Cotabambas Project
Apurimac, Perú
NI 43-101 Technical Report on Updated Preliminary Economic Assessment

Prepared for:
Panoro Minerals Ltd

Prepared by:
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Luis Vela, CMC, Panoro Minerals Ltd.

Effective Date: 22 September 2015

Project Number: 181089

CERTIFICATE OF QUALIFIED PERSON

I, Stewart Twigg, P.Eng., am employed as a Project Manager with Amec Foster Wheeler (Perú) S.A. (Amec Foster Wheeler).

This certificate applies to the technical report titled "Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Updated Preliminary Economic Assessment" that has an effective date of 22 September, 2015 (the "technical report").

I am a Professional Engineer of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated from Queens University in 1995 with a Bachelor of Science in Mechanical Engineering.

I have practiced my profession for 17 years. I have been directly involved in the design, construction, start up and operation of zinc and lead refineries including Teck Cominco's metallurgical facilities in Trail, British Columbia and Votorantim's Refineria de Cajamarquilla in Lima, Peru for eight of my 17 years. I have participated in studies and the execution of infrastructure projects including detailed construction planning and execution of major mining and infrastructure projects in the Middle East, Canada, Chile and Peru.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Cotabambas Project.

I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.5, 1.17, 1.19, 1.20, 1.22, 1.23; Section 2; Section 3; Section 4; Section 5; Section 18; Section 20; Sections 21.1, 21.2.1, 21.2.4, 21.2.5, 21.2.6, 21.2.7, 21.3; Sections 24.1 and 24.2; Sections 25.1, 25.2, 25.11, 25.12, 25.14, 25.15, 25.17; Section 26.1.5, 26.1.6, 26.2; and Section 27 of the technical report.

I am independent of Panoro Minerals Ltd as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Cotabambas Project during the preparation of the updated Preliminary Economic Assessment and the technical report that is based on the updated Preliminary Economic Assessment. I have previously co-authored a technical report on the Cotabambas Project, entitled:

Twigg, S., Munoz, S., Colquhoun, W., Khera, V., Morrison, R., Hirst, J., Daigle, P., and Searston, S., 2015: Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared by Amec Foster Wheeler and Tetra Tech WEI Inc. for Panoro, effective date 9 April, 2015, 318 p.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 6 November, 2015

"signed and sealed"

Stewart Twigg, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, William Colquhoun FSAIMM, Pr Eng, am employed as a Principal Metallurgical Consultant with Amec Foster Wheeler (Perú) S.A. (Amec Foster Wheeler).

This certificate applies to the technical report titled "Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Updated Preliminary Economic Assessment" that has an effective date of 22 September, 2015 (the "technical report").

I am a Fellow of the South African Institute of Metallurgy and a registered Professional Engineer of the Engineering Council of South Africa. I graduated from Strathclyde University with a Bachelor of Science Degree in Chemical and Process Engineering in 1982.

I have practiced my profession for 29 years. I have been directly involved in mining and base metal processing operations, metallurgical consulting and engineering studies in Africa, Europe, Australia, Far East and North and South America including Peru.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Cotabambas Project.

I am responsible for Sections 1.2, 1.12, 1.16, 1.18, 1.19, 1.20, 1.22, 1.23; Section 2; Section 3; Section 13; Section 17; Section 19; Sections 21.1, 21.2.1, 21.2.3, 21.2.5, 21.2.6, 21.2.7, 21.3; Sections 24.1 and 24.2; Sections 25.7, 25.10, 25.13, 25.14, 25.15, 25.17; Section 26.1.3; and Section 27 of the technical report.

I am independent of Panoro Minerals Ltd as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Cotabambas Project since 2012 and participated in the preparation of the initial Preliminary Economic Assessment, and the updated Preliminary Economic Assessment. I previously co-authored the following technical reports on the Cotabambas Project:

Wright, C., and Colquhoun, W., 2012: Cotabambas Copper-Gold Project, National Instrument 43-101 Technical Report on a Mineral Resource Estimate, Apurímac, Perú: report prepared by AMEC (Perú) S.A. for Panoro, effective date 24 July, 2012, 120 p.

Twigg, S., Munoz, S., Colquhoun, W., Khera, V., Morrison, R., Hirst, J., Daigle, P., and Searston, S., 2015: Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared by Amec Foster Wheeler and Tetra Tech WEI Inc. for Panoro, effective date 9 April, 2015, 318 p.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 6 November 2015

"signed"

William Colquhoun, FSAIMM, Pr Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Vikram Khera, P.Eng., am employed as a Senior Financial Analyst with Amec Foster Wheeler (Perú) Americas Ltd. (Amec Foster Wheeler).

This certificate applies to the technical report titled "Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Updated Preliminary Economic Assessment" that has an effective date of 22 September 2015 (the "technical report").

I have practiced my profession for over 11 years.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Cotabambas Project.

I am responsible for Sections 1.2, 1.18, 1.21; Section 2; Section 3; Section 19; Section 22; Sections 25.13 and 25.16; and Section 27 of the technical report.

I am independent of Panoro Minerals Ltd as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Cotabambas Project during the preparation of the initial Preliminary Economic Assessment and the update to the Preliminary Economic Assessment. I have previously co-authored a technical report on the Cotabambas Project:

> *Twigg, S., Munoz, S., Colquhoun, W., Khera, V., Morrison, R., Hirst, J., Daigle, P., and Searston, S., 2015: Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared by Amec Foster Wheeler and Tetra Tech WEI Inc. for Panoro, effective date 9 April, 2015, 318 p.*

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 6 November 2015

"signed and sealed"

Vikram Khera, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Jesse Aarsen, P.Eng., am a Senior Associate Mining Engineer with Moose Mountain Technical Services, with a business address of 1975-1st Avenue South, Cranbook, BC.

This certificate applies to the technical report titled "Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Updated Preliminary Economic Assessment" that has an effective date of 22 September, 2015 (the "technical report").

I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#38709) and the Association of Professional Engineers and Geoscientists of Alberta (#74969).

I graduated from the University of Alberta in April 2002 with a Bachelor of Science degree in Mining Engineering Co-op.

I have practiced my profession for 11 years. I have been directly involved in an open pit copper project in central BC and worked on other open-pit copper and gold projects in central BC and Mexico.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Cotabambas Project.

I am responsible for Sections 1.15; Section 15; Section 16; Sections 21.2.2, 25.9, and 26.1.4 of the technical report. I am co-responsible for the following sections, where the information used in the section pertains to the mine plan or capital and operating costs arising from the proposed mine plan: Sections 1.19, 1.20, 1.22, 1.23; Sections 21.1.1, 21.1.2, 21.1.4, 21.1.6, 21.1.8, 21.2.1, 21.2.7; and Sections 25.14, 25.15, and 25.17. I am also co-responsible for Section 2.5, Section 3, and Section 27.

I am independent of Panoro Minerals Ltd as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Cotabambas Project during the preparation of the updated Preliminary Economic Assessment and the technical report that is based on the updated Preliminary Economic Assessment.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 6 November, 2015

"signed and sealed"

Jesse Aarsen, B.Sc., P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Luis Vela, CMC., am employed as the Vice President, Exploration, with Panoro Minerals Ltd., (Panoro).

This certificate applies to the technical report titled "Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Updated Preliminary Economic Assessment" that has an effective date of 22 September 2015 (the "technical report").

I am a Registered Member of the Chilean Mining Commission, (CMC #0173).

I graduated from the Universidad Nacional de San Agustin in Perú with a Bachelor of Geology Sciences degree in 1991 and a Geologic Engineer degree in 1996.

I have practiced my profession for 24 years. I have experience in copper porphyries, precious metals, exploration/mining and pre-feasibility studies in México, Chile, Bolivia and Perú. Prior to Panoro I worked as the Vice President and Exploration Manager for companies such as Trafigura Beheer Group, Andean Gold Ltd, Minera Penoles, Andean American Mining, Minera Aurifera Retamas among others and have acted in a consulting capacity.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have regularly visited the Cotabambas Project, with the two most recent visits being 1 to 3 July, 2015 and 6 to 9 September, 2015.

I am responsible for Sections 1.4, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 1.13, 1.14, 1.22, 1.23; Section 2; Section 3; Section 4, Section 5, Section 6, Section 7, Section 8, Section 9, Section 10, Section 11, Section 12, Section 14, Section 20.6; Section 23; Section 24.3; Sections 25.2, 25.3, 25.4, 25.5, 25.6, and 25.8; Sections 26.1.1, 26.1.2 and 26.2, and Section 27 of the technical report.

I am not independent of Panoro Minerals Ltd as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Cotabambas Project since August 2011 in my capacity as Vice President, Exploration, with Panoro.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 6 November 2015

"signed"

Luis Vela, CMC

IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for Panoro Apurímac S.A. (Panoro) by Amec Foster Wheeler Perú S.A. (Amec Foster Wheeler) and Moose Mountain Technical Services (MMTS), collectively the "Report Authors". The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Panoro subject to terms and conditions of its respective individual contracts with the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1.0 SUMMARY

1.1 Introduction

Panoro Minerals Ltd. (Panoro) requested that Amec Foster Wheeler Americas Ltd (Amec Foster Wheeler) and Moose Mountain Technical Services (MMTS) prepare a preliminary economic assessment (PEA) report (the Report), based on a Mineral Resource estimate undertaken by Tetra Tech WEI Inc. (Tetra Tech) in July 2014, for the Cotabambas copper–gold project (the Project), located in Apurimac, Perú.

The Report was prepared to support the disclosure of the financial estimates resulting from the updated PEA disclosed by Panoro in the news release entitled "Panoro Reports Updated Preliminary Economic Asessment Results for Cotabambas Copper-Gold-Silver Project, Peru"dated 22 September, 2015.

1.2 Principal Outcomes

The after-tax financial summary is provided in Table 1-1.

Table 1-1: After-Tax Financial Summary

Item	Unit	Total
Payable Metal		
Cu	M lb	2,638.3
Au	M oz	1.6
Ag	M oz	17.3
Cash Costs (C1)		
Total cash costs	$/lb	2.16
Secondary metal credit	$/lb	(0.94)
Cash costs net of credits	$/lb	1.22
Cash Costs (C2)		
C1 - Net Direct Cash Cost	$/lb	1.22
Depreciation	$/lb	0.72
C2- Production Cost	$/lb	1.94
Financial		
Cumulative net cash flow	$M	1,900
Internal rate of return	%	16.7%
Net present value @ 7.5%	$M	684
Mine life	Years	19
Payback period	$M	3.6
Total start-up capital	$M	1,486
Total life-of-mine capital	$M	1,963

The capital expenditure of $1.53 B stated in the press release of September 22, 2015 includes both initial capital and estimated closure costs. The initial capital required, without provision for closure, is estimated to be $1.49 B.

1.3 Project Description and Location

The Project is located 545 km southeast of Lima, the capital city of Perú, 50 km southwest of Cusco, 60 km east of Abancay, capital of Apurimac Region, and 1 km south of the village of Ccalla and 500 m to the northwest of the town of Cotabambas.

The Cotabambas Project can be accessed by road from Cusco following the paved highway (3S), 32 km from Cusco west to the town of Anta and then via a gravel road (3SF) for approximately 115 km to the town of Cotabambas. Travel time is typically five hours.

There are regular flights to and from Cusco. The flight time from Lima to Cusco is typically one hour. The nearest railhead is in Izcuchaca, a town about 20 km west of Cusco.

The region's climate is typical of the Southern Peruvian Andes. There are two main seasons. Mining activities should be capable of being conducted year-round.

The Cotabambas Project is located in mountainous terrain of the high Andean Cordillera. Elevations on the property vary between approximately 3,000 and 4,000 masl. The region is characterized by deeply incised river valleys and canyons such as the Apurimac River valley that is 2,000 m below the Cotabambas Project area. The Project physiography is dominated by northeast-trending ridges, separated by quebradas or ravines.

The area is vegetated by tough mountain grasses and shrubs, with portions being cultivated by local farmers. In general, the property is above the tree line with the only trees being the non-indigenous Eucalyptus and pine, which have been planted around communities and on hill slopes and along roadways to control erosion.

1.4 Mineral Tenure, Surface Rights, Royalties, and Agreements

The Cotabambas mining and exploration concessions are 100% held by Panoro through its indirectly wholly-owned Peruvian subsidiary Panoro Apurimac S.A. The 17 permits have an area of 15,900 ha. The concessions cover the Ccalla and Azulccacca deposits which are situated mainly within Concessions 10077493 (Maria Carmen-1993) and 10214793 (Maria Carmen 1993 Dos).

Panoro has paid the concessions fees (annual fees and penalty) in respect of all of the Cotabambas mineral concessions until the year 2015, which has met the requirements to keep the concessions current until June 2017. Concession payments are US$3/ha. Panoro does not currently own any surface rights on the Property. Surface rights ownership is held by the communities of Ccochapata, Ccalla and Guaclle and individual surface rights holders. To date Panoro has been able to negotiate access to the Project with the appropriate surface rights holders in support of exploration and drilling activities.

Successful consultation to date is also evidenced by the minimal interruptions and opposition from local communities during exploration activities that have been conducted by Panoro.

The Peruvian government currently levies a sliding-scale royalty on gross sales from mining operations that ranges between 1% and 12%, and which is imposed on operating mining income. Panoro considers that a minimum royalty of 1% of mining sales would be applicable to the Cotabambas Project at a PEA stage of evaluation.

A semi-detailed environmental impact assessment (SDEIA) was completed and subsequently expanded to allow Panoro to drill up to 200 drill holes on the Cotabambas property.

1.5 Environment, Permitting, and Socio-Economics

Currently, the environmental liabilities are restricted to those expected to be associated with an exploration-stage project, and include drill sites and access roads.

Based on work completed in 2012, air and water quality are reported to be below national guidelines for particulates, gasses and dissolved metals. Air quality and noise measurements conducted in the Project area show levels are typical of rural areas in Perú and comply with environmental quality standards. No archaeological sites were identified in the area affected by the exploration activities.

A water quality program covering surface water (i.e., creeks) and springs present in the Ccalla and Duraznomayo catchments potentially affected by the Project has established that the water is suitable for agricultural and human consumption. However, exceedances in surface water were identified for parameters such as iron, aluminium and manganese, potentially associated with the presence of the mineralized zones located within the catchments. The presence of coliforms potentially linked to the raising of animals has also been identified in some of the surface water bodies in the Project area.

No fish species were identified in the Ccalla or Duraznomayo streams. During the baseline investigations conducted in 2012, one wildcat (*leopardus jocobitus*) and one deer (*hippocamelus antisensis*) were observed. These two species are listed as endangered and vulnerable, respectively, under Peruvian regulations.

Preliminary geochemical testing has been conducted for waste rock, indicating that the rock does not have the potential to generate acid. Leaching tests show some potential for the release of manganese. Segregation of waste rock is currently not considered; its placement is proposed in one facility in the waste rock facility (WRF) to be located to the north of the main open pit in the Duraznomayo Creek.

Site run-off will constitute the main source of water for the Project. Mine water will be recycled as much as possible and evaporation and seepage losses will be minimized

in order to reduce fresh water requirements for the Project and avoid potential effects on other surface and groundwater users in the vicinity of the Project. The Project will require a water license for use of fresh water and may also require an authorization to discharge liquid effluents, if applicable.

Potential effects on water quantity downstream of the Project site due to dewatering activities and the use of contact water in mining activities needs to be quantified during future studies. In addition, the assumption in the PEA that water quality downstream of the Project will remain unaffected by the discharge of mining effluents needs to be verified. An assessment of the likely effects on stakeholders in the Project area of influence will need to be conducted, together with an appropriate assessment of permittability, social impacts, financial impacts, and mitigation measures that may be applicable.

A comprehensive environmental and social impact assessment (EIA) will be necessary for the Project in order to obtain necessary permits for construction, operations, and closure. This assessment will be conducted in compliance with Peruvian regulations.

The EIA for the Project will include a conceptual closure plan to obtain the environmental approval. A detailed Mine Closure Plan must be submitted within one year after EIA approval. A provision of $50 M has been made in the Project financial model to take the closure costs into account.

Once the environmental and social impact assessment is approved by Peruvian authorities, a variety of permits, licenses, and authorizations will be required to proceed with the construction and operations of the Project.

The majority of Project components (i.e., open pit, waste rock facility (WRF), plant, water dams and camp, and a portion of the tailings storage facility (TSF)) will be located in the District of Cotabambas. The remaining portion of the TSF would be located in the District of Coyllurqui.

There is likely to be a requirement to relocate some stakeholders from the communities of Ccalla, Ccochapata, the town of Cotabambas, and the surrounding district. At this stage of evaluation, the estimations of the area affected and number of stakeholders is preliminary and will need to be refined during future more detailed studies.

Mitigation measures to avoid, reduce, or compensate for potential Project effects will need to be developed and supported by comprehensive environmental and social baseline investigations and engineering studies.

1.6 Geology and Mineralization

The Ccalla and Azulccacca zones of the Cotabambas deposit are considered to be examples of porphyry copper deposits. The two host porphyries cover an area about 2.5 km long and 1.5 km wide.

The deposit is hosted in the Andahuaylas–Yauri belt, which is dominated by the Andahuaylas–Yauri batholith which is exposed for approximately 300 km between the towns of Yauri in the southeast and Andahuaylas in the northwest, and Mesozoic to Early Cenozoic clastic and marine sediment sequences.

Mineralization occurs in hypogene, supergene enrichment and oxide zones within the host porphyries and surrounding diorites. A well-developed leached cap hosts the oxide mineralization. Sulphide mineralization occurs below the base of the leached cap.

Hypogene mineralization in the Project area has been intersected at depths from approximately 20 m from surface to depths of over 500 m from surface. Mineralization occurs as disseminated chalcopyrite and pyrite, pyrite-chalcopyrite stringers or veinlets and quartz–chalcopyrite–pyrite veinlets. Chalcopyrite mineralization intensity decreases and disseminated pyrite mineralization increases distal to the higher grade parts of the hypogene zone. Sulphide mineralization consists of chalcopyrite and pyrite and gold grades are strongly correlated to copper grades in the hypogene zone. Some occurrences of bornite have been noted in deeper portions of the hypogene zones. Silver grades are not as strongly correlated to copper grades as they are to gold grades, but are generally elevated where copper–gold mineralization is present.

Zones of high-grade chalcocite mineralization with lesser covellite and chalcopyrite occur at the top of the hypogene sulphide mineralization, and at the base of the leached cap. This type of mineralization is interpreted to be a zone of supergene enrichment. Supergene zones occur at Ccalla and Azulccacca and are characterized by high chalcocite content, correspondingly high cyanide-soluble copper assay grades and total copper grades that are generally >1%.

Oxide mineralization occurs in the leached cap of the Ccalla and Azulccacca zones. Iron oxides and oxy-hydroxides replace pyrite, and oxide copper–gold mineralization occurs as patches of green copper oxides, typically chrysoscolla, malachite and broncanthite. Copper oxides occur as coatings on disseminated chalcopyrite grains and as fill in fractures and veinlets. Oxide gold mineralization has been defined in a lens in the Azulccacca area, but has also been intersected in short, isolated 1 to 5 m intervals in other parts of the leached cap of the deposit.

1.7 Exploration

In 1995, Anaconda Perú S.A., a Peruvian subsidiary of Antofagasta Plc (Antofagasta), carried out mapping, soil and rock geochemical sampling programs. In total, 24 drill holes totaling 8,538 m were completed from 1996 to 2000, with numerous mineralized intervals intersected. No exploration was known to be conducted between 2000 and 2002.

