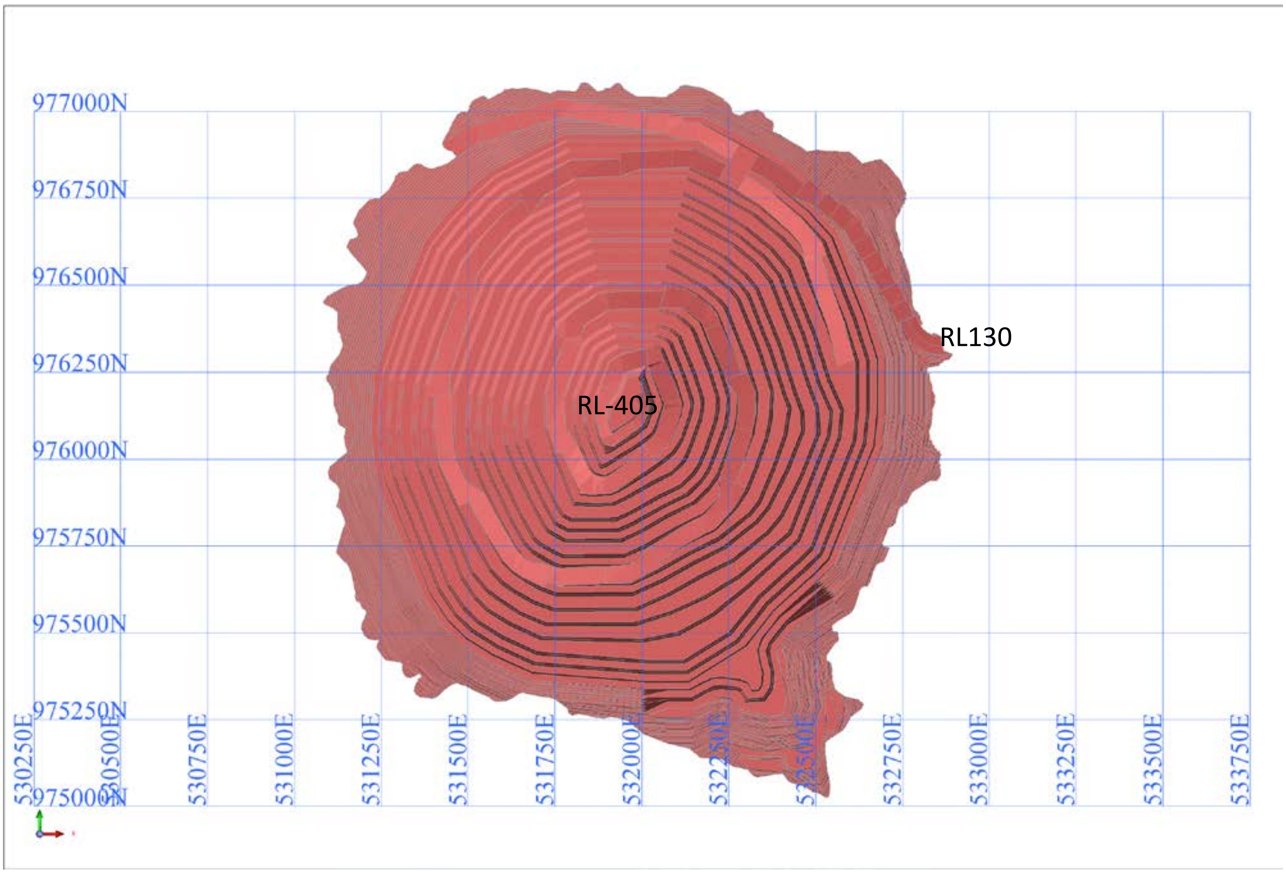
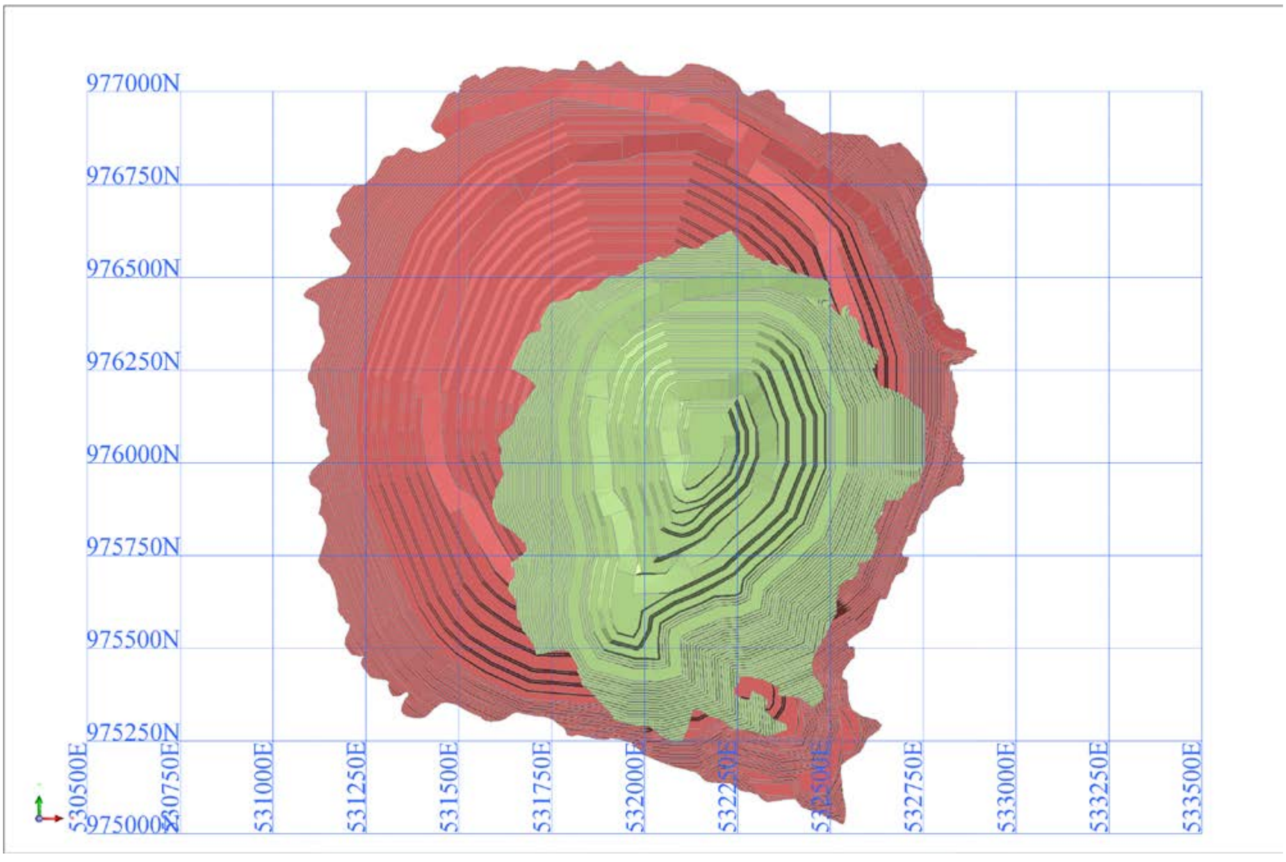


Figure 15-29 Balboa Pit design showing all stages



BABR

Figure 15-30 BABR Pit phase 1 design

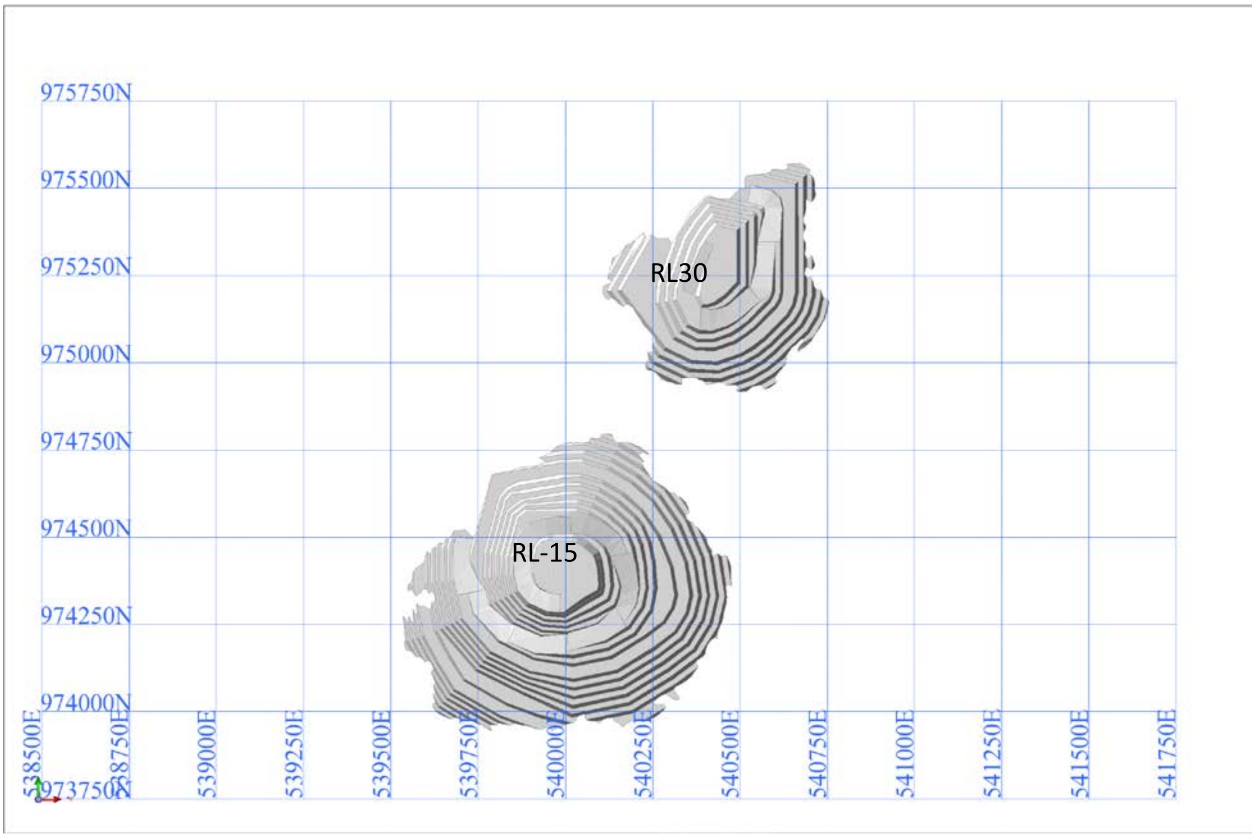


Figure 15-31 BABR ultimate pit design

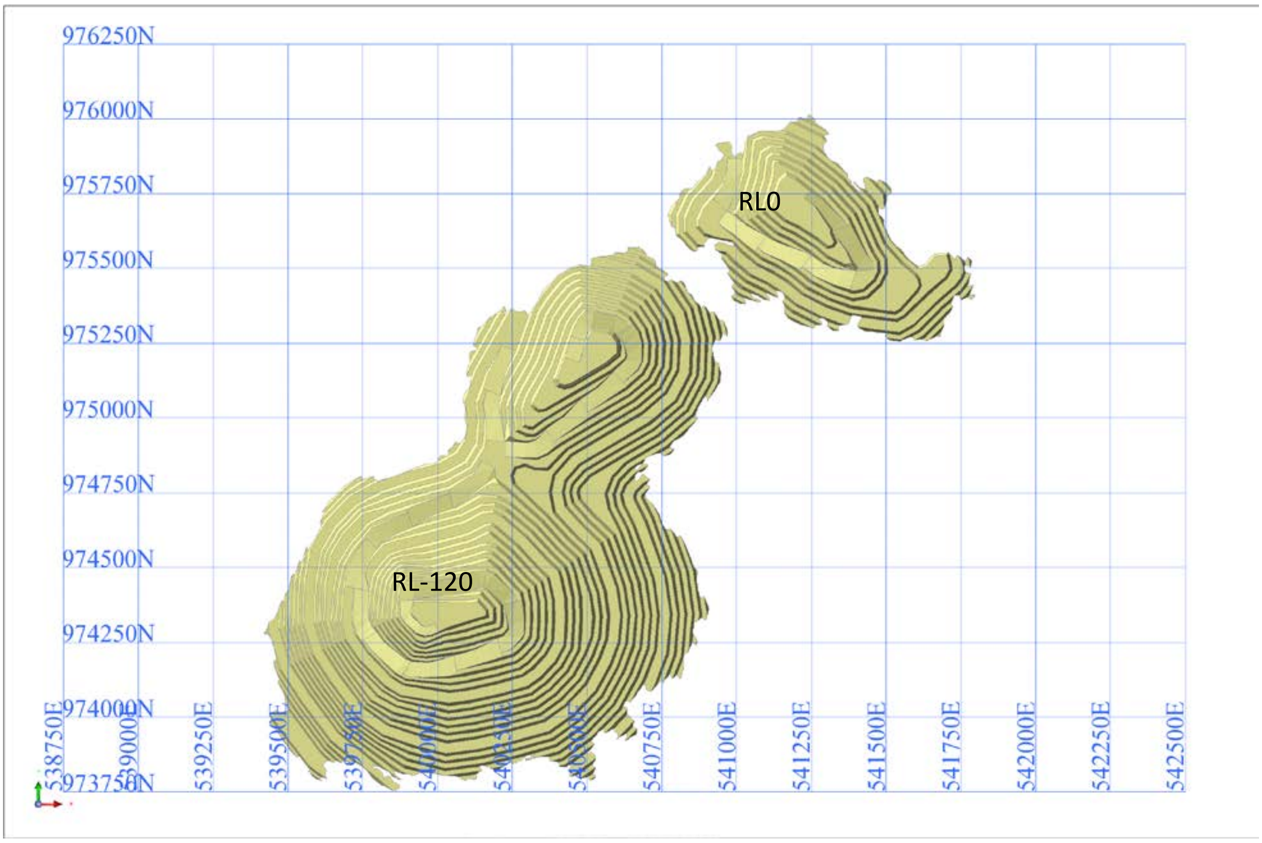
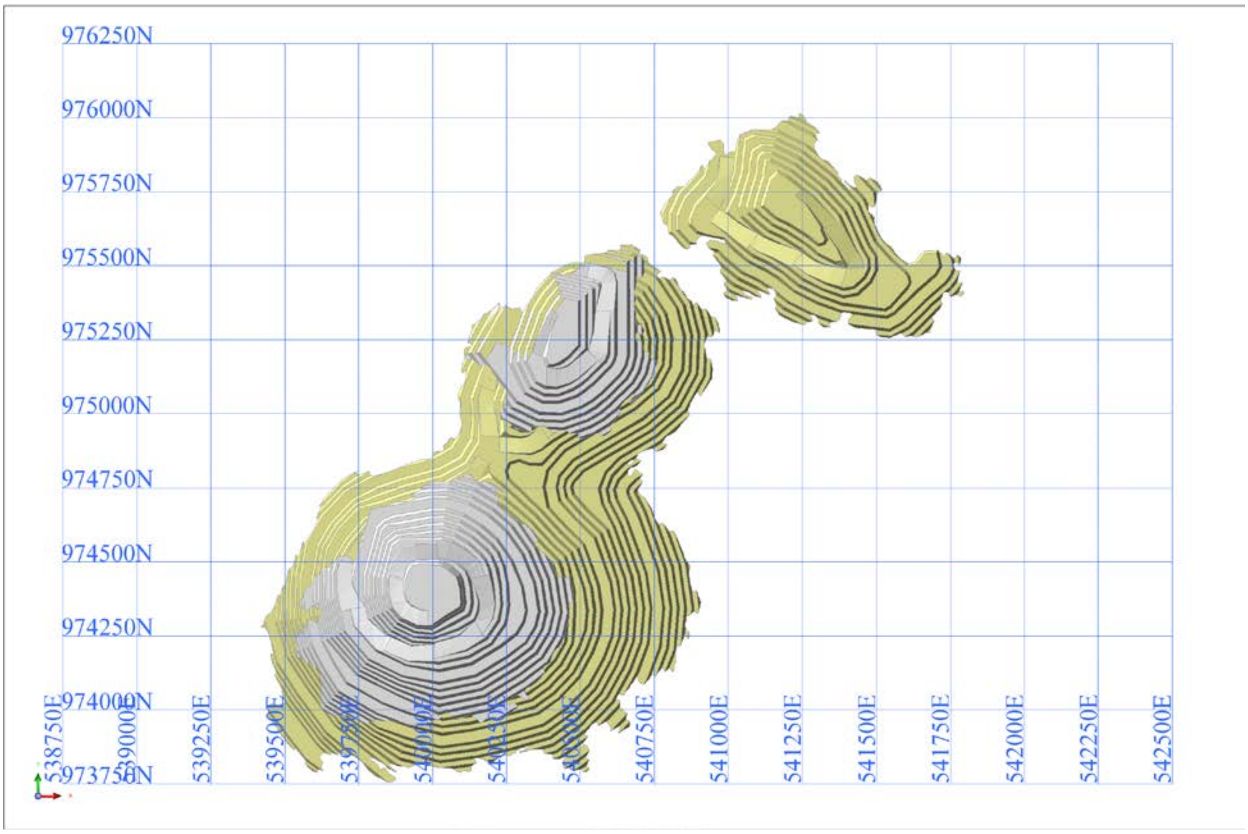


Figure 15-32 BABR Pit design showing all stages



15.6.3 Mine site layout

Figure 15-33 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

15.7 Mineral Reserve statement

The total Mineral Reserve is estimated as 3,182.5 million tonnes at 0.38% TCu. The estimate is entirely within the Measured and Indicated Mineral Resource estimate reported in Table 14-10. A breakdown by pit and classification is provided in Table 15-12.

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 phased pit designs and the mining production schedule described in Item 16.11.2.

Table 15-13 lists the recovered metal by pit and classification, accounting for unplanned mining dilution and recovery, and variable processing recovery. The processing recovery percentages are average figures determined from the production schedule.

Figure 15-33 Ultimate pit designs, waste dumps and ore conveyors

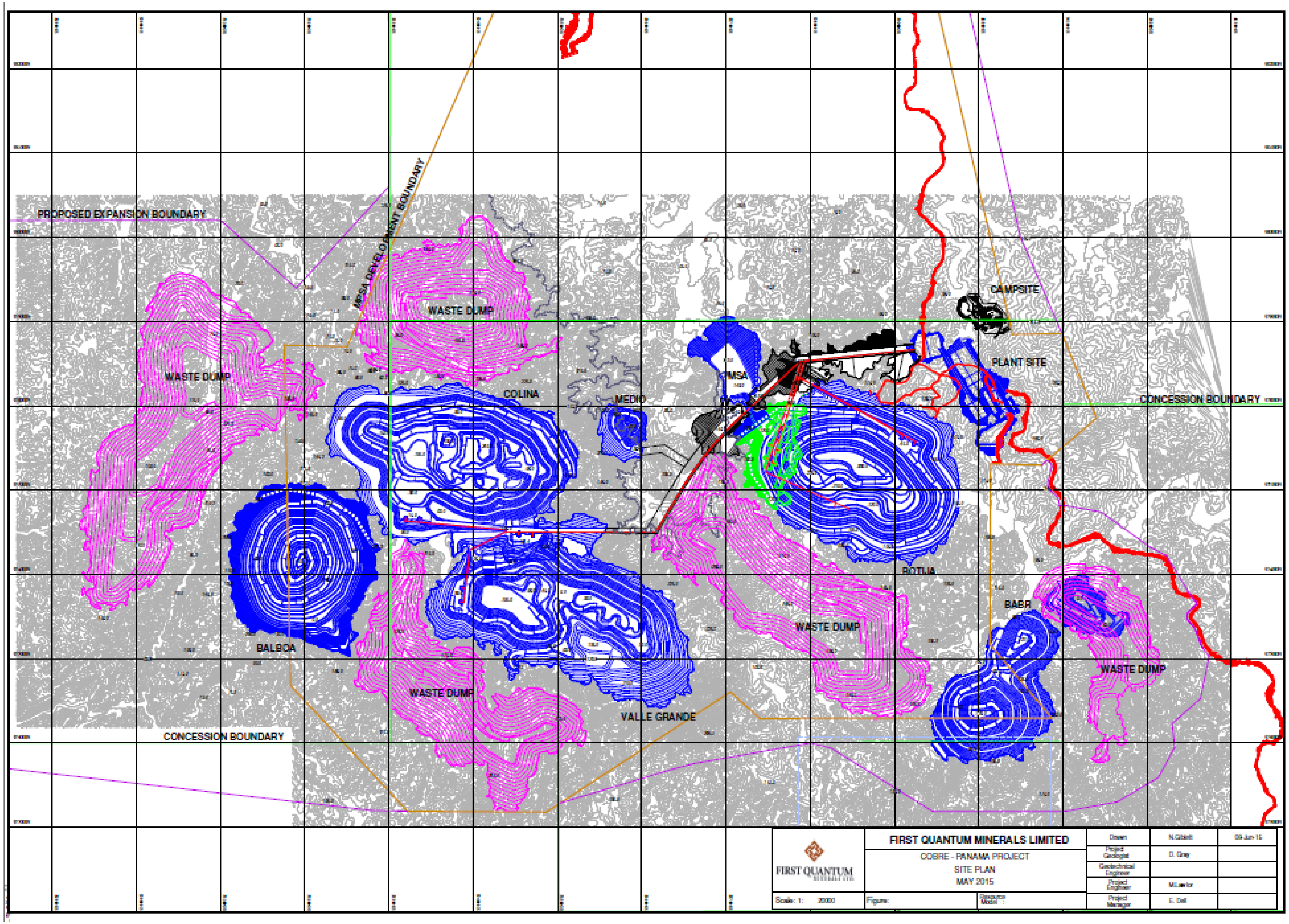


Table 15—12 Cobre Panamá Project Mineral Reserve statement, at June 2015

MINERAL RESERVE AT 30th JUNE 2015 (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	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 &	Proved									
MEDIO	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	603.5	0.35	70.79	0.07	1.10	2,124.5	42.7	1,289.8	21,377.4
	Probable	2,579.0	0.38	56.95	0.07	1.41	9,876.4	146.9	5,651.1	117,024.7
	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**

Table 15—13 Cobre Panamá Project Mineral Reserve, estimated recovered metal

MINERAL RESERVE AT 30th JUNE 2015 (Cu = \$3.00/lb, Mo=\$13.50/lb, Au=\$1,200/toz, Ag=\$16/toz)													
		After Mining Dilution & Recovery				After Processing Recovery							
Pit	Class	TCu metal (kt)	Mo metal (kt)	Au metal (koz)	Ag metal (koz)	Rec. TCu (%)	Rec. Mo (%)	Rec. Au (%)	Rec. Ag (%)	TCu metal (kt)	Mo metal (kt)	Au metal (koz)	Ag metal (koz)
BOTIJA	Proved	1,519.2	25.4	1,099.5	14,494.7	92%	55%	60%	47%	1,391.3	13.9	658.6	6,814.6
	Probable	2,082.0	41.9	1,264.0	20,949.8	90%	55%	53%	47%	1,879.3	23.0	666.8	9,875.3
	Total P+P	**3,601.2**	**67.2**	**2,363.5**	**35,444.6**	**91%**	**55%**	**56%**	**47%**	**3,270.7**	**36.8**	**1,325.5**	**16,689.9**
COLINA &	Proved												
MEDIO	Probable	3,820.8	65.6	1,994.2	50,575.7	90%	55%	52%	47%	3,450.8	35.8	1,039.6	23,718.9
	Total P+P	**3,820.8**	**65.6**	**1,994.2**	**50,575.7**	**90%**	**55%**	**52%**	**47%**	**3,450.8**	**35.8**	**1,039.6**	**23,718.9**
VALLE GRANDE	Proved												
	Probable	1,995.2	37.2	821.1	24,773.3	90%	51%	46%	47%	1,794.8	19.1	378.3	11,596.7
	Total P+P	**1,995.2**	**37.2**	**821.1**	**24,773.3**	**90%**	**51%**	**46%**	**47%**	**1,794.8**	**19.1**	**378.3**	**11,596.7**
BALBOA	Proved												
	Probable	1,478.8	6.9	1,104.4	18,784.8	90%	55%	69%	40%	1,337.8	3.8	763.8	7,513.9
	Total P+P	**1,478.8**	**6.9**	**1,104.4**	**18,784.8**	**90%**	**55%**	**69%**	**40%**	**1,337.8**	**3.8**	**763.8**	**7,513.9**
BABR	Proved												
	Probable	864.9	8.9	518.8	6,055.6	88%	55%	50%	30%	758.3	4.9	258.9	1,813.8
	Total P+P	**864.9**	**8.9**	**518.8**	**6,055.6**	**88%**	**55%**	**50%**	**30%**	**758.3**	**4.9**	**258.9**	**1,813.8**
TOTAL	Proved	2,082.0	41.9	1,264.0	20,949.8	90%	55%	53%	47%	1,879.3	23.0	666.8	9,875.3
	Probable	9,678.8	143.9	5,538.0	114,684.2	90%	54%	56%	45%	8,733.0	77.4	3,099.3	51,458.0
	Total P+P	**11,760.1**	**185.8**	**6,802.0**	**135,634.0**	**90%**	**54%**	**55%**	**45%**	**10,612.3**	**100.4**	**3,766.1**	**61,333.3**

The Mineral Reserve listed in Table 15-12 represents mining recovery of 86% and 96% of the Measured plus Indicated Mineral Resource tonnage and insitu metal estimate, respectively. In total, 18% of the combined Mining Inventory tonnage is classified as Proven with the remainder classified as Probable.

15.7.1 Mineral Reserve estimate checklist

The Proven and Probable Mineral Reserve estimate has been reported to conform with the CIM Standards on Mineral Resources and Reserves (CIM, 2005) of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM). This standard is generally consistent with reserves reported using the guidelines of the JORC Code (JORC, 2012). Appendix B sets-out the JORC Code checklist responses for estimation and reporting of an Ore Reserve, which is equivalent to a Mineral Reserve under the CIM standards.