Antofagasta and Companhia do Rio Vale Doce (CVRD) formed a joint venture company called Cordillera de las Minas (CDLM) in 2002, and transferred ownership of several groups of exploration concessions in southern Perú to CDLM. From 2002 to 2006, CDLM carried out additional mapping, surface rock and soil geochemical sampling, induced polarization (IP) surveying, magnetometer surveying, and diamond drill testing of previously identified geological, geochemical, and geophysical anomalies. In total, 10 drill holes totalling 3,252 m were drilled.

From 2011 to present, Panoro completed additional mapping, surface rock and stream sediment geochemical sampling, IP surveying, and magnetometer surveying. Panoro also conducted diamond drill testing of geological, geochemical, and geophysical anomalies in Ccalla and Azulccacca deposits. In total, 121 drill holes (63,198 m) have been completed by Panoro. Mineral Resources were estimated in 2009 and updated in 2012 and 2014.

1.8 Exploration Potential

The Project retains exploration potential, and Panoro have a number of geophysical, geochemical, structural and principal component analysis targets that could support further work. Panoro considers nine targets to have exploration potential for porphyry and skarn style mineralization.

1.9 Drilling

Drilling completed on the Project to the cessation of the current drilling program in February 2014 comprises 144 core holes for 70,042 m. Eleven holes were completed after the database closure for the Mineral Resource estimate.

Logging of core follows industry-standard practices. Panoro has re-logged the CDLM and Antofagasta core to provide a coherent lithological logging framework. Geotechnical and geological logging are carried out on whole core by the Panoro geology team. Standardized geological and geotechnical logs are filled out by hand and then entered into a Microsoft Excel® drill hole log template. The log sheets capture interval lengths, lithology code, alteration mineralogy and intensity, sulphide and oxide mineralogy, intensity and occurrence, and major structures.

In 2012, Panoro carried out a collar re-survey program visiting historic and current drill platforms to obtain high-precision total station GPS locations for all drill holes. All subsequent drill collars have been surveyed using the total station instrument.

Down hole surveys were acquired using Eastman and Sperry Sun photographic tools at approximately 100 m intervals for drill holes drilled during the Antofagasta and CDLM drill campaigns. For the Panoro campaigns, down hole surveys have been performed using a single-shot magnetic tool, typically at 50 m down hole intervals.

In relatively competent and fractured rock, core recovery is greater than 95%. In intervals crossing strong faults of less than 5 m, generally intersected once or twice per drill hole, core recovery is poor, ranging from as low as 30 to 75% and loss of chalcopyrite from fractures resulting in a possible decrease in apparent grade for these zones.

Depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths during the Panoro programs are typically greater than true widths. Drill orientations from the Panoro drilling are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.

1.10 Sample Analysis and Security

1.10.1 Sampling

The details of drill core sampling methods for the Antofagasta and CDLM campaigns are not known; however, re-logging by Panoro and a review of the database has led to some conclusions regarding sampling practices.

For Panoro programs, the sampling interval is nominally 2 m, but samples are broken at major contacts in lithology and mineralization type. Samples are divided so that the minimum sample length is approximately 0.5 m and the maximum sample length is 3.0 m.

1.10.2 Density

There are three sources of density data for the Cotabambas Project:

- Pre-Panoro data for 3,125 samples for which individual weights are not recorded and density determination protocols are unknown. These data were rejected for use in resource estimation
- Cellophane film-sealed water immersion density determinations on 1,443 samples with sealed and un-sealed weights in water and air carried out by Panoro

- 107 density validation determinations carried out on behalf of Panoro by ALS Chemex in Lima using a wax-sealed water immersion method.

A least-squares linear regression equation was derived to relate cellophane-sealed bulk density to dry insitu bulk density and the full suite of Panoro density determinations were used to estimate dry insitu bulk density for each domain.

1.10.3 Sample Preparation and Analysis

During the Antofagasta drill campaign, samples were prepared at ALS Geolab Perú in Arequipa and analyzed by ALS Geolab, an independent assay facility in Lima. Results were reported for total copper by atomic absorption (AA) and gold by fire assay. Preparation and analysis for the CDLM campaigns were carried out by CIMM Perú in Lima. Results were reported for total copper, sulphuric acid soluble copper (CuAS), silver by atomic absorption (AA) and gold by fire assay. Accreditations for ALS Chemex and CIMM Perú are unknown for the time of the pre-Panoro campaigns

Samples collected during the Panoro campaigns were prepared by the ALS Minerals (formerly ALS Chemex) sample preparation facility in Arequipa. Dry samples were crushed to better than 70% passing -2 mm. A 250 g sub-sample of the crushed sample was taken and pulverized to better than 85% passing 75 µm. Samples were analyzed at the ALS Minerals chemical laboratory in Lima by AA with the AA62 package for total copper, molybdenum, lead, zinc, and silver, and fire assay for gold. Trace mercury was assayed using Hg-CV41 package and 33 elements were assayed by ME-ICP61 package. ALS Global is an independent laboratory with ISO 9001:2000 and ISO 17025 certifications at its facilities in Perú.

1.10.4 Sample Security

Sample security is performed in accordance with exploration best practices and industry standards. Core is taken from the core tube and placed in core boxes at the drill site to a locked sampling facility under the supervision of Panoro geologists.

Samples and reference materials are stored in a locked container until shipping in a truck to the ALS warehouse in Cusco from which point ALS Minerals takes responsibility for chain of custody. Drill core, coarse reject and pulps are archived at Panoro's core storage warehouse in Cusco.

Historic data have been validated via a check assaying program and the high reproducibility of the original results indicates to Amec Foster Wheeler that there was no tampering of the original results or the stored pre-Panoro core and coarse reject material.

1.10.5 Quality Assurance and Quality Control

During the Antofagasta and CDLM campaigns quality assurance practices relied on internal laboratory controls and do not meet current industry standard practices.

1.11 Data Verification

Tetra Tech reviewed the drill core to the drill logs and reviewed the sampling and logging protocols from the various drill programs and found that they meet or exceed industry standards. Assay analyses and quality assurance QA/QC sampling was also reviewed and found to be adequate for this type of deposit. Tetra Tech found no significant errors in the database and that the data is acceptable for resource estimation.

1.12 Metallurgical Testwork

Preliminary comminution, hydrometallurgical and flotation test work have been carried out for Cotabambas since 2012. Amec Foster Wheeler designed and supervised the most recent 2014 testwork program. The 2012 (Certimin laboratory (Certimin)) test program investigated the metallurgical responses of four mineralization zones to acid leach, cyanidation and flotation concentration. The 2013 (Peacocke laboratory) test work, using reject samples from the 2012 test work, focused on investigating the metallurgical responses of the mineralization to gravity concentration, although preliminary scoping tests using acid leach, cyanidation and flotation procedures were also conducted on the tailings produced from the gravity concentration tests. Additional work was performed in 2014 by Certimin on comminution, batch flotationand batch gravity, together with some preliminary locked cycle testwork. During 2014 the Aminpro laboratory completed acid leach and cyanide leach tests, whereas the Outotec laboratory did preliminary investigations of rheology, settling and filtration.

Test work to date has focused on the global metallurgical characterization of composites of the main mineralization types, and no variability testing has been conducted at this stage.

Bond ball mill work index (BWi) test work in 2012 indicated that gold oxide, copper- gold oxide and secondary sulphide mineralization types are relatively soft and hypogene sulphide mineralization is of moderate hardness. The BWi for the Cu–Au and Cu–Mo composites tested in 2014 were reported to be 14.55 and 16.55 kwh/TM respectively, which are considered to be moderately competent to hard for grinding. The Cu–Mo result is 15% higher than that noted in 2012, suggesting some inherent variability exists in hardness in the deposit for copper metallurgy.

No deleterious elements that could have a significant effect on potential economic extraction were noted. Mo is indicated in the concentrate but the head grade is too low to result in the production of a saleable by-product or credit.

Projected recoveries are summarized in Table 1-2.

1.13 Mineral Resource Estimates

Tetra Tech completed a block model, grade interpolation and mineral resource estimate on the Cotabambas deposit (this report). Drill hole data and assay data were supplied by Panoro, including preliminary geological and topographic wireframes. The resource model and estimate were completed using Datamine™ software (version 3.20.6140.0). The estimation utilized Ordinary Kriging (OK) method of interpolation on density, copper, gold, silver, and molybdenum to provide an estimate based on copper equivalent (Cueq) cut-offs.

Mineral Resources were estimated as contained within a conceptual pit shell developed based on a set of bench-marked input parameters (mining, processing and general and administrative (G&A) costs), metal prices and preliminary metallurgical recovery data. An Indicated Resource was established based on data density, interpolation strategy, statistics, and proximity to surface.

Inferred and Indicated Mineral Resources for the Project are tabulated in Table 1-3. Mineral Resources were prepared by Tetra Tech. The Qualified Person for the estimate is Mr Luis Vela, CMC, a Panoro employee.

At a 0.2% Cueq cut-off, Tetra Tech estimated a total Indicated Mineral Resource of 117 Mt at 0.42% copper, 0.23 g/t gold, 2.74 g/t silver and 0.0013% molybdenum, and a total Inferred Mineral Resource of 605 Mt at 0.31% copper, 0.17 g/t gold, 2.33 g/t silver and 0.0019% molybdenum.

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The following factors could affect the Mineral Resources: commodity price and exchange rate assumptions; pit slope angles and other geotechnical factors; assumptions used in generating the conceptual pit shell, including metal recoveries, and mining and process cost assumptions, and the ability to obtain relevant permits and social licence to operate.

Table 1-2: Projected Recovery by Mineralization Type (Zone)

Mineralization Type	Recovery			Concentrate Grade		
	Cu %	Au %	Ag %	Cu %	Au g/t	Ag g/t
Hypogene Sulphide	87.5	62.0	60.4			
Supergene Sulphide	87.5	62.0	60.4			
Mixed Oxide Cu-Au	60.0	55.0	—			
Oxide High Au	—	65.0	—			
Overall Life-of-Mine Average *	**80.4**	**61.3**	**52.2**	**27.0**	**11.5**	**130**

Table 1-3: Mineral Resource Statement

Resources Category	Zone	Cut-Off Grade% Cueq	Million Tonnes	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu (Blb)	Au (Moz)	Ag (Moz)	Mo (Mlb)
Indicated	Hypogene Sulphide	0.2	84.2	0.37	0.21	2.73	0.0018	0.69	0.58	7.39	3.43
	Supergene Sulphide	0.2	8.9	0.73	0.31	3.07	-	0.14	0.09	0.88	0.01
	Oxide Copper-Gold	0.2	23.8	0.49	0.24	2.63	-	0.26	0.18	2.01	0.01
	Oxide Gold	Na	0.2	-	0.66	3.74	-	-	0	0.02	-
	Total		**117.1**	**0.42**	**0.23**	**2.74**	**0.0013**	**1.09**	**0.86**	**10.3**	**3.45**
Inferred	Hypogene Sulphide	0.2	521	0.29	0.18	2.41	0.0021	3.36	2.94	40.35	24.22
	Supergene Sulphide	0.2	7.4	0.73	0.18	1.93	0.0007	0.12	0.04	0.46	0.11
	Oxide Copper-Gold	0.2	75.8	0.41	0.15	1.82	0.0003	0.68	0.37	4.44	0.5
	Oxide Gold	Na	1.2	-	0.61	3.27	-	-	0.02	0.12	-
	Total	0.2	**605.3**	**0.31**	**0.17**	**2.33**	**0.0019**	**4.16**	**3.38**	**45.37**	**24.83**

Note: Mineral Resources have an effective date of June 20, 2013 and were prepared by Robert Morrison, P.Geo. (APGO, 1839). The qualified Person for the estimate is Mr Luis Vela, CMC, a Panoro employee. The estimate is based on 56,813 meters of drilling by Panoro and 9,923 meters of drilling from legacy campaigns. Copper equivalent (Cueq) is calculated using the equation: Cueq = Cu + 0.4422 Au + 0.0065*Ag, based on the differentials of long range metal prices net of selling costs and metallurgical recoveries for gold and copper and silver. Mineralization would be mined from open pit and treated using conventional flotation and hydrometallurgical flow sheets. Rounding in accordance with reporting guidelines may result in summation differences. CuEQ cut-offs were used to report almost all of the resource. These cut-offs are a function of metal price and recoveries. In the in situ resource, estimated gold, silver and molybdenum are then converted to US dollars and combined. The combined funds are re-converted to copper and added to the in situ copper values. The following metal prices are used: copper – US$3.20/lb; gold – US1,350/troy oz; silver – US23.00/troy oz; molybdenum – US$12.50/lb. The following metal recoveries were applied to the in situ resource: molybdenum – 40%; gold – 64%; silver – 63%. As the resource is reported as in situ, no recovery is applied to copper.

1.14 Mineral Resource Re-tabulation

As metallurgical sample characterization and testing progressed it was noted that the oxide samples tested were biased towards a mixed mineralization containing relatively

higher proportions of chalcopyrite than that indicated by the global characterization of the oxide zone by copper sequential analysis in the resource model. One of the main contributing factors to this was a change in oxide domaining based on geological logging implemented during the last resource estimate update (Morrison et al., 2014) on which oxide metallurgical sampling was based versus previous estimates that were based on copper speciation.

For the purposes of the PEA, Tetra Tech restated the oxide resource model tabulations to include mixed Cu–Au and oxide Cu–Au zones that were defined by copper speciation parameters provided by Amec Foster Wheeler. As flotation testing of the global samples of oxide tested in the most recent program (2014) indicated relatively low copper recoveries (attributable mainly to the contained sulphide mineralization with little or no copper oxide recovery) this was done with the objective of estimating suitable mixed sulphide and oxide Cu–low Au mineralization within the oxide resource (Mixed Cu–Au) that would have a reasonable prospect of economic copper and gold recovery by blending with hypogene sulphide in the mine plan. Some oxide (oxide–Au) mineralization, either with no recoverable copper or grade, with a high gold grade (where the net smelter return was higher than the cut-off grade) would also be blended for gold recovery by flotation.

The Qualified Person for the re-tabulation is Mr Luis Vela, CMC, a Panoro employee.

1.15 Proposed Mine Plan

The PEA mine plan is based on a subset of the Mineral Resources. No mining dilution is incorporated in the plan. Indicated and Inferred Mineral Resources are considered in the mine design and production scheduling. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized.

1.15.1 Throughput

The throughput selected for the PEA is 80,000 t/d.

1.15.2 Pit Optimization and Design

In the pit optimization process, Amec Foster Wheeler defined an area beneath the town of Cotabambas that could not be mined. As a result, the blocks below this area are not considered in the nested pit process.

Pit optimization considered the following metal prices: US$3.25/lb for copper, US$1,300/oz for gold and US$20.5/oz for silver. These prices differ slightly from the

metal prices used in the financial analysis. The overall slope angle (OSA) used in the pit optimization was 42º. Twelve-metre bench heights are assumed.

A total of 36 nested pit shells are generated in the pit optimization process for different revenue factors (RF), which varied from 0.3 to 1.0 using intervals of 0.02. To select the final pit, Amec Foster Wheeler performed a skin analysis. This methodology quantifies the impact of adding neighboring layers to the pit shell selected in terms of net present value (NPV). Pit 21 is selected for the PEA design.

The final pit design and the phase design are developed with MineSight® software. Parameters for the detailed pit design such as minimum mining width, ramp slope angle, bench height, inter-ramp slope angle and others are based in the mine equipment selected.

The mine plan comprises two pits; one termed the North Pit, containing the Ccalla porphyry, and the second, smaller South Pit, containing the Azulccacca porphyry. Each pit is broken into smaller phases to allow a smoother progression of waste stripping and mill feed tonnes through the mine schedule. Small starter phases allow higher grade material to be exposed earlier. The North Pit has four phases and the South Pit has three.

1.15.3 Net Smelter Return Considerations

The net smelter return (NSR value is used to define waste and mill feed material. NSR considers off-site costs, mill process recoveries and in-situ grade to determine the value of material and define whether material is classified as mill feed or waste. For the project an NSR cut-off grade of 6 is used.

1.15.4 Provisional Mine Schedule

The proposed Cotabambas mine has a mine life of 18 years, comprised of one year of pre-stripping followed by 17 years of mill feed. A maximum bench advance (sinking rate) of 12 benches per year is used. During pre-stripping the sinking rate nominally reaches 15 but the topmost 3–4 benches are mined with dozers pushing material downslope.

Marginally-economic material is placed into a stockpile throughout the mine life. This strategy results in increased mill feed grades during the first years of production, and higher material movement during the first years of production. Mining in the North and South Pits is completed at the end of Year 13. Years 14–17 of mill production are undertaken with stockpile reclaim material.

1.15.5 Waste Rock Storage Facility (WRF)

The WRF will be located northeast from the pit in Guaclle Creek and will have a life-of-mine (LOM) capacity of 369 Mm^3.

1.15.6 Stockpile

Material that is stockpiled will temporarily be placed on the WRF from where it will be reclaimed during the last years of production. The maximum stockpile size is approximately 119 Mt. At the end of the mine life, the stockpile is fully reclaimed.

1.15.7 Equipment and Manpower

1.15.8 Equipment Considerations

The Cotabambas mine fleet will consist of a conventional diesel fleet with the capacity to mine approximately 116 Mt of total material per year operating on 12 m benches.

Equipment requirements were estimated using first principle calculations. Equipment sizing and numbers were based on operational factors, including fleet life, averageavailabilities, and efficiency percentages. A schedule of 365 days per year, with two shifts of 12 hours per day, and seven days per work week was used.

Variable equipment availabilities were provided by vendors to better estimate trucks, shovel and drill equipment requirements throughout the life of the mine. Availability was presented in 6,000 hour increments.

Because rock strength test and compression test work have not been done for Cotabambas deposit, MMTS used estimated powder factor values of 0.30 kg/t and 0.28 kg/t for mill feed and for waste, respectively.

1.15.9 Equipment Requirements

For the pre-strip mining six drills will be required, and for the peak years of production, seven drills are required. Drill equipment with 251 mm drill diameter size is assumed.

The primary production loading fleet for mineralized material and waste will consist of five 34 m^3 capacity hydraulic shovels.

The pre-stripping stage will require 35 haul trucks, and during the LOM the truck fleet will reach a peak of 40 during the first two years of production. Trucks will have a 227 t capacity. The truck fleet will be comprised of a mix of Owner-operated and rental units with a maximum Owner fleet size of 25 trucks. Rental units will be operated by the Owner.

The maximum mine support fleet estimated includes four CAT D10 dozers, four CAT 884 wheeled dozers, three CAT 16M graders, four CAT 777G 35,000 L capacity water trucks, one 18,000 L capacity fuel truck, and one lube vehicle. Other miscellaneous support equipment such as service trucks, cranes, excavators and crew vehicles are also included.

1.15.10 Manpower

The mine plan assumes that Cotabambas will operate seven days a week, 24 hours per day, with four crews rotating to fill the mine roster of 12 hours per shift.

1.15.11 Mine Consumables

Main consumables for mine operations include diesel fuel, ANFO, emulsion and tires. The yearly average maximum requirement occurs in Years 1-6 and is as follows:

- Diesel – 48 ML/a
- ANFO and emulsion – 19 ktt/a
- Tires – 210 per year.

1.16 Proposed Recovery Methods

The plant design consists of a plant with a nominal processing capacity of 80,000 t/d and includes crushing, grinding, flotation, concentrate dewatering and tailings disposal.

It is planned that 80 kt/d of mill feed will be crushed in a primary crushing circuit, and milled through a primary grind semi-autogenous grind (SAG) mill and secondary grind ball mills to produce fine material at P_{80} 106 µm.

Milled feed will be processed in a conventional rougher, regrind and cleaner copper flotation plant to produce about 250,000 t/a of copper concentrate with a life-of-mine average concentrate grade of 27.0% Cu, 11.5 g/t Au and 130 g/t Ag (refer to Table 1-2). Concentrate will be dewatered to about 8% moisture, and tailings will be pumped to a TSF at 62 wt% solids for disposal.

The concentrate road transport option selected for the basis of this PEA considers a Cotabambas–Challhuahuacho–Espinar–Imata–Arequipa–port of Matarani route, with a total distance of 598 km.

About 80% of process water in the plant flotation tailings stream will be recovered and recycled to the process from tailings thickeners (75%) located at the plant site and as reclaim from the thickened tailings storage facility (5%). Water make-up requirements for the plant are estimated to be about 364 L/s.

Plant power and consumables were estimated. The power requirements have been factored from the installed power indicated in the major equipment list, and the consumables have been estimated from a combination of mass balances and metallurgical testwork, as well as comparable industry benchmarks.

1.17 Infrastructure

Two routes are proposed to be used to transport goods to and from the Project. The route from the port of Callao will be utilized to transport personnel and minor equipment during construction and through-out operations. The Matarani port route will be utilized to transport major equipment during construction. Copper concentrate will also be transported by truck utilizing this route to Matarani port where it will be offloaded and shipped to the end user.

Tailings disposal assumes that thickened high density tailings will be pumped to a facility with a capacity of 483 Mt. This would be located within the adjacent basin of the Ccayarayoc hill, located 6.2 km from the process plant, at an elevation of 1,000 m above the process plant (4 255 m). This configuration of the TSF has three retent dams with a dam volume of 38.23 Mm^3 obtained at the minimum crest elevation of the retention dams at 4 302 m, and a dam: tailings volume ratio of 0.10. Due to the TSF location within a different basin, a constant operating lake of 1 Mm^3, which is a conservative pond volume, was assumed, together with a tailings surface with a slope of 1.2% discharge from two opposite points to the upstream face of the retention dams. Tailings deposition is opposite the dams and will require further evaluation during more detailed studies.

The conceptual water management was developed based on water balance use of the non-contact water pond (fresh water pond). The pond is located at the north east from the open pit and downstream from the process plant. The pond capacity will be approximately 3 Mm^3 and the minimum volume will be 0.1 Mm^3. The main purpose will be to complement make up water supply for the plant, due that the main source will come from the tailings pond recycle water. The non-contact water pond will be supplied from surface runoff water and direct precipitation. Contact water will be collected below the waste rock facility or within an event pond within the plant footprint. No allowance for a water treatment plant has been considered at this time.

On-site infrastructure will include a security office, training building for 15 persons, medical centre, 400-person accommodations camp, diner, fuel station, administrative plant offices, plant warehouse, maintenance workshop, process plant sample warehouse, laboratories, mine administrative offices, mine operation offices, mine control booth, explosive warehouse, six-bay truck shop (including spares warehouse), mine and geology sample warehouse, and a provisional laydown area

An annual power consumption of 120 MW has been estimated. The Project will be connected to the national electrical network through the extension of an existing substation located in Abancay. The Project considers this extension and a 61 km long 220 kV transmission line from Abancay to Cotabambas. The electrical design assumes that space will be available for expansion of the existing Abancay substation.

1.18 Marketing

No market studies have specifically been conducted on the Cotabambas Project. The final sulfide copper concentrate produced is expected to be shipped to smelters located in China, Japan or India. The copper concentrate is also expected to realize important payable silver and gold credits at various times in the mine life. No credit for Mo has been considered.

Testwork results indicate the copper concentrate will be relatively clean and can reasonably be expected to be marketable with a Cu grade in the range of 25 to 28%.

No deleterious elements that could have a significant effect on potential economicextraction have been detected in the concentrates produced to date by the various test programs.

No contracts are in place for the Project.

At this stage of Project development, it is expected that future sales contracts would be negotiated such that the sales contracts would be typical of, and consistent with, standard industry practice, and be similar to contracts for the supply of copper–gold concentrate elsewhere in the world.

1.19 Capital Cost Estimates

The accuracy of this estimate is considered to be within -35% to +50%. The level of definition expressed as a percentage of total engineering is in the range of 0 to 2% and engineering completion is within a range of 0 to 1%.

The capital cost estimate is expressed in US dollars for the third quarter of 2014. Costs associated with escalation beyond the third quarter of 2014, currency fluctuations, interest during construction, and property acquisition or taxes are excluded from this estimate.

The capital cost estimate consists of estimates of the direct and indirect costs for the open pit mine, mineral process plant, on site and off site infrastructure, including auxiliary buildings, TSF, WRF, camp site, electrical power supply, water management, and access road upgrade.

The capital cost estimate is provided in Table 1-4.

Table 1-4: Capital Cost Estimate Summary

Description	Cost Estimate (US$M)
Initial Capital Cost	
Mining	381.2
Tailings Disposal	78.2
Process Plant	505.2
Site Infrastructure	67.2
Off Site Infrastructure	27.5
Owners Cost	40.0
Indirect Cost	152
Contingency	235
Total Capital Cost	***1,486.2***
Sustaining Capital Cost	
Mine	21.8
Tailings Storage	405.5
Total Sustaining Capital	**427.2**
Closure Cost	
Closure Cost	45.5
Contingency	4.5
Total Closure Cost	**50.0**
Total Capital Cost Estimate	***1,963.4***

The capital expenditure of $1.53 B stated in the press release of September 22, 2015 includes both initial capital and estimated closure costs. The initial capital required, without provision for closure, is estimated to be $1.49 B.

1.20 Operating Cost Estimates

The operating cost estimate includes the open pit mining, minerals processing, and general and administrative operating costs of the Project. The accuracy of this estimate is considered to be within ±35% and no contingency has been included. All operating costs estimates are expressed in US dollars for the third quarter of 2014. Costs associated with escalation beyond the third quarter of 2014, currency fluctuations, marketing and sales costs, royalties, treatment charges/refining charges (TC/RCs), product shipping costs, interest charges, licenses or taxes are excluded from the estimate.

The operating cost estimate is provided in Table 1-5.

Table 1-5: Operating Cost Estimate Summary

Description	LOM Cost Estimate	LOM Cost Estimate (US$M)
Mining	$1.59/t of material mined (excluding pre-stripping)	1,732
	$3.59/t of mill feed processed	
Processing	$4.38/t of mill feed processed	2,116
General and Administration	$0.41/t of mill feed processed	197
Total Operating Cost Estimate		**4,045**

1.21 Economic Analysis

The following section is preliminary in nature and partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Dates cited are for illustrative purposes only, as a decision to proceed with mine construction will require formal approvals from appropriate regulatory authorities and from Panoro's Board, and will require support from additional, more detailed, studies, and declaration of Mineral Reserves.

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Information that is forward-looking includes:

- Mineral Resource estimates
- Assumed commodity prices and exchange rates
- The proposed mine production plan
- Projected recovery rates
- Selection of sites for key infrastructure such as the WRF and TSF
- Infrastructure construction costs and proposed operating costs
- Assumptions as to closure costs and closure requirements

- Project development schedules, including resettlement activities
- Assumptions that Panoro will be able to permit the design and operate to the projected design assumptions
- Assumptions that Panoro will be able to acquire the social licence to construct the envisaged Project and operate it under the design assumptions
- Financial analysis of the Cotabambas Project was carried out using a discounted cash flow (DCF) approach using "beginning of period" discounting.

Taxation was applied at the Project level, and assumed:

- A minimum royalty of 1% of mining sales would be applicable to the Cotabambas Project at a PEA stage of evaluation
- The corporate income tax rate in Perú is scheduled to be 26% in the year that production is planned to commence
- Profit sharing in Perú is currently 8%.

A preliminary mine schedule and operating and capital costs were established for the Cotabambas Project operating at the prescribed throughput rate of 29.3 Mt/a.

The Project is forecast to provide a pre-tax a net present value (NPV) at 7.5% discount rate of US$1,052.5 M, an internal rate of return (IRR) of 20.4% and a payback period of 3.2 years.

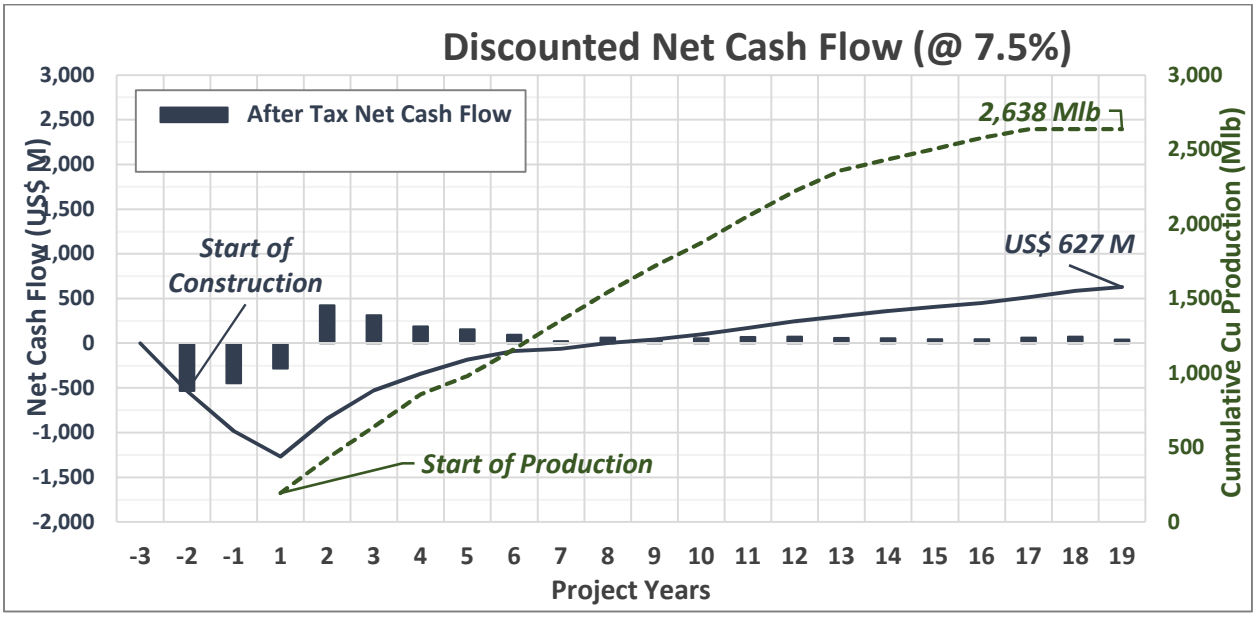
Financial analysis showed the afte- tax Project NPV at 7.5% discount to be US$684 M and the IRR to be 16.7%. The project payback period after tax is 3.6 years.

Figure 1-1 provides the after-tax undiscounted cashflow. Table 1-1 earlier provided a summary of the LOM cashflow predictions.

The life of mine cash cost per pound of payable copper, after secondary metal credits, is US$1.22/lb.

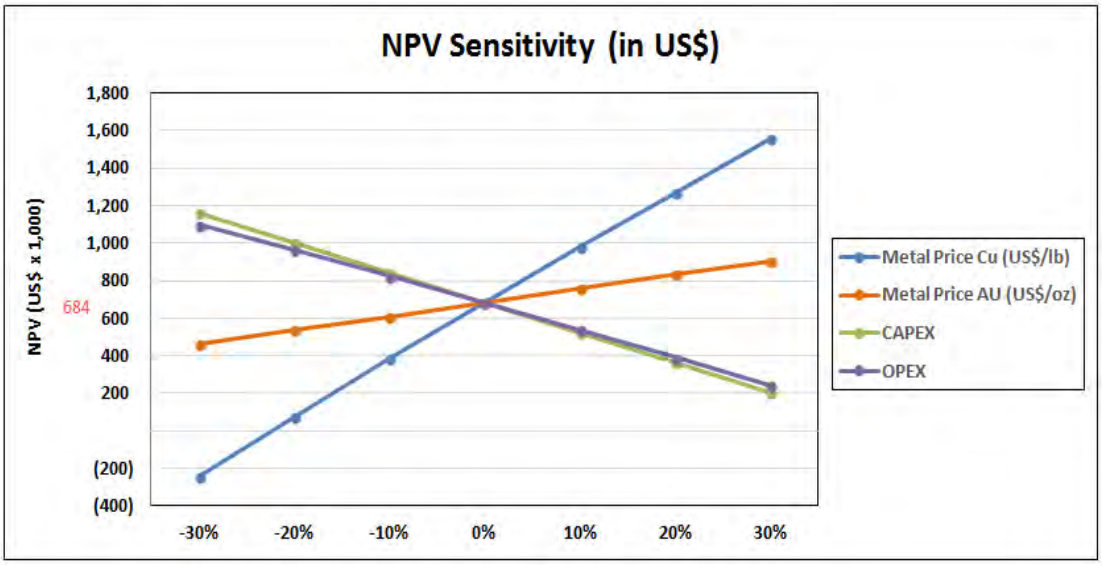
Sensitivity analysis was performed taking into account variations in copper metal prices, copper grade, gold metal prices, gold grade, operating costs and capital costs. The results from the analysis showed that the project NPV sensitivity is (in order from highest to lowest) copper metal price, copper grade, capital costs, operating costs, gold metal price, and gold grade. Gold grade and gold metal price have almost the same effect on the NPV as do copper metal price and copper grade (Figure 1-2).

Figure 1-1: After Tax Net Cash Flow (Undiscounted)



Note: Figure prepared by Amec Foster Wheeler, 2015

Figure 1-2: Sensitivity of After-Tax NPV Discounted at 7.5% (in US$)



Note: Figure prepared by Amec Foster Wheeler, 2015

1.22 Interpretation and Conclusions

Using the parameters and assumptions described in this Report, the Project shows positive economics. There is upside potential for the Project if additional exploration can identify mineralization that can support estimation of Mineral Resources.

1.23 Recommendations

Amec Foster Wheeler has proposed a two-phase work program for the Project. The first phase consists of the required data collection activities to support a pre-feasibility study (PFS) and support for future environmental permitting through an environmental and social impact assessment (EIA), and the second phase consists of the study completion.

Phase 1 would include:

- Exploration drilling and reconnaissance valuation of selected regional targets; infill and step-out drilling at the Ccalla and Azulccacca zones
- Update and generation of additional models, including core recovery and geometallurgy, refinement of existing lithology, structural and alteration models, and additional refinement of estimation domains
- Completion of additional metallurgical and process-related testwork, including baseline/variability work and economic grind recovery trade-off studies, plant siting options analysis, comminution configuration options, evaluation of the potential for gravity gold concentrate production, and the feasibility/economics of incorporating oxide leaching
- Reviews of the impact of various scenarios in regards to community relocations on the mine plan; potential use of the South pit as either a WRF or a TSF; examination of alternatives for cut-off grade optimization; identification of areas and alternatives for stockpiles; consideration of the potential for underground operations
- Completion of additional infrastructure-related studies, including hydrogeological studies and geo-hazard assessments; development of a site water balance, contact water and fresh water strategy; reviews and updates to assumed WRF and TSF location and design, TSF pipeline routing and design, and water management for these infrastructures; a route study that further defines the upgrades required to the access road; and confirmation of available port facilities and costs associated with concentrate handling at port
- A comprehensive set of baseline studies to encompass environmental and social baseline data that can be used in the PFS, and to provide the basis for an EIA.

Phase 2 would comprise the completion of a pre-feasibility study, using the information

collected in Phase 1.

Phase 1 is estimated at approximately $26 M, and Phase 2 at US$2–3 M.

2.0 INTRODUCTION

Panoro Minerals Ltd. (Panoro) requested that Amec Foster Wheeler Americas Ltd (Amec Foster Wheeler) and Moose Mountain Technical Services (MMTS) prepare a preliminary economic assessment (PEA) report (the Report) based on a Mineral Resource estimate prepared by Tetra Tech WEI Inc. (Tetra Tech) in July 2014 for the Cotabambas copper–gold project (the Project) located in Apurimac, Perú.

The Project location is shown in Figure 2-1 and Figure 2-2.

2.1 Terms of Reference

The Report was prepared to support the disclosure of the financial estimates resulting from the updated PEA disclosed by Panoro in the news release entitled “Panoro Reports Updated Preliminary Economic Asessment Results for Cotabambas Copper-Gold-Silver Project, Peru”dated 22 September, 2015.

All measurement units used in this Report are metric, and currency is expressed in US dollars unless stated otherwise. The Report uses Canadian English.

2.2 Qualified Persons

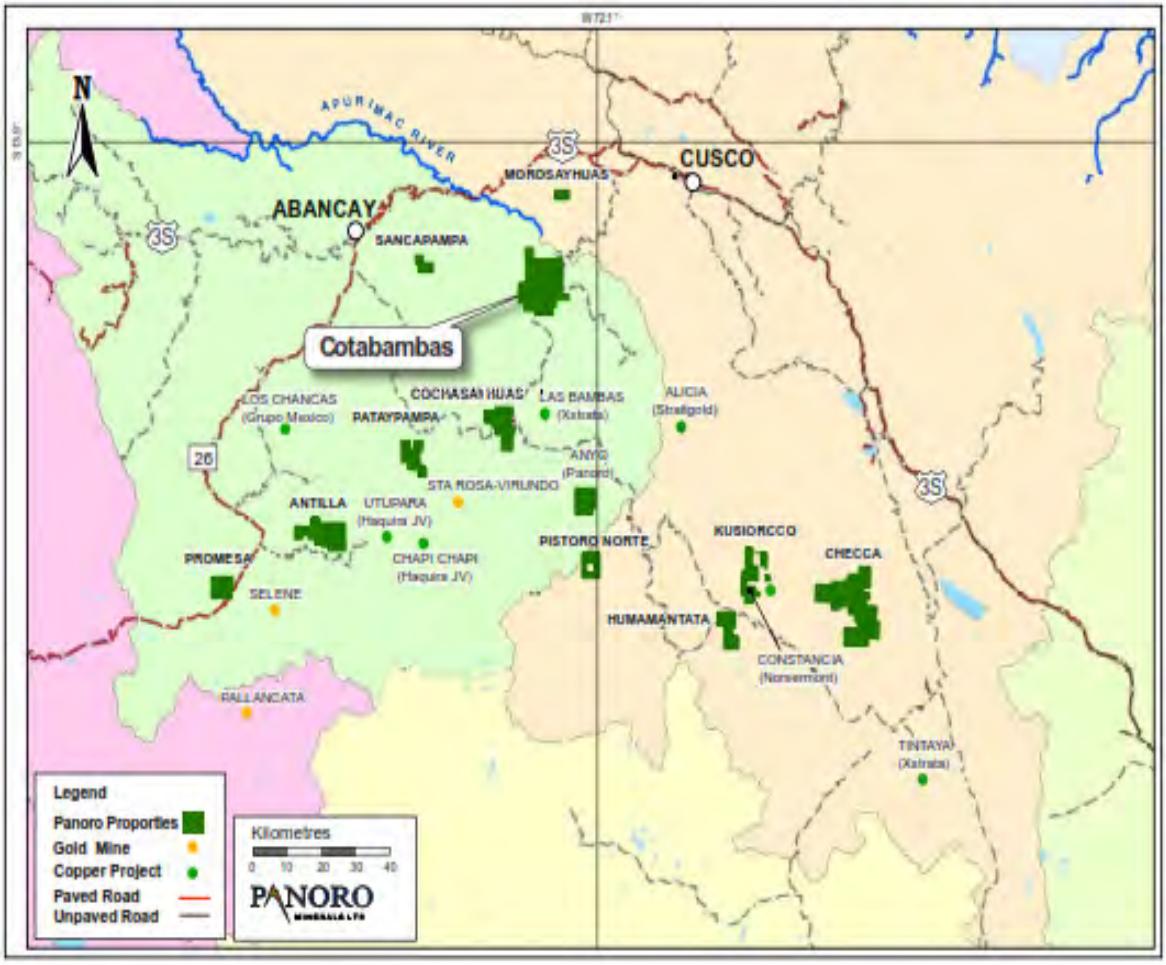
The following serve as the qualified persons (QPs) for this Technical Report as defined in National Instrument 43-101, *Standards of Disclosure for Mineral Projects*, and in compliance with Form 43-101F1:

- Stewart Twigg, P.Eng., Manager of Projects, Amec Foster Wheeler Lima
- William Colquhoun, Pr Eng, FSAIMM, Principal Process Manager, Amec Foster Wheeler Lima
- Vikram Khera, P.Eng., Senior Financial Analyst, Amec Foster Wheeler Oakville
- Luis Vela, CMC, Vice President Exploration, Panoro Minerals Ltd
- Jesse Aarsen, P.Eng., Senior Associate, Moose Mountain Technical Services.