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 will proceed in phases 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 pushed out and deepened in successive phases.

Building upon the technologies developed at other FQM operations, the Project will feature 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.

Furthermore, the haulage fleet is likely to be converted for trolley-assisted (TA) haulage. Where possible, the current designs incorporate pit ramps of sufficient width and alignment to accommodate this. In places, these ramps could be extended onto the waste dumps.

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

The primary crushing circuit will comprise five semi-mobile, independent, gyratory crushers (ThyssenKrupp KB63 x 89) 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 16.5.4), crusher productivity was determined on the basis of the parameters shown in Table 16-1.

Table 16—1 In-pit crusher capacity

Nominal hours	tph	3,600	for each crusher
Availability	%	96%	routine crusher maintenance
Available hours	tph	3,456	
Utilisation	%	75%	18 hours per day (not continuous feed)
Daily capacity	tpd	62,208	
Operational downtime	days/yr	24.0	adjacent blasting; belt maintenance etc
Annual capacity	**Mtpa**	**21.2**	

16.1.2 Trolley-assisted haulage

Trolley-assisted haulage is a concept that will be assessed during the early life of operations, although it is not expected to be implemented before the start-up of the concentrator. The primary truck haulage fleet will be “trolley-assist ready”. 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.

16.1.3 Waste dumping

The 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 36° batter angle, a 30 m batter height, 42 m width berms, and 50 m wide ramps at 1:10 gradient. The overall angle of each dump slope is approximately 20°.

16.1.4 Ore stockpiling and reclaiming

The life of mine (LOM) production schedule described in Item 16.5 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 block value and other criteria for the adopted ore stockpiles. Mineralised waste is considered as ore for crusher feed only when scheduled to be mined direct to the crushers and not rehandled from a stockpile. The lithology types are 100 = granodiorite, 200 = porphyry, 300 = andesite, 500 = saprock and 600 = saprolite.

Table 16—2 Stockpile grade ranges

		Block Value without Rehandle Cost	Block Value with Rehandle Cost	Lithology Type	Resource Class	Block NSR ($/t)
Ore						
	Saprock_ore	>0	>0	500	Meas+Ind	any
	High grade (HG)	>0	>0	>0 & <500	Meas+Ind	>=15
	Low grade (LG)	>0	>0	>0 & <500	Meas+Ind	>=10 & <15
	Ultra low grade (ULG)	>0	>0	>0 & <500	Meas+Ind	>0 & <10
Waste						
	Saprolite	any	any	600	Meas+Ind+Inf	any
	Saprock_waste	<=0	<=0	500	Meas+Ind+Inf	any
	Waste	<=0	<=0	<500	Meas+Ind+Inf	any
Other						
	Mineralised Waste	>0	<=0	>0 & <500	Meas+Ind	any

The high and low grade ore stockpiles are considered to be “active” throughout the mine life and space adjacent to the Botija Pit western crest has been set aside as an initial location for these (refer to Figure 15-28). The maximum size of these stockpiles is 10.9 Mt, reached in 2019. The maximum reclaim of 11.7 Mt from these piles occurs in 2020 when there are at least two crushers remaining near-surface in the Botija starter pit.

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

The ultra low grade and saprock ore stockpiles are developed over the life of the mine and are not reclaimed until the final ten years of operations. The location for these has not been identified explicitly although it is imagined that they would form separate and discrete volumes within the ultimate waste dumps and close to the final near-surface crusher positions.

16.1.5 Grade control

Conventional open pit grade control practices are envisaged, incorporating RC drilling and sampling on a suitably designed drilling pattern and over multiple bench horizons. Multi element sample assaying will be carried out on site. A grade control modelling process will be implemented as the basis for designing dig blocks.

16.1.6 Drilling and blasting

Near-surface saprolite material will be mined essentially as free-dig. Below this horizon, production drilling and blasting will take-place in rock conditions requiring a range of drilling/charging patterns and powder factors (Table 16-3).

Table 16—3 Drilling and blasting parameters

	Ore	Waste
Burden (m)	6.4	8.4
Spacing (m)	7.3	9.7
Bench Height (m)	15	15
Sub-Drill (m)	2	2
Blasthole Diameter (mm)	270	311
Powder Factor (kg/t)	0.40	0.28

Controlled blasting will be undertaken on final walls to prevent blast damage and to maintain wall control. In order to minimise vibration and fly-rock damage, appropriately engineered blast designs will be required in the vicinity of in-pit crushers and conveyors.

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 showed the following required excavation and installation dimensions for each semi-mobile crusher installation:

- excavated height of crusher pocket = 19.4 m
- dimensions (area) at base of crusher pocket = 18.0 m wide x 25.5 m deep
- pocket batter angle = 75°

- bin capacity above crusher = 300 m^3
- bin capacity below crusher and above transfer conveyor = 550 t
- length of draw- bridges = 12.98 m
- width of draw-bridges = 14.92 m at opening, closing to 11.8 m at tip-head
- clearance to under-side of overhead crane rails = 10.8 m

In relation to the conveyor specifications, the following parameters were considered:

- length of transfer conveyor (ie, between the crusher and discharge conveyor) = 42.5 m minimum
- nominal minimum length of discharge conveyor (ie, between transfer points) = 1,000 m
- width of discharge conveyor belt = 1.6 m belt width; 2.5 m structural width
- width of service corridor alongside discharge conveyor = 4.0 m
- transfer conveyor must be positioned at 90° to discharge conveyor

Additional IPCC design considerations include:

- Installations are to be established near-surface initially, but must proceed to deeper positions or to new locations in other pits, at (notional) relocation intervals of up to five years.
- Where possible, new crusher positions must provide for ease of relocation and logical extension of conveyor belt infrastructure.
- Conveyor belt transfer point(s) between the pit crests and the plant are fixed.
- Additional transfer points and haul road/belt crossovers are to be minimised.
- Avoid situations where there will be blasting immediately above or adjacent to crushers and conveyors. Enlarge the pit cutbacks and ensure separation.
- Where possible, design for long, straight discharge conveyor routes.
- Minimise haul ramp / conveyor belt crossovers. Minimise in-pit transfer points.

16.2.2 Mine geotechnical engineering

Botija Pit

Consultants Call & Nicholas Inc. (CNI) completed mine geotechnical evaluations for the proposed Botija Pit (CNI, March 2013) and for generic saprolite and saprock excavated slopes (CNI, April 2013).

Figure 16-1 shows the recommended slope design sectors for the Botija Pit (based on earlier optimisations and designs), whilst Table 16-4 summarises the corresponding bench configuration for this and other pits. In geotechnical terminology, the overall slope angle (OSA) is the angle between the surface crest and the ultimate pit toe, inclusive of haulroads. The interramp slope angle (ISA) is the angle between bench toe positions.

CNI pit slope stability analyses were completed assuming drained slope conditions (depressurised), on the basis that facture frequency would be great enough in mine benches to preclude pore-pressure build up (CNI, March 2013). CNI also assumed that natural depressurisation would occur due to blasting damage and mining of overburden.

CNI considers that the only design sectors which would benefit from dewatering to increase the overall slope angle, would be Sectors 1 and 3 to 7 (Figure 16-1). Bench scale design criteria dictate the recommended overall slope angles in all other design sectors.

Separate geotechnical investigations for the saprolite and saprock zones at Botija were documented by CNI in April 2013. The recommended pit slope design parameters for these zones are also summarised in Table 16-4.

Colina Pit

Consultants AMEC Earth & Environmental (AMEC) completed mine geotechnical evaluations for the proposed Colina Pit (AMEC, February 2010). Figure 16-2 shows the recommended slope design sectors for the Colina Pit (based on earlier optimisations and designs), whilst Table 16-4 summarises the corresponding bench configuration for the pit.

Slope design parameters for the pits other than Colina were extrapolated from the Botija parameters (Table 16-4).

Figure 16-1 Botija Pit slope design sectors (CNI, March 2013)

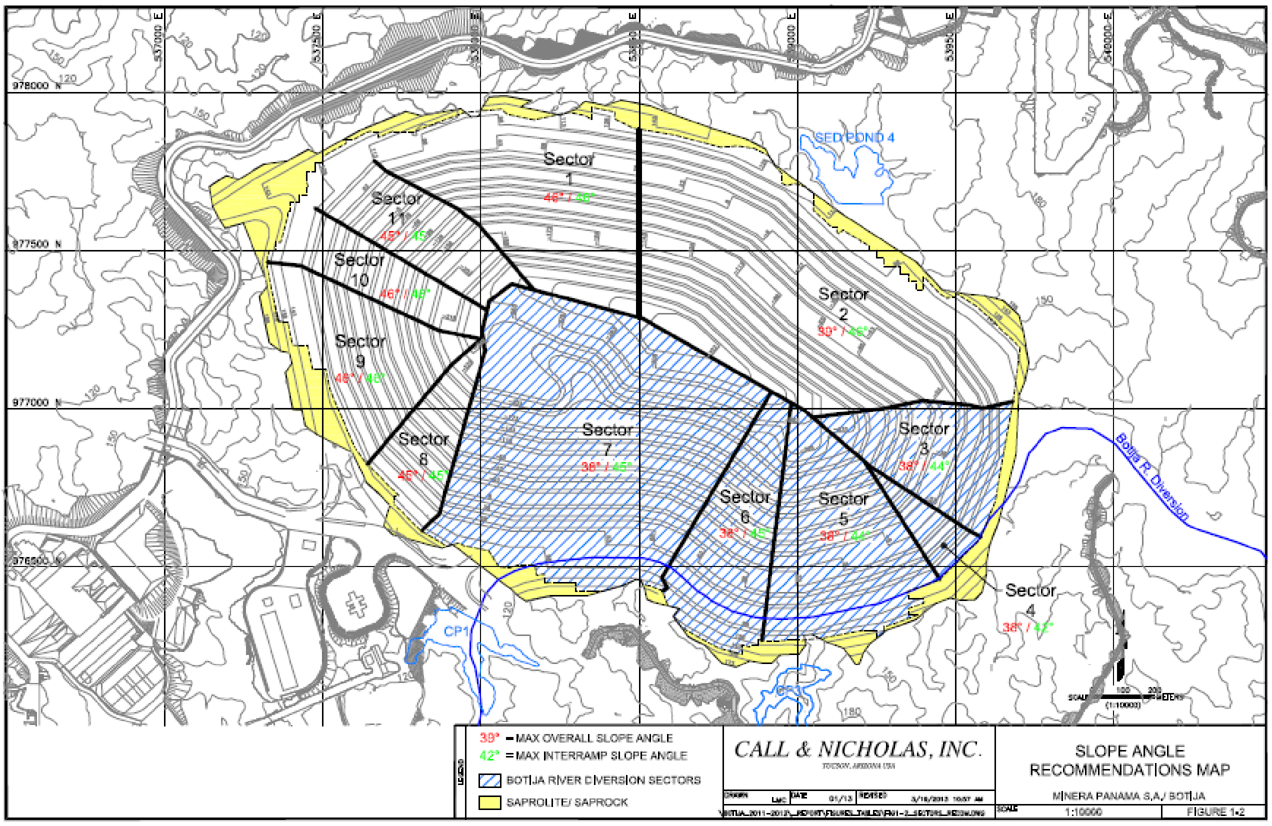
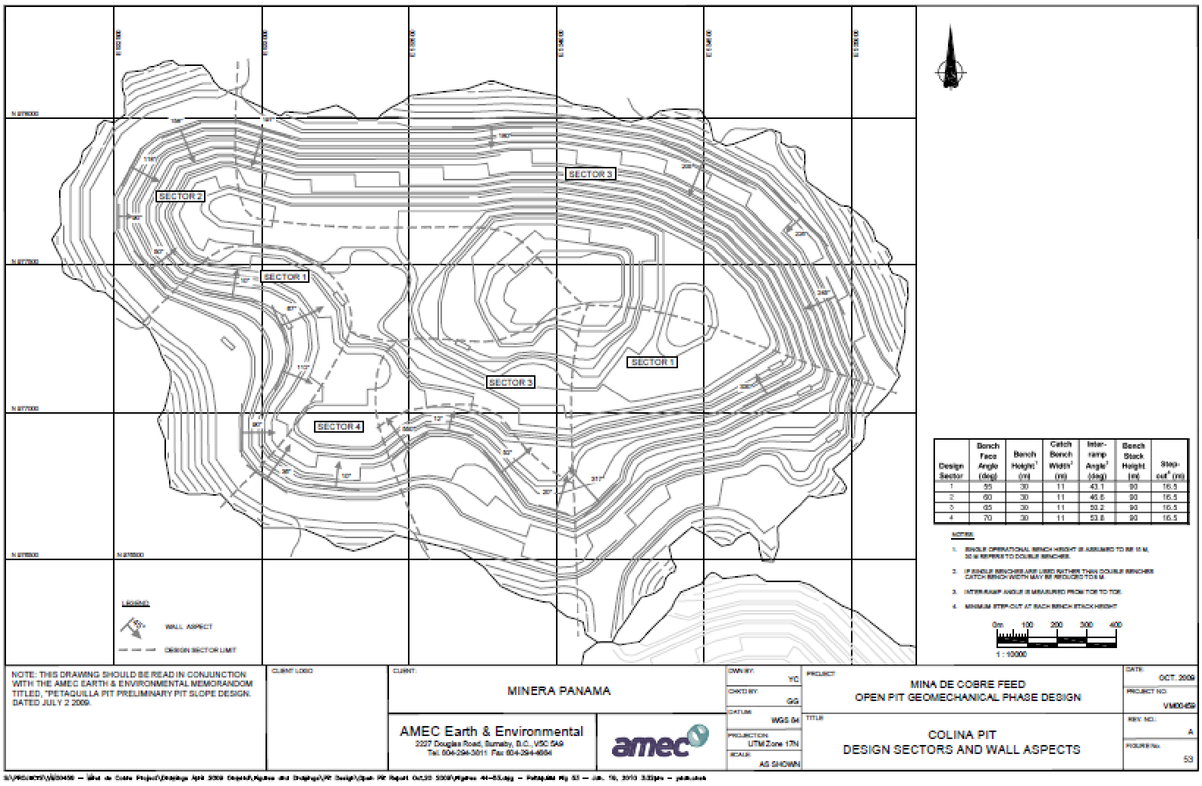


Figure 16-2 Colina Pit slope design sectors (AMEC, February 2010)



Postscript on mine geotechnical engineering

Further structural geological information was made available to CNI following the completion of the detailed pit designs for Botija. This information, in the form of three-dimensional planes defining five major faults (ie, Sante Fe, East Andesite, Strike Central, West Pit and Botija Faults), was reviewed by CNI in relation to a preliminary Botija ultimate pit design (Ver 9; cf Ver 18 in Figure 15.21). The CNI (2014) review conclusions are summarised as follows:

1. The presence of interpreted north dipping structures, including faults and sympathetically oriented pervasive joints, may result in the south wall of the Botija Pit having to be flattened to a 30^{O} overall slope angle to achieve a minimum 1.2 safety factor. These conclusions are based on preliminary low confidence analyses that utilise limited data. CNI (2014) recommended that the design change should be contemplated only if geotechnical mapping of the initial phase pits confirms the persistence of these structures. Effective dewatering of the south wall will help mitigate a potential reduction in the overall slope angle. The Botija waste dump toe behind the pit south wall, however, should be moved further back in anticipation of a potentially flatter slope angle.

Table 16—4 Compiled pit slope design parameters

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	41.8	31.6	90	46.0	Double Benching 30 meters	70	18.1
		2	42.0	31.1	90	46.0		70	18.1
	South Wall River Diversion Diversion Sectors	3	40.3	32.5	90	44.0		69	19.5
		4	38.4	34.2	90	42.0		67	20.6
		5	40.3	31.5	90	44.0		67	18.3
		6	41.0	31.9	90	45.0		69	18.5
		7	41.0	31.6	90	45.0		68	17.9
	West Wall	8	41.1	31.7	90	45.0		69	18.5
		9	41.9	31.5	90	46.0		70	18.1
		10	42.0	31.2	90	46.0		70	18.1
		11	41.1	32.8	90	45.0		71	19.7
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	-	-	-	33.0		68	5.7
		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
Working									
37.5 degree	Bot 1, VG		-	-	-	37.5	15	55	9
40.0 degree	Bot 2, Col 1		-	-	-	40.0	15	60	9
43.0 degree	Bot 3, Col 2		-	-	-	43.0	15	65	9
45.0 degree	Bot 3, Col 3-4		-	-	-	45.0	15	65	9
42.0 degree	Balboa		-	-	-	42.0	30	68	21

* 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

2. These same north dipping structures may provide a toppling failure mechanism along the north wall of the Botija Pit. This potential risk could be managed now by redesigning the upper crusher (in Design Sector 1, but now removed from the designs), conveyor layout and slope immediately below the north wall crest. Once again, however, a design change to the north wall overall slope should be contemplated only if geotechnical mapping of the initial phase pits confirms the persistence of these structures.

3. There is a potential wedge failure structure above the crusher positions at the south east of Design Sector 2. Depending on geotechnical mapping, the overall slope may need to be flattened in this vicinity during mining of the phase 3 pushback (Figure 15-17).

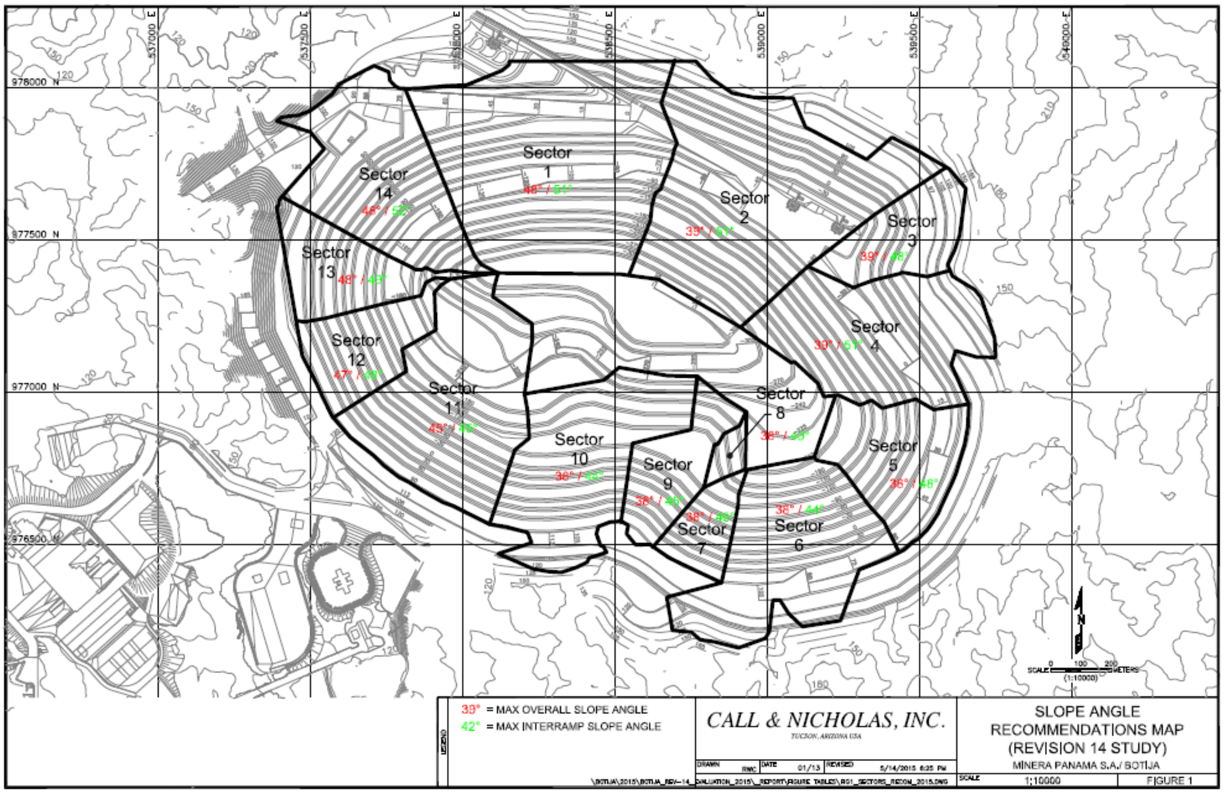
4. The overall slope is two degrees too steep in Design Sector 4. This is a relatively small sector (refer to Figure 16-1) and the design will be reassessed from geotechnical mapping and before committing to the phase 3 pushback.

A further review of the Botija Pit was completed by CNI (2015), based on an updated ultimate pit design (Ver 14; cf Ver 18 in Figure 15.21). This version of the ultimate pit design is very similar to the current design. Figure 16-3 shows that the configuration of design sectors has changed as a result of the inclusion of new data and revised geotechnical modelling/analysis criteria. The new information has resulted in the recommendation of increased overall and interramp slope angles in the western and northwestern sectors of the pit. Conversely, recommended interramp angles in the southern walls have been decreased slightly.

CNI noted that the interpreted attitude of the East Sante Fe fault poses a risk to the north wall due to its proximity to the crusher pockets designed into Sector 2.

As in their 2014 report, CNI recommends that the final wall limits designs be reviewed again when batter exposures are available for mapping of structural persistence. The phased stage designs for Botija will allow mapping and refinement of the geotechnical model before committing to the ultimate north wall slope limits. The south wall infrastructure (ie, waste dump and ore stockpile) have been set-back from the crest for a distance of 200 to 250 m. The southern crest limits and the dump extents can be modified following geotechnical mapping and model refinements during the development of the phase 1 pit.

Figure 16-3 Revised Botija Pit slope design sectors (CNI, May 2015)



16.2.3 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.

These piezometers confirmed the elevated phreatic surface level at approximately 20 m below surface. The apparent gradient of the phreatic surface indicates regional flow towards the Botija River.

Preliminary groundwater modelling was completed in order to assess the impact on regional groundwater levels due to open pit mining and potential seepage from the TMF. From estimated hydraulic parameters interpreted from the bore testing, Golder determined moderate hydraulic conductivity in the upper 100 m, comprising alluvium, saprolite, saprock and upper bedrock, and a relatively low hydraulic conductivity in the granodiorite and andesite bedrock below 100 m.

Only baseline groundwater studies relating to mine dewatering requirements have been completed to date. As part of the proposed future mine geotechnical investigations, however, there are plans to complete a dewatering study for the Botija Pit which will include aquifer testing, an updated hydrogeological model and a preliminary design for vertical and/or horizontal dewatering wells.

16.2.4 Pit water management

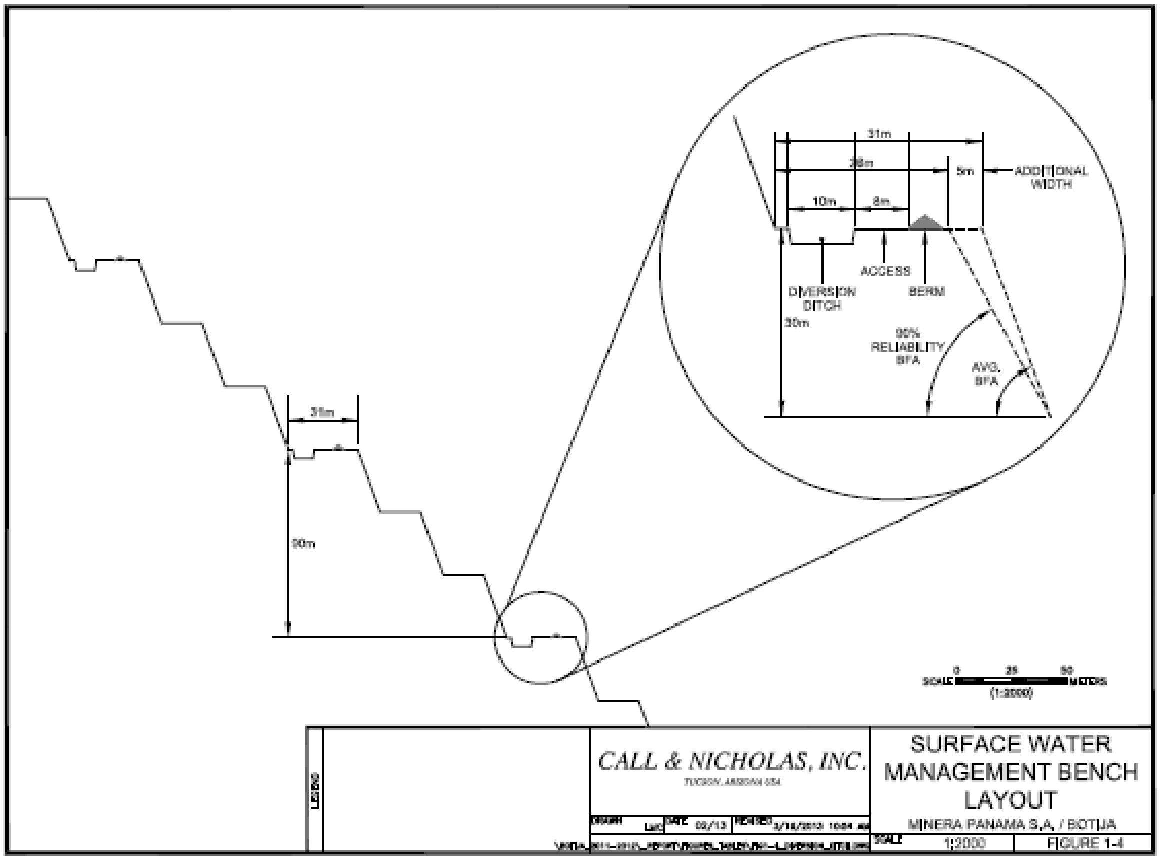
The pit slope design parameters (Table 16-4) also account for catchment ditches and sumps for management of surface water, currently designed to be 26 m wide and spaced at 90 m vertical intervals, as shown in Figure 16-4.

Contact water from the pit sumps will be pumped to the process water tanks for use in the plant.

16.2.5 Surface water diversion

Diversion of the Botija River is planned along the upper southeast wall of the Botija Pit (Sectors 3 to 7 in Figure 16-1). A 30 m wide diversion channel has been incorporated into the pit design at the +90 metre level. Diversion of the Petaquilla River is planned along the upper southwest wall of the Colina Pit.