Figure 2-1: Location Map

Note: Figure courtesy Panoro and Tetra Tech, 2014.

Figure 2-2: Property Location Map



Note: Figure courtesy Panoro and Tetra Tech, 2014.

2.3 Site Visits and Scope of Personal Inspection

Mr Luis Vela has been to site on numerous occasions since he first became involved in the Project in 2011, the most recent being 1 to 3 July, 2015 and 6 to 9 September, 2015. During his site visits, Mr Vela has reviewed the process of data acquisition and viewed interpreted geological sections and grade shell models. While no drilling was underway during the most recent site visits, Mr Vela was on site during the 2011–2013 programs, and reviewed drilling and sampling practices in the field, and sample security measures.

Mr Vela has also inspected core in the core storage warehouse in Cusco, and reviewed core handling, security and storage protocols and practices.

No other QPs have been to site.

2.4 Effective Dates

The Report has a number of effective dates as follows:

- Date of the Mineral Resource estimate: 20 June, 2013
- Date of the information included in the report on the most recent drilling program: 17 February, 2014
- Date of the latest information on mineral tenure, surface rights and Project ownership: 1 April 2015
- Date of the updated mine plan that supports the updated PEA: 22 September, 2015
- Date of the financial analysis: 22 September, 2015.

The overall report effective date is taken to be the date of the financial analysis and is 22 September, 2015.

2.5 Information Sources and References

Information used to support this Report was also derived from previous technical reports on the Project, and from the reports and documents listed Sections 3 and 27. Additional information was sought from Panoro personnel where required.

Mr Alvaro Paredes and Mr Carlos Barrantes of Amec Foster Wheeler provided specialist information to Mr Twigg in relation to social, permitting and environmental aspects of the Project that were used in Section 20, Section 24.1, and Section 26.1.6 of the Report.

Mr Sergio Munoz and Mr Antonio Peralta of Amec Foster Wheeler provided specialist information to Mr Twigg, Mr Colquhoun, and Mr Khera in relation to risks and opportunities in relation to potential mining operations that were used in Section 24.1 and Section 24.2 of the Report.

2.6 Previous Technical Reports

Panoro has previously filed technical reports on the Project as follows:

- Lee, C. Nowak, M. and Wober, H. H., 2007: Independent Technical Report on the Mineral Exploration Properties of the Cordillera de las Minas S.A. Andahuaylas-Yauri Belt, Cusco Region, Perú: report prepared by SRK Consulting (Canada) Inc. for Panoro, effective date 9 March 2007, 125 p.
- Wright, C., Arrias, W., and Vilela, M., 2009: Restated, Amended Technical Report for the Antilla Property Apurimac, Perú: report prepared by AMEC (Perú) S.A. for Panoro, effective date 1 June, 2009, amended 23 August, 2009, 100 p.

- Wright, C., and Colquhoun, W., 2012: Cotabambas Copper-Gold Project, National Instrument 43-101 Technical Report on a Mineral Resource Estimate, Apurímac, Perú: report prepared by AMEC (Perú) S.A. for Panoro, effective date 24 July, 2012, 120 p.
- Morrison, R.S., Huang, J., and Daigle, P., 2014: Technical Report and Resource Estimate of the Cotabambas Copper-Gold Project, Perú: report prepared by Tetra Tech WEI Inc. for Panoro, effective date 29 October 2013, amended 7 July 2014, 130 p.
- Twigg, S., Munoz, S., Colquhoun, W., Khera, V., Morrison, R., Hirst, J., Daigle, P., and Searston, S., 2015: Cotabambas Project, Apurimac, Perú, NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared by Amec Foster Wheeler and Tetra Tech WEI Inc. for Panoro, effective date 9 April, 2015, 318 p.

3.0 RELIANCE ON OTHER EXPERTS

The QP authors of this Report state that they are qualified persons for those areas as identified in the "Certificate of Qualified Person" for each QP, as included in this Report. The QPs have relied, and believe there is a reasonable basis for this reliance, upon the following other expert reports, which provided information regarding mineral rights, surface rights, and environmental status in sections of this Report as noted below.

3.1 Mineral Tenure, Surface Rights and Agreements

The QPs have fully relied upon, and disclaim responsibility for, information derived from legal experts for the information on mineral tenure, surface rights, and agreements through the following documents:

- Martinez, H., 2012a: Legal Opinion Concessions Cotabambas: independent legal opinion from Rosseló Attorneys at Law on the Cotabambas Property exploration concessions, underlying agreement, termination of agreement, injunction by Chancadora Centauro S.A. and opinion of likely decision by arbitrator in favour of Panoro, 15 May, 2012, 6 p.
- Martinez, H., 2012b: Absolution of Questions Regarding the Cotabambas Property: independent legal opinion from Rosseló Attorneys at Law on the Cotabambas Property, update on arbitration, mineral concession status, third party or government royalty, drilling permits, surface rights ownership and community agreements, 3 October, 2012, 3 p.
- Martinez, H., 2015: Legal Opinion: preliminary independent legal opinion from Rosseló Attorneys at Law on the Cotabambas Property, 1 April 2015, 4 p.
- Martinez, H., 2015: Legal Opinion: independent legal opinion from Rosseló Attorneys at Law on the Cotabambas Property, 14 May 2015, 6 p.
- Shaheen, L., 2015: Email to Amec Foster Wheeler, 6 November 2015 regarding confirmation of information used in the report in relation to Project access, mineral tenure and surface rights.

This information is used in Section 4 of the Report and in the summary and interpretations and conclusions in Sections 1 and 25.

3.2 Environmental, Permitting and Social Licence

The QPs have not reviewed the environmental, archaeological, social and community status of the Project. The QPs have fully relied upon, and disclaim responsibility for, information derived from experts for this information through the following documents:

- MEM, 2012. Resolucion Directoral N° 194-2012-MEM-AAM. Ministry of Energy and Mines approval of Panoro's modification to semi-detailed EIA for a drill program on the Property. Signed and sealed by Dr. Manuel Castro Baca, Director General Asuntos Ambientales Mineros of the MEM. 10 p.
- Schlumberger Water Services, 2012. EIA Semi Detallado de la Modificación del Proyecto Cotabambas-Ccalla. Semi-detailed environmental impact assessment study completed by Schlumberger Water Services for Panoro dated July 2012.

This information is used in Sections 4, 5 and 20 of the Report, and in the summary and interpretations and conclusions in Sections 1 and 25.

3.3 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information derived from Panoro for the information on taxation applicable to the Cotabambas Project at a project level through the following document:

- Panoro Minerals Ltd. 2015: Cotabambas Project: Preliminary Economic Assessment NI 43-101 Technical Report – Taxation Notes: Notes and reports collected by Panoro Minerals Ltd for Amec Foster Wheeler (Perú) S.A., 11 May, 2015, 45 p.

This information is used in Sections 4 and 22 of the Report and in the summary and interpretations and conclusions in Sections 1 and 25 that relate to the financial analysis.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Project is located 545 km southeast of Lima, the capital city of Perú, 50 km southwest of Cusco, 60 km east of Abancay, capital of Apurimac Region, and 1 km south of the village of Ccalla and 500 m to the northwest of the village of Cotabambas.

The centre of the Project is located at coordinates 784,500 mN and 8,480,000 mE, Universal Transverse Mercator (UTM) Zone 18.

The Project location is indicated in Figure 4-1.

4.1 Project Ownership

The Cotabambas Property mining and exploration concessions are 100% held by Panoro through its indirectly wholly-owned Peruvian subsidiary Panoro Apurimac S.A.

4.2 Mineral Tenure

The Project area consists of 17 concessions, and covers a total land area of 15,900 ha (Figure 4-1).

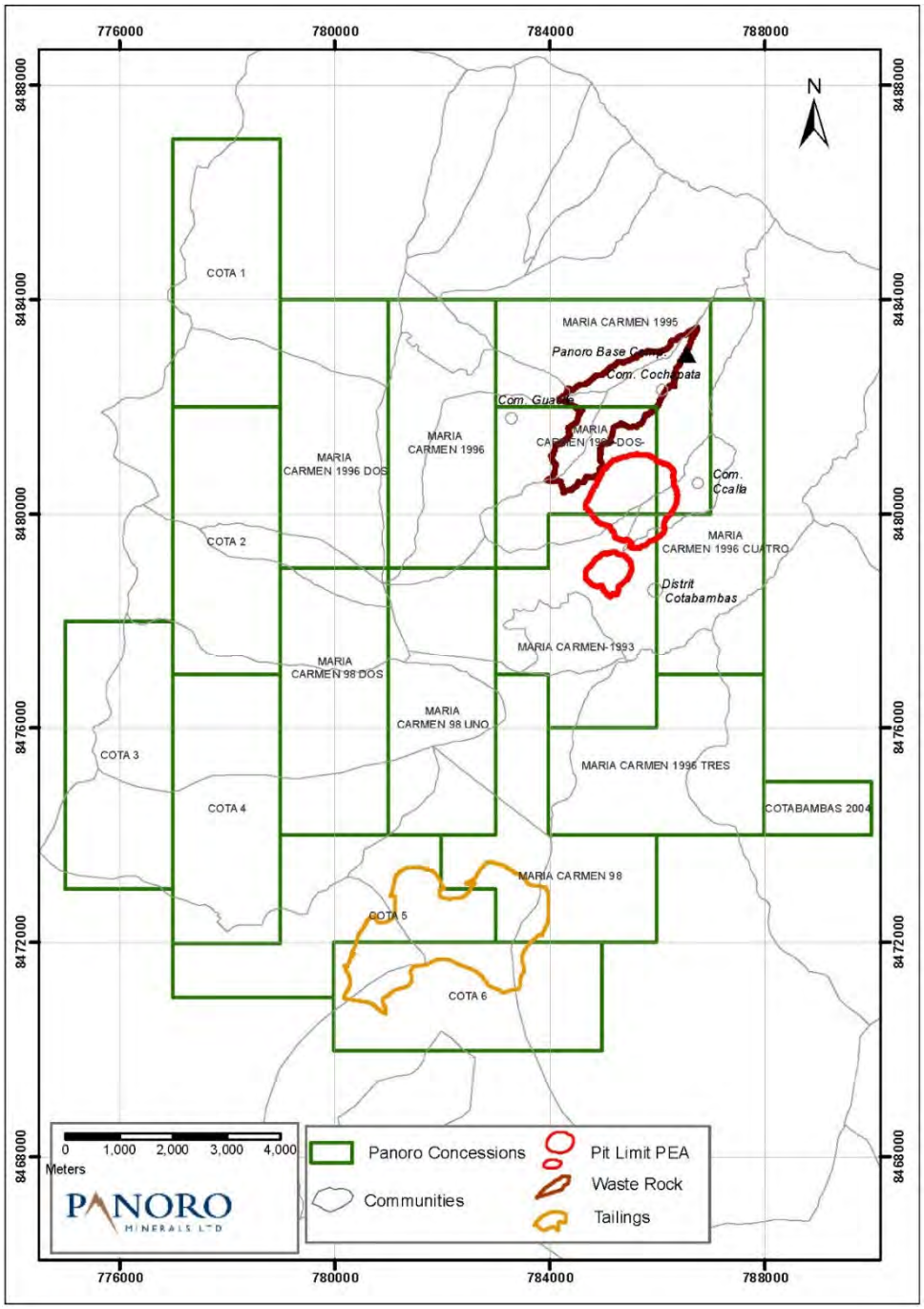
Concessions are located and registered by UTM coordinates and there is no requirement for physical location on the ground.

The concessions cover the Ccalla and Azulccacca deposits which are situated mainly within Concessions 10077493 (Maria Carmen-1993) and 10214793 (Maria Carmen 1993 Dos).

Panoro has paid the concession fees (annual fees and penalty) in respect of all of the Cotabambas concessions until the year 2015, which has met the requirements to keep the concessions current until June 2017 (Rosseló, 2015). Each year that Panoro meets the fee payments will extend the concession currency by an additional year (i.e. payment of the fees by June 2016 will extend the currency to June 2018 etc.). Current annual concession fees payable on the 15,900 ha total US$47,700 or US$3/ha.

Mineral tenure is summarized in Table 4-1.

Figure 4-1: Cotabambas Concession Map



Note: Figure courtesy Panoro, 2015. Infrastructure locations shown in the figure are conceptual locations determined for the purposes of the PEA.

Table 4-1: Cotabambas Exploration Concessions

Concession No.	Concession Name	Area (ha)	Date Compliant	Date of Certification	Entry Record Number	Departmental Resolution Record	Valid To
10077493	Maria Carmen 1993	1,000	21/05/1993	27/10/1994	20001871	6752-1994-RPM	June 2017
10214793	Maria Carmen 1993 Dos	700	04/10/1993	21/07/1994	20001865	3659-1994-RPM	June 2017
10221295	Maria Carmen 1995	1,000	02/01/1995	25/06/1996	20002348	3396-1996-RPM	June 2017
10128796	Maria Carmen 1996	1,000	02/05/1996	24/07/1996	20002372	3924-1996-RPM	June 2017
10142696	Maria Carmen 1996 Cuatro	1,000	14/05/1996	26/07/1996	20002374	4069-1996-RPM	June 2017
10142496	Maria Carmen 1996 Dos	1,000	14/05/1996	24/07/1996	20003121	3928-1996-RPM	June 2017
10142596	Maria Carmen 1996 Tres	1,000	14/05/1996	26/07/1996	20002373	4064-1996-RPM	June 2017
10087098	Maria Carmen 1998	1,000	04/05/1998	31/08/1998	11014198	2824-1998-RPM	June 2017
10086398	Maria Carmen 1998 Dos	1,000	04/05/1998	21/08/1998	11014261	2563-1998-RPM	June 2017
10086898	Maria Carmen 1998 Uno	1,000	04/051998	31/08/1998	11015294	2826-1998-RPM	June 2017
10230704	Cotabambas 2004	200	01/07/2004	27/10/2004	11036655	3815-2004-RPM	June 2017
10138512	COTA 1	1,000	09-04-2012	07/09/2012	11128518	3086-2012-INGEMET	June 2017
10138612	COTA 2	1,000	09/04/2012	05/02/2013	11131465	5092-2012-INGEMET	June 2017
10138712	COTA 3	1,000	09/04/2012	10/10/2012	11128522	3592-2012-INGEMET	June 2017
10138412	COTA 4	1,000	09/04/2012	07/09/2012	11128523	3178-2012-INGEMET	June 2017
10138812	COTA 5	1,000	09/04/2012	07/09/2012	11128696	3056-2012-INGEMET	June 2017
10138912	COTA 6	1,000	09/04/2012	07/09/2012	11128698	3153-2012-INGEMET	June 2017
Total		**15,900**					

4.3 Surface Rights

Panoro does not currently own any surface rights on the Property. Surface rights ownership is held by the District of Cotabambas, the communities of Ccochapata, Ccalla and Guaclle, and individual surface rights holders (see also the discussion in Section 4.5).

To date Panoro has been able to negotiate access to the Project with the appropriate surface rights holders in support of exploration and drilling activities. Project advancement will require continued cooperation with these surface rights holders for surface rights access.

4.4 Royalties and Encumbrances

The Mining Royalty and Special Mining Tax are tributes paid to the Peruvian Government for the exploitation of mineral resources which were developed by government and industry association and implemented at the end of 2011. The Cotabambas Project would be subject to these royalties once in production (Martinez, 2012b).

The Peruvian government currently levies a sliding-scale royalty on gross sales from mining operations that ranges between 1% and 12%, and which is calculated based on operating margin. Panoro considers that a minimum royalty of 1% of mining sales would be applicable to the Cotabambas Project at a PEA stage of evaluation.

4.5 Property Agreements

Agreements have previously been negotiated with the surrounding communities of Ccochapata, Ccalla, and Guaclle, as well as individual surface rights holders in the District of Cotabambas, which have allowed Panoro to conduct exploration activities from 2010 through 2013.

In 2012 the agreement with the community of Ccalla was renewed for three more years, starting in November 1st 2012 and expiring 29 October, 2015.

The agreement with the Guaclle community was not renewed in 2013 as the land covered by the agreement was not considered a priority target for the focus of exploration activities at the time.

The agreement with the Ccochapata community expired in July 2014.

In 2014, agreements with the communities of Ccaranca and Chaupec were negotiated in support of exploration programs, both had a validity of six months, and ended in March and April of 2015 respectively.

When exploration activities recommence, access agreements with the affected communities where exploration is planned would be negotiated as required.

4.6 Permits

A semi-detailed environmental impact assessment (SDEIA) was completed and subsequently expanded to allow Panoro to drill up to 200 drill holes on the Cotabambas property (SWS, 2012). This permit is still valid.

Exploration activities were permitted for water extraction at the rate of 0.1 L/second.

Panoro does not to date have any agreements with the other communities for exploration outside the permitted areas, and would need to be obtained, for example, for any future geotechnical and geomechanical drill hole campaigns for the proposed tailings and waste dump locations.

Additional information on permitting assumptions is included in Section 20. The permitting assumptions and permissions may need to be revisited if the scope of the exploration activities changes in order to support pre-feasibility or feasibility studies.

4.7 Environment, Environmental Liabilities and Social Licence

Currently, the environmental liabilities are restricted to those expected to be associated with an exploration-stage project, and include drill sites, and access roads.

Air and water quality are reported to be below national guidelines for particulates, gasses and dissolved metals (SWS, 2012). Closure plans for the ongoing drill program have been approved by the Peruvian environmental agency. This applies only for the areas of the exploration programs, it does not apply for the rest of the proposed areas of the project, such as tailings, waste dump, water dams, process plant and camp.

No archaeological sites were identified in the area affected by the exploration activities (SWS, 2012).

Environmental and social base line studies and potential environmental and social effects of development and operation of the Project are discussed in Section 20 of this Report.