Figure 16-4 Botija Pit water management (CNI, March 2013)



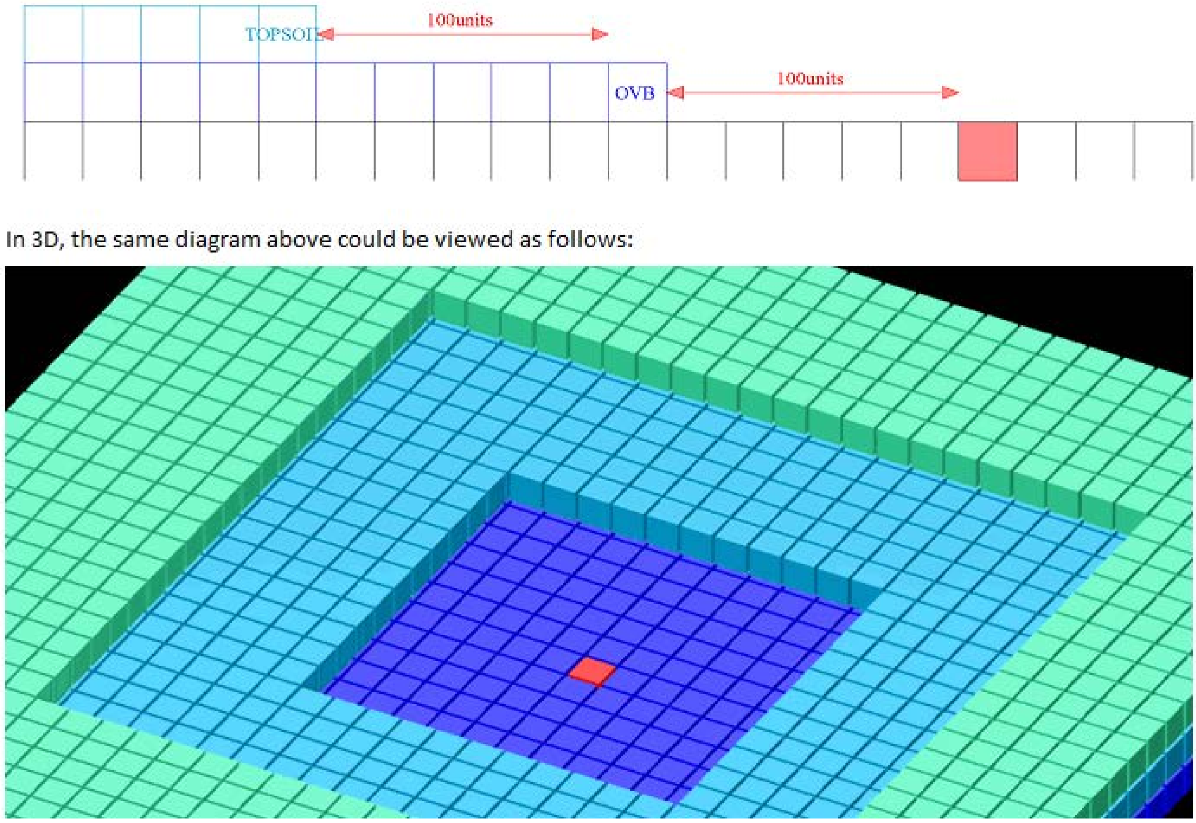
16.3 Mining and processing schedules

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

- minimum mining block size (X x Y x Z) for Saprolite = 100 x 75 x 5 m and non-Saprolite (Granodiorite + Andesite +Porpyhry + Saprock) = 200 x 125 x 15 m
- the size in MineSched was selected to minimise equipment movement (minimum block to be mined)
- mining flitch height for Saprolite = 5 m and non Saprolite = 15 m
- sinking rate for Saprolite = 40 to 60 m/year and non Saprolite = 45 – 60 m/year
- terrace mining with horizontal lag distance of 100 m for Saprolite and 200 m for non Saprolite.

The horizontal lag distance is as explained in Figure 16-5.

Figure 16-5 MineSched scheduling – mining lag criteria



The schedule level of detail is monthly from 2015 to 2017, quarterly for 2018 to 2020, and annually from 2021.

16.3.1 Elevated cut-off strategy

Item 15.5 described an optimisation sensitivity analysis on elevated cut-off grades above the marginal cut-off grade. The rationale for this exercise was to assess the potential for improving the plant feed grade, particularly in the initial production years. By so doing, plant feed classified as ultra low grade (ULG) (ie, with an average grade between the marginal cut-off grade and the selected elevated cut-off grade) can be consigned to a stockpile for deferred reclaim.

In combination with the optimisation results listed in Table 15-9, a number of production schedules were evaluated to assess the impact on total metal production and Project economics. An elevated cut-off grade of 0.2%Cu was selected in consideration of a less than 5% relative impact on Project NPV, but with an improved metal production profile in the first half of the Project life. Table 16-5 shows a comparison of schedules run with a marginal cut-off grade inventory and with an elevated 0.2% Cu cut-off grade inventory. The cumulative metal production information in this table confirms the improved production profile in the elevated cut-off scenario.

Table 16—5 Comparison of LOM schedule inventories, at different cut-off grades

Schedule v38g marginal cut/off		2015 to 2024	2025 to 2034	2035 to 2044	2045 to 2054	from 2055	Total
Pit Production	Mt	1,335.8	1,897.2	1,704.1	1,450.9	78.4	6,466.3
Ore Mined from Pit	Mt	730.6	1,001.2	1,032.7	621.7	75.3	3,461.5
%Cu Ore Mined from Pit	%	0.37	0.38	0.35	0.30	0.41	0.36
Ore to Mill direct	Mt	554.7	884.7	887.5	585.7	72.6	2,985.2
%Cu Ore to Mill direct	%	0.43	0.41	0.38	0.31	0.42	0.39
Ore to SP	Mt	175.9	116.5	145.2	36.0	2.7	476.3
%Cu Ore to SP	%	0.17	0.15	0.17	0.19	0.14	0.17
Rehandle Ore SP to Mill	Mt	28.0	13.7	10.9	147.9	90.5	291.0
%Cu Rehandle Ore SP to Mill	%	0.25	0.24	0.28	0.18	0.24	0.21
Crusher Feed	Mt	582.7	898.4	898.4	733.6	163.1	3,276.2
%Cu Feed	%	0.42	0.41	0.38	0.28	0.32	0.37
Cu Production	**kt**	**2,179**	**3,288**	**3,034**	**1,804**	**425**	**10,729**
Cumulative Cu Production	**kt**	**2,179**	**5,466**	**8,501**	**10,304**	**10,729**	
Au Production	**koz**	**831**	**1,238**	**682**	**809**	**252**	**3,812**
Cumulative Au Production	**koz**	**831**	**2,068**	**2,750**	**3,559**	**3,812**	
Schedule v38h elevated cut/off		**2015 to 2024**	**2025 to 2034**	**2035 to 2044**	**2045 to 2054**	**from 2055**	**Total**
Pit Production	Mt	1,336.4	1,872.5	1,900.1	1,357.0	0.0	6,466.0
Ore Mined from Pit	Mt	782.7	1,043.0	1,046.9	623.0	0.0	3,495.6
%Cu Ore Mined from Pit	%	0.36	0.39	0.34	0.32	0.00	0.35
Ore to Mill direct	Mt	551.4	881.3	856.1	496.4	0.0	2,785.2
%Cu Ore to Mill direct	%	0.44	0.43	0.38	0.36	0.00	0.40
Ore to SP	Mt	231.3	161.7	190.8	126.6	0.0	710.4
%Cu Ore to SP	%	0.17	0.16	0.16	0.14	0.00	0.16
Rehandle Ore SP to Mill	Mt	23.2	11.2	36.4	276.0	50.5	397.3
%Cu Rehandle Ore SP to Mill	%	0.27	0.29	0.20	0.18	0.19	0.19
Crusher Feed	Mt	574.6	892.6	892.5	772.4	50.5	3,182.5
%Cu Feed	%	0.44	0.43	0.37	0.30	0.19	0.38
Cu Production	**kt**	**2,245**	**3,408**	**2,915**	**1,976**	**67**	**10,612**
Cumulative Cu Production	**kt**	**2,245**	**5,654**	**8,569**	**10,545**	**10,612**	
Au Production	**koz**	**851**	**1,221**	**699**	**954**	**42**	**3,768**
Cumulative Au Production	**koz**	**851**	**2,073**	**2,772**	**3,726**	**3,768**	

16.3.2 LOM schedule

Features of the LOM mining and production schedule as listed in Table 16-6 are as follows:

- Mining (ie, the pre-strip period) commences in July 2015 and processing commences in July 2017. The Project life is 40 years to 2055.
- The total material mined from all pits amounts to 6,465.9 Mt (2,516.8 Mbcm), of which 3,182.5 Mt is ore (including saprock ore, high grade ore, low grade ore) and 3,283.4 Mt is waste (including saprolite, saprock waste and mineralised waste).
- The direct feed ore is 2,785.2 Mt at a grade of 0.40%Cu and 397.3 Mt at a grade of 0.19% Cu is stockpile reclaim.
- The total HG and LG ore mined to stockpiles and reclaimed throughout the Project life is 49.9 Mt at a grade of 0.27% Cu.
- In addition, 223.9 Mt of ultra low grade ore at a grade of 0.17% Cu is mined to stockpile and reclaimed in the final years of the Project.
- The crusher feed ramps up from 2017 to 74 Mtpa by 2019, at which level it remains until 2021. The feed rate then ramps up to 90 Mtpa by 2023, at which level it stays until 2046.
- The rate drops to 74 Mtpa between 2047 and 2054.

- In terms of total plant feed, the average copper grade is 0.42%Cu for the first twenty years, and then 0.32%Cu for the remaining Project life.
- Ignoring the 2017 commissioning year, the average annual copper metal production in the first twenty years is 328,000 tonnes. Thereafter, the annual average is 228,000 tonnes.
- The annual average by-product production is approximately 2,570 tonnes of molybdenum, 97 thousand ounces of gold and 1,570 thousand ounces of silver.
- The total waste mined includes 313.1 Mt of mineralised waste at a grade of 0.12% Cu.
- The overall life of mine strip ratio (tonnes) is 1 : 1.

Figures 16-6 to 16-10 depict the LOM schedule graphical results.

Table 16—6 LOM scheduling – summary of results

Year	Mined Ore	Grade	Metal	Crusher Feed	Grade	Metal	Cu Metal Recovery		Stockpile Movements												Waste		Ore+Waste	
	(t)	(%TCu)	(t Cu)	(t)	(%TCu)	(t Cu)	(%)	(t)	HG/LG On (t)	HG/LG Off (t)	Balance (t)	ULG On (t)	ULG Off (t)	Balance (t)	Saprock On (t)	Saprock Off (t)	Balance (t)	MW On (t)	MW Off (t)	MW Balance (t)	(t)	Strip Ratio	(bcm)	(t)
2015	19,050	0.10	19												19,050		19,050	128,700		128,700	19,571,030	1,027.4	11,158,053	19,590,080
2016	6,309,300	0.15	9,492						873,840		873,840	636,990		636,990	4,798,471		4,817,521	1,136,520		1,265,220	34,840,827	5.5	19,734,430	41,150,127
2017	41,583,647	0.24	100,939	17,461,029	0.33	57,729	87.9	50,741	2,604,374	1,939,193	1,539,021	8,975,311		9,612,300	14,482,126		19,299,647	9,417,630		10,682,850	56,945,941	1.4	44,643,999	98,529,587
2018	88,877,238	0.36	318,825	70,052,091	0.41	284,658	89.3	254,126	2,209,739	1,539,021	2,209,739	15,356,076		24,968,376	2,798,353		22,098,000	11,139,337		21,822,188	34,092,718	0.4	46,969,152	122,969,956
2019	98,087,634	0.41	399,705	73,995,186	0.47	348,592	90.1	314,210	12,528,701	3,883,053	10,855,388	10,750,975		35,719,351	4,695,825		26,793,825	7,801,088		29,623,276	51,804,102	0.5	63,037,449	149,891,736
2020	80,075,822	0.45	362,927	74,202,840	0.49	364,749	90.4	329,846	2,251,725	11,671,722	1,435,390	6,762,199		42,481,550	8,530,780		35,324,605	5,424,938		35,048,213	78,248,441	1.0	66,999,720	158,324,263
2021	88,234,181	0.43	383,060	74,314,748	0.49	360,592	90.4	325,849		1,435,390		8,026,478		50,508,028	7,328,344		42,652,949	6,954,610		42,002,823	87,118,375	1.0	68,930,187	175,352,556
2022	94,859,219	0.38	355,758	86,999,940	0.39	339,977	89.3	303,607	2,724,384		2,724,384	4,410,995		54,919,023	723,900		43,376,849	8,827,490		50,830,314	89,529,915	0.9	70,541,498	184,389,134
2023	109,021,499	0.36	391,188	89,627,154	0.40	354,418	89.1	315,919		2,724,384		13,574,804		68,493,827	8,543,925		51,920,774	8,487,224		59,317,538	83,791,932	0.8	75,237,179	192,813,431
2024	109,448,816	0.40	433,836	87,956,361	0.44	391,347	89.7	351,179				10,533,103		79,026,930	10,959,353		62,880,127	6,879,529		66,197,066	83,892,871	0.8	75,449,461	193,341,688
2025	96,918,769	0.43	418,962	88,012,816	0.44	387,224	89.8	347,800				81,881		79,108,811	8,824,072		71,704,199	5,508,131		71,705,198	95,778,112	1.0	73,993,347	192,696,881
2026	93,894,143	0.41	381,224	89,999,875	0.41	369,508	89.4	330,345	740,103		740,103	887,215		79,996,026	2,266,950		73,971,149	8,572,411		80,277,609	98,919,288	1.1	74,447,215	192,813,431
2027	92,969,879	0.48	444,966	89,999,875	0.49	439,264	90.4	396,936	1,276,903		2,017,007	902,526		80,898,552	790,575		74,761,724	5,723,455		86,001,064	79,232,260	0.9	67,602,953	172,202,139
2028	97,595,045	0.42	412,442	90,246,450	0.44	397,960	89.8	357,460	3,294,459		5,311,466	1,701,460		82,600,013	2,352,675		77,114,399	8,772,463		94,773,527	95,746,643	1.0	75,503,021	193,341,687
2029	87,126,228	0.39	342,538	88,675,319	0.40	350,402	89.2	312,433		5,311,466				82,600,013	3,762,375		80,876,774	5,189,925		99,963,452	99,664,962	1.1	73,430,441	186,791,190
2030	97,973,705	0.39	378,701	89,999,875	0.40	363,283	89.1	323,596	74,230		74,230	6,423,224		89,023,237	1,476,375		82,353,149	8,505,113		108,468,565	93,545,717	1.0	72,425,643	191,519,422
2031	105,097,338	0.43	448,732	89,999,875	0.46	414,368	89.9	372,509	5,856,166		5,930,396	9,184,147		98,207,384	57,150		82,410,299	9,589,954		118,058,519	85,533,122	0.8	71,842,574	190,630,461
2032	97,609,013	0.38	367,101	90,246,450	0.40	359,728	89.0	320,204		2,956,789	2,973,607	10,319,352		108,526,736			82,410,299	6,598,796		124,657,315	95,130,349	1.0	72,946,498	192,739,362
2033	101,838,220	0.37	372,613	89,999,875	0.40	356,236	89.0	316,961		1,490,532	1,483,075	11,052,402		119,579,138	2,276,475		84,686,774	7,889,400		132,546,715	67,822,007	0.7	64,395,694	169,660,226
2034	102,022,641	0.38	388,631	85,397,402	0.43	368,430	89.5	329,794		1,483,075				119,579,138	18,108,314		102,795,088	3,604,343		136,151,059	88,122,239	0.9	72,456,248	190,144,880
2035	102,085,272	0.37	375,821	88,838,824	0.39	349,889	88.9	311,176						119,579,138	13,246,448		116,041,536	4,830,908		140,981,966	86,787,189	0.9	71,617,284	188,872,461
2036	93,898,233	0.39	370,191	90,246,450	0.40	361,976	89.2	322,845	1,362,478		1,362,478	611,368		120,190,506	1,677,937		117,719,473	5,982,167		146,964,133	99,443,458	1.1	72,990,184	193,341,691
2037	96,800,878	0.38	365,295	89,999,875	0.39	349,856	89.1	311,767	4,511,700		5,874,178	1,941,641		122,132,147	347,662		118,067,136	7,162,594		154,126,728	93,784,009	1.0	72,577,652	190,584,887
2038	90,193,244	0.39	347,466	86,431,517	0.40	344,577	88.7	305,513		5,874,178		8,854,856		130,987,002	781,050		118,848,186	5,154,171		159,280,898	102,487,616	1.1	74,185,403	192,680,861
2039	111,518,207	0.35	385,034	89,999,875	0.38	341,087	88.0	300,200	4,601,580		4,601,580	14,878,402		145,865,404	2,038,350		120,886,536	9,846,721		169,127,619	81,295,225	0.7	73,542,974	192,813,432
2040	103,764,362	0.35	361,341	90,246,450	0.38	339,295	87.6	297,204		4,362,897	238,683	16,705,770		162,571,175	1,175,039		122,061,574	9,896,746		179,024,365	89,577,326	0.9	74,036,274	193,341,688
2041	91,427,895	0.36	332,994	89,999,875	0.37	330,185	87.7	289,736	566,059		804,742	513,000		163,084,175	348,961		122,410,536	6,576,245		185,600,611	98,042,842	1.1	74,622,109	189,470,738
2042	86,467,158	0.34	297,662	87,114,738	0.34	299,637	87.8	263,043		804,742				163,084,175	157,162		122,567,698	10,347,130		195,947,740	92,371,771	1.1	71,690,430	178,838,929
2043	108,818,666	0.32	344,432	89,369,362	0.35	310,645	88.1	273,565				19,273,092		182,357,266	176,212		122,743,911	10,088,333		206,036,073	83,858,011	0.8	74,908,367	192,676,677
2044	81,422,701	0.32	259,671	90,246,450	0.30	275,063	87.3	240,181				15,998,034	25,329,738	173,025,563	507,954		123,251,864	10,635,308		216,671,381	106,059,049	1.3	77,294,068	187,481,750
2045	69,296,456	0.33	227,004	89,999,875	0.29	262,913	86.8	228,282				13,248,326	34,048,629	152,225,260	96,884		123,348,748	8,792,778		225,464,159	123,516,973	1.8	79,725,231	192,813,428
2046	66,422,986	0.36	236,597	89,999,875	0.31	277,831	86.7	240,886				100,610	23,691,786	128,634,084	14,288		123,363,036	12,229,443		237,693,603	112,501,931	1.7	67,396,955	178,924,917
2047	65,864,370	0.36	239,887	74,000,100	0.34	254,114	87.1	221,292					8,135,730	120,498,354			123,363,036	8,927,789		246,621,392	116,386,338	1.8	68,318,448	182,250,708
2048	39,904,568	0.42	169,020	74,202,840	0.31	228,998	86.2	197,461					34,298,272	86,200,082			123,363,036	3,757,261		250,378,653	135,029,889	3.4	65,366,810	174,934,456
2049	34,908,075	0.27	94,046	74,000,100	0.22	162,484	85.6	139,129	19,575		19,575	559,575	39,671,175	47,088,481			123,363,036	14,329,912		264,708,565	118,265,735	3.4	57,400,874	153,173,810
2050	40,556,599	0.28	114,718	74,000,100	0.23	173,244	86.7	150,194	10,125	19,575	10,125	380,363	33,814,413	13,654,430			123,363,036	14,489,437		279,198,002	105,443,395	2.6	54,845,505	145,999,994
2051	69,207,848	0.32	224,647	74,000,100	0.31	232,988	88.3	205,745		10,125			4,782,127	8,872,303			123,363,036	17,102,587		296,300,589	76,209,680	1.1	54,822,161	145,417,528
2052	82,287,223	0.35	288,469	74,202,840	0.36	270,574	88.8	240,165	4,438,367		4,438,367	3,646,015		12,518,319			123,363,036	12,565,800		308,866,389	36,814,691	0.4	45,111,696	119,101,913
2053	58,143,485	0.44	254,949	74,000,100	0.39	286,375	88.8	254,215		4,438,367	0	7,708,500	19,126,747	1,100,071			123,363,036	4,216,387		313,082,776	6,229,237	0.1	24,603,750	64,372,723
2054				74,000,100	0.19	143,023	69.1	98,898					1,100,071	0		72,900,029	50,463,007			313,082,776				
2055				50,463,007	0.19	97,685	68.9	67,333								50,463,007	0			313,082,776				
Undiluted	**3,182,549,611**	**0.38**	**12,000,903**	**3,182,549,611**	**0.38**	**12,000,903**	**88.4**	**10,612,345**	**49,944,509**	**49,944,509**		**223,998,689**	**223,998,689**		**123,363,036**	**123,363,036**		**313,082,776**	**0**		**3,283,435,217**	**1.03**	**2,516,800,941**	**6,465,984,829**
Diluted	**3,181,276,591**	**0.37**	**11,760,885**	**3,181,276,591**	**0.37**	**11,760,885**	**90.2**	**10,612,345**	**49,924,531**	**49,924,531**		**223,909,090**	**223,909,090**		**123,313,690**	**123,313,690**		**312,957,543**	**0**		**3,284,708,237**	**1.03**	**2,516,800,941**	**6,465,984,829**

Figure 16-6 LOM scheduling – annual ore and waste mined tonnes

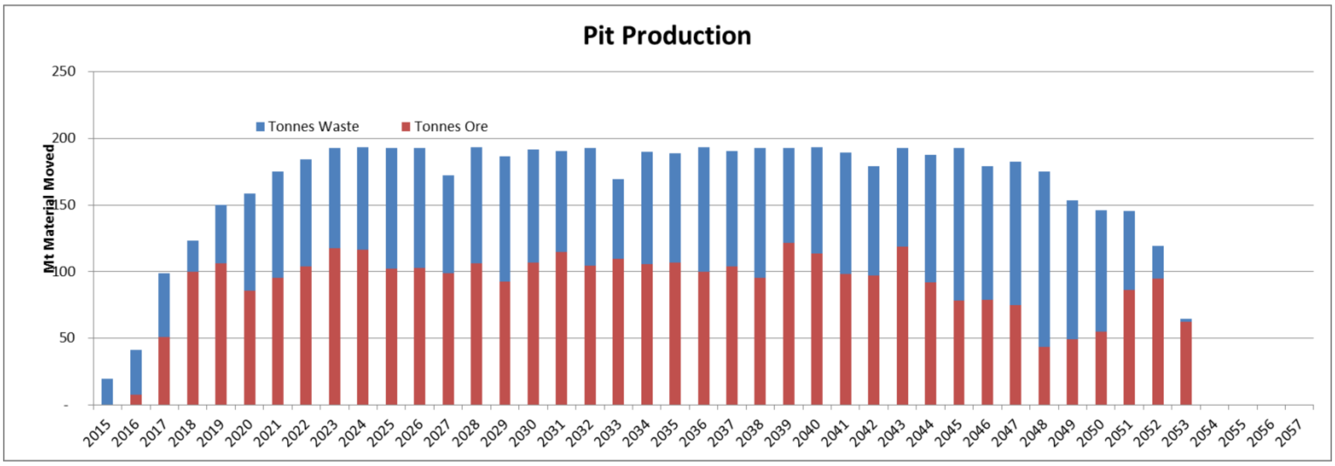


Figure 16-7 LOM scheduling – annual plant feed tonnes

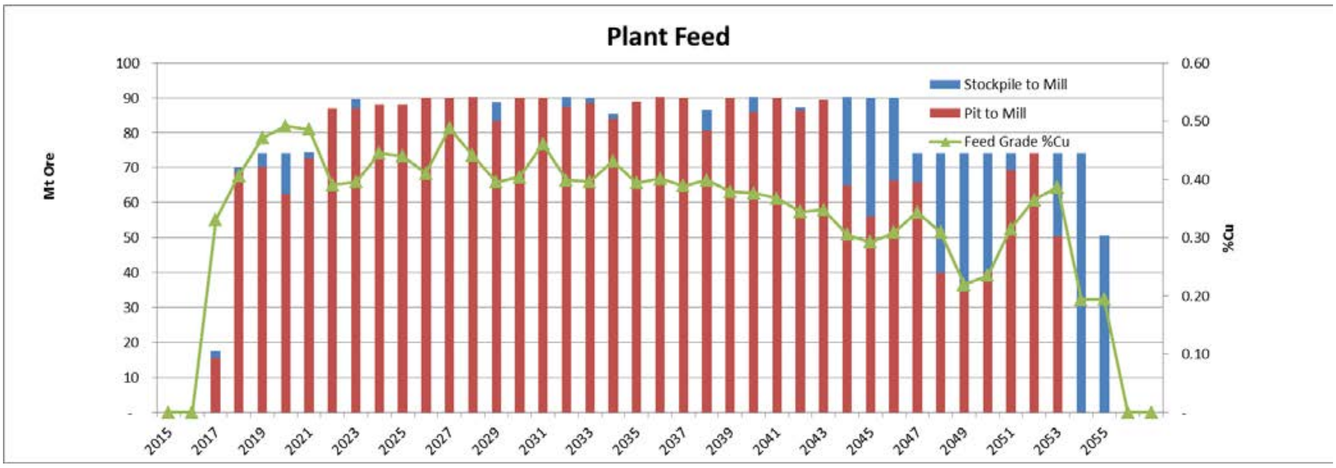


Figure 16-8 LOM scheduling – annual copper metal production

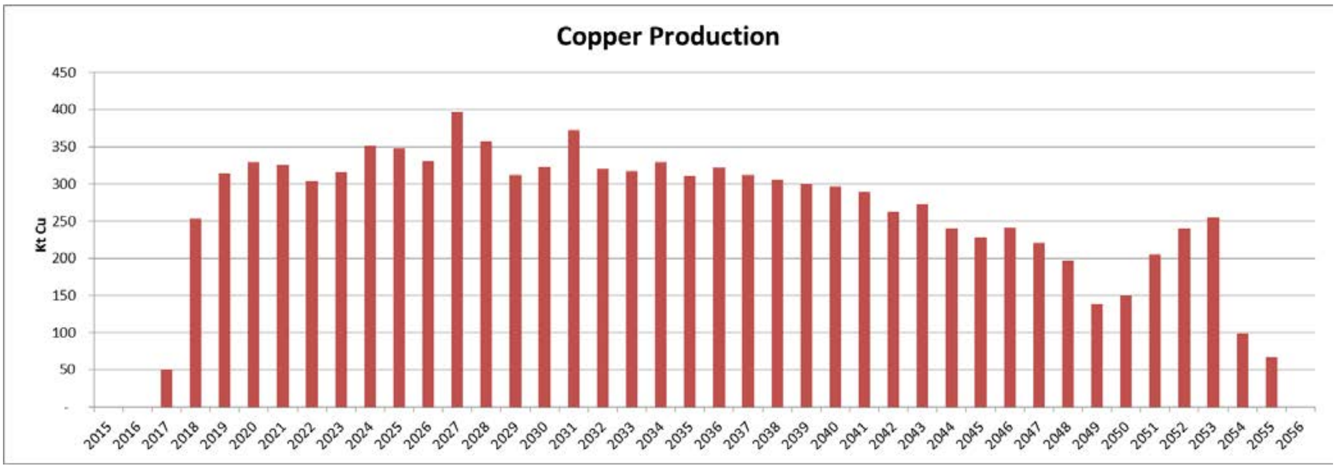


Figure 16-9 LOM scheduling – annual stockpile balance tonnes

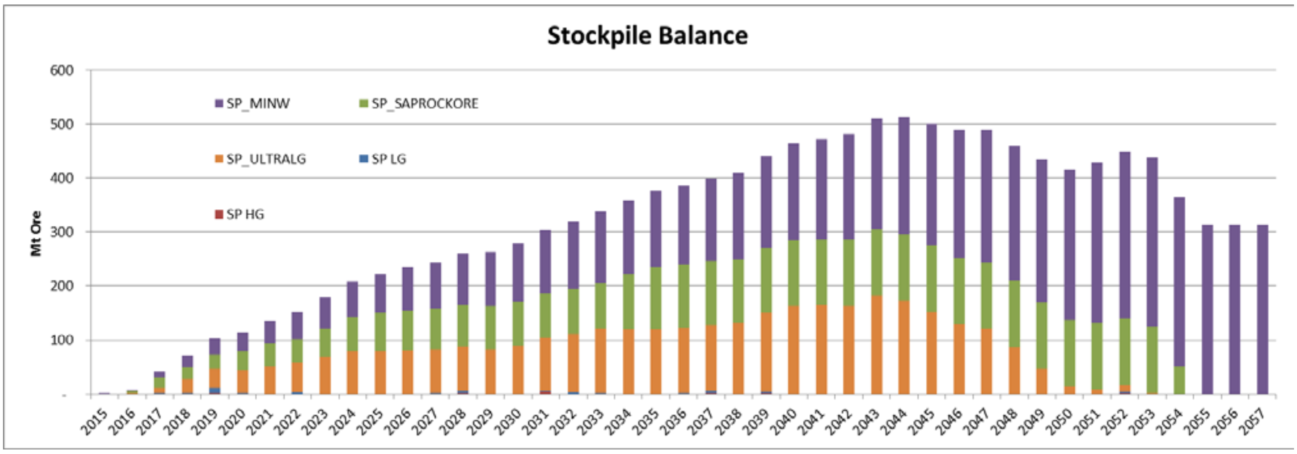
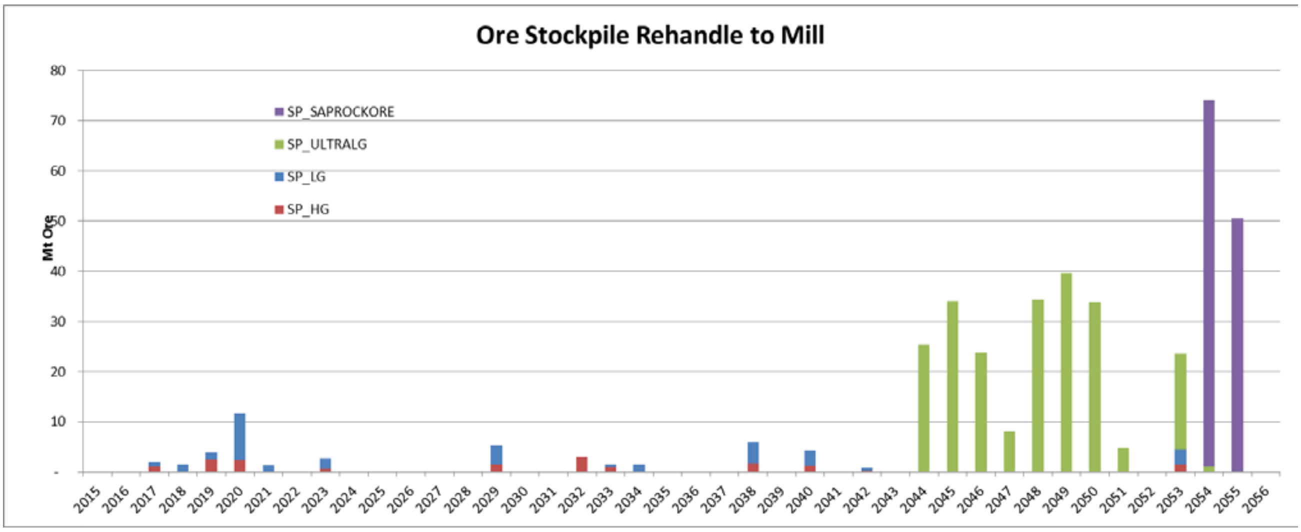


Figure 16-10 LOM scheduling – annual ore stockpile tonnes reclaimed into plant

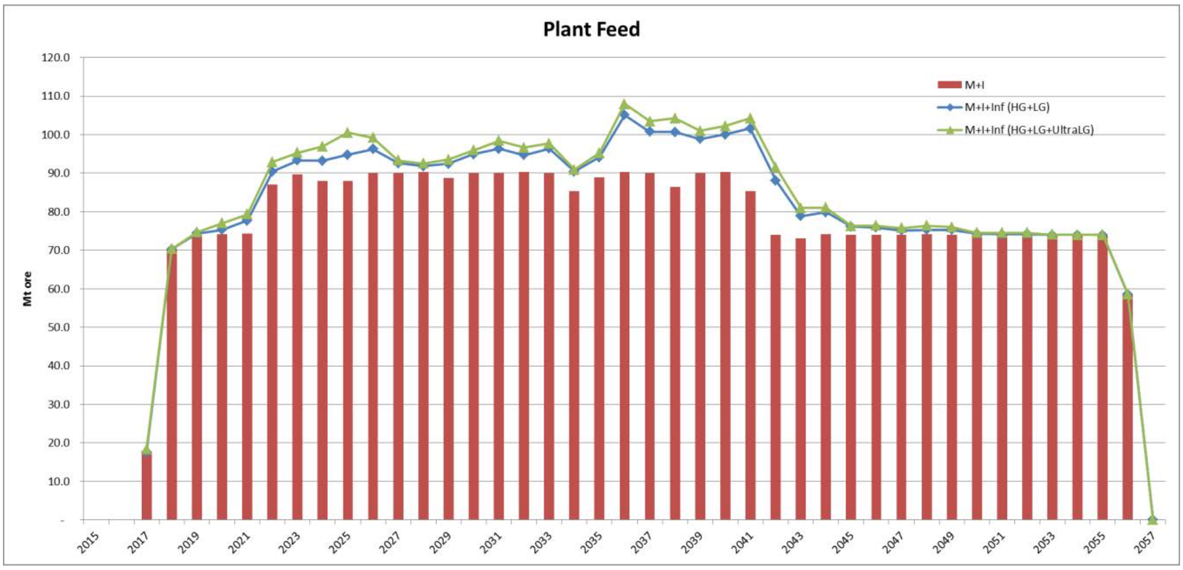


16.3.3 Scheduling for future plant capacity

In the event that the expanded plant capacity can process up to 100 Mtpa, there is the potential for currently Inferred Mineral Resource (above marginal cut-off grade) to be included as future plant feed on the proviso that this resource can be upgraded to at least an Indicated classification. Alternatively, the 90 Mtpa rate could be maintained over a longer period.

Figure 16_11 shows a notional increased annual plant feed profile if Inferred Mineral resource (approximately 237 Mt additional over the life of mine) was able to be reclassified as additional feed. This notional profile is for information only and the related inventory forms no part of the Mineral Reserve estimation process.

Figure 16-11 LOM scheduling – Plant feed profile with the inclusion of currently Inferred Mineral Resource



16.3.4 Mining sequence

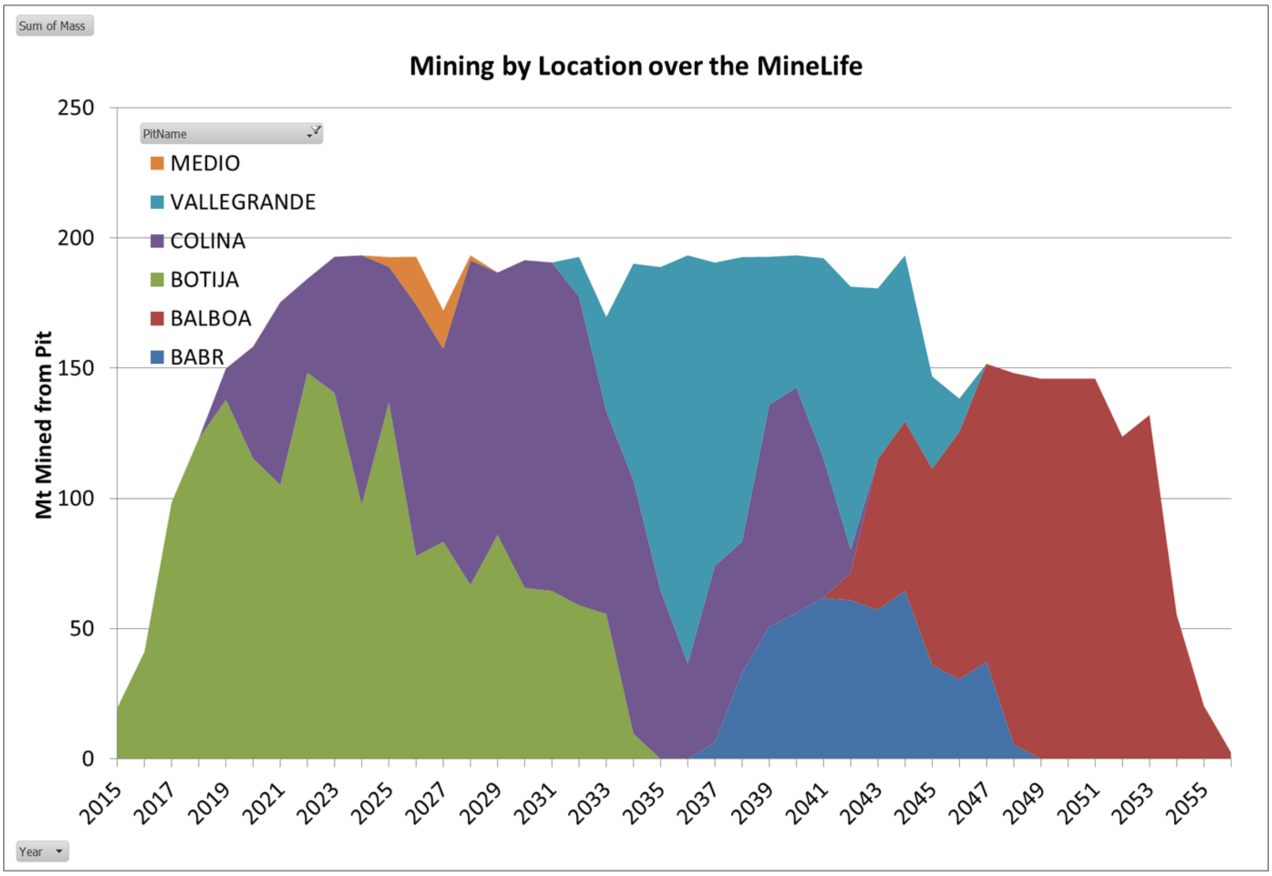
The sequencing concept for mining of the several Cobre Panamá pits is dictated by the relative grade distribution across the pits, the location of the starter pit boxcut at Botija in close proximity to the processing plant (to minimise initial overland conveying infrastructure), the minimisation of initial 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 will start 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 will be stockpiled for reclaim at the commencement of processing.
3. As part of the pre-strip, crusher pockets within the Botija boxcut will be mined to enable the first IPCs to be available by early 2017, for first ore production in Q3 2017.
4. Commencing with the boxcut, the Botija Pit will be mined in phases, ie phase 1 from 2017 to 2021, phase 2 from 2017 to 2022, phase 3 from 2017 to 2030, and the ultimate pit phase from 2025 to 2034.
5. Mining in Colina will commence in 2019 in phase 1 and phase 2, continuing to 2022 and 2031, respectively. Phase 3 will be mined from 2025 to 2035, followed by the ultimate pit phase from 2027 to 2042.
6. The Medio Pit will be mined netween 2025 and 2028.
7. The Valle Grande Pit will be mined in two phases from 2032 to 2045.
8. The BABR Pits will be mined in two phases from 2037 to 2049.
9. The Balboa Pit will be mined in two phases from 2041 to 2055.

Figure 16-12 shows the mining sequence and production profile. From 2044 onwards, the strip ratio increases to above 1:1, and Cu production decreases to less than 250 Mtpa as the average grade of the plant feed decreases to about 0.3%Cu. This period coincides with the reclaim of ULG ore stockpiles to keep the plant producton rate at 74 Mtpa.

Figure 16-12 Graph showing mining sequence



16.3.5 IPCC sequence

The IPCC layout plan (Figure 16-13) shows the proposed locations of the crushers over time.

Figure 16-14 shows the number of crushers required each year and their relocations relative to the plan in Figure 16-13.

The Balboa ore would be hauled to the crushers in the Colina 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 at a site to be determined. This same crusher could be used on ULG and saprock stockpile reclaim.

Figure 16-13 IPCC layout plan

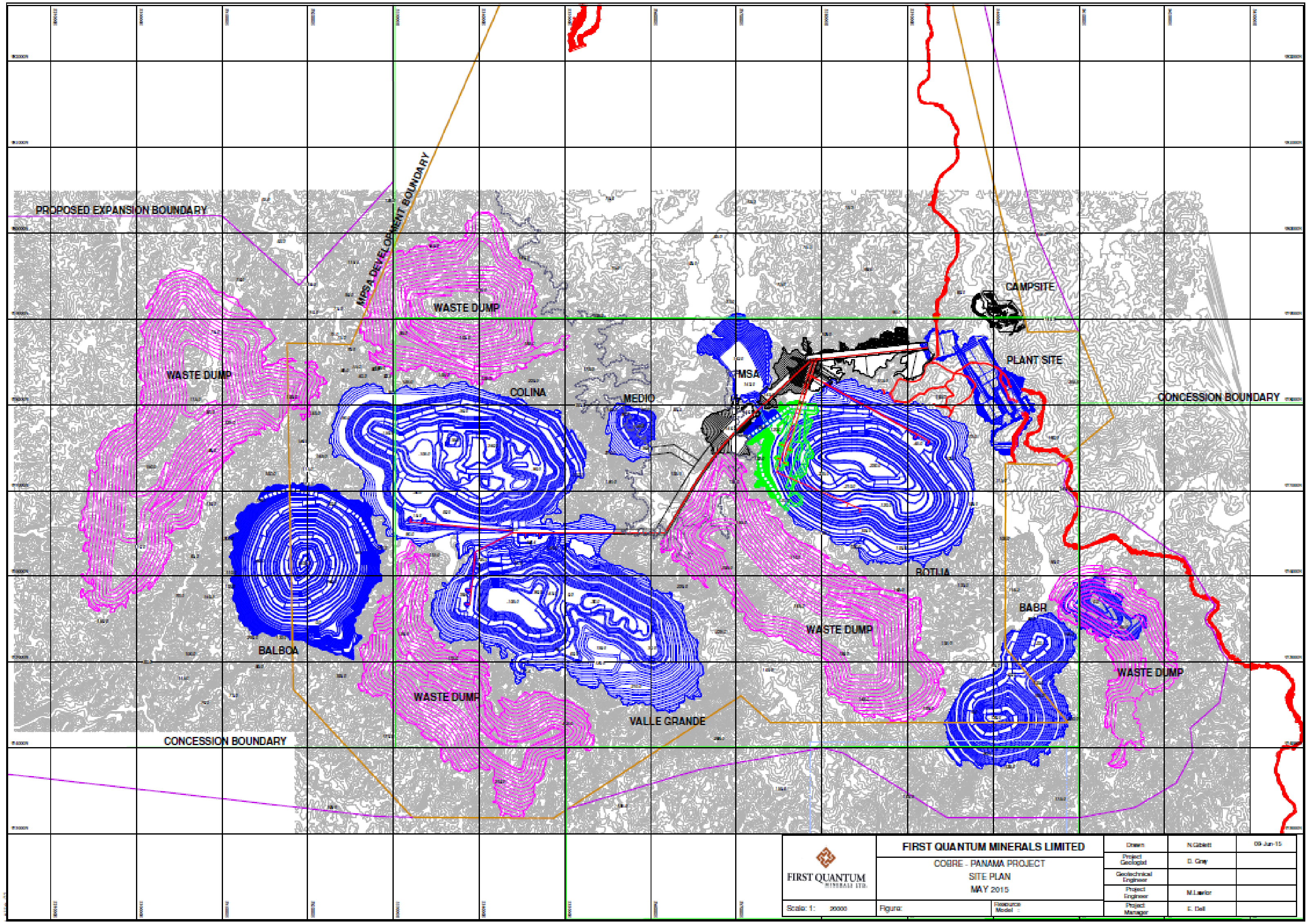
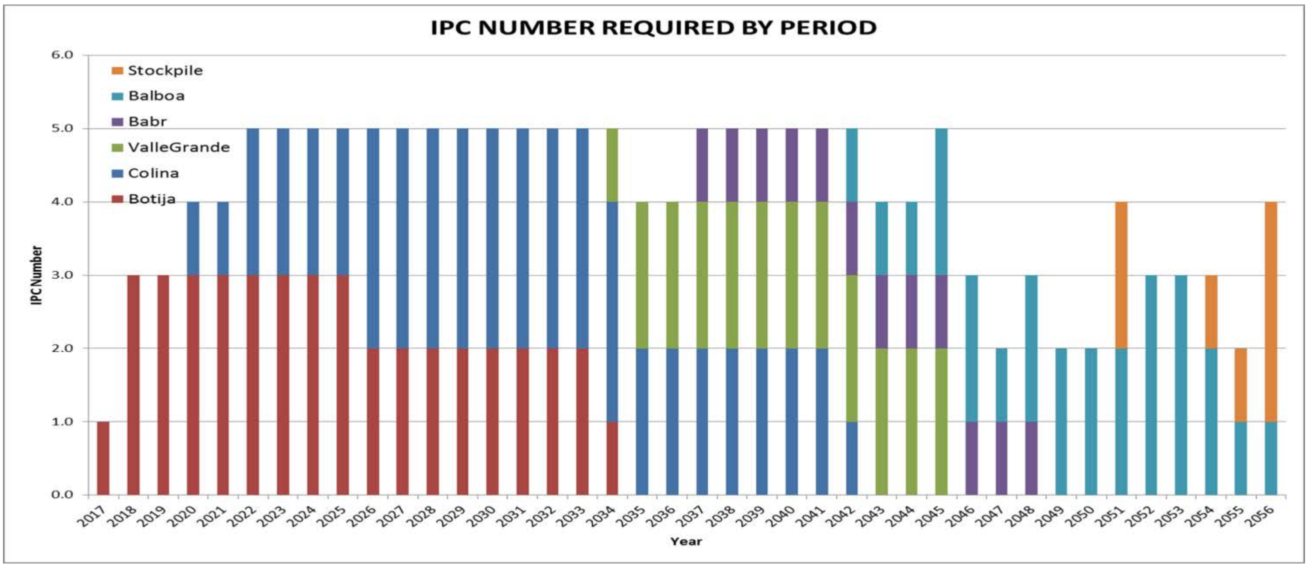


Figure 16-14 In-pit crusher requirements and relocations over time



16.3.6 Waste dumping schedule

Table 16-7 shows the waste dumping destinations relative to the site layout plan shown in Figure 15-34. During the pre-stripping prior to mid 2017, approximately 46.5 Mbcm of waste will be mined from the Botija Pit and used for construction purposes as described in Item 16.3.7. An additional 9.5 Mlcm of material required for initial infrastructure will be sourced from the Botija Pit following the pre-strip period.

Thereafter, Botija Pit waste would be hauled and dumped onto the Botija dump located adjacent and to the south west. This same dump would receive waste from the BABR Pits. The Colina dump would receive waste from the Colina and Medio Pits, whilst the Valle Grande dump would receive waste from the Valle Grande Pit only, etc.

Table 16—7 LOM scheduling – waste dump schedule

Pit	Insitu Pit Volumes			Dump Design Volumes		Dump Destination Volumes						
	Waste (Mbcm)	Min. waste (Mbcm)	Total insitu (Mbcm)	Insitu (Mbcm)	Swelled 30% (Mlcm)	Infrastructure (Mlcm)	Botija (Mlcm)	Colina (Mlcm)	Valle Grande (Mlcm)	BABR (Mlcm)	Balboa (Mlcm)	TOTAL (Mlcm)
Botija	283.9	38.0	321.9	321.9	**418.5**	57.9	360.6					418.5
Colina	284.6	20.8	305.4	315.3	**409.8**			292.7			104.4	397.1
Medio	9.7	0.1	9.8					12.8				12.8
Valle Grande	188.4	14.8	203.3	203.3	**264.2**				264.2			264.2
BABR	104.6	9.2	113.8	113.8	**148.0**		43.3			104.7		148.0
Balboa	320.8	35.2	356.0	356.0	**462.8**						462.8	462.8
TOTAL	**1,192.1**	**118.2**	**1,310.3**	**1,310.3**	**1,703.4**	**57.9**	**403.9**	**305.4**	**264.2**	**104.7**	**567.2**	**1,703.4**

16.3.7 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 (PAF). In the Ecometrix report, the Valle Grande waste is said to be significantly non-PAF (NAF)[7].

[7] Ore stockpiles are also described as being significantly PAF.

The Ecomterix report provides 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.

The design of the surface water management layout and control dams remains in progress. Further updates to the mine plan and site layout will consider the segregation of PAF and NAF waste and the control of run-off from the waste dumps.

16.3.8 Pre-stripping schedule

A detailed pre-stripping schedule was developed within the LOM schedule monthly framework, in order to plan and sequence the mining and delivery of waste for construction purposes. Aspects of this schedule are as follows:

- The Botija Pit pre-strip will start in July 2015. This means that before this date, trees will have already been cut down and road access established to the appropriate pre-stripping and waste dumping areas.
- There is 46.5 Mbcm of waste available (48.4 Mlcm) for construction in the period from July 2015 to June 2017. This quantity includes Waste, Saprolite, Saprock Waste, Mineralised Waste and Ultra low grade, all of which are considered suitable for use across the site during the pre-strip construction period, for the construction locations shown on Figure 16-15.
- The priorities for pre-strip period waste dumping are:
    1. Area 2 is first priority for the initial equipment assembly area (required whilst the mining services area (MSA) is being developed. The first of the primary haul trucks will be required in August 2016, before the MSA is complete. Area 2 includes the adjacent conveyor corridor.
    2. The MSA is second priority and required to be completed by March 2017. From March 2017, civils construction will commence at this site.
    3. Area 3 is third priority to prevent ponding and to provide additional equipment assembly and storage areas. Area 3 includes the adjacent conveyor corridor.
    4. The Stage 1 WRD is the fourth priority to provide a levelled ore stockpile site to accommodate ore mined during the pre-strip period. The location of this dump and stockpile site is adjacent to the initial in-pit crusher locations in the Botija Pit boxcut. There is 4.4 Mbcm of HG and LG ore to be stockpiled during the pre-strip period, in addition to 0.5 Mbcm of Saprock ore.
    5. Area 1, the Colina conveyor corridor (Stages 1 and 2) and the Stage 2 WRD follow in priority.

Figure 16-16 and Figure 16-17 show the pre-strip waste mining volumes (in bcm and by material type), and the waste dumping volumes (in lcm and dumping destination), respectively.