4.8 Comments on Section 4

In the opinion of the QP responsible for Section 4, the information discussed in this section supports the declaration of Mineral Resources, based on the following:

- Information from legal experts and Panoro experts supports that Panoro holds 100% of the Project

- Information from legal experts supports that the mining tenure comprising the Property is valid and is sufficient to support declaration of Mineral Resources
- Panoro holds no surface rights in the area, and any future mining activities will require agreements to be negotiated with both local communities and individual surface rights holders
- Exploration activities to date have been completed under the appropriate Peruvian permits; additional agreements and permits will be required when exploration activities recommence at the Project site
- Panoro will need to apply for additional permits as appropriate under applicable Peruvian laws to allow future mining operations
- Based on information provided by Panoro and third-party experts retained by Panoro, to the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project
- Notwithstanding the information contained above in this Section 4, there is no guarantee that title to any of the Project will not be challenged or impaired, and third parties may have valid claims affecting the Project, including prior unregistered liens, agreements, transfers or claims, including aboriginal land claims, and title may be affected by, among other things, undetected defects. As a result, there remains a risk that there may be future constraints on Panoro's ability to operate the Project, or Panoro may be unable to enforce rights with respect to the Project.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Cotabambas Project can be accessed by road from Cusco following the paved highway (3S), 32 km from Cusco west to the town of Anta and then via a gravel road (3SF) for approximately 115 km to the town of Cotabambas. The gravel road makes a very steep descent to the Apurimac River, crosses it using the Huallpachaca bridge, and then climbs back up to the Town of Cotabambas (Figure 5-1).

The drive from Cusco to the Project area is typically 3.5 hours.

The (3SF) gravel road continues south approximately 83 km to the Las Bambas Mine operated by MMG Limited. From this point, the roads have been recently upgraded and join the Matarani port via the following routes:

- A 97 km long paved road from Las Bambas to Quiñacota
- A 114 km long paved road from Quiñacota to Espinar
- A 139 km long gravel road in good condition from Espinar to Imata
- A 251 km long paved road from Imata to the port of Matarani.

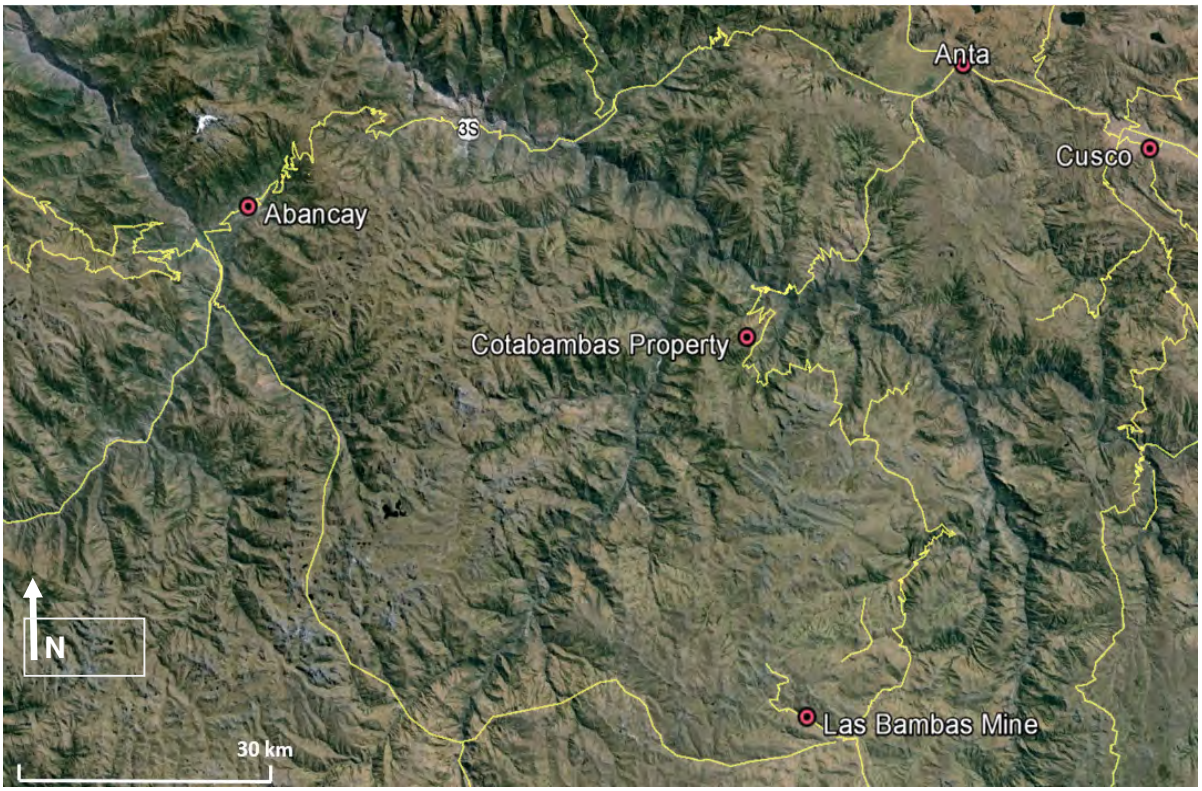
This second access is planned in the PEA to be used for mineral exportation and major equipment supply to the Project. To be fully operational, the 83 km portion from Cotabambas to Las Bambas Mine will need to be upgraded. Refer also to discussions in Section 18 of the Report on infrastructure and access.

There are regular flights to and from Cusco. The flight time from Lima to Cusco is typically one hour. The nearest railhead is in Izcuchaca, a town about 20 km west of Cusco.

5.2 Climate

The region's climate is typical of the Southern Peruvian Andes. There are two main seasons. The Andean summer is May to October and is characterized by very low rainfall, sunny days and cool nights with temperatures ranging from night-time lows of 5°C to 10°C and daytime high temperatures of 18°C to 24°C. Most of the region's rainfall occurs in the Andean winter from November to April. Winter temperatures ranges are not as large as those of the summer season, and range from lows near 10°C to highs around 18°C.

Figure 5-1: Cotabambas Access Map



Note: Figure prepared by Amec Foster Wheeler, 2015 using a Google Earth backdrop. The Las Bambas mine is operated by third parties and is not part of the Cotabambas Project.

The average annual precipitation in the area is close to 1,000 mm, with an average temperature of 13°C.

5.3 Local Resources and Infrastructure

Cusco, with a 2013 population of 413,000, is the closest major town to the Project and can provide most supplies for the base camp. Basic supplies, food, and fuel to support exploration-level activities can be sourced from the surrounding villages.

Unskilled labour to support exploration and for future construction and operations could be sourced locally. More specialized skilled labour is available in the cities of Cusco and Arequipa. A significant mining labour force supporting both small scale and large scale mining and mineral processing activities exists in the region.

The town of Cotabambas has a population of approximately 2,000. Personnel accommodations are available in Cotabambas. The town is connected to the national telephone network. Electrical power is supplied to the town from the Cachimayo electrical station through a series of distribution lines at 138 to 33 kV (DEPMEM, 2004).

Outside of the town of Cotabambas, approximately 500 people live in small settlements around the Project area. Approximately 2 km north of Cotabambas is the community of Ccalla. One kilometre north of Ccalla is the community of Ccochapata. To the west of Ccalla and Ccochapata is the community of Guaclle.

Panoro rents facilities that are used as a permanent base camp next to the Cochapata village; these facilities comprise fixed buildings for offices and core logging, sampling and storage facilities. The base camp also serves as equipment storage depot and garage. A second base of operations is rented in the village of Ccalla, consisting of offices, a kitchen, and a second drill core logging and sampling facility.

Infrastructure within the Cotabambas Project is basic and limited to a small network of access roads. There is intermittent cellular telephone coverage within the Project. Electrical power is presently supplemented by portable generators which are sufficient for exploration activities however; a high tension line will be required to bring sufficient power to the site for large scale mining and milling operations. Water for exploration activities is sourced from nearby streams.

Additional information on infrastructure relevant to the PEA is included in Section 18.

5.4 Physiography

Cotabambas is located in mountainous terrain of the high Andean Cordillera. Elevations on the property vary between approximately 3,000 and 4,000 masl. The Project physiography is dominated by northeast-trending ridges separated by *quebradas* or ravines.

The Azulccacca area is to the south and occurs on a high ridge separated from the Ccalla area to the north by the Quebrada Azulccacca (Figure 5-2). The Ccalla area is approximately 500 m lower in elevation than the Azulccacca area, but on a similar northeast-trending ridge.

The region is characterized by deeply incised river valleys and canyons such as the Apurimac River valley which is 2,000 m below the Cotabambas Project area.

Figure 5-2: Photograph of the Cotabambas Deposit Area

Note: Photo taken from Azulccacca north-west to Ccalla with the Apurimac River valley in the background. Photo taken by Amec Foster Wheeler in May, 2012. Width of field of view in mid-ground is 2,000 m.

The area is vegetated by tough mountain grasses and shrubs, with portions being cultivated by local farmers. In general, the property is above the tree line with the only trees being the non-indigenous Eucalyptus and pine, which have been planted around communities and on hill slopes and along roadways to control erosion.

5.5 Comments on Section 5

In the opinion of the QP responsible for Section 5:

- Mining activities should be capable of being conducted year-round
- There is sufficient suitable land available for any future tailings disposal, mine waste disposal, and related mine infrastructure within the mineral claims. The infrastructure assumptions in the PEA are within the claim boundaries.

6.0 HISTORY

There is no known production from the Project.

6.1 Antofagasta Minerals (1995 to 2002)

In 1995, Anaconda Perú S.A., a Peruvian subsidiary of Antofagasta Plc (Antofagasta), carried out mapping, soil and rock geochemical sampling programs, and geophysical surveys over the Ccalla, Ccochapata, Azulccacca, and Guaclle areas of the Project.

The first diamond drill testing of the surface soil and rock geochemical and geophysical anomalies occurred in July 1996. Intermittent drilling continued until April 2000. In total, 24 drill holes totaling 8,538 m were completed, with numerous mineralized intervals intersected. The results of these drill campaigns were reported in internal company reports by Val d'Or (1996) and Perello et al. (2001).

No exploration was known to be conducted between 2000 and 2002.

6.2 Cordillera de las Minas (2002 to 2006)

Antofagasta and Companhia do Rio Vale Doce (CVRD) formed a joint venture company called Cordillera de las Minas (CDLM) in 2002, and transferred ownership of several groups of exploration concessions in southern Perú to CDLM.

From 2002 to 2006, CDLM carried out additional mapping, surface rock and soil geochemical sampling, induced polarization (IP) surveying, magnetometer surveying, and diamond drill testing of previously identified geological, geochemical, and geophysical anomalies. In total, nine drill holes totalling 3,252 m were drilled.

Drill holes from the CDLM campaigns were logged for descriptive rock type and alteration using graphic logs and geotechnical data such as fracture density, recovery and RQD were recorded. Samples were sent for analysis to the CIMM laboratory in Lima. Analyses for total copper, arsenic, silver, gold, lead and zinc and sequential soluble copper were carried out at CIMM. No independent QA/QC procedures were followed for this assaying. Density determinations were also made on a systematic basis; however, details about the procedures and the original measurements are unknown.

6.3 Panoro Minerals (2007 to Present)

In March 2007, Panoro paid US$16.6 million to acquire all outstanding shares of CDLM on the Lima exchange. The deal saw Panoro acquire 13 properties, including the Cotabambas Project.

From 2011 to present, Panoro completed additional mapping, surface rock and stream sediment geochemical sampling, IP surveying, and magnetometer surveying. Panoro also conducted diamond drill testing of geological, geochemical, and geophysical anomalies in Ccalla and Azulccacca deposits. In total, 121 holes (63,198 m) have been completed, with the most recent drill hole completed in February 2014.

Mineral Resources were estimated in 2009, and updated in 2012 and 2014.

A preliminary economic assessment (PEA) was prepared in April 2015. An update to the PEA is presented in the later sections of this Report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Andahuaylas–Yauri belt is located immediately south of the Abancay deflection of the cordillera where thrust faulting, oriented dominantly north–south, is deflected to strike northwest–southeast (Figure 7-1). At the deflection, the normal subduction of southern Perú and northern Chile changes to flatter subduction below central and northern Perú.

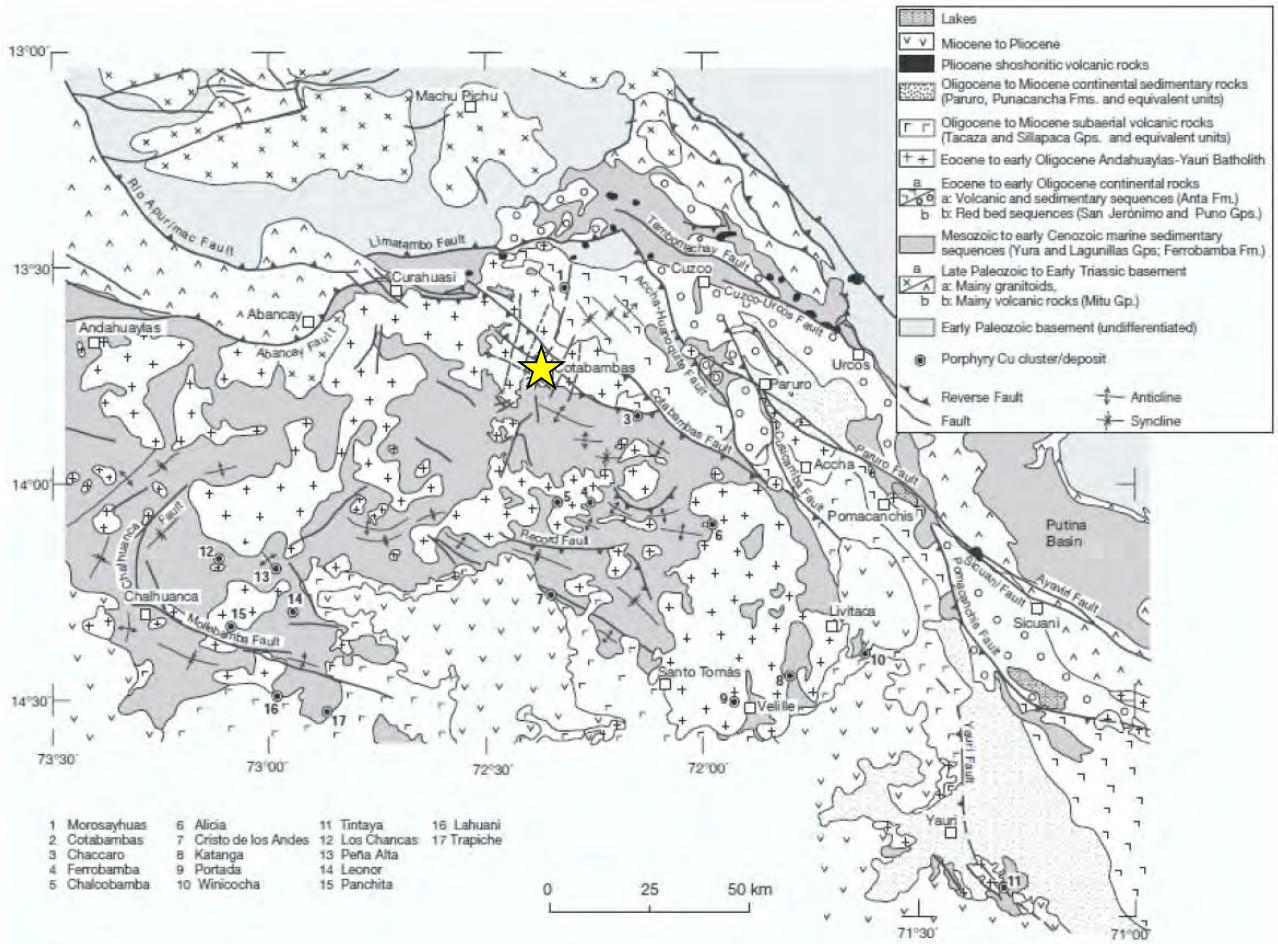
The geology of the Andahuaylas–Yauri belt is dominated by the Andahuaylas–Yauri batholith which is exposed for approximately 300 km between the towns of Yauri in the southeast and Andahuaylas in the northwest, and Mesozoic to Early Cenozoic clastic and marine sediment sequences (Figure 7-2). The batholith ranges from 25 km wide at the east end to 130 km wide near Abancay, and is composed of early mafic to intermediate intrusive with cumulate textures, grading to intermediate intrusive rocks with equigranular to porphyritic textures. The batholith intrudes Precambrian to Palaeozoic basement rocks which are exposed to the northeast. The basement sequence culminates in Permian to Early Triassic age Mitu Group volcaniclastic and clastic rocks.

The basement is overlain by Mesozoic and Cenozoic sediments deposited in the Eastern and Western Peruvian basins. The eastern basin is made up of marine clastic and carbonate rocks. The northeastern edge of the western basin includes the Lagunilla and Yura Groups, made up of middle to late Jurassic quartz–arenite, quartzite and shale, that grades upward to massive micritic limestone, shale and chert of the Mara and Ferrobamba Formations. At the top of the Yura Group is the Soraya Formation, composed of arenite, quartz arenite and quartzite. The Yura Group hosts the Cotabambas deposit.

Eocene and Oligocene stratigraphy is dominated by the sedimentary San Jerónimo Group and the dominantly volcanic Anta Formation, which un-conformably overlie the Mesozoic and Cenozoic sediments. Miocene and Pliocene volcanics and sediments overlie Oligocene sediments. A discontinuous veneer of Pleistocene fluvio-glacial sediments and re-worked gravels overlie the region.

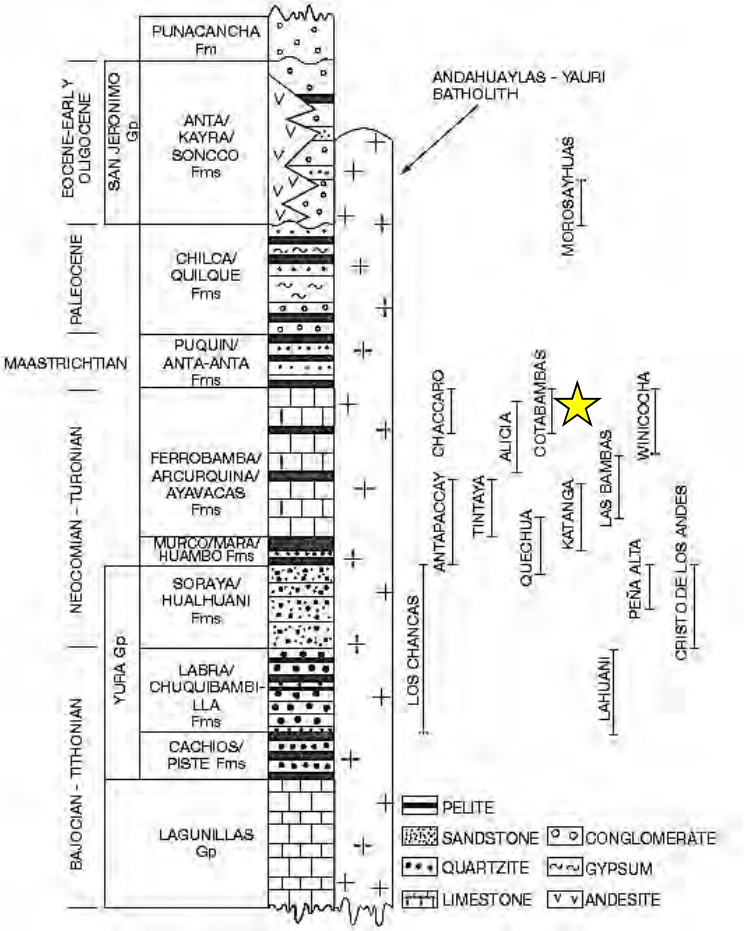
Major mineralization styles in the region include porphyry copper (± Mo ± Au), iron-copper skarn, replacement, and sediment-hosted oxide zinc deposits, and minor epithermal vein- style mineralization.