Figure 16-15 Pre-strip schedule period – location of waste dumping sites

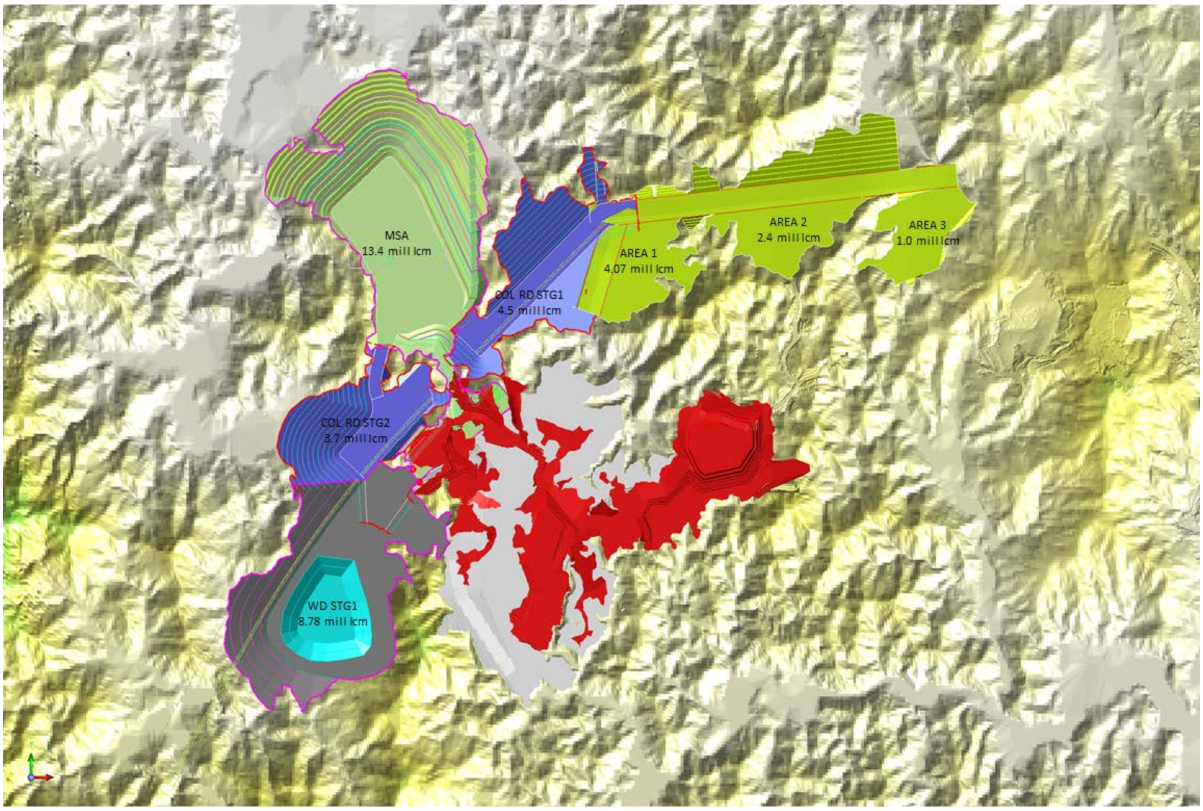


Figure 16-16 Pre-strip period waste mining volumes

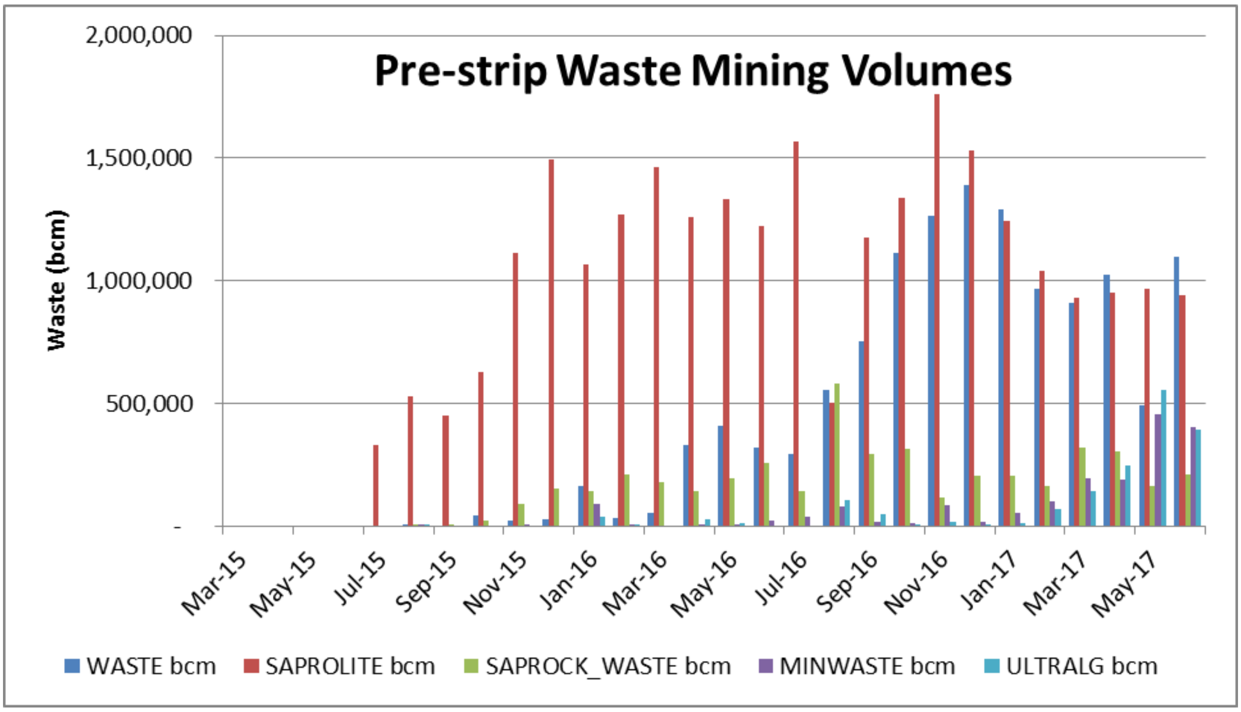
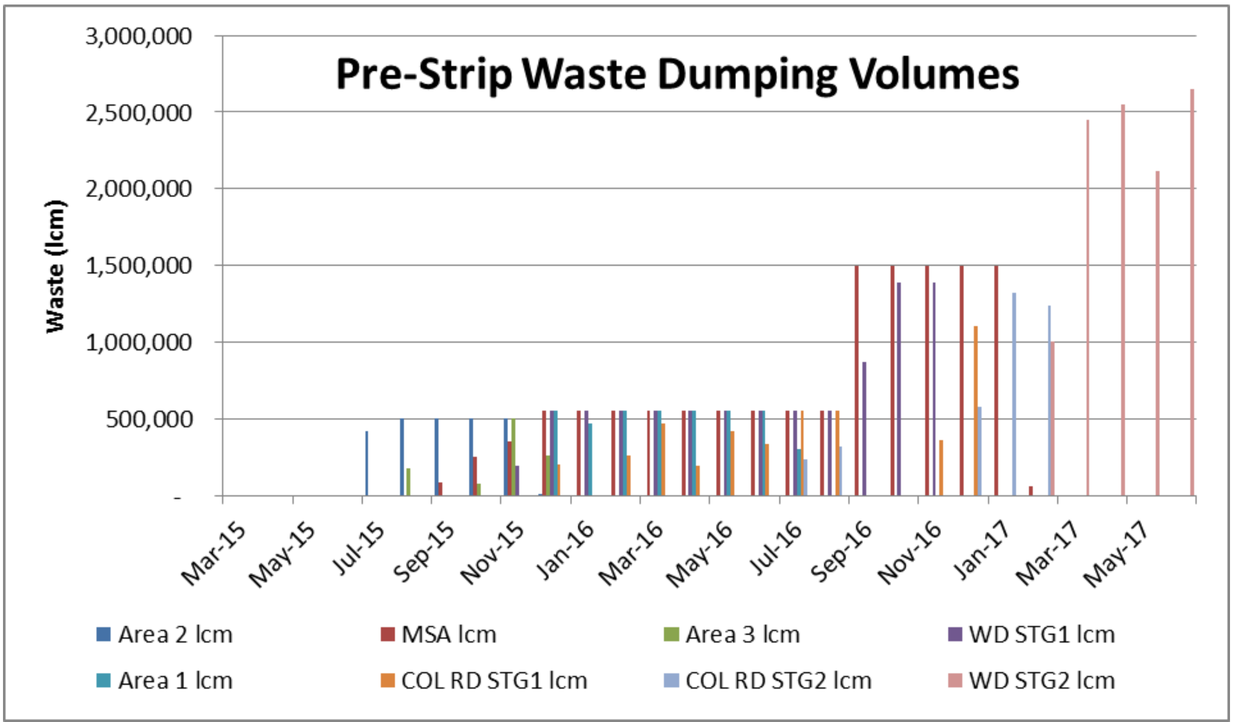


Figure 16-17 Pre-strip period waste dumping volumes



16.3.9 Mining equipment

Table 16-8 lists the primary mining fleet and support equipment for the Project[8]. The primary mining fleet will comprise the following:

- up to 7 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 46 x Komatsu 960E (or equivalent), 360 t haul trucks

The secondary mining fleet fleet will comprise:

- diesel hydraulic shovels
- 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 mining support equipment (track and wheel dozers, graders, service trucks etc) will be in place at Cobre Panamá.

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

Table 16—8 Mining equipment

Mining Equipment	Model	2017 to 2026		2027 to 2036		2037 to 2046		2047 to end	
		Average #	Maximum #	Average #	Maximum #	Average #	Maximum #	Average #	Maximum #
Drills									
Furukawa	DCR1500-EDII	4	6	4	5	5	6	4	6
Caterpillar	MD6640	4	6	5	6	5	7	4	7
Excavators									
Fleet 1	PC1250	2	2	2	2	2	2	2	2
Fleet 2	PC2000	2	2	2	2	2	2	2	2
Fleet 3	PC8000	3	3	3	3	3	4	2	3
Fleet 4	P&H4100	3	4	4	4	4	5	3	4
Trucks									
Fleet 1	A40F	6	9	7	9	7	8	6	9
Fleet 2	777G	6	9	7	9	7	8	6	9
Fleet 3	960E	10	15	13	16	13	15	11	16
Fleet 4	960E_2	16	29	24	30	23	27	21	29
Ancillary Plant									
Dozer	D475A	4	5	4	5	5	6	4	6
Dozer	D375A	5	7	6	7	6	8	5	7
Dozer	WD900	3	4	4	4	4	6	3	5
Dozer	WD600	4	6	5	6	5	7	4	6
Grader	16M	5	7	6	7	6	8	5	7
Grader	24M	5	7	6	7	6	8	5	7
WaterBowser	HD785-7WT	3	4	3	4	3	4	3	4
Excavator	PC800	1	2	1	2	2	2	1	2
Loader	WA600	1	1	1	1	1	1	1	1
Cable Reeler	WA600-CABLE	2	3	3	3	3	4	2	3
Rockbreaker	Volvo 120L	3	4	4	4	4	5	3	4
Wash Truck	WS	1	2	1	2	2	2	1	2
GET Truck	GT	1	2	1	2	2	2	1	2
Grease Truck	GS	1	1	1	1	2	2	1	2

In addition to the items listed in Table 16-8, other mining related equipment for the Project includes:

- telescopic handlers
- crew buses for mine operations and maintenance personnel
- vibratory compactors for mine road construction and maintenance
- cable reelers
- 150 t and 250 t lowbed trailers and prime movers
- pit shovel motivator
- lighting towers
- 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

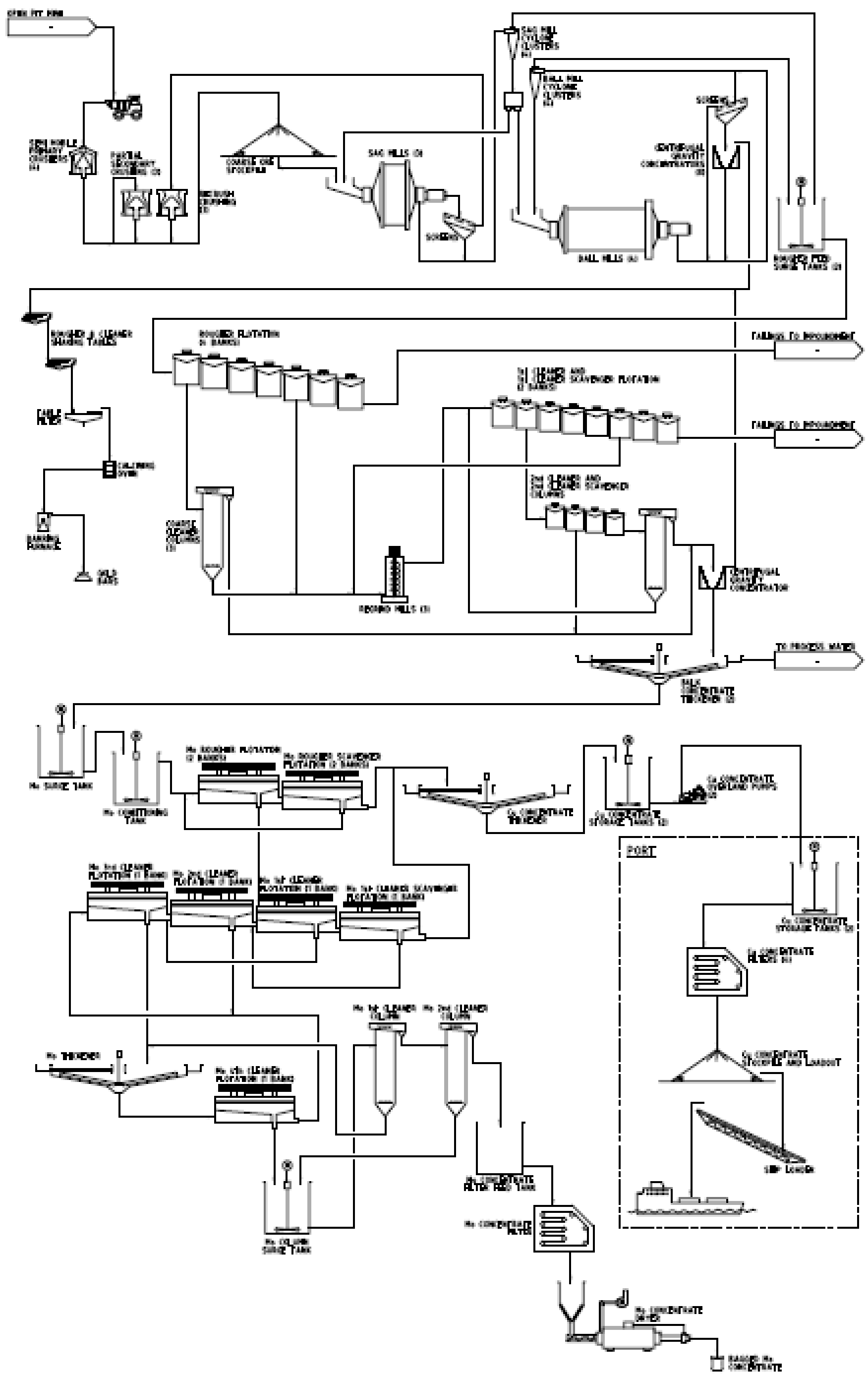
17.2 Process description

in 2018, in the year following start-up, the concentrator will treat nominally 190,000 t/day of ore supplied from the Botija Pit; this rate will increase up to 203,600 t/day by 2021. 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 0.65% Cu and 0.023% Mo. These levels are higher than the highest sustained head grades of 0.48% Cu and 0.008% Mo scheduled to be mined in 2027 and 2021, 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 primary gyratory crushers will be semi-mobile in-pit installations, as described in Item 16.

Crushed ore will be conveyed out of the pit to a surface transfer point, and thence by overland conveyor where it will discharge into either secondary crusher feed bins or bypass direct to a coarse ore stockpile at the concentrator. Provision will be made at the transfer point to accept mill feed from future crushed ore 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 feeders and conveyors will draw ore from below the coarse ore stockpile and feed two parallel wet-grinding lines, each consisting of a semi-autogenous grinding (SAG) mill and two ball mills, all equipped with gearless drives. A third train of 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.

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. The pebble crushing circuits will include pebble bins, cone crushers, and a bypass arrangement. Crushed pebbles will return to the SAG mills via the stockpile feed conveyors. The pebble crushing plant is to be located adjacent to the secondary crushers.

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 a surge tank prior to pumping to the flotation area.

Linked to the ball mill circuit will be a gravity gold recovery plant. A proportion of the ball mill discharge will be pumped to a gravity gold circuit 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, and gold 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 regrind mills, and then cleaned in two stages of mechanical cells followed by a one column stage to produce a final bulk concentrate.

From 2025 an increase in rougher capacity will be required to accommodate the increase in throughput, but the amount of copper will be the same; therefore, no change to the existing downstream regrind and cleaning capacity will be needed. The circuit has been designed to accomodate these additional requirements.

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.

17.2.3 Concentrates

The molybdenum concentrate will be filtered, dried, and packaged in bulka bags 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 port site.

If the molybdenum head grade is very low, the molybdenum separation plant can be bypassed.

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 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.

17.2.4 Tailings disposal and process water reclaim

For the first twenty years of the operation, tailings containing silicate, iron sulphide and other minerals from the rougher and cleaning steps will be deposited in the TMF located north of the mine and plant.

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.

In the original Inmet concepts, and after twenty 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 the depleted Colina Pit.

The timeframes for the tailings backfill will be revised as a function of new and updated mining and processing schedules produced when production commences.

Flyash run-off liquor from the power plant will also be deposited into the TMF after being pumped up from the port site..

Reclaim water will be pumped from the TMF to the process water pond 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 pond, water will either be used in the milling plant, or boosted for general in-plant use. Excess water from the TMF will be overflowed via a decant tower 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.

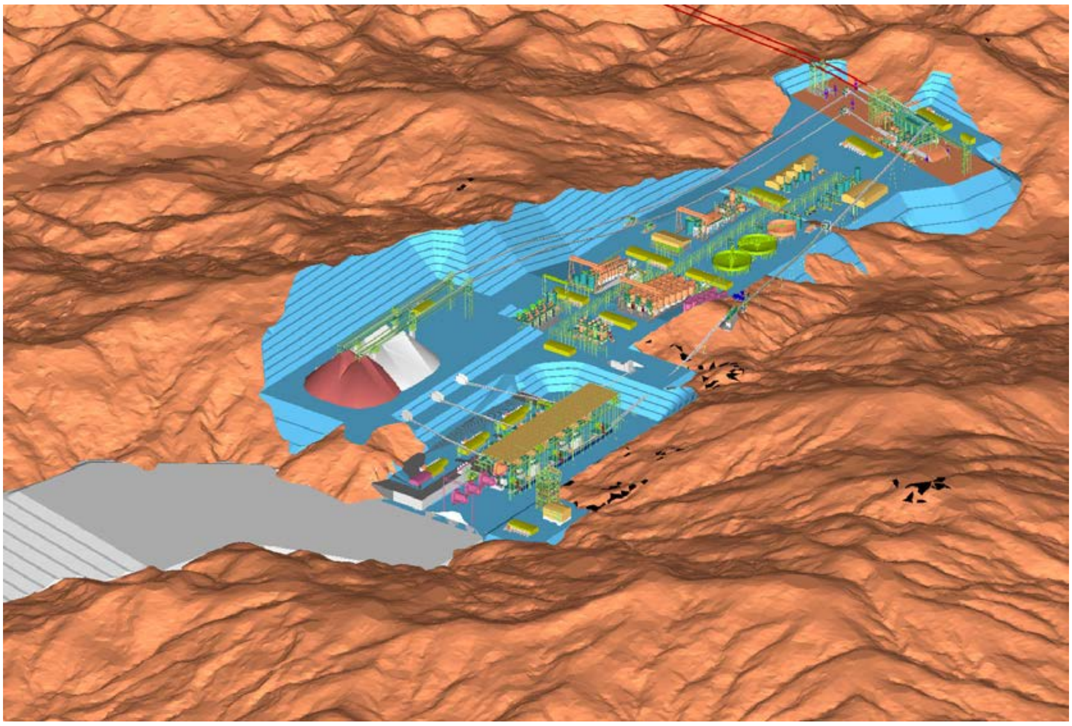
ITEM 18 PROJECT INFRASTRUCTURE

The following information is largely reproduced from the 2010 ESIA (Golder, 2010) with updates reflecting changes to the Project scale and scope following FQM's acquisition of the Project.

18.1 Plant site facilities

The plant site location has changed from that proposed by Inmet and described in the 2010 ESIA. Figure 15-34 shows the new location, whilst Figure 18-1 shows a schematic arrangement of the plant site.

Figure 18-1 Plant site schematic



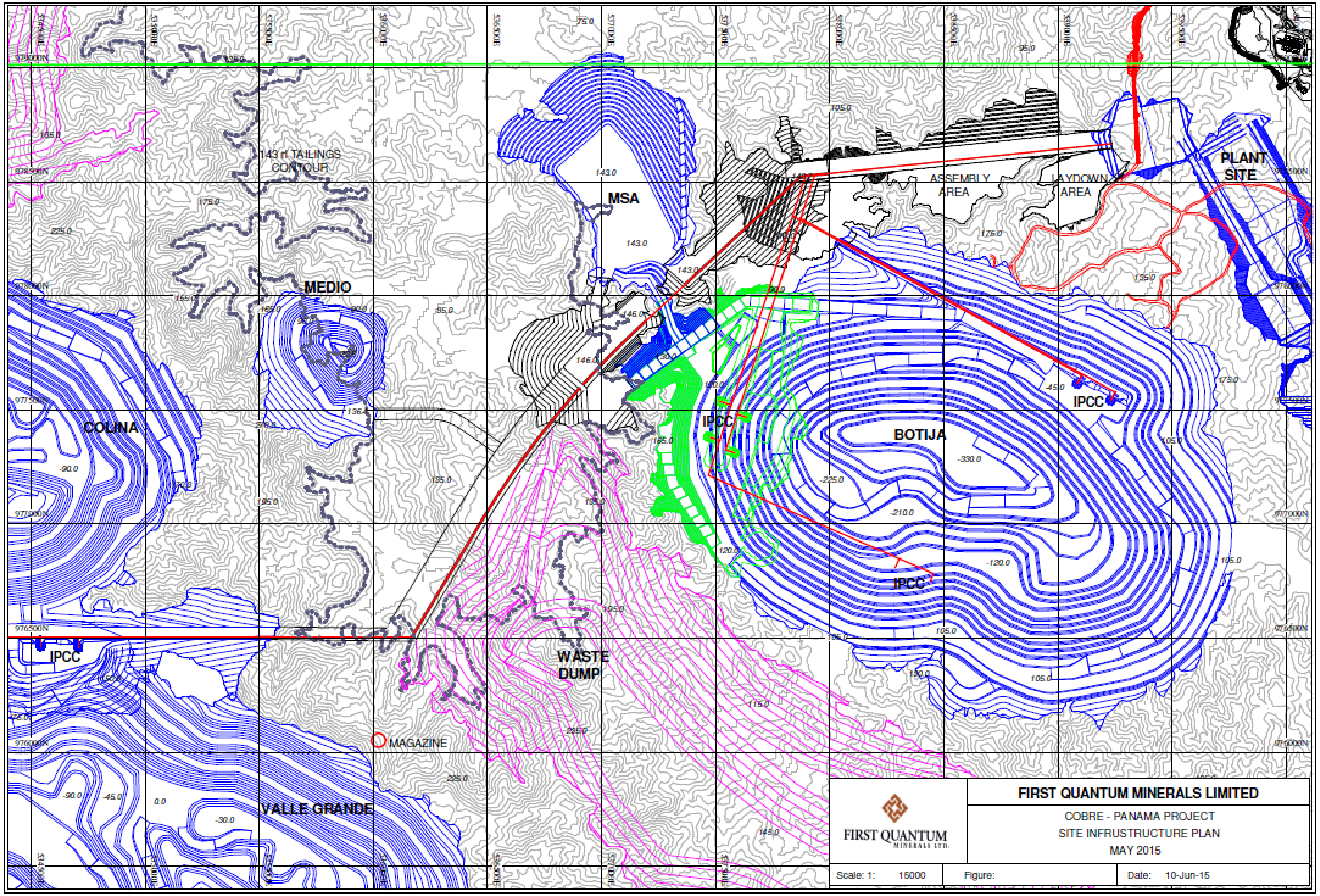
18.2 Mining facilities

The mining facilities include the mobile equipment workshops (ie, the mine services area or MSA), and the explosives preparation and storage magazines. Figure 18-2 shows the locations of these facilities, in addition to:

- the location of the initial equipment assembly area (required before completion of the MSA)
- the location of the equipment laydown area
- 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

Fuel storage and dispensing facilities will be located at the MSA.

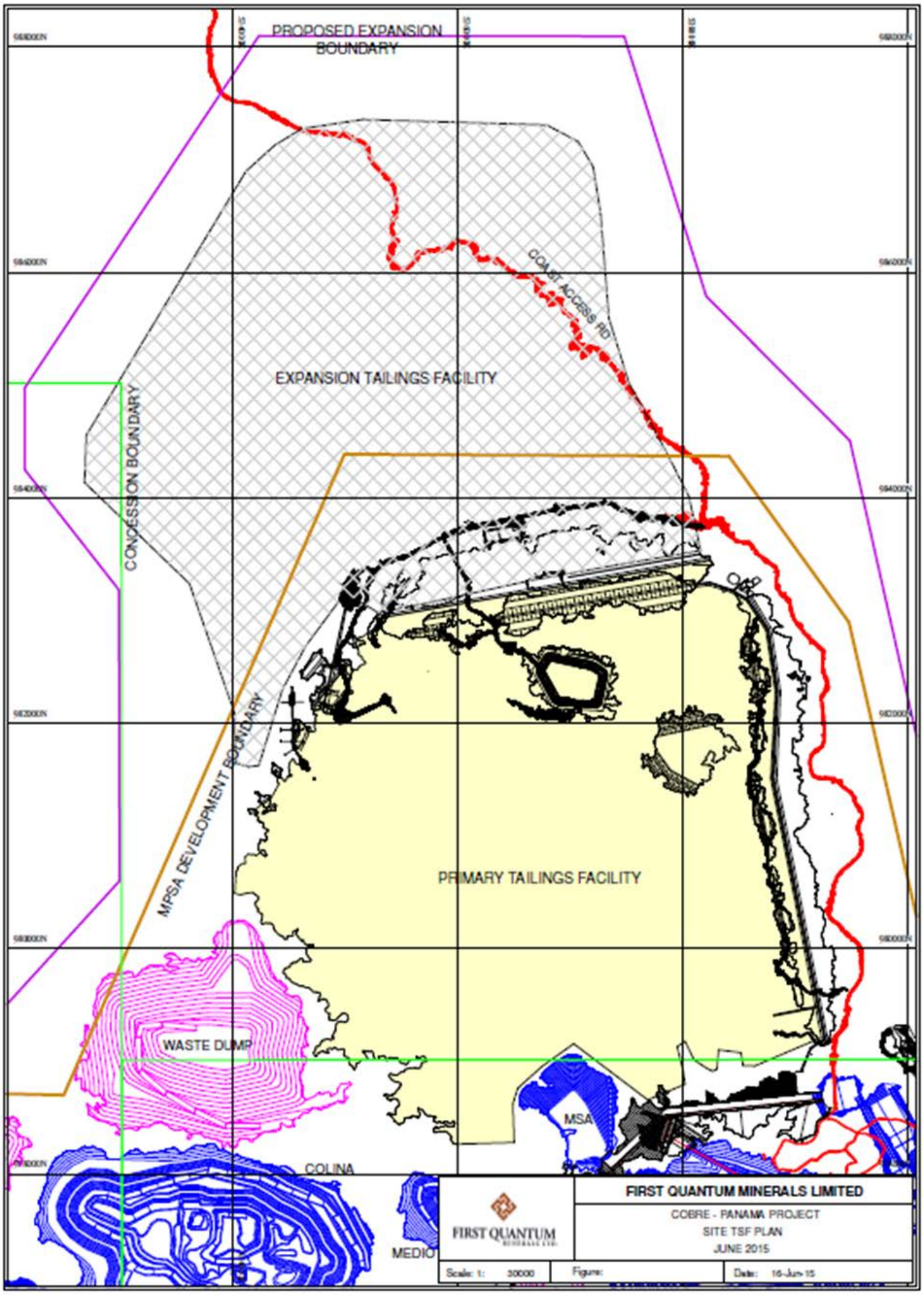
Figure 18-2 Mining facilities



18.3 Tailings management facility site

Whilst the location of the tailings management facility (TMF) has not changed, details of the design have been optimised from the arrangement devised by Inmet. Figure 18-3 shows the footprint of the new arrangement.