Figure 7-1: Regional Geology of the Yauri–Andahuaylas Belt



Note: Figure from Perelló et.al, 2003. Star indicates the location of the Cotabambas Project.

Figure 7-2: Regional Stratigraphic Sequence



Source: Perelló et.al. (2003). The yellow star highlights the approximate stratigraphic position of the Cotabambas deposit.

7.2 Project Geology

Based on an understanding of the regional geology of the area and Project-wide 1:10,000 and 1:5,000 scale mapping, the geology of the Cotabambas Project is dominated by:

- Andesite of the Eocene to early Oligocene Anta Formation
- Diorite related to the Eocene to early Oligocene Andahuaylas–Yauri Batholith
- Later, altered, mineralized monzonite porphyry, also related to the Eocene to early Oligocene Andahuaylas–Yauri Batholith
- Late dacite volcanic dome and associated latite dikes.

The emplacement of the quartz monzonite porphyry and later latite dykes are controlled by a system of strong sub-vertical fault and shear zones that have an azimuth of approximately 030°. A second set of structures, perpendicular to the 030° system and parallel to the regional thrust fault systems with azimuth 120° runs between the Ccalla area and the Guaclle area to the west (Figure 7-3).

Figure 7-4 provides a detailed geological interpretation compiled by Panoro in the Ccalla and Azulccacca porphyry areas. The two porphyries cover an area about 2.5 km long and 1.5 km wide.

7.2.1 Mineralization

Mineralization occurs in hypogene, supergene enrichment and oxide zones. A well-developed leached cap hosts the oxide mineralization. Sulphide mineralization occurs below the base of the leached cap. This zonation is considered to be typical of porphyry-style copper and porphyry-style copper–gold deposits (Figure 7-5).

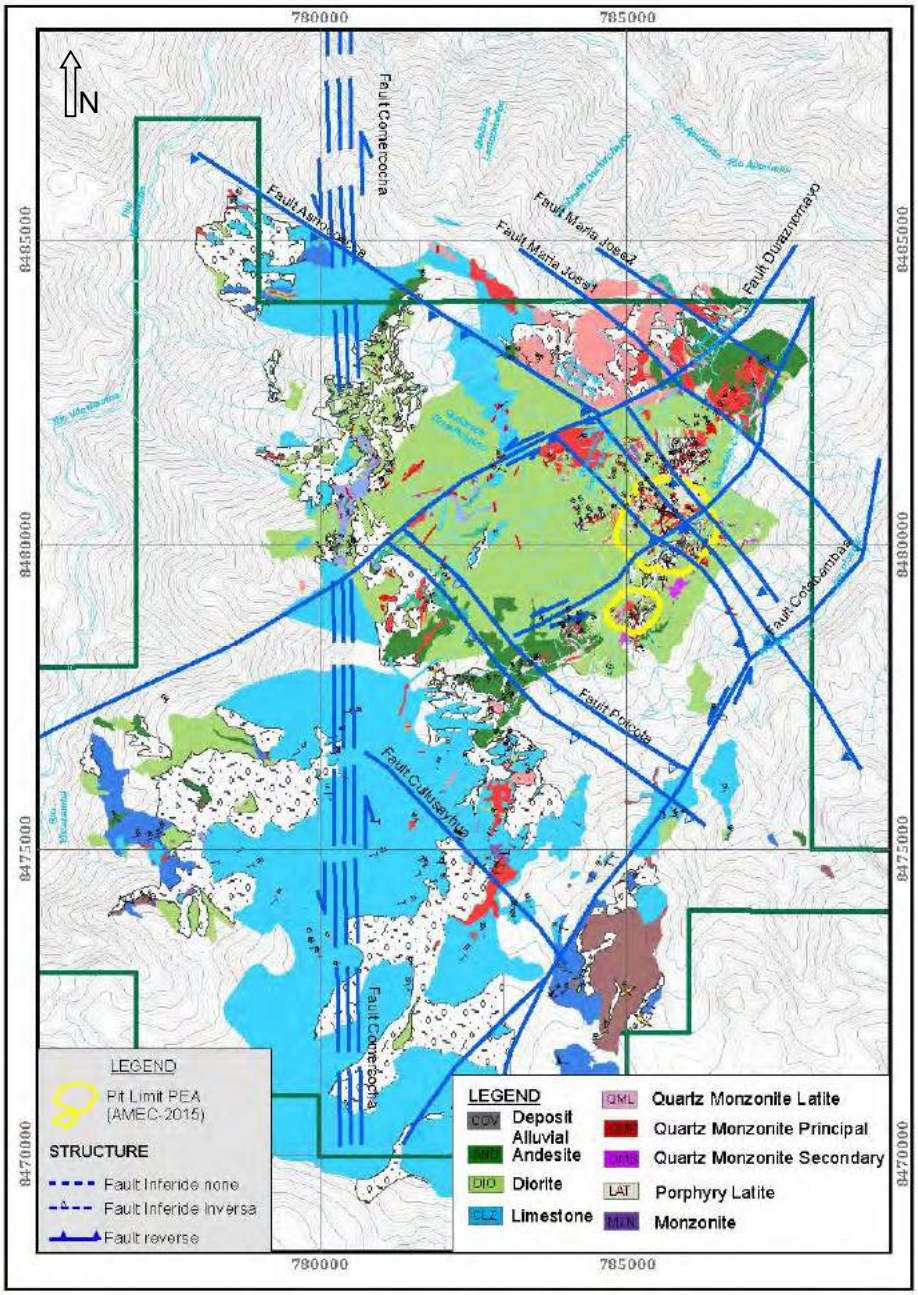
Figure 7-6, Figure 7-7 and Figure 7-8 show the distributions of gold, copper, and geochemical and mineralogical assemblages for cross section 12E through the Cotabambas deposit (refer to Figure 7-4 for the section location).

Hypogene Mineralization

Hypogene mineralization in the Project area has been intersected at depths from approximately 20 m from surface to depths of over 500 m from surface.

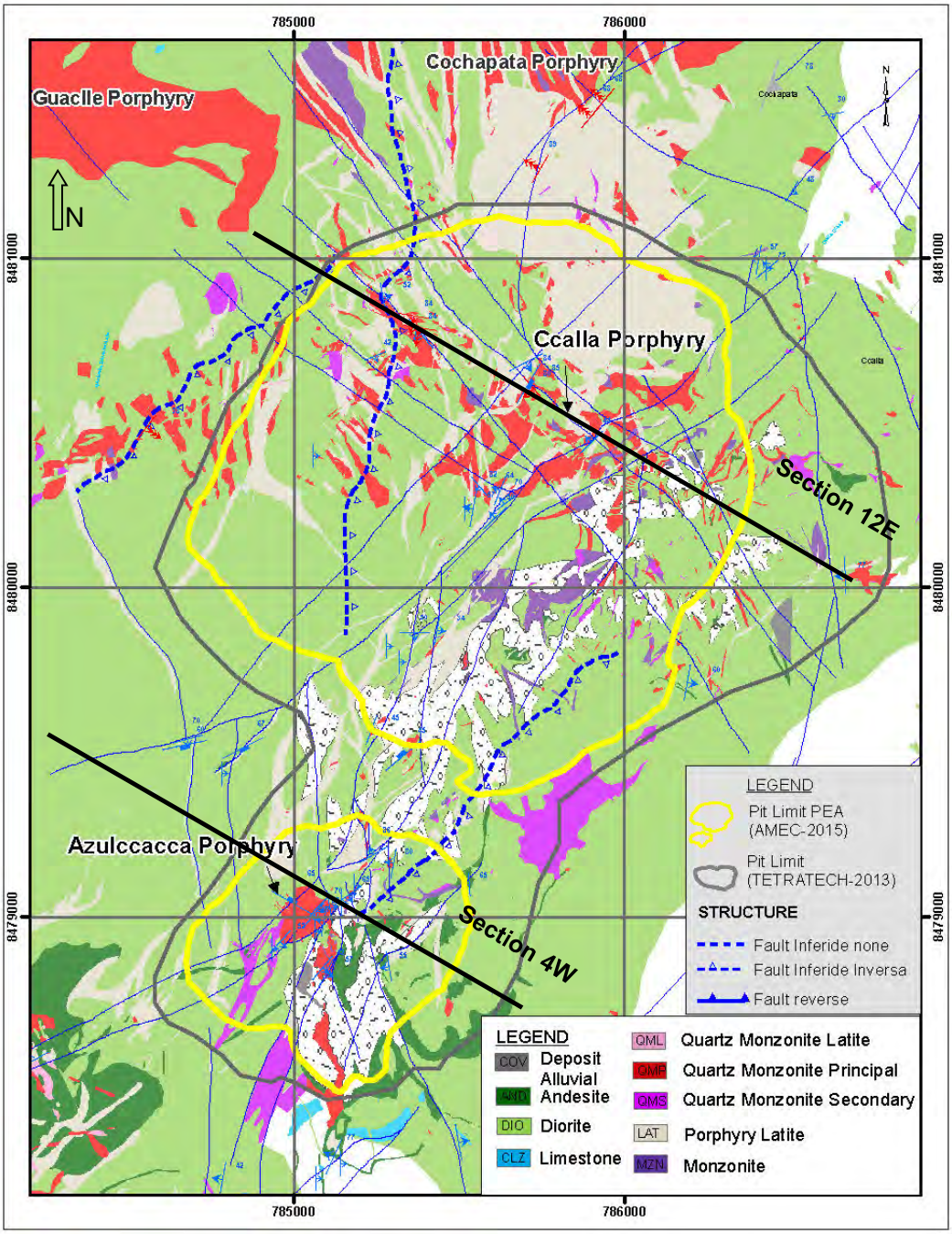
Hypogene copper–gold–silver mineralization is best developed with pyrite mineralization in quartz–sericite-altered quartz monzonite porphyry dykes running parallel to the north north-east trending structural corridors at Ccalla and Azulccacca. Mineralization occurs as disseminated chalcopyrite and pyrite, pyrite-chalcopyrite stringers or veinlets and quartz chalcopyrite pyrite veinlets. Local patches of hypogene mineralization are developed in diorite, peripheral to the quartz monzonite porphyry, where the north–northeast-trending structural system passes within 10 to 20 m of the diorite–porphyry contact. Chalcopyrite mineralization intensity decreases and disseminated pyrite mineralization increases distal to the higher grade parts of the hypogene zone.

Figure 7-3: Project Geology Plan



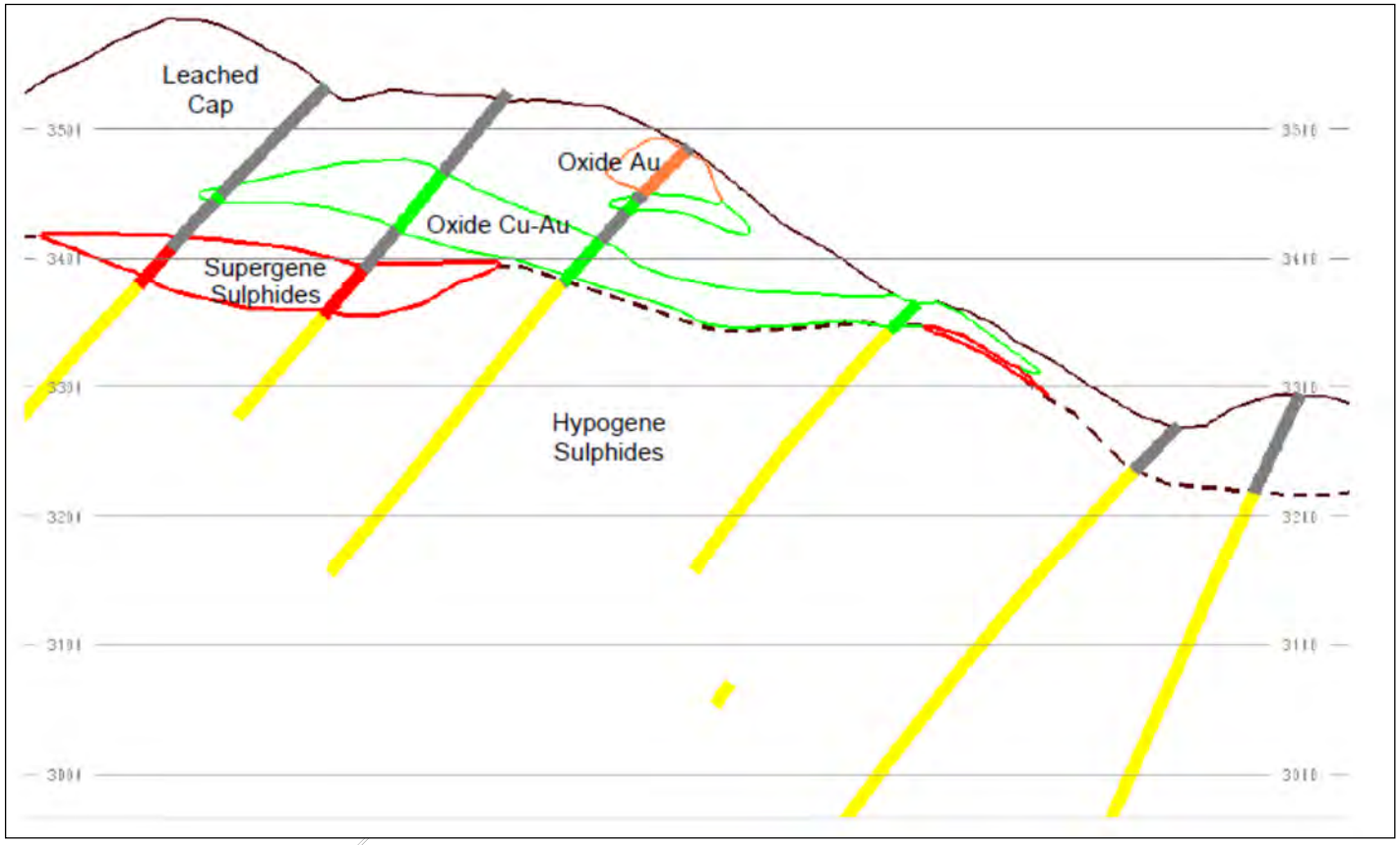
Note: Figure courtesy Panoro, 2015. In the legend, "Fault Inferide none" = inferred fault (unknown sense of slip), and "Fault Inferide inversa" = inferred reverse fault

Figure 7-4: Deposit Geology Plan



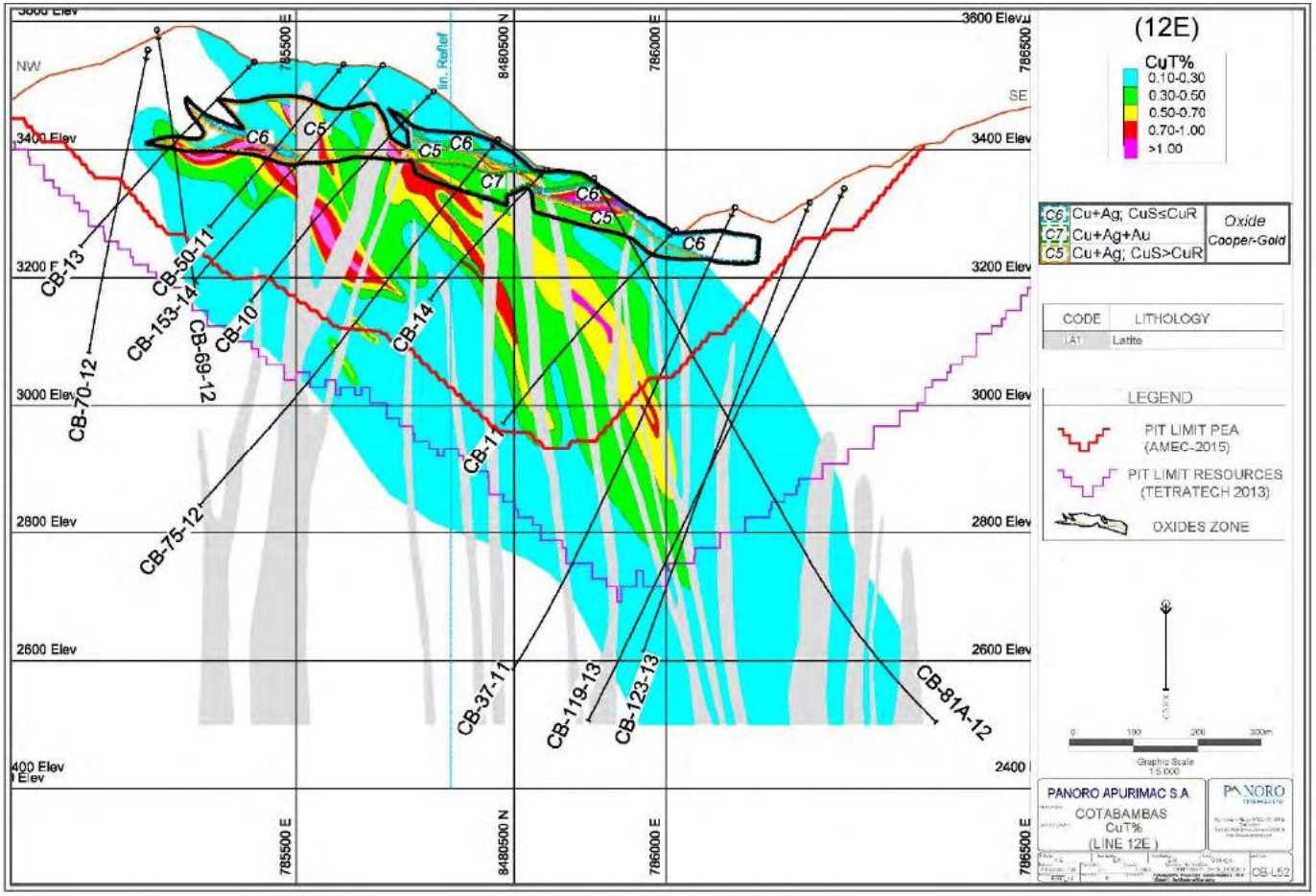
Note: Figure courtesy Panoro, 2015. In the legend, "Fault Inferide none" = inferred fault (unknown sense of slip), and "Fault Inferide inversa" = inferred reverse fault

Figure 7-5: Mineralization Zonation Schematic



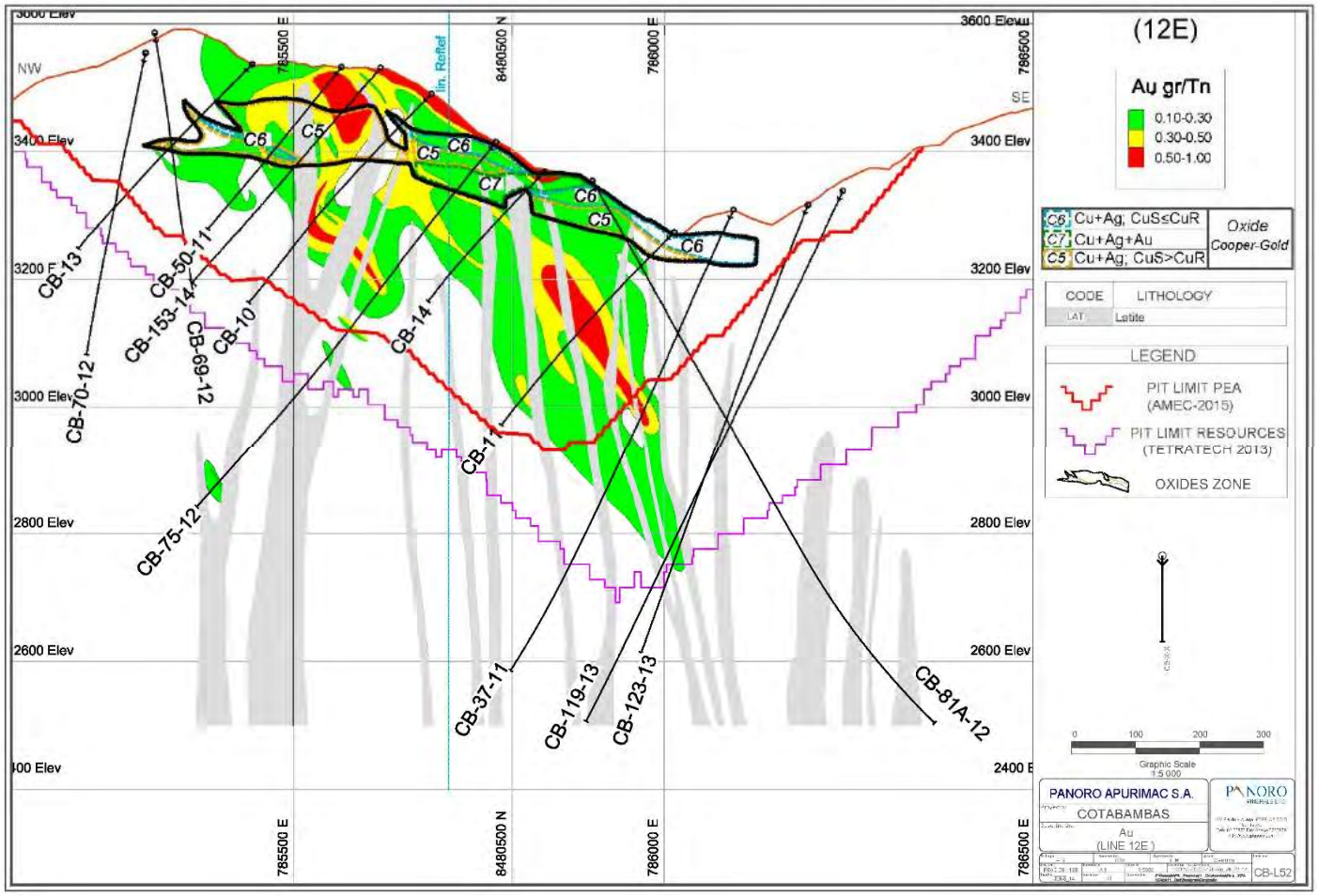
Note: Yellow designates hypogene mineralization; red designates supergene mineralization; green designates oxide mineralization; orange designates leach-cap mineralization. Figure prepared by Amec Foster Wheeler, 2012.