Figure 18-3 TMF layout plan



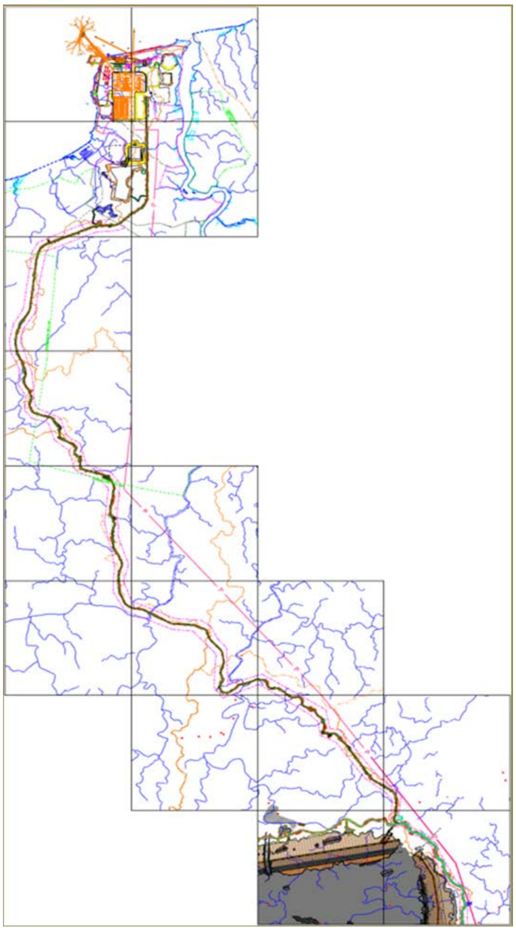
18.4 Port site

The port site is located at Punta Rincón on the Caribbean Sea (Figure 18-4) and will include 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 about 25 km.

Coal will be received at the port into a discharge hopper on shore. From there, the coal will be conveyed (under cover) to storage stockpiles at the power plant.

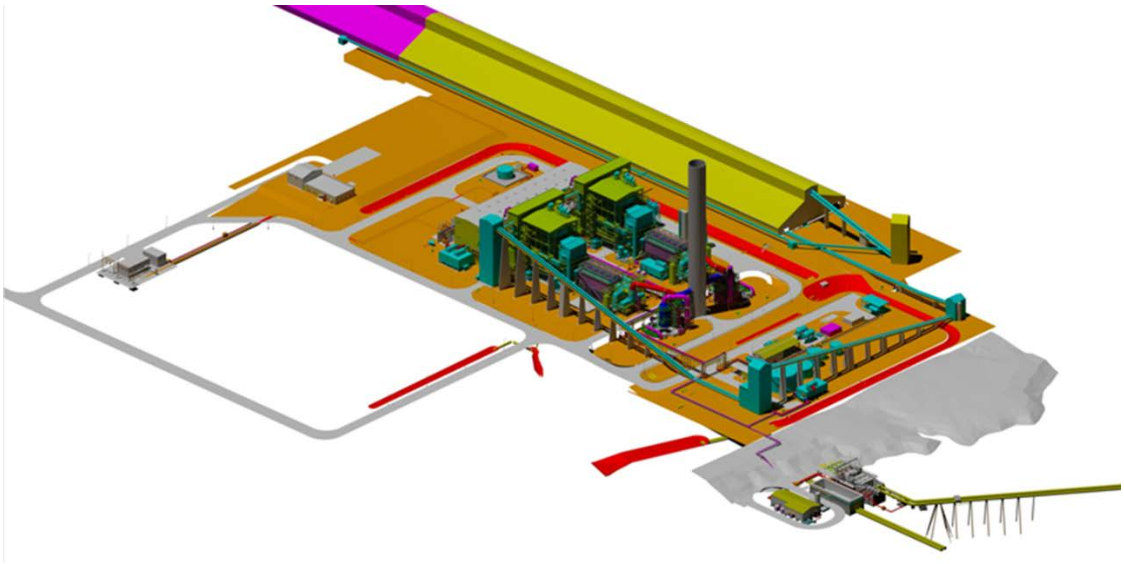
Figure 18-4 Port site location



18.5 Power station

The coal fired power station at the port site will operate continuously, 24 hours per day. The power station facilities at the port site will include two 150 MW coal-fired generators (with provision for future additional trains). A schematic arrangement is shown in Figure 18-5.

Figure 18-5 Power station schematic



18.6 Site access road

There will be three access roads serving the Project site:

1. The existing road accesses the site from the south, at a turnoff from the Pan-American Highway at Penonomé.
2. The new site access road constructed between the port site and the plant site.
3. A new eastern access road (being constructed) to replace (1) above which will not be usable once mining commences.

18.7 Power transmission

A 230 kV power transmission line will be erected between the power station at the port and the mine site. There will be interconnections to the plant switchyard and onward to the Llano Sanchez substation (ie, the connection point to the national grid).

18.8 Accommodation facilities

Temporary accommodation facilities will be built for the construction phase along with permanent facilities for the operations workforce.

ITEM 19 MARKET STUDIES AND CONTRACTS

19.1 Agreements for sale of concentrate

A copper concentrate marketing study was prepared for Inmet in September 2009. The study document provided a view on the then prevailing copper price projections, metal supply and demand trends, likely treatment and refining charges (TCRC's), freight rates and product marketability.

With the exception of marketability, much of the reported information is now out-dated and no further marketing studies have been undertaken.

Due to the volatility of the concentrate markets, it is too soon to progress beyond preliminary shortlisting of potential customers and the development of a marketing strategy. To that end, FQM has:

- commenced in January 2013, a marketing strategy involving first round meetings with nine international groups (ie, from which detailed term sheet negotiations can commence with selected smelters);
- received stated interest from several of these groups; and
- reviewed, from two further groups, letters of interest entered into with Inmet (ie, with follow up negotiations yet to be carried out).

In relation to molybdenum concentrate, a now out-dated marketing study was also prepared for Inmet in 2009. The current marketing strategy includes meetings with potential customers commencing in 2013.

19.2 Product marketability

Indications from the September 2009 marketing study are that, based on Project metallurgical information, the product appears to 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.

As part of the 2013 marketing strategy and arising from the first round meetings, it is likely that further confirmatory assays will be required by some of the smelting groups in order to alleviate any concerns regarding potential concentrate contaminants.

19.3 Other contracts and agreements

The supply of thermal coal into the Project, for power generation, will be a major contract and one which is yet to be negotiated. Other than a supply analysis completed for Inmet in November 2011, no further work on a coal procurement strategy and eventual supply agreement has commenced.

The 2011 analysis did identify that USA and Colombian supply options could be attractive due to delivered price and relative proximity.

The supply of diesel fuel into the Project, as another major contract, is also yet to be assessed and negotiated.

Contracts for the supply of primary mining equipment are in progress.

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. Discussions are in progress to effect changes to the existing security and reporting requirements of this agreement.

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 Rose et al (March 2013), with updates authored by Michael Lawlor (QP) and reviewed by Alberto Casas, Environmental Director of MPSA.

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) on 28th December 2011 for the Project as then envisaged.

The approved Project was for a mining operation comprising three open pits, at Botija, Colina and Valle Grande.

Since then, and prior to and following the acquisition of Inmet by FQM, 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, ie at Balboa and Botijo Abajo/Brazo
- formation of additional waste rock storage facilities and realignment of approved facilities
- changes in material routing and scheduling in the mine plan
- a production rate increase
- expansion of the tailings management facility
- increased site clearing for development of overland conveyors connecting to in-pit crushers

The expected timeframe for submitting the new ESIA for approval is the end of 2016. Under the new Ministry of Environment Category III, an ESIA has to be reviewed within six months of submittance.

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
- 30 km of road between the mine and the port

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

Through the construction and operational phases, these EMS plans will be updated to reflect changes in site and operational conditions.

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.

20.4 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, SO2, 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.

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 up to 75% of 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 an 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 2 hours.

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 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 rural electrification.

20.6 Resettlement

To date, there have been three community resettlement projects involving two indigenous communities at Petaquilla and Chicheme, and 46 farming families. For all but nine of the 46 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 has followed the International Finance Corporation Performance Standards. All 42 Chicheme families have been relocated to a new community, whilst seven of the fourteen Petaquilla families have been relocated and the other seven will be relocated during 2015. The resettlement plan has provided for the construction of entirely new communities, including new houses (house for house was replaced), land title for the new lands (land for land was replaced) and compensation for the farms and crops belonging to each family.

Following compensation and resettlement, all 46 families have participated in a three year resettlement monitoring programme and are exceeding development goals. As part of the three year programme, MPSA is assisting each family with the development of new subsistence farming areas. The new communities now have a school, a communal meeting house and sports fields. Each house has a water supply, sewerage and solar panels installed.

20.7 Land

Parallel to the environmental/social studies and permitting requirements, access to land has been part of the rights to develop the Project, mainly by means of the following:

- acquisition of titled private properties and possession rights (1,475 ha), plus coastal national land (70 ha) at the Port Site,
- acquisition (via lease) of 7,477.54 ha of national land at the Mine Site,
- easement rights for a number of roads over some 70 km (Molejón bypass; La Pintada bypass; Llano Grande road; Eastern Access road),
- easement right for 115 km of the 230 kV transmission line and a landplot for substation at Llano Grande, and
- acquisition of 1,150 ha as repository land for resettlement of two indigenous communities and 2,140 ha for resettlement of campesinos.

20.8 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 will be decommissioned, demolished and/or removed. Diverted water courses are to be reinstated where possible and roads that are no longer required will 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.

Mine closure provisions are listed in Item 21.1.2.

ITEM 21 CAPITAL AND OPERATING COST ESTIMATES

21.1 Capital costs

21.1.1 Project development capital

Table 21-1 lists the development capital costs included in the Project cashflow model of Item 22. Included in the total $6,425 million estimated capex is $913 million that was incurred prior to FQM's acquisition of the Project.

Table 21-1 Development capital cost provisions

Capital Item	Incurred Pre-Acquisition (US$M)	Incurred at 30/12/2014 (US$M)	Capex Spend 2015 to 2018 (US$M)	Total Capex Spend (US$M)	Expansion Capex Spend (US$M)
Mine, Port & Infrastructure	480.0	78.2	1,465.0	2,023.2	
Process Plant	62.0	175.3	1,061.6	1,298.9	500.0
Power Plant	209.0	115.6	339.5	664.1	415.0
Indirect Cost	162.0	912.6	1,364.2	2,438.8	
Contingency	-	-	-	-	
Total Project	**913.0**	**1,281.7**	**4,230.3**	**6,425.0**	**915.0**

Included within the capital spend for the mine, port and infrastructure is the cost of primary mining equipment ($340.5M) and mining support related equipment ($148.5M).

21.1.2 Mine closure provisions

Table 21-2 lists the closure cost provisions included in the Project cashflow model for the final years of operational life. These costs provide for rehabilitation of the entire site, including mining areas, TMF, access roads and the port site.

Table 21—2 Closure cost provisions

Description	Unit	Quantity	Av. Unit Cost (US$)	Total (US$)	Year 35 2051	Year 36 2052	Year 37 2053	Year 38 2054	Year 39 2055	Year 40 2056
Direct Costs										
Surface Preparation	ha	631.9	$18,705	$11.8	$2	$2	$2	$3	$3	$0
Capping	ha	1,895.8	$15,401	$29.2	$5	$4	$6	$9	$7	$0
Revegetation	ha	631.9	$16,501	$10.4	$2	$1	$2	$3	$2	$0
Reforestation	ha	1,263.9	$6,601	$8.3	$1	$1	$2	$2	$2	$0
Demolition	ls	1.0	$16,000	$0.0	$0	$0	$0	$0	$0	$0
Misc. reclamation costs	ls	1.0	$231,000	$0.2	$0	$0	$0	$0	$0	$0
Post closure monitoring	yr	5.0	$334,001	$1.7	$0	$0	$0	$0	$0	$2
Subtotal				**$61.7**	**$9.5**	**$8.0**	**$11.5**	**$17.4**	**$13.7**	**$1.7**
Indirect Costs										
Employee indemnification	ls	1.0	$1,300,000	$1.3	$0.2	$0.2	$0.2	$0.2	$0.2	$0.2
Administration	ls	1.0	$9,255,516	$9.3	$1.5	$1.5	$1.5	$1.5	$1.5	$1.5
Final closure plan preparation	ls	1.0	$75,000	$0.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Consultant services	ls	1.0	$50,000	$0.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Contractor mobilisation and demobilisation	ls	1.0	$40,000	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Contingency (10% of total direct costs)	ls	1.0	$6,170,344	$6.2	$1.0	$1.0	$1.0	$1.0	$1.0	$1.0
Subtotal				**$16.9**	**$2.8**	**$2.8**	**$2.8**	**$2.8**	**$2.8**	**$2.8**
TOTAL				**$78.6**	**$12.3**	**$10.8**	**$14.3**	**$20.2**	**$16.5**	**$4.5**

The closure cost estimates were prepared by MPSA in 2013 and are built up in detail from a comprehensive table of areas to be rehabilitated, multiplied by unit costs ($/ha) for surface preparation, loading/hauling of capping material, placement and revegetation. The estimates also include provision for reclamation of land fill areas and borrow pits/trenches, plus administrative costs. Whilst they may be reviewed and updated in time, given the relative magnitude of these costs and their timeframe for expenditure, they have been accepted as is for inclusion in the cashflow model for this Technical Report.

21.2 Operating costs

21.2.1 Processing and G&A costs

Project operating costs have been 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 21-3 and 21-4 for processing and G&A costs, respectively. The estimates are shown in unit cost terms, for the purposes of mine planning and for cashflow modelling to support the Mineral Reserves estimate.

Table 21—3 Process operating costs

Processing costs	Consumption rates		Unit costs		$'000	$/t
Grinding media					**2,699,923**	**0.81**
SAG mill	0.40	kg/t milled	0.94	$/kg	1,257,163	0.38
Ball mill	0.38	kg/t milled	0.94	$/kg	1,194,462	0.36
Regrind mill	0.08	kg/t milled	0.94	$/kg	248,298	0.07
Liners					**831,360**	**0.25**
Gyratory crusher mantle liners	0.00	kg/t milled	4.43	$/kg	12,543	0.00
Gyratory crusher concave liners	0.00	kg/t milled	3.38	$/kg	11,879	0.00
Pebble crusher mantle liners	0.00	kg/t milled	4.25	$/kg	17,312	0.01
Pebble crusher bowl liners	0.00	kg/t milled	4.31	$/kg	16,654	0.00
SAG Mill liners	0.05	kg/t milled	3.38	$/kg	574,185	0.17
Ball Mill Liners	0.02	kg/t milled	2.87	$/kg	198,788	0.06
Reagents					**1,985,662**	**0.59**
Sodium Hydrosulphide (NaHS)	10.17	kg/t dry concentrate	1.16	$/kg (0.0137 t conc / t ore)	538,207	0.16
Lime	0.99	kg/t ore	0.22	$/kg	727,511	0.22
Flotation Frother (MIBC)	0.03	kg/t ore	3.23	$/kg	299,856	0.09
Collector (SIPX)	0.03	kg/t ore	1.88	$/kg	195,659	0.06
Promoter (A3302)	0.01	kg/t ore	4.93	$/kg	165,661	0.05
Fuel Oil	0.03	kg/t dry concentrate	0.74	$/kg (0.0137 t conc / t ore)	955	0.00
Flocculant Plant and Port (AF303)	0.07	kg/t dry concentrate	3.03	$/kg (0.0137 t conc / t ore)	9,431	0.00
NaOH	0.00	kg/t ore	1.91	$/kg	885	0.00
Carbon Dioxide (CO2)	0.04	kg/t dry concentrate	0.48	$/kg (0.0137 t conc / t ore)	835	0.00
Anti-Scalant	0.01	kg/t ore	1.74	$/kg	46,662	0.01
Plant general operating supplies			0.05	$/t av. assumption	**167,606**	**0.05**
Electricity					**4,022,544**	**1.20**
Mill	20.60	kWh/t milled	0.055	$/kWh	3,809,858	1.14
Site & Services	0.45	kWh/t milled	0.055	$/kWh	83,225	0.02
Port	0.70	kWh/t milled	0.055	$/kWh	129,461	0.04
Maintenance parts			0.68	$/t av. assumption	**2,279,442**	**0.68**
Mobile equipment			0.10	$/t av. assumption	**319,835**	**0.10**
Labour	34.0	$M/annum	100.0	Mtpa	**1,139,721**	**0.34**
Assay laboratory	1.1	$M/annum	100.0	Mtpa	**36,088**	**0.01**
Other to tie to Basic Engineering						
Subtotal, plus						**4.02**
sustaining capex allowance	606.7	$M	3,352.1	Mt		**0.18**
power cost credit						
			TOTAL PROCESS OPERATING COST			**4.20**

In the MPSA estimates, an additional power cost was included for provision of excess power back into the national grid. In the updated cost estimates, in circumstances of increased plant throughput and power consumption, this cost has been removed.

Further process operating and G&A cost estimates were completed in January 2015 to reflect updated estimates of:

- consumption rates for grinding media, liners and reagents
- electrical power consumption rates
- annual labour costs
- annual assay costs

- unit costs for grinding media, liners and reagents, electrical power and maintenance parts

The sustaining capital allowance was also increased to 5% of the operating and G&A costs. The impact of these updates is a revision of the processing operating costs from \$4.20/t milled to \$3.92/t milled, and a revision of the G&A costs from \$1.25/t milled to \$0.85/t milled (on the basis of maintaining the same fixed costs for the increased plant throughput).

Table 21—4 G&A operating costs

General & Administration Costs	$M/annum	Consumption rates		Unit costs		$'000	$/t
Finance						**1,249,000**	**0.37**
Labour	4.41					176,322	0.05
Services	3.16					126,375	0.04
Insurance	18.55					742,159	0.22
Other Operating Costs	5.10					204,144	0.06
External relations						**121,346**	**0.04**
Labour	1.16					46,259	0.01
Services	0.89					35,532	0.01
Other Operating Costs	0.99					39,555	0.01
Environment						**373,091**	**0.11**
Labour	1.39					55,645	0.02
Services	7.52					300,685	0.09
Other Operating Costs	0.42					16,761	0.01
Security						**288,282**	**0.09**
Labour	0.64					25,476	0.01
Services	6.57					262,806	0.08
Human Resources						**550,083**	**0.16**
Labour	3.35					134,085	0.04
Services	8.90					355,995	0.11
Other Operating Costs	1.50					60,003	0.02
Executive						**313,089**	**0.09**
Labour	3.28					131,068	0.04
Mobile Equipment	0.58					23,130	0.01
Services	0.71					28,493	0.01
Other Costs to tie to Basic Engineering	2.49					99,558	0.03
Other Operating Costs	0.77					30,840	0.01
Site Services						**1,001,949**	**0.30**
Labour	11.73					469,297	0.14
Mobile Equipment	2.45					97,882	0.03
Fixed Equipment	3.46					138,443	0.04
Mine Allocation	4.11					164,589	0.05
Marine Services	3.29					131,738	0.04
Other to tie to basic engineering							
Power						**152,389**	**0.05**
G&A		0.8	kWh/t milled	0.055	$/kWh	115,206	0.03
Site services		0.3	kWh/t milled	0.055	$/kWh	37,183	0.01
Subtotal, plus							**1.20**
sustaining capex allowance		181.0	$M	3,352.1	Mt		**0.05**
power cost credit							
				TOTAL G&A OPERATING COST			**1.25**

21.2.2 Mining costs

Variable mining costs comprising drill, blast, load and haul costs, on a bench by bench basis, were derived by MPSA in 2013. 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.

The 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 21-5), and
- an incremental mining cost, comprising haulage beyond the reference bench level, and to respective ore tipping and waste dumping destinations (Table 21-5).

Table 21—5 Base and average incremental mining costs

Cost Item	Mining Costs - with IPCC			
	Waste ($/t)	Ore ($/t)	Total ($/t)	Prop'n (%)
Management	$0.00	$0.00	**$0.00**	**0.0%**
Operations O/H	$0.05	$0.04	**$0.04**	**2.2%**
Engineering	$0.02	$0.02	**$0.02**	**1.1%**
Geology	$0.03	$0.02	**$0.03**	**1.3%**
Technical Services	$0.01	$0.01	**$0.01**	**0.3%**
Drilling	$0.16	$0.14	**$0.15**	**7.2%**
Blasting	$0.37	$0.32	**$0.34**	**16.8%**
Loading	$0.18	$0.16	**$0.17**	**8.3%**
Hauling to Reference Bench Level	$0.65	$0.61	**$0.63**	**31.1%**
Stockpile Rehandle	$0.03	$0.03	**$0.03**	**1.6%**
Services	$0.20	$0.18	**$0.19**	**9.3%**
Dewatering	$0.08	$0.07	**$0.07**	**3.6%**
Contract Clearing & Pre-stripping	$0.18	$0.16	**$0.17**	**8.2%**
Reclamation	$0.01	$0.01	**$0.01**	**0.4%**
Subtotal Base Mining Cost	**$1.98**	**$1.77**	**$1.86**	**91.3%**
Average Incremental Haulage	$0.22	$0.15	**$0.18**	**8.7%**
Total Average Mining Cost	**$2.20**	**$1.92**	**$2.04**	**100.0%**

Table 21—5 indicates that approximately 40% of the total average mining costs relate to haulage. Table 21-6 shows the calculation basis for the incremental haul cost factors, for uphill and downhill hauls.

Table 21—6 Incremental haul costs

Item	Units	Up hill loaded	Down hill loaded
Hourly costs			
Hauling Labour Cost	$/hour	$26.02	$26.02
Contract Services Cost	$/hour	$0.00	$0.00
Diesel Cost	$/hour	$206.63	$206.63
Antifreeze, Oil, and Grease Cost	$/hour	$5.31	$5.31
Other Fuels And Lubricants Cost	$/hour	$0.00	$0.00
Tyre Cost	$/hour	$88.51	$88.51
Maintenance Parts & Supplies Cost	$/hour	$105.87	$105.87
MARC Fee	$/hour	$2.52	$2.52
Auxiliary Equipment Cost	$/hour	$3.52	$3.52
Other Operating Costs	$/hour	$0.00	$0.00
Subtotal Cost	**$/hour**	**$438.38**	**$438.38**
Truck cycle time parameters			
up hill loaded	km/hour	10	20
down hill empty	km/hour	25	25
Ramp grade	%	10	10
Bench Height	m	15	15
Cycle time calculation for one bench			
up hill loaded	minutes	0.90	0.45
down hill loaded	minutes	0.36	0.36
total cycle time	minutes	1.26	0.81
Incremental mining cost estimate			
Cost for haulage truck	$/t/bench	0.032	0.020
Additional road maintenance	%	10	10
Total Cost	**$/t/bench**	**0.035**	**0.023**

The total mining cost estimate at each mining bench level was then determined in consideration of reference bench elevations beyond which a varying incremental cost would be incurred. 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, and no consideration was made for trolley-assisted haulage. Table 21-7 lists the total mining costs at varying pit levels, the algorithms for calculating which are as follows:

- 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

Table 21—7 Total mining costs varying by depth

Mining RL (m)	Incremental Mining Cost		Base Mining Cost		
	Up hauls	Down hauls	Waste	Ore	Total
	($/t/bench)	($/t/bench)	($/t)	($/t)	($/t)
210 to -330	0.035	0.023	$1.98	$1.77	$1.86
Mining RL (m)	**Reference Elevation**		**Total Mining Cost**		
	Waste RL	**Ore RL**	**Waste**	**Ore**	**Total**
	(m)	**(m)**	**($/t)**	**($/t)**	**($/t)**
210	125	120	$2.11	$1.91	$2.11
195	125	120	$2.08	$1.88	$2.08
180	125	120	$2.06	$1.86	$2.06
165	125	120	$2.04	$1.84	$2.02
150	125	120	$2.02	$1.81	$1.99
135	125	120	$1.99	$1.79	$1.96
120	125	120	$1.99	$1.77	$1.93
105	125	120	$2.02	$1.80	$1.95
90	125	120	$2.06	$1.84	$1.96
75	125	120	$2.09	$1.87	$1.98
60	125	50	$2.13	$1.78	$1.94
45	125	50	$2.16	$1.78	$1.96
30	125	50	$2.20	$1.81	$1.98
15	125	50	$2.23	$1.85	$2.01
0	125	50	$2.27	$1.88	$2.02
-15	125	50	$2.30	$1.92	$2.06
-30	125	50	$2.34	$1.95	$2.09
-45	125	50	$2.37	$1.99	$2.13
-60	125	50	$2.41	$2.02	$2.15
-75	125	-60	$2.44	$1.80	$2.02
-90	125	-60	$2.48	$1.84	$2.02
-105	125	-60	$2.51	$1.87	$2.05
-120	125	-60	$2.55	$1.91	$2.07
-135	125	-60	$2.58	$1.94	$2.06
-150	125	-60	$2.62	$1.98	$2.09
-165	125	-60	$2.65	$2.01	$2.14
-180	125	-60	$2.69	$2.05	$2.17
-195	125	-60	$2.72	$2.08	$2.21
-210	125	-60	$2.76	$2.12	$2.25
-225	125	-60	$2.79	$2.15	$2.28
-240	125	-60	$2.83	$2.19	$2.31
-255	125	-60	$2.86	$2.22	$2.35
-270	125	-60	$2.90	$2.26	$2.32
-285	125	-60	$2.93	$2.29	$2.34
-300	125	-60	$2.97	$2.33	$2.35
-315	125	-60	$3.00	$2.36	$2.38
-330	125	-60	$3.04	$2.40	$2.40
Grand Total		**Average**	**$2.20**	**$1.92**	**$2.04**

Although Table 21-5 shows a small allowance for ore rehandling (in this case assumed to be ROM Pad reclaim over a short tramming distance), an additional cost was included in the Reserves scheduling process and cashflow modelling to account for longer term ore stockpile rehandling to suitably located IPC positions. An allowance of $1.30/t ore reclaimed was derived from the costs listed in Table 21-5 (ie, by deletion of drill/blast costs).