Figure 7-6: Section 12E Showing Distribution of Copper Values Note:



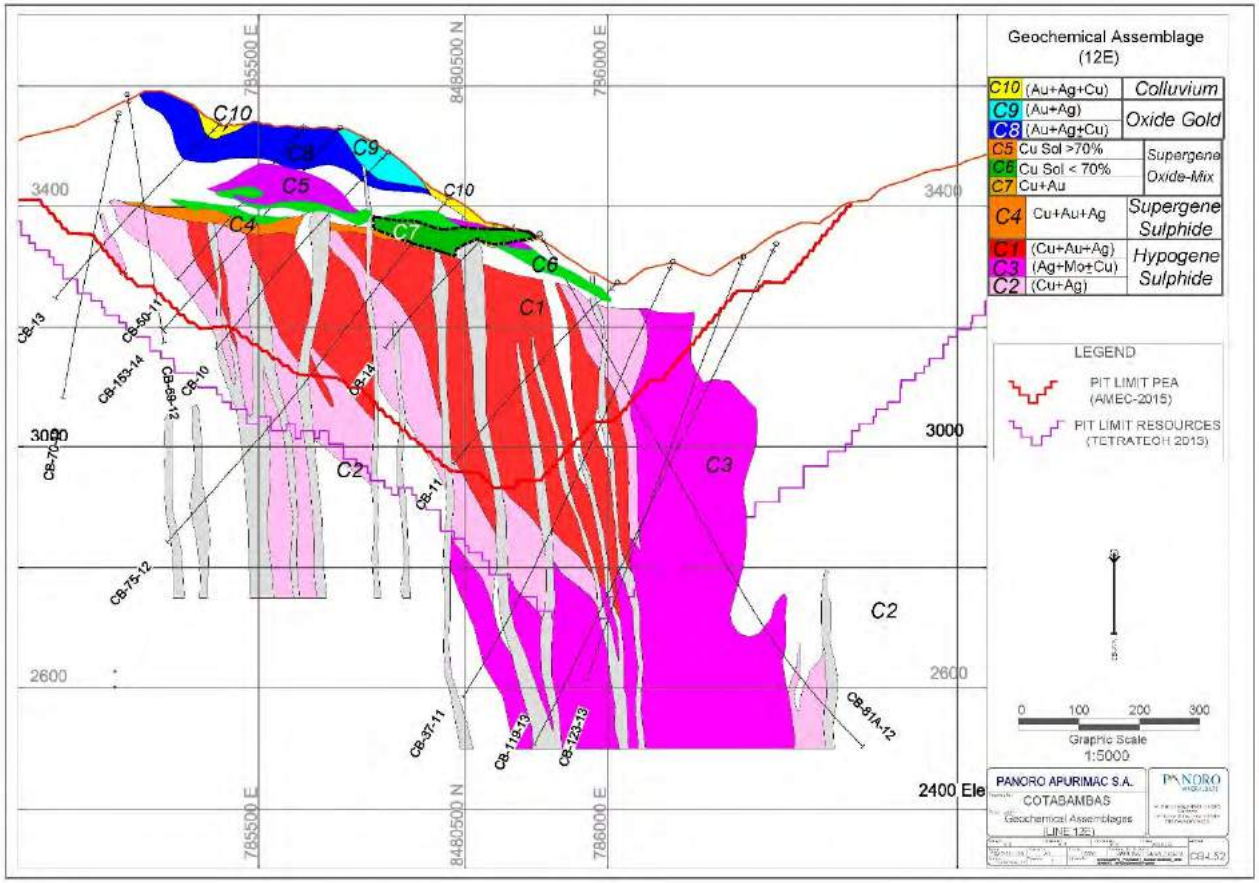
Note: The thicker black line is the copper oxide/gold blanket. Figure courtesy Panoro, 2015

Figure 7-7: Section 12E Showing Distribution of Gold Values



Note: The thicker black line is the copper oxide/gold blanket. Figure courtesy Panoro, 2015

Figure 7-8: Section 12E Showing Geochemical and Mineralogical Assemblages



Note: Figure courtesy Panoro, 2015

Sulphide mineralization consists of chalcopyrite and pyrite and gold grades are strongly correlated to copper grades in the hypogene zone. Some occurrences of bornite have been noted in deeper portions of the hypogene zones. Silver grades are not as strongly correlated to copper grades as they are to gold grades, but are generally elevated where copper–gold mineralization is present.

Hypogene mineralization (Figure 7-9, Figure 7-10 and Figure 7-11) occurs as disseminated stringers and in the form of four different types of veinlets:

- A1: quartz, anhydrite, magnetite, chalcopyrite and pyrite
- A2: quartz, magnetite, chalcopyrite, pyrite
- B: quartz, chalcopyrite, molybdenite
- D: quartz, pyrite, galena and sphalerite.
- Veinlet types A1 and A2 are part of the early potassic alteration phase. Veinlets of type B are part of a transitional phyllic alteration phase. Type D mineralization is interpreted to be part of that late alteration stage.

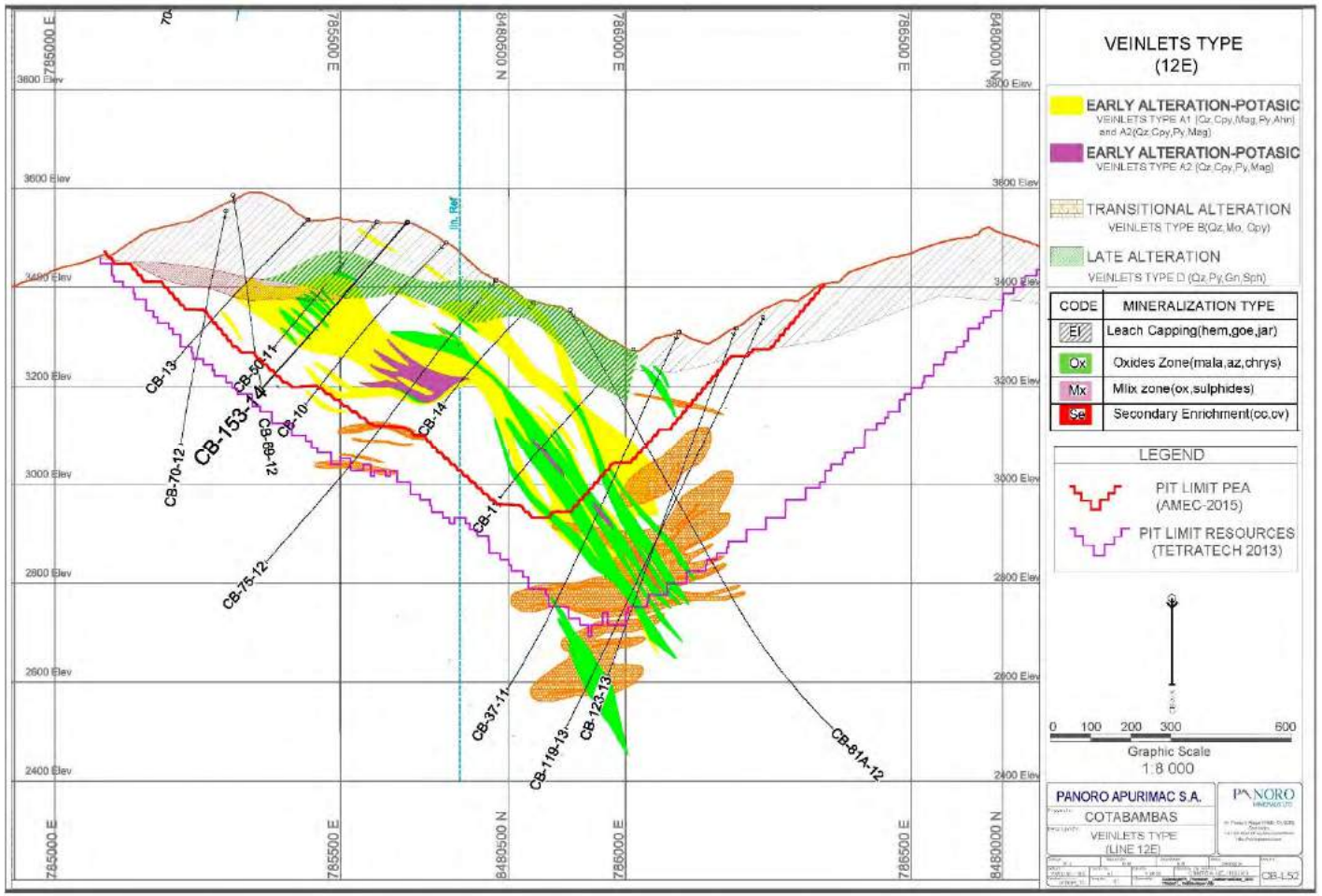
Figure 7-12 includes a set of core photographs which illustrate the various mineralization types.

Supergene Sulphide Enrichment Zone

Zones of high-grade chalcocite mineralization with lesser covellite and chalcopyrite occur at the top of the hypogene sulphide mineralization, and at the base of the leached cap. This type of mineralization is interpreted to be a zone of supergene enrichment that typically forms in porphyry copper deposits where low pH argillic and advanced argillic alteration at the top of the porphyry system leach primary copper mineralization above the paleo-water table and re-deposit it as chalcocite at the water-table surface (refer to Figure 7-10).

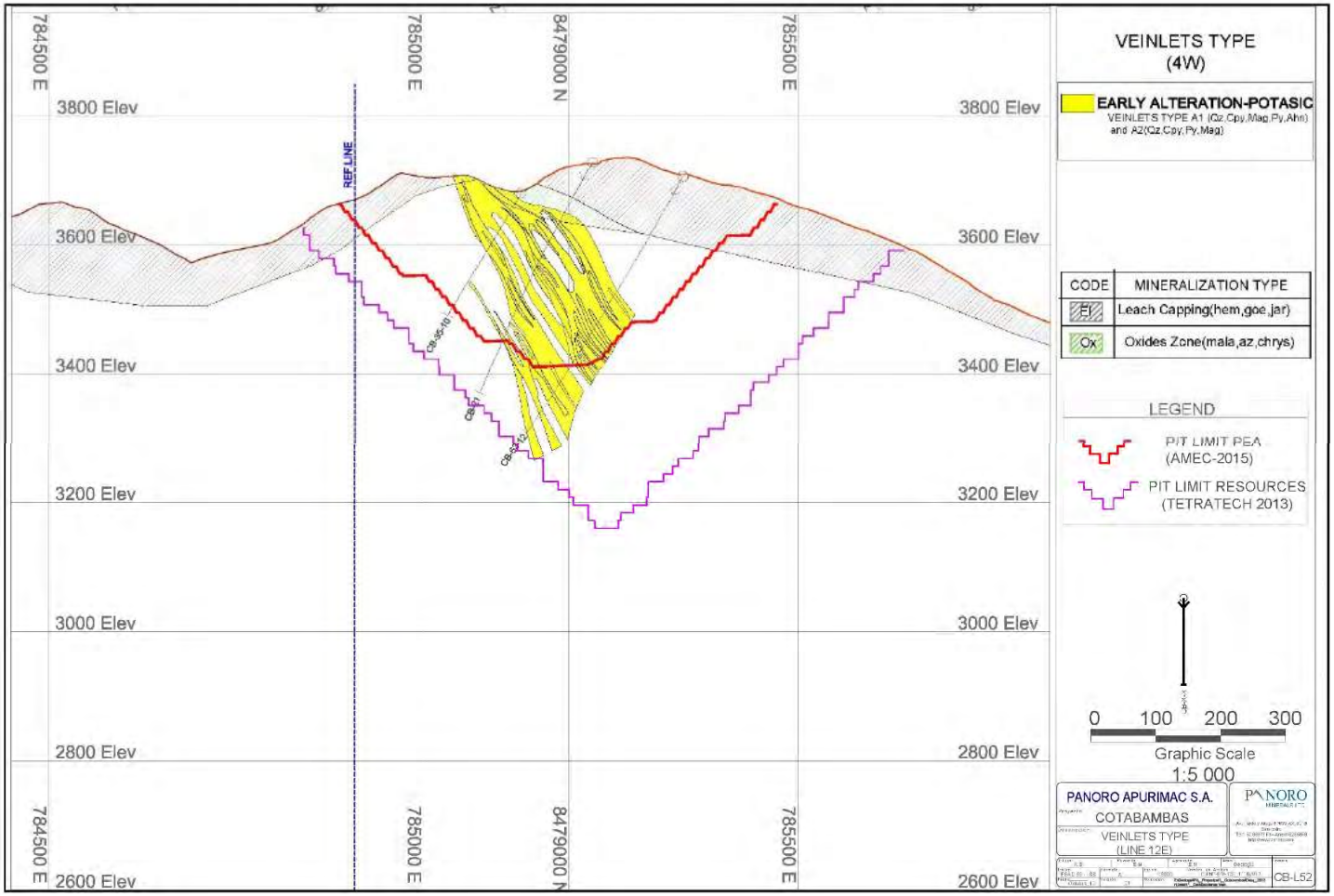
Supergene zones occur at Ccalla and Azulccacca and are characterized by high chalcocite content, correspondingly high cyanide-soluble copper assay grades and total copper grades that are generally >1%.

Figure 7-9: Section 12E Showing Mineralization Zones and Veinlet Types in the Ccalla Deposit



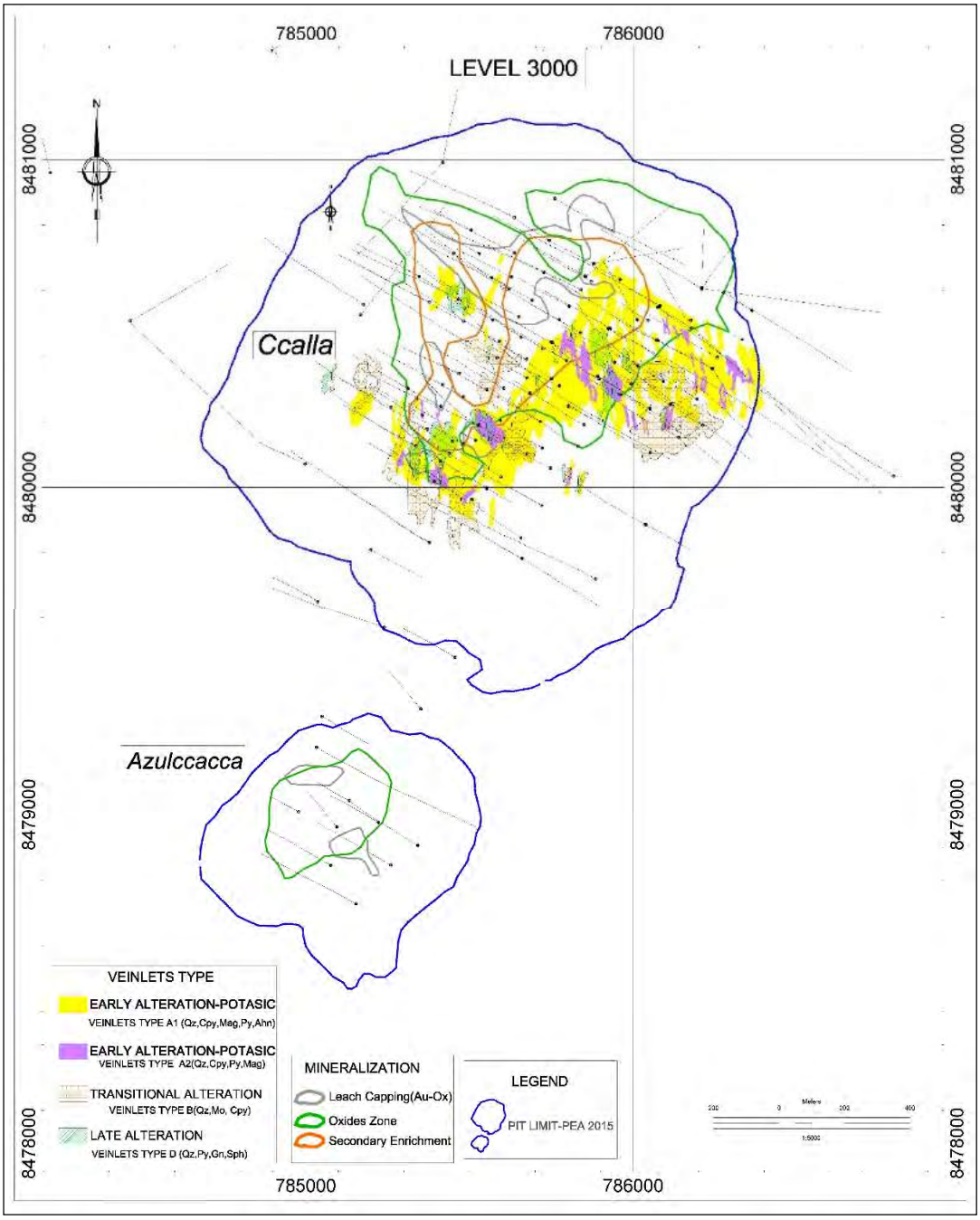
Note: Figure courtesy Panoro, 2015

Figure 7-10: Section 4W Showing Mineralization Zones and Type "A" Veinlets in the Azulccacca Deposit