With the completion of the detailed mine design layouts described in Item 15, inclusive of modelled ore haul profiles to IPC installations within each pit, and waste hauls to designed dumps, the mining cost estimates were updated in Q1 2015.

The base mining cost for ore was reduced from $1.77/t as shown in Table 21-6, to $0.30/t, by including only management, overheads and labour costs. The large variable cost component in Table 21-5 for loading and hauling was substituted into the incremental cost calculations. In the same way, the base mining cost for waste was reduced from $1.98/t to $0.30/t. Table 21-8 shows the revised algorithms used for calculating the incremental mining costs for each pit.

The equivalent overall average mining costs are shown in Table 21-9. 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—9 shows the unadjusted overall average mining costs derived from the new incremental relationships, in addition to adjusted overall 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—8 Revised mining costs ($/t) relationship, varying by depth

	Waste	Ore
All Pits	2.1732 - (Zelev * (0.0403/15))	2.0535 - (Zelev * (0.0412/15))
Botija	2.3223 - (Zelev * (0.0403/15))	2.2480 - (Zelev * (0.0412/15))
Colina	2.0939 - (Zelev * (0.0403/15))	1.9213 - (Zelev * (0.0412/15))
Medio	1.9804 - (Zelev * (0.0403/15))	2.2450 - (Zelev * (0.0412/15))
Valle Grande	2.1604 - (Zelev * (0.0403/15))	1.9934 - (Zelev * (0.0412/15))
Babr	2.2156 - (Zelev * (0.0403/15))	2.2877 - (Zelev * (0.0412/15))
Balboa	2.0508 - (Zelev * (0.0403/15))	2.0971 - (Zelev * (0.0412/15))

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—9 Revised overall average mining costs

Pit	Units	Unadjusted total cost	Adjusted for trolley-assist		
			Haulage cost	TA saving	Total cost
BOTIJA	$/t waste	2.29	0.92	0.23	2.06
	$/t ore	2.33	0.93	0.12	2.21
	$/t total	2.31	0.92	0.35	1.96
COLINA	$/t waste	1.9	0.76	0.19	1.71
	$/t ore	1.91	0.76	0.10	1.81
	$/t total	1.91	0.76	0.29	1.62
MEDIO	$/t waste	1.6	0.64	0.16	1.44
	$/t ore	1.98	0.79	0.10	1.88
	$/t total	1.75	0.70	0.26	1.49
VALLE GR/	$/t waste	1.77	0.71	0.18	1.59
	$/t ore	1.67	0.67	0.08	1.59
	$/t total	1.72	0.69	0.26	1.46
BABR	$/t waste	1.92	0.77	0.19	1.73
	$/t ore	2.04	0.82	0.10	1.94
	$/t total	1.98	0.79	0.29	1.69
BALBOA	$/t waste	1.93	0.77	0.19	1.74
	$/t ore	2.09	0.84	0.10	1.99
	$/t total	1.99	0.80	0.30	1.69
ALL PITS	**$/t waste**	**1.99**	**0.80**	**0.20**	**1.79**
	$/t ore	**1.98**	**0.79**	**0.10**	**1.88**
	$/t total	**1.98**	**0.79**	**0.30**	**1.68**

Table 21-10 lists the concentrate charges and payable percentages adopted for the pit optimisation described in Item 15. This metal cost information was adopted from information available in 2014. Table 21-10 also lists updated information that was available in Q1 2015 following a cost review by Metal Corp Trading AG.

The updated estimates following the 2015 review were:

- 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 (checked against WoodMackenzie/AME data)
- inclusion of a copper removal charge from the molybdenum concentrate (previously omitted)
- inclusion of a molydenum 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

Table 21—10 Concentrate charges

	2014	2015
Copper Concentrate		
Cu con - Cu grade, %	25.00%	28.00%
Cu con overland freight, $/dmt	$0.00	$2.75
Cu con ocean freight, $/dmt	$40.00	$41.10
Cu con treatment, $/dmt	$70.00	$95.00
Cu refining, $/lb payable	$0.07	$0.090
Au refining, $/oz payable	$5.50	$5.50
Ag refining, $/oz payable	$0.40	$0.40
Cu payable, %	96.43%	96.43%
Au payable, %	92.00%	86.00%
Ag payable, %	90.00%	80.00%
Cu metal cost, $/lb Cu payable	**$0.277**	**$0.323**
Molybdenum Concentrate		
Mo con - Mo grade, %	52.00%	52.00%
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 Cu	$0.00	$0.69
Mo refining, $/lb payable	$0.00	$0.55
Mo payable, %	86.20%	86.20%
Mo metal cost, $/lb Mo payable	**$0.091**	**$1.334**

The Panamánian government levies a 5% royalty on base metals recovered and 4% royalty on precious metals recovered[9]. From the above table, therefore:

- The copper metal cost:
  = metal price x royalty + Cu metal cost
  = $3.00 x 5% + $0.277 = $0.43/lb Cu (according to the 2014 optimisation inputs)
  = ($3.00 x 96.43% - $0.323) x 5% + $0.323 = $0.45/lb Cu (according to the 2015 updates for the cashflow model)
- The molybdenum metal cost:
  = metal price x royalty + Mo metal cost
  = $13.50 x 5% + $0.091 = $0.77/lb Mo (according to the 2014 optimisation inputs)
  = ($13.50 x 86.2% - $1.334) x 5% + $1.334 = $1.85/lb Mo (according to the 2015 updates for the cashflow model)
- The gold metal cost:
  = metal price x royalty + Au refining cost x Au payable
  = $1,200 x 5% + $5.50 x 92% = $65.06/oz ($948.77/lb Au) (according to the 2014 optimisation inputs)
  = (($1,200 x 86%) – ($5.50 x 86%)) x 4% + ($5.50 x 86%) = $45.82/oz ($668.20/lb Au) (according to the 2015 updates for the cashflow model)
- The silver metal cost:
  = metal price adjusted x royalty + Ag refining cost x Ag payable

[9] 5% royalty adopted in the pit optimisation for all metals. Refer to Item 4.6 for further information on Project royalties.

= \$16.00 x 5% + \$0.40 x 90% = \$1.16/oz Ag (\$16.92/lb Ag) (according to the 2014 optimisation inputs)
= ((\$16.00 x 80%) – (\$0.40 x 80%)) x 4% + (\$0.40 x 80%) = \$0.82/oz (\$11.96/lb Au) (according to the 2015 updates for the cashflow model)

21.3 Other cost

Allowances for Project sustaining capital costs are included as a provision within the total processing operating costs and within the total G&A operating costs (Table 21-3 and Table 21-4). The updated estimates are based on 5% of the respective operating costs.

An allowance of \$0.26/t mined for mining equipment sustaining capital was determined by MPSA in January 2013 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

This allowance was included in the optimisation inputs and the cashflow model.

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 Reserves estimate, and in order to demonstrate a positive cashflow for each year of mining and processing. The development capital costs and longer term rehabilitation costs are included in the analysis for completeness.

The cashflow model forms part of a more comprehensive Project financial model which extends to depreciation, tax, financing and inter-company cashflows. Consequently, net present value (NPV) and internal rate of return (IRR) are not reported for the undiscounted cashflow model presented in Table 22-1.

22.2 Production schedule

The production schedule forming the basis of the cashflow model is the same as that listed in Table 16-6 of Item 16.

22.3 Cashflow model

The cashflow model to support the Mineral Reserves estimate is listed in Table 22-1. The annual revenues are calculated from the same metal prices as used in the pit optimisation process (Item 15):

- Copper = US$3.00/lb (US$6,615/t)
- Molybdenum = US$13.50/lb (US$29,762/t)
- Gold = US$1,200/oz
- Silver = US$16.00/oz

The payable metal factors are:

- Copper = 96.43%
- Molybdenum = 86.20%
- Gold = 86.00%
- Silver = 80.00%

The unit operating costs equate to the same costs as summarised in Item 21:

- Mining waste = $1.79/t waste (overall average for all pits)
- Mining ore = $1.88/t ore (overall average for all pits)
- Processing (including sustaining costs) = $3.92/t ore
- G&A (including sustaining costs) = $0.85/t ore
- Stockpile reclaim costs = included in processing costs

An additional cost of $1.30/t ore reclaimed has been adopted for reclaim from longer term ore stockpiles. Mining sustaining costs of 0.26/t (mined) have also been included (as an overall equivalent to varying annual capital expenditures).

The metal costs (including TCRC's and royalties) equate to the same costs as summarised in Item 21:

- Copper = US$0.45/lb
- Molybdenum = US$1.85/lb
- Gold = US$45.82/oz
- Silver = US$0.82/oz

The recovery values shown in Table 22-1 are average figures resulting from the application, on a mining model block by block basis, of the variable relationships listed in Item 15.

The Project is cashflow positive from 2018 and payback on the $6,425M capital spend occurs in 2024.

All mining expenditure incurred prior to commercial production is included in the $6,425M capital estimate.

22.4 Sensitivity analysis

A sensitivity analysis was completed as part of the pit optimisation work described in Item 15.4.5. The most sensitive variable is metal price (and recovery, since the magnitude of impact is is the same). Based on the undiscounted cashflow model Table 22—2 summarises the impact of varying the copper price, operating costs and metal costs by +/- 10% and confirms this analysis.

Table 22—1 Mineral Reserves undiscounted cashflow model summary

		TOTAL	<2016 <Year -1	2017 to 2026 Year 1 to 10	2027 to 2036 Year 11 to 20	2037 to 2046 Year 21 to 30	2047 to end Year 31 to end
MINING							
Total ore (inluding direct feed and s/pile reclaim)	Mt	**2,958.6**	5.7	821.6	938.0	814.6	378.6
Total waste (incl. MW not reclaimed)	Mt	**3,507.4**	55.0	839.5	931.2	1,075.0	606.7
Strip ratio		**1.2**	9.7	1.0	1.0	1.3	1.6
TOTAL FEED TO PLANT (before mining dil'n & recovery)							
TOTAL	t	**3,182.5**	0.0	752.6	893.7	893.4	642.9
Cu	%	**0.38**	0.00	0.43	0.42	0.35	0.29
Mo	ppm	**59.57**	0.00	72.24	73.65	54.99	31.54
Au	ppm	**0.07**	0.00	0.08	0.07	0.05	0.07
Ag	ppm	**1.35**	0.00	1.36	1.47	1.34	1.20
TOTAL FEED TO PLANT (after mining dil'n & recovery)							
TOTAL	t	**3,181.3**	0.0	752.3	893.3	893.1	642.6
Cu	%	**0.37**	0.00	0.42	0.41	0.34	0.28
Mo	ppm	**58.40**	0.00	70.82	72.21	53.91	30.92
Au	ppm	**0.07**	0.00	0.08	0.07	0.05	0.06
Ag	ppm	**1.33**	0.00	1.33	1.45	1.31	1.18
AVERAGE RECOVERIES							
Cu	%	**90.2**	0.0	91.6	91.3	89.6	86.9
Mo	%	**54.0**	0.0	55.0	54.6	53.4	51.0
Au	%	**55.4**	0.0	56.6	54.5	50.3	60.2
Ag	%	**45.2**	0.0	47.3	47.3	44.6	39.9
METAL RECOVERED							
Cu	kt	**10,612.3**	0.0	2,923.6	3,363.9	2,750.4	1,574.4
Mo	kt	**100.4**	0.0	29.3	35.2	25.7	10.1
Au	koz	**3,767.6**	0.0	1,123.9	1,091.5	746.0	806.3
Ag	koz	**61,333.4**	0.0	15,208.4	19,633.4	16,788.8	9,702.8
METAL PAYABLE							
Cu	kt	**10,233.3**	0.0	2,819.2	3,243.8	2,652.1	1,518.2
Mo	kt	**86.5**	0.0	25.3	30.4	22.2	8.7
Au	koz	**3,240.2**	0.0	966.5	938.7	641.6	693.4
Ag	koz	**49,066.7**	0.0	12,166.7	15,706.7	13,431.1	7,762.2
GROSS REVENUE							
Cu	$M	**67,681.8**	0.0	18,645.8	21,453.9	17,540.9	10,041.2
Mo	$M	**2,575.4**	0.0	751.8	903.6	659.9	260.0
Au	$M	**3,888.2**	0.0	1,159.8	1,126.4	769.9	832.1
Ag	$M	**785.1**	0.0	194.7	251.3	214.9	124.2
subtotal	**$M**	**74,930.5**	**0.0**	**20,752.2**	**23,735.2**	**19,185.6**	**11,257.5**
CAPITAL COSTS							
Development capex	$M	**6,425.0**	4,995.9	1,429.1	0.0	0.0	0.0
Expansion capex	$M	**915.0**	0.0	915.0	0.0	0.0	0.0
Sustaining capex	$M	**0.0**	0.0	0.0	0.0	0.0	0.0
Closure and reclamation	$M	**78.6**	0.0	0.0	0.0	0.0	78.6
subtotal	**$M**	**7,418.6**	**4,995.9**	**2,344.1**	**0.0**	**0.0**	**78.6**
OPERATING COSTS							
Mining waste	$M	**6,183.2**	0.0	1,503.5	1,667.8	1,925.3	1,086.6
Mining ore	$M	**5,554.3**	0.0	1,545.5	1,764.4	1,532.3	712.1
Mining sustaining	$M	**1,665.4**	0.0	431.9	486.0	491.3	256.2
Processing (incl sustaining)	$M	**12,471.1**	0.0	2,949.2	3,501.8	3,500.9	2,519.1
G&A (incl sustaining)	$M	**2,709.2**	0.0	640.7	760.7	760.5	547.3
Stockpile reclaim	$M	**516.5**	0.0	30.2	14.6	122.3	349.4
subtotal	**$M**	**29,099.7**	**0.0**	**7,101.0**	**8,195.4**	**8,332.7**	**5,470.6**
METAL COSTS (INCLUDING ROYALTIES)							
Cu	$M	**10,568.9**	0.0	2,911.7	3,350.2	2,739.1	1,568.0
Mo	$M	**409.2**	0.0	119.5	143.6	104.9	41.3
Au	$M	**172.6**	0.0	51.5	50.0	34.2	36.9
Ag	$M	**50.2**	0.0	12.5	16.1	13.8	7.9
subtotal	**$M**	**11,201.1**	**0.0**	**3,095.1**	**3,559.8**	**2,891.9**	**1,654.2**
CASHFLOW	**$M**	**27,211.1**	**-4,995.9**	**8,212.0**	**11,980.0**	**7,961.0**	**4,054.1**

Table 22—2 Undiscounted cashflow model sensitivity analysis

Cu price		**$M**	
2.70	$/lb	20,443.0	75%
3.00	**$/lb**	**27,211.1**	**100%**
3.30	$/lb	33,979.3	125%
Operating costs		**$M**	
8.27	$/t processed	30,121.1	111%
9.19	**$/t processed**	**27,211.1**	**100%**
10.01	$/t processed	24,301.2	89%
Cu metal costs		**$M**	
0.41	$/lb	28,331.2	104%
0.45	**$/lb**	**27,211.1**	**100%**
0.50	$/lb	26,091.0	96%

ITEM 23 ADJACENT PROPERTIES

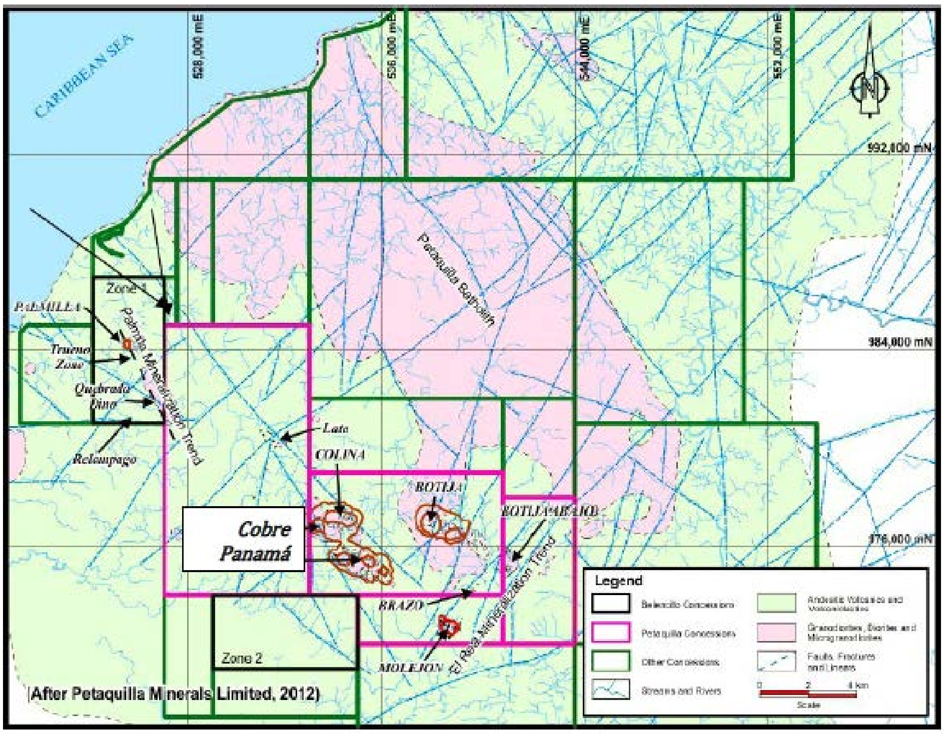
23.1 Introduction

The following information is reproduced in part from Rose et al (March 2013). It has been updated to include more recent publicly reported information (SGS Canada Inc., 2013). 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[10].

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

[10] MPSA now owns the PML rights to the Palmilla deposit.

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 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.

Preproduction at Molejón began in July 2009 and commercial production was achieved in January, 2010 at a mining rate of 1,500t/day (PML news release, January 11 2010). In 2012, PML reportedly produced 68,000 oz of gold at a cash cost of \$550/oz to \$600/oz (PML news release, July 3 2012). 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 project is currently at an intermediate stage of exploration, with more than 4,900 m of surface trenching completed, and more than 13,000 m of cored drilling in 67 holes (SGS Canada Inc., 2013). 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**								

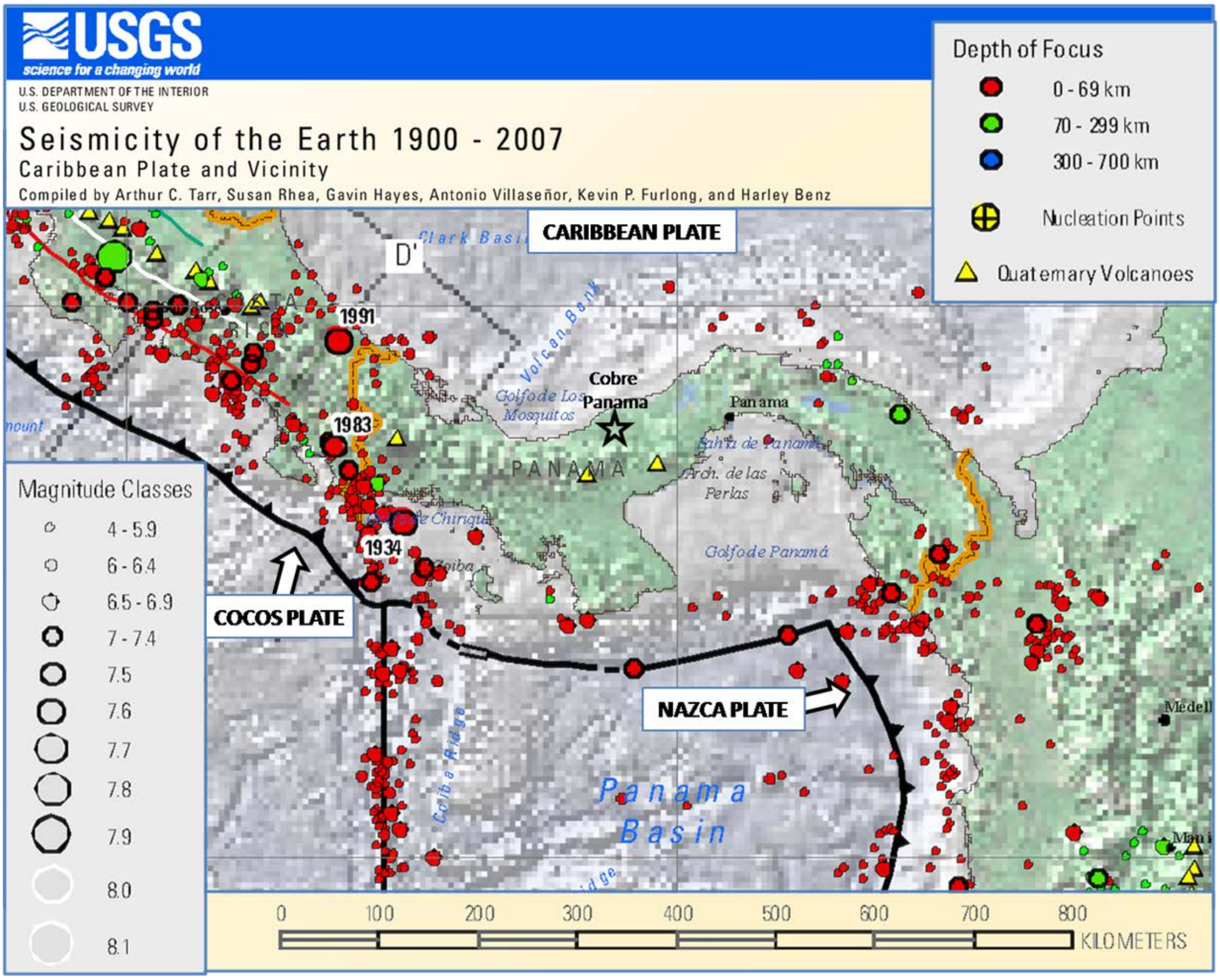
- Numbers may differ due to rounding
- Gold equivalent (Au Eq.) is calculated using Au, Cu and Ag prices and recoveries
- Gold \$1,400/oz., Copper \$ 3.50/lb., Silver \$ 30 /oz.
- Recoveries : Au 85%, Ag and Cu 75%
- Costs : \$2.1/t mined, processing and G&A \$14.00/t processed
- Fixed density of 2.67 t/m^3
- Royalties Au 4% and Cu 5% NSR

ITEM 24 OTHER RELEVANT DATA AND INFORMATION

24.1 Seismicity

Figure 24-1 indicates that relative to other areas of the isthmus, the Project site is located in an area relatively unaffected by seismicity.

Figure 24-1 Seismicity map (Tarr et al, US Geological Survey)



Work by the Universidad de Panamá, Instituto de Geociencias (IGC, 2013), confirms the tectonic setting shown in Figure 24-1, 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 dam and structures at the plant and port sites.

ITEM 25 INTERPRETATIONS AND CONCLUSIONS

25.1 Mineral Resource modelling and estimation

The Mineral Resource estimates for all of the major deposits at Cobre Panamá (Botija, Colina, Valle Grande, Botija Abajo, Brazo and Medio) have been updated prior to the issue of this Technical Report. The resource estimates were preceded by significant 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 largely unchanged from previous estimates and are based upon data quality, assay quality, sample spacing, geological continuity and estimation metrics. In comparison with the 2010 estimate the amount of contained copper in Measured plus Indicated Resources has increased by 21% while the Inferred resource contained copper has dropped by 64%. This increase is due to the addition of the Balboa deposit while the decrease acknowledges the levels of extrapolation in previous models which have been reduced in this estimate. 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 total quantity of Mineral Resource at Cobre Panamá is very large by world standards, this has been split over 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 the previous estimate, 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 Mineral Reserve estimation

The Mineral Reserve estimate for the Cobre Panamá Project is the product of a thorough and conventional process reflecting detailed staged and ultimate pit designs constrained by appropriate optimal pit shells. Volume comparisons between the design ultimate pits and the corresponding pit shells indicate acceptable minor differences.

The optimisation process incorporates the best available information, including variable processing recovery relationships determined from metallurgical testwork and analysis. Both planned and unplanned mining dilution were considered in the modelling and optimisation process, respectively.

The pit designs take account of the desired IPCC concept and incorporate detailed crusher pocket layouts, haulage/tipping access and suitable in-pit conveyor routes. Waste dumps and ore stockpiles have been included into the mine site layout plan and haulage simulations have been undertaken to optimise the mining fleet requirements and the mining sequence.

In the opinion of Michael Lawlor (QP), therefore, the Mineral Reserve estimate reflects an achievable mining plan and production sequence and one which has taken account of staged mining and processing capacity, “smoothed” equipment usage profiles, longer term in-pit crusher relocations, optimised waste haulage profiles and a practical stockpile building and reclaim strategy.

25.2.1 Uncertainty and risk

Mining and processing costs

Operating costs and metal cost inputs (ie, accounting for royalties, concentrate transport and refining charges) for optimisation have been determined from MPSA cost modelling. The processing costs include general and administration costs (G&A costs).

The unit ore mining cost estimates do vary by depth increment and do reflect short hauls to the notional in-pit crusher locations considered at the time of the estimates. The revised cost estimates make allowance for potential trolley-assisted savings (ie, faster cycle times and reduced fuel consumption based on experience at the Company’s Kansanshi operations in Zambia). Life of mine planning is an iterative process and hence mining costs and optimisation inputs will be refined in future Mineral Reserve estimation work, reflecting the detailed pit designs and haulage layouts that are now available.

Processing costs (including G&A costs) estimates were originally prepared in detail by MPSA and account for:

- consumable rates for such as grinding media, liners, reagents and power
- maintenance parts
- personnel levels/numbers, and labour rates/allowances

These were updated for the optimisation to reflect:

- increased power costs (~32% equivalent unit cost increase)
- increased maintenance parts costs (~94% equivalent unit cost increase)
- processing labour fixed costs increased to $34 M/annum for a larger 100 Mtpa operation (~36% equivalent unit cost increase)
- increased administration labour fixed costs (~27% to 34% equivalent unit cost increase for various administration functions)

In terms of operating cost uncertainty, the optimisation sensitivity analyses indicate an approximate -3% and -4.5% impact on net value due to a 10% increase in mining and processing operating costs, respectively.

At these impact levels there is minimal risk to the selected pit shell as the basis for all following pit design and production scheduling work. The ultimate pit shell size and mining inventory varies by only +/- 2.5% and hence the impact of these cost variances could be assessed as a cashflow model sensitive variable.

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 the consultants to review the designs in the context of these structures and modify their design recommendations accordingly.

Due to the staged sequence of mining the Botija Pit, it will be possible to manage geotechnical uncertainty by the mapping of structural exposures, the monitoring of actual slope performance, and then modifying the slope design parameters as necessary for the subsequent phase cutbacks.

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.

The Botija optimisation sensitivity analyses indicate a 4% reduction in value for a 4% flattening of the overall slope.

Hydrogeology

The hydrogeological work to date has focussed on the Botija Pit and has enabled a preliminary characterisation of hydrogeological conditions and the identification of potential aquifers impacting on that open pit. Hydrogeological uncertainty will need to be addressed throughout the operational life by the collection and analysis of piezometric data, numerical modelling and the drilling of suitably located dewatering bores, as and where necessary.

The pit slope design parameters assume drained slope conditions and hence vertical perimeter (and/or in-pit) bores and possibly horizontal drain holes will be required.

In terms of managing the risk of operational delays, the water inflows to the pit (ie, from groundwater and rainwater run-off) will be managed by in-pit sumps in the floor(s) of the advancing excavations and staged pumping to surface via sumps located at intervals up the overall slope. The pit designs have allowed for sufficient berm/bench width in places to enable these sumps to be located.

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.

Major surface water courses have been considered during the open pit and surface layout designs so as to mitigate the risk of inundation.

Extending from the mining geotechnical considerations mentioned above, the critical in-pit crusher positions for the Botija starter pit have been positioned below the saprolite horizon. Even though the individual crusher pockets are expected to be excavated into more competent saprock and fresh rock benches, the steep pocket batters will likely require extensive support, reinforcement and shotcreting.

Instability risk will be managed by the controlled blasting and ground support procedures as adopted by FQM for similar in-pit facilities in the Company's Zambian operations.

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.

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) in December 2011. The approved Project was for a mining operation comprising three open pits, at Botija, Colina and Valle Grande. Since then, and prior to and following the acquisition of Inmet by FQM, the Project definition and development scope has changed and the ESIA studies and documentation are currently being revised. The expected timeframe for the submission of a new ESIA is the end of 2016.

An environmental management system (EMS) has been developed to include the environmental and social management plans and commitments. Through the construction and operational phases, these EMS plans will be updated to reflect changes in site and operational conditions.

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

It is recommended that extensional drilling is carried out to resolve the mineralisation boundaries of the major orebodies, especially at Botija and in the Colina-Medio area. This is important to determine the ultimate pit shells and thus the mining and infrastructure (stockpiles, etc.) footprint.

26.2 Cost modelling

As an item of continuous improvement, it is recommended that all operating costs be reviewed and analysed in terms of the production scale and profile described in this Technical Report. In particular, the mining 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.

Furthermore, and now that detailed designs have been prepared, the mining operating costs should be re-assessed to take account of potential trolley-assisted haulage routes within the pits and up onto the waste dumps.

26.3 Geotechnical engineering

The review items and recommendations by consultants CNI (2014, 2015) should be contemplated following confirmatory geotechnical mapping and observations of all actual exposures in the Botija Pit.

26.4 Hydrogeology

The proposed hydrogeological programme should proceed with the recommended aquifer testing, hydrogeological modelling and design/specification of dewatering wells.

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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 Technical Report”, dated effective 30th June 2015 (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 registered Professional Natural Scientist with the South African Council for Natural Scientific Professions (SACNASP).

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 June 2014.

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 30th day of June, 2015 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 Technical Report", dated effective 30th June 2015 (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 form 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 June 2014.

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 30th day of June, 2015 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 Technical Report", dated effective 30th June 2015 (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 May 2015.

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 30th day of June, 2015 at West Perth, Western Australia, Australia.

Robert Stone

Appendix A JORC Table 1, Sections 1-3

Section 1 Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	*Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.*	Almost exclusively diamond drilling – some trenching
	Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used	Half core samples taken, cut mostly with a diamond saw
	Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information	Diamond drilling, average 1.5 m samples HQ diameter core.
Drilling techniques	*Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	Diamond drilling from surface, occasionally with triple tube in areas of poor ground.
Drill sample recovery	*Method of recording and assessing core and chip sample recoveries and results assessed*	Measurement of core recovery for each core run.
	Measures taken to maximise sample recovery and ensure representative nature of the samples	Diamond drill core with very good recovery; samples are considered to be very representative.
	Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	No known relationship (positive or negative) between core recovery.
Logging	*Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	A record is made of RQD and core recovery for every metre of core. Point load testing carried out approximately every 5-10 m on core. A significant number of holes were drilled primarily for geomechanical testing. Density measurements using Archimedes method (and some wax coating) taken every 10-20 m.
	Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	Logging is both qualitative (rock, lithological, alteration descriptions) and semi-quantitative (estimates of sulphide percentages)
	The total length and percentage of the relevant intersections logged	All holes are completely logged.
Sub-sampling techniques and sample preparation	*If core, whether cut or sawn and whether quarter, half or all core taken.*	Half core taken using a diamond saw. Core splitter used prior to MPSA work. Adrian/Teck used a core splitter.
	If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	All samples used in the resource are from diamond drill core.

Criteria	JORC Code explanation	Commentary
	For all sample types, the nature, quality and appropriateness of the sample preparation technique.	Crushing, sub-sampling and grinding of half core following drying.
	Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.	Sieve testing of second crushing stages to check that particle size has been achieved (80% passing 2 mm).
	Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.	A number of twinned holes were drilled across the project. Coarse crush duplicates are submitted as part of the QAQC stream. Quarter core duplicates of half core were taken.
	Whether sample sizes are appropriate to the grain size of the material being sampled.	A 250 g aliquot is split out and sent through to ALS for grinding and assaying.
Quality of assay data and laboratory tests	*The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	
	For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	Handheld XRF (Olympus) for validation of assays. Magnetic Susceptibility measurement. ASD measurements on a set of cores (Analytic Spectral Device) for qualitative assessments of clay mineral spectra. TerraSpec 4 ASD device being used (www.asdi.com). ASD data being processed.
	Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.	Blanks, Standards, Field blanks (coarse blanks), coarse crush duplicates and umpire programmes all in place.
Verification of sampling and assaying	*The verification of significant intersections by either independent or alternative company personnel.*	Twinning programme in place. No scissor holes as most holes are vertical.
	The use of twinned holes.	Several full twinned hole programmes.
	Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	Electronic data capture of logging information, also data entry of sampling information. Storage in spreadsheets, but also now being uploaded to Industrial Strength Database (Datashed)
	Discuss any adjustment to assay data.	No adjustments. Some top cuts (caps) applied during resource estimation. Some corrections to collar co-ordinates as a result of check surveys.
Location of data points	*Accuracy and quality of surveys used to locate drillholes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	The majority of holes picked up by surveyors with accurate instrument. Some holes picked up by DGPS or GPS. Detailed topography (LIDAR) survey used for elevation control. A number of errors in old surveys have been identified and are being corrected; in general these are not material to the open pit mineral resource
	Specification of the grid system used.	Combination of NAD27 (Canal Zone) and WGS84 – all collars stored in WGS84 system.
	Quality and adequacy of topographic control.	Site wide LIDAR survey with a 2-3m accuracy – used for vertical control on the holes.
Data spacing and distribution	*Data spacing for reporting of Exploration Results.*	Spacing varies from 200 x 200 down to a staggered 50 x 50.
	Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	Yes – more details to be applied.
	Whether sample compositing has been applied.	Yes – more details to be applied.

Criteria	JORC Code explanation	Commentary
Orientation of data in relation to geological structure	*Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	Holes are drilled vertical in the main (
	If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	Massive structures- no bias considered.
Sample security	*The measures taken to ensure sample security.*	Geologists transport core. Sample preparation is carried out at an MPSA facility on site and anonymous samples are shipped as coarse samples to the commercial laboratory in Lima.
Audits or reviews	*The results of any audits or reviews of sampling techniques and data.*	Review by Dominique Francois-Bongarcon and by Freeport. External compilation of QAQC data by Optiro.

Section 2 Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.	Site concession granted. Mining allowed within an ESIA (Environmental and Social Impact Area) which is being expanded subject to permitting.
	The security of the tenure held at the time of reporting along with any known impediments to obtaining a license to operate in the area.	Act of Panamánian Parliament granting access to the project to MPSA
Exploration done by other parties	Acknowledgment and appraisal of exploration by other parties.	Multiple phases of exploration by UN, PMRD (Japanese consortium), Teck/Adrian, PTC (Petaquilla Copper), MPSA (Inmet) and FQM.
Geology	Deposit type, geological setting and style of mineralisation.	Low suphidation porphyry system with subsidiary gold and molybdenum.
Drill hole Information	A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: • easting and northing of the drill hole collar • elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar • dip and azimuth of the hole • down hole length and interception depth • hole length.	Many thousands of holes have been drilled at the property. Most of these have been publically reported over a period of 10-15 years.
Data aggregation methods	In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.	Composite intersections of copper, gold and molybdenum are reported (without capping).
	Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.	Most samples are 1 – 1.5 m. Large intercepts are aggregated by length weighting.
	The assumptions used for any reporting of metal equivalent values should be clearly stated.	Copper, gold and molybdenum grades only reported.
Relationship between mineralisation widths and intercept lengths	These relationships are particularly important in the reporting of Exploration Results. If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').	Most holes are interpreted as drilling perpendicular to the mineralisation. At some deposits (e.g. Balboa) the angle of intercept is poorer than at others.
Diagrams	Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	Previous reports have summarised sections and tabulations of intercepts.
Balanced reporting	Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	The database contains over XX holes. Misleading reporting has not been practiced.

Criteria	JORC Code explanation	Commentary
Other substantive exploration data	Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	There has been sufficient drilling to delineate all of the orebodies. Other substantive exploration data is not required.
Further work	The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large-scale step-out drilling). Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive	The orebodies have largely been delineated in most directions; however, some of the deposits are open in some directions, including down plunge. Ongoing sterilisation, geotechnical and hydrological as well as exploration drilling is ongoing.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary
Database integrity	Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.	Input of digital data from as many sources as possible, minimizing manual data entry. Spot checks of data by Chief Geologist. Validation of MPSA database by FQM.
	Data validation procedures used.	Database validations included: • Visual investigation and checks of the relative magnitudes of downhole survey data were completed in order to identify improperly recorded downhole survey values. • 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.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* *If no site visits have been undertaken indicate why this is the case.*	The CP/QP visited site in June 2014 and inspected geology, core, data gathering and sample preparation procedures. Some data validation activities were carried out during this visit.
Geological interpretation	*Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.*	The mineralisation at the Cobre project tends to occur within the porphyritic rocks and extends into the contact area of the plutonic rocks. For each deposit examination of the drillhole intersections and grade inflections in log-probability plots were used to select nominal cut-off grades for interpretation of low grade (nominal cut-off grade of 0.15%) and medium grade (nominal cut-off grades of 0.4% or 0.5%) copper mineralisation. An additional high grade copper domain (nominal cut-off grade of 0.8%) was defined at Botija and gold domains were defined at Balboa and Colina. The mineralisation interpretations are robust and well-supported by the drilling. Lithological models were developed that were used to assign density values. Variogram ranges are significant, suggesting good grade continuity.
	Nature of the data used and of any assumptions made.	
	The effect, if any, of alternative interpretations on Mineral Resource estimation.	
	The use of geology in guiding and controlling Mineral Resource estimation.	
	The factors affecting continuity both of grade and geology.	
Dimensions	*The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource*	The deposits occur over an area measuring approximately 10 km east-west by 4 km north-south. The individual deposits range from 0.5km to 3 km in strike length and from 0.5 km to 1 km across strike. The deposits generally extend to 300 m below the surface, and Balboa extends to 800 m below surface
Estimation and modelling techniques	*The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.*	Industry-standard modelling techniques were used. Estimation was by Ordinary Kriging into large panels, followed by Uniform Conditioning assuming a relevant SMU size. Local Uniform Conditioning to the SMU scale was applied as a post-processing step. A reconciliation was carried out with the previous Mineral Resource Estimates (2011).
	The assumptions made regarding recovery of by-products.	

Criteria	JORC Code explanation	Commentary
	Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation).	Deleterious elements As (ppm), Bi (ppm), Fe (%), S (%), Pb (ppm) and Zn (ppm) were estimated by ordinary kriging.
	In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	Parent block dimensions are 50 mE by 50 mN by 15 mRL; this dimension was supported by a kriging neighbourhood study and is about half the drillhole spacing.
	Any assumptions behind modelling of selective mining units.	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; this is considered appropriate for the expected scale of mining.
	Any assumptions about correlation between variables.	Copper, gold, silver and molybdenum generally have poor correlations with each of the defined domains. These elements were estimated independently of each other.
	Description of how the geological interpretation was used to control the resource estimates.	Wireframed solid domains based upon the geological and mineralogical controls were generated.
	Discussion of basis for using or not using grade cutting or capping.	Top-cuts for copper, gold, molybdenum and gold at 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 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.
	The process of validation, the checking process used, the comparison of model data to drillhole data, and use of reconciliation data if available.	Validation of the OK parent block grades included: 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; and examining trend plots of the input data and estimated block grades. Validation of the LUC copper and gold models included: 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; that metal correlations in the SMU blocks were comparable with the correlations observed in the drillhole data; and 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. Mining has not commenced and so reconciliation data is not available.
Moisture	*Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnes are estimated on a dry basis
Cut-off parameters	*The basis of the adopted cut-off grade(s) or quality parameters applied*	A reporting cut-off of 0.11% copper was applied for reporting, based upon mining costing and assumed realistic metal prices.
Mining factors or assumptions	*Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	Large-scale open pit mining methods were assumed; leading to the choice of the parent block size, SMU size and reporting cut-off grade.

Criteria	JORC Code explanation	Commentary
Metallurgical factors or assumptions	*The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	No relevant metallurgical factors were applied. An industry-standard flowsheet is envisaged.
Environmental factors or assumptions	*Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made*	There are no environmental assumptions which have impacted on the Mineral Resource estimation.
Bulk density	*Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.*	Density measurements were made on a large number of samples of diamond core. Average values were determined based on the rock type and were assigned to the resource model.
	The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit,	
	Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	
Classification	*The basis for the classification of the Mineral Resources into varying confidence categories*	The resources were classified on the basis of data density, data quality and geological continuity.
	Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).	All relevant data has been considered in the classification.
	Whether the result appropriately reflects the Competent Person's view of the deposit.	The classification appropriately reflects the Competent Persons (QPs) view of the deposits.
Audits or reviews	*The results of any audits or reviews of Mineral Resource estimates.*	Estimates have been reviewed by Optiro and by FQM staff.
	Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate	The estimate and its classification is believed to be appropriate for scheduling on a quarterly basis initially, followed by scheduling on an annual basis.

Criteria	JORC Code explanation	Commentary
	The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used	
	These statements of relative accuracy and confidence of the estimate should be compared with production data, where available	Mining has not commenced and so production data is not available.

Appendix B JORC Table 1 Section 4

Cobre Panamá Project Mineral Reserve Summary, Checklist of Assessment and Reporting Criteria

Criteria	JORC Code Explanation	Commentary
Mineral Resource estimate for conversion to Ore (Mineral) Reserves	*Description of the Mineral Resource estimate used as a basis for the conversion to an Ore (Mineral) Reserve.*	The Mineral Resource models used for optimisation and design were current in Q3 2014, when the mine planning models were prepared as the basis of the Mineral Reserve estimate.
	Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore (Mineral) Reserves.	The oxide Measured and Indicated Mineral Resource estimates are reported **inclusive** of the Mineral Reserves.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.*	Michael Lawlor visited site between 23rd and 25th June 2014. Robert Stone visited site April 2013, October 2014 and May 2015.
	If no site visits have been undertaken indicate why this is the case.	n/a.
Study status	*The type and level of study undertaken to enable Mineral Resources to be converted to Ore (Mineral) Reserves.*	The level of mine planning completed for this Study is commensurate with the requirements for a Feasibility Study to bankable standard. Mine planning and design has included conventional Whittle Four-X optimisation of the Mineral Resources, detailed staged and ultimate pit designs inclusive of geotechnical design recommendations, and comprehensive life-of-mine production schedules (on a monthly and annual basis).
	The Code does not require that a final feasibility study has been undertaken to convert Mineral Resources to Ore (Mineral) Reserves, but it does require that appropriate studies will have been carried that will have determined a mine plan that is technically achievable and economically viable, and that all Modifying Factors have been considered.	The mine plan is considered to be technically achievable and economically viable, based on the analyses that have been completed as a check on the expected cashflows. In the conversion of Mineral Resources to Mineral Reserves, modifying factors such as mining/metallurgical parameters, economic, product marketing, legal, environmental (in respect of waste dump design), social and government considerations have been taken into account.
Cut-off parameters	*The basis of the cut-off grade(s) or quality parameters applied.*	Marginal and elevated cut-off grades have been considered during the Whittle Four-X optimisation process.
Mining factors or assumptions	*The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore (Mineral) Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).*	A conventional approach has been adopted in the use of optimisation followed by detailed mine design and production scheduling.
	The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.	The selected shovel/truck mining method is considered to be appropriate to the style of mineralisation and to the operating conditions.
	The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.	Geotechnical investigations and analyses for pit slope stability have been carried out, and these have included drilling/logging investigations, mapping and inspections by a geotechnical consultant, laboratory testing and comprehensive kinematic analyses. The work as yet, has not been supported by equally comprehensive groundwater investigations, supporting the recommendation and adoption of drained overall pit slope angles. Further work is required to confirm the groundwater

		drawdown (abstraction) requirements and impacts on mining vertical advance and pit slope stability.
	The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).	A mine planning model was produced from each of the Datamine Mineral Resource models. For the purposes of the optimisation software, each mine planning model was regularised to an agreed smallest mining unit (SMU) block size, suitable for the scale of proposed primary mining equipment.
	The mining dilution, recovery and minimum mining width factors used.	Geological losses were built into the regularised mine planning models and are considered as allowances for "planned dilution". Adopted "umplanned" mining dilution and recovery factors are considered to be appropriate for this Technical Report, based on the size of the mining equipment, the mining flitch height and the geometry of the ore/waste contacts. This also applies to the adopted minimum mining dimensions.
	The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.	Inferred Mineral Resources are not included in the mining studies for this Technical Report.
	The infrastructure requirements of the selected mining methods.	All infrastructure to support the shovel/truck and IPCC mining method has been designed/planned. The first of the IPC positions will be mined in late 2016.
Metallurgical factors or assumptions	*The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.*	Comprehensive metallurgical testwork based on representatively selected samples has been carried out to support sulphide flotation processing. The choice of processing method has been endorsed by independent metallurgical advice and a programme of confirmatory testwork.
	Whether the metallurgical process is well-tested technology or novel in nature.	Copper flotation processing is considered to be well-tested and proven worldwide.
	The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.	The metallurgical recovery factors have been based on representative ore samples and testwork from each of the several deposits, supported by metallurgical simulation.
	Any assumptions or allowances made for deleterious elements.	There are no seriously deleterious elements, and the available water supply for processing is suitable without treatment.
	The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.	No pilot scale testwork has been carried out subsequent to change of ownership (but was prior to ownership change). However the behaviour of the oretypes as presented to a crushing and milling circuit is well understood from similar operations and the confirmatory testwork confirmed this.
	For minerals that are defined by a specification, has the Ore (Mineral) Reserve estimation been based on the appropriate mineralogy to meet the specifications?	There is only one principal type of plant feed mineralogical source; there is no oxide or mixed ore feed. The minreralogy is conventional with no uncommon mineral types. Recovery equations are based on variable relationships to elemental grades and orebodies.
Environmental	*The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	The Project ESIA was approved in December 2011 for the Project as then envisaged. There are over 600 impacts and management commitments addressed in the ESIA. The Project definition and devlopment scope has now changed and particular aspects will need to be addressed in a revised and resubmitted ESIA, scheduled for the end of 2016. Geochemical characterisation of ore and waste types was carried out in 2014 and water balance/management studies also completed.

		Further work on the mine site layout plan and the segregation/encapsulation of waste types is required in future planning ahead of production. This work is to tie-in with water management controls and containment.
Infrastructure	*The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.*	In addition to the plant site currently under construction, required infrastructure at the port and power generation site is also designed and under construction. There are camps already in existence and expanded facilities are being considered.
Costs	*The derivation of, or assumptions made, regarding projected capital costs in the study.*	Capital costs have been estimated in detail by the Company, based on detailed itemisations and vendor quotes and tenders.
	The methodology used to estimate operating costs.	Mining operating costs have been built up from first principles. Process operating costs have also been built up from first principles and take account of detailed estimates of manning numbers, consumables usage and cost.
	Allowances made for the content of deleterious elements.	Assay smelter deductions have been included in the net revenue estimates, in addition to payable percentages.
	The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co-products.	Optimisations have been completed on a conservative long-term copper price projection.
	The source of exchange rates used in the study.	n/a.
	Derivation of transportation charges.	Advice provided by in-house metals trading division.
	The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.	Product transport and marketing charges have been considered in the optimisations and cashflow models. Metal cost information provided by in-house metals trading division.
	The allowances made for royalties payable, both Government and private.	Royalties payable to the Panamánian government have been considered in the optimisations and cashflow models.
Revenue factors	*The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.*	Revenue factors have taken account of all relevant TCRCs, smelter returns.
	The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	Optimisations and cashflow modelling have been completed on a conservative long-term metal price projections.
Market assessment	*The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.*	The Company's research indicates that medium to longer term demand exceeds supply. Competition from other suppliers is a threat, but it is expected that there will be a ready market for the sale of Cobre Panamá product.
	A customer and competitor analysis along with the identification of likely market windows for the product.	The Company is working to establish offtake agreements.
	Price and volume forecasts and the basis for these forecasts.	The Company is working to establish offtake agreements.
	For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	The Company is working to establish offtake agreements.
Economic	*The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.*	A detailed NPV study has not been undertaken. Whilst a comprehensive financial model exists for the Project, the undiscounted cashflow anlyses undertaken for the Technical Report are consdered sufficient to indicate economic viability.
	NPV ranges and sensitivity to variations in the significant assumptions and inputs.	Sensitivities considered in the optimisation process.
Social	*The status of agreements with key stakeholders and matters leading to social licence to operate.*	As part of the social impact programme, two indigenous communities (300 people) and

		dispersed Latino people (190) have been subject to physical and economic resettlement in relation with the acquisition of the land polygon of 7,477.54 ha of the Mine Site
Other	*To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore (Mineral) Reserves:*	Risk mitigation strategies have been identified by FQM management, and these will continue into the Project implementation and operations phase.
	Any identified material naturally occurring risks.	Not identifed to date.
	The status of material legal agreements and marketing arrangements.	None yet in place; other than a precious metals stream agreement with a subsidiary of Franci-Nevada Corporation.
	The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	There are potential government, social and security related risks for a company operating in Panama. The Company considers that, by virtue of its in-country experience to date, the social and security risks are manageable.
Classification	*The basis for the classification of the Ore (Mineral) Reserves into varying confidence categories.*	Subject to the application of modifying factors and the Mineral Reserve estimation process, Measured Mineral Resource has been translated to Proved Mineral Reserve; in the same way, Indicated Mineral Resource has been translated to Probable Mineral Reserve.
	Whether the result appropriately reflects the Competent Person's view of the deposit.	Yes it does.
	The proportion of Probable (Mineral) Ore Reserves that have been derived from Measured Minerals Resources (if any).	The Mineral Reserves have been classified as Proven or Probable, according to the corresponding Measured and Indicated Mineral Resources.
Audits or reviews	*The results of any audits or reviews of Ore (Mineral) Reserve estimates.*	The mine planning leading up to the Mineral Reserve estimates was completed by the Company's mining engineers. Audits and reviews of the technical input to and progress of this work were completed by the QP.
Discussion of relative accuracy/confidence	*Where appropriate a statement of the relative accuracy and confidence level in the Ore (Mineral) Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.*	n/a
	The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	n/a
	Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore (Mineral) Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.	Following initial optimisations in 2014, further operating cost estimates/reviews were completed in 2015, particularly in respect of processing costs and TCRCs. Cost savings (lower) were identified for the former and ommitted costs were identified and included (TCRCs) for the latter. Optimisations and cashflow anlayses confimed neglible net impact on the fundamental selection of pit shells for

		mine planning.
	It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	n/a