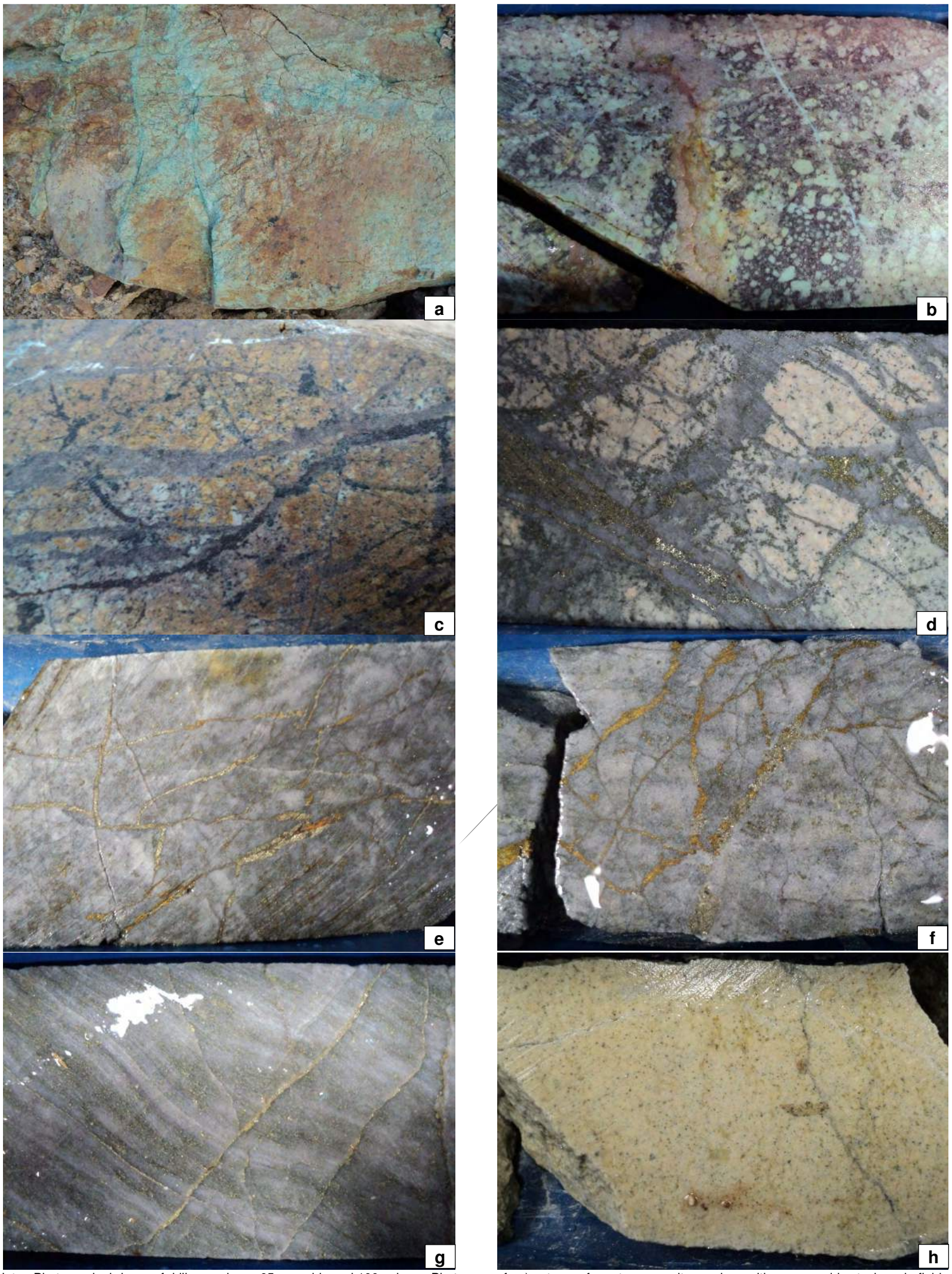
Note: Figure courtesy Panoro, 2015

Figure 7-11: Plan View of Level 3000 Showing Mineralization Zones and Veinlet Types in the Ccalla and Azulccacca Deposits



Note: Figure courtesy Panoro, 2015

Figure 7-12: Photographs of Mineralization



Note: Photographs b-h are of drill core pieces 65 mm wide and 100 m long. Photos are of: a) outcrop of quartz monzonite porphyry with copper oxide stock work, field of view 1.5 m wide, b) porphyry with quartz vein and chalcocite, c) quartz monzonite porphyry with chalcocite stringers and cross-cutting quartz veinlet, d) quartz monzonite breccia with quartz-pyrite-chalcopyrite matrix, e) intensely silicified quartz monzonite with chalcopyrite stockwork, f) intensely silicified quartz monzonite with pyrite-chalcopyrite stockwork, g) sheared porphyry with cross-cutting and disseminated chalcopyrite, h) barren latite dyke. Photographs courtesy Panoro, 2012.

Oxide Copper–Gold Mineralization

Oxide mineralization occurs in the leached cap of the Ccalla and Azulccacca deposits. The leached cap is characterized by abundant limonite, goethite and manganese wad, and a characteristic mottled orange brown colour. Iron oxides and oxy-hydroxides replace pyrite, and oxide copper–gold mineralization occurs as patches of green copper oxides, typically chrysoscolla, malachite and broncanthite. Copper oxides occur as coatings on disseminated chalcopyrite grains and as fill in fractures and veinlets.

Lenses of oxide copper–gold mineralization having lateral extents of 100 to 200 m and thicknesses of 10 to 50 m have been mapped in outcrop and intersected in core drill holes. These lenses typically occur over hypogene and secondary sulphide mineralization; however, isolated drill hole intersections indicate that oxide copper–gold mineralization may also overlie low-grade hypogene mineralization. This could be indicative of possible remobilization of copper mineralization in the leached cap.

Oxide Gold Mineralization

Oxide gold mineralization has been defined in a lens in the Azulccacca area, but has also been intersected in short, isolated 1 to 5 m intervals in other parts of the leached cap of the deposit. Oxide gold mineralization is associated with limonite and occurs near major structures cutting the hypogene sulphide zone that are associated with the quartz monzonite porphyry.

7.3 Comments on Section 7

In the opinion of the QP for Section 7, the geological understanding of the deposit and the mineralization setting are acceptable to support Mineral Resource estimation.

8.0 DEPOSIT TYPES

8.1 Porphyry Copper Deposits

The following discussion of the typical nature of porphyry copper deposits is based on classification compilations prepared by Sillitoe (2010), Singer et al. (2008), and Sinclair (2006).

8.1.1 Geological Setting

Porphyry copper systems commonly define linear belts, some many hundreds of kilometres long, as well as occurring less commonly in apparent isolation. The systems are closely related to underlying composite plutons, at paleo-depths of 5 km to 15 km, which represent the supply chambers for the magmas and fluids that formed the vertically elongate (>3 km) stocks or dyke swarms and associated mineralisation.

Commonly, several discrete stocks are emplaced in and above the pluton roof zones, resulting in either clusters or structurally controlled alignments of porphyry-copper systems. The rheology and composition of the host rocks may strongly influence the size, grade, and type of mineralisation generated in porphyry-copper systems. Individual systems have life spans of circa 100,000 years to several million years, whereas deposit clusters or alignments, as well as entire belts, may remain active for 10 million years or longer.

Deposits are typically semicircular to elliptical in plan view. In cross-section, ore-grade material in a deposit typically has the shape of an inverted cone with the altered, but low-grade, interior of the cone referred to as the “barren” core. In some systems, the barren core may be a late-stage intrusion.

The alteration and mineralisation in porphyry-copper systems are zoned outward from the stocks or dyke swarms, which typically comprise several generations of intermediate to felsic porphyry intrusions. Porphyry copper–gold–molybdenum deposits are centered on the intrusions, whereas carbonate wall rocks commonly host proximal copper–gold skarns and less commonly, distal base metal and gold skarn deposits. Beyond the skarn front, carbonate-replacement copper and/or base metal–gold deposits, and/or sediment-hosted (distal-disseminated) gold deposits can form. Peripheral mineralisation is less conspicuous in non-carbonate wall rocks, but may include base metal- or gold-bearing veins and mantos. Data compiled by Singer et al. (2008) indicate that the median size of the longest axis of alteration surrounding a porphyry copper deposit is 4–5 km, while the median size area of alteration is 7–8 km^2.

High-sulphidation epithermal deposits may occur in lithocaps above porphyry-copper deposits, where massive sulphide lodes tend to develop in their deeper feeder

structures, and precious metal-rich, disseminated deposits form within the uppermost 500 m.

Figure 8-1 shows a schematic section of a porphyry copper deposit illustrating the relationships of the lithocap to the porphyry body, and associated mineralisation styles.

8.1.2 Mineralisation

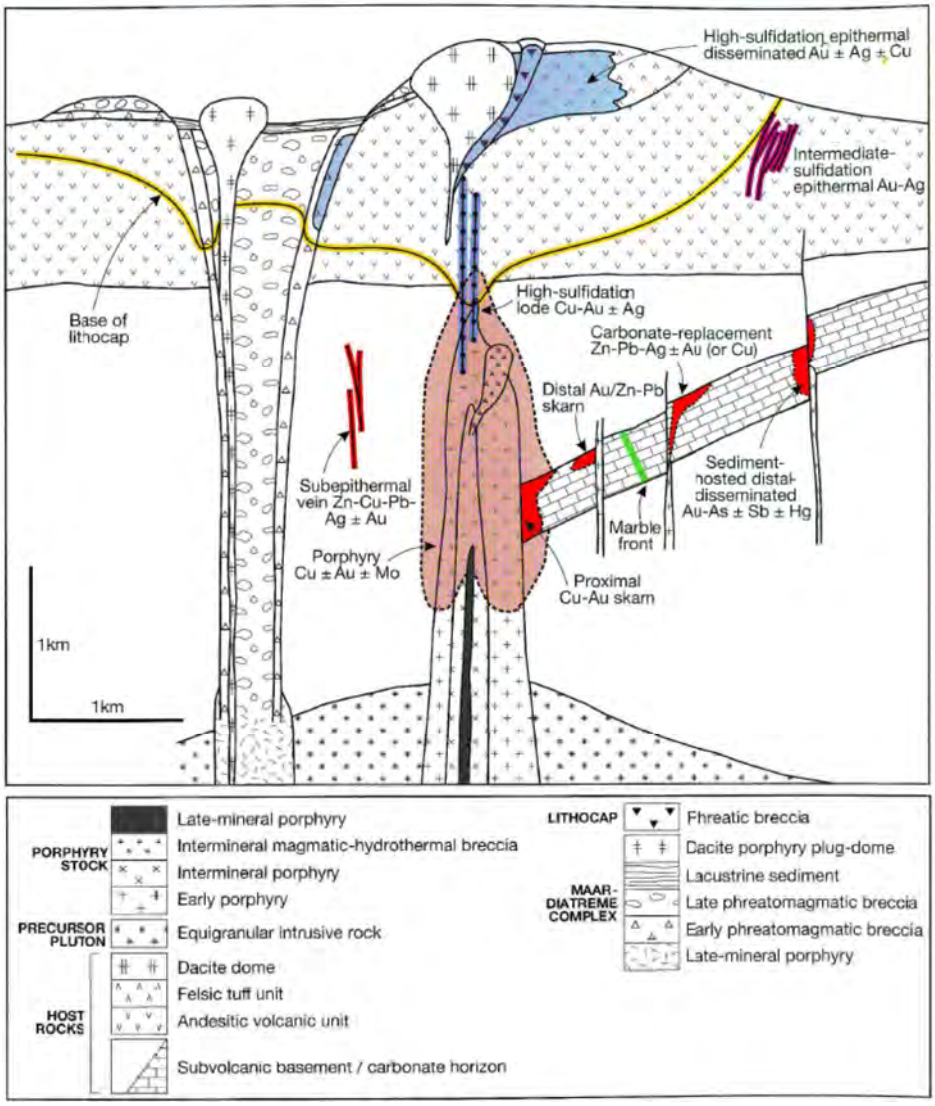
Porphyry-copper mineralisation occurs in a distinctive sequence of quartz-bearing veinlets as well as in disseminated forms in the altered rock between them. Magmatic-hydrothermal breccias may form during porphyry intrusion, with some breccias containing high-grade mineralisation because of their intrinsic permeability. In contrast, most phreatomagmatic breccias, constituting maar–diatreme systems, are poorly mineralized at both the porphyry copper and lithocap levels, mainly because many such phreatomagmatic breccias formed late in the evolution of systems, and the explosive nature of their emplacement fails to trap mineralising solutions.

Copper–ore mineral assemblages are a function of the chemical composition of the fluid phase and the pressure and temperature conditions affecting the fluid. In primary, unoxidized or non-supergene-enriched ores, the most common ore–sulphide assemblage is chalcopyrite ± bornite, with pyrite and minor amounts of molybdenite. In supergene-enriched ores, a common assemblage can comprise chalcocite + covellite ± bornite, whereas, in oxide ores, a characteristic assemblage could include malachite + azurite + cuprite + chrysoscolla, with minor amounts of minerals such as carbonates, sulphates, phosphates, and silicates. Typically, the principal copper sulphides consist of millimetre-scale grains, but may be as large as 1–2 cm in diameter and, rarely, pegmatitic (larger than 2 cm).

8.1.3 Alteration

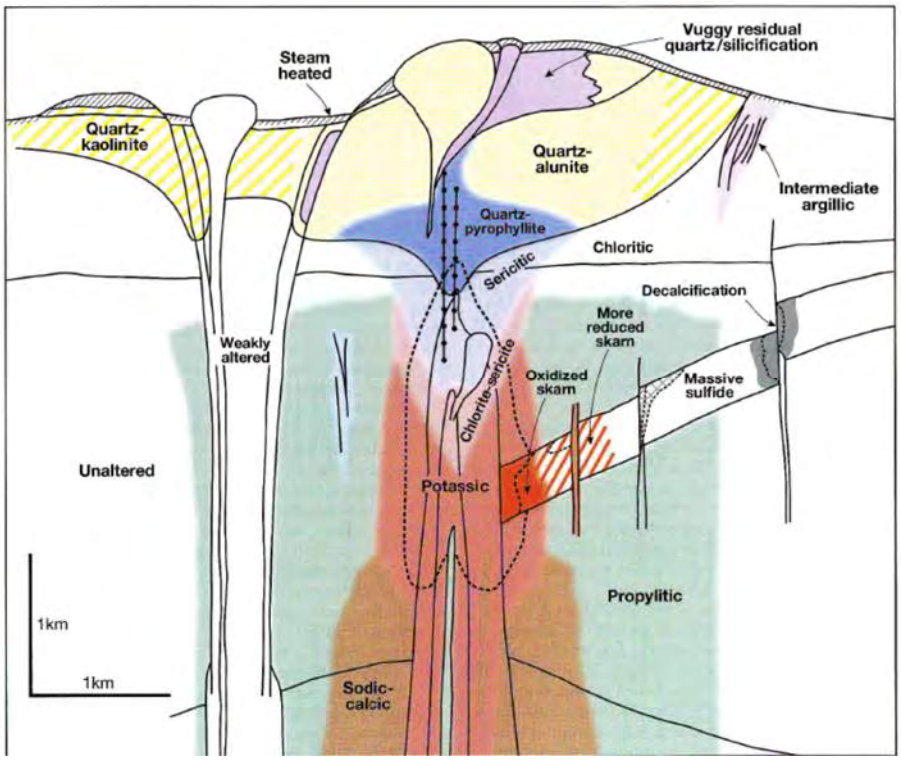
Alteration zones in porphyry copper deposits are typically classified on the basis of mineral assemblages. In silicate-rich rocks, the most common alteration minerals are K-feldspar, biotite, muscovite (sericite), albite, anhydrite, chlorite, calcite, epidote, and kaolinite. In silicate-rich rocks that have been altered to advanced argillic assemblages, the most common minerals are quartz, alunite, pyrophyllite, dickite, diaspore, and zunyite. In carbonate rocks, the most common minerals are garnet, pyroxene, epidote, quartz, actinolite, chlorite, biotite, calcite, dolomite, K-feldspar, and wollastonite. Other alteration minerals commonly found in porphyry-copper deposits are tourmaline, andalusite, and actinolite. Figure 8-2 shows the typical alteration assemblage of a porphyry copper system.

Figure 8-1: Schematic Section, Porphyry Copper Deposit



Note: Figure from Sillitoe, 2010.

Figure 8-2: Schematic Section Showing Typical Alteration Assemblages



Note: Figure from Sillitoe (2010)

Porphyry copper systems are initiated by injection of oxidized magma saturated with sulphur- and metal-rich, aqueous fluids from cupolas on the tops of the subjacent parental plutons. The sequence of alteration–mineralisation events is principally a consequence of progressive rock and fluid cooling, from >700° to <250°C, caused by solidification of the underlying parental plutons and downward propagation of the lithostatic–hydrostatic transition. Once the plutonic magmas stagnate, the high-temperature, generally two-phase hyper-saline liquid and vapour responsible for the potassic alteration and contained mineralisation at depth and early overlying advanced argillic alteration, respectively, gives way, at <350°C, to a single-phase, low-to-moderate-salinity liquid that causes the sericite–chlorite and sericitic alteration and associated mineralisation. This same liquid also is a source for mineralisation of the peripheral parts of systems, including the overlying lithocaps.

The progressive thermal decline of the systems combined with syn-mineral paleo-surface degradation results in the characteristic overprinting (telescoping) and partial to total reconstitution of older by younger alteration–mineralisation types. Meteoric water is not required for formation of this alteration–mineralisation sequence although its late ingress is commonplace.

8.2 Comments on Section 8

In the opinion of the QP for Section 8, the key characteristics of mineralization within the Project are:

- Mineralization is hosted by quartz–monzonite porphyry intrusions
- Alteration is predominantly pervasive quartz–sericite alteration with quartz–sulphide veining. Distal alteration includes weak chloritization, epidotization and sulphidation (pyrite) of un-mineralized diorite
- The deposit consists of hypogene sulphide, enriched secondary sulphide and oxide copper-gold and gold-only mineralization types
- Hypogene mineralization consists of chalcopyrite and pyrite with locally important chalcocite, and copper silicates, oxides and carbonates. Gold mineralization is disseminated and generally associated with copper sulphides, and with iron–oxy-hydroxides such as limonite in the leached cap of the deposit
- Gold grades are associated with copper grades but are higher than those typically observed in the Andahuaylas-Yauri belt. Silver grades are approximately 10:1 to gold grades and are also higher than typical for the district
- There is a strong structural control on mineralization with the most intense mineralization associated with strong north–northeast-trending faults and shears.

These characteristics are considered to be typical of porphyry-style copper deposits of the South American Cordillera. The South American Cordillera porphyry deposits are a subset of the porphyry group discussed in Section 8.1.

In the northwest part of the Project area, there is local evidence of polymetallic (lead–zinc–silver) mineralization hosted in calcareous sediments that may be associated with a skarn or replacement-type system.

9.0 EXPLORATION

The Cotabambas Project area is relatively large and access to parts of the property can be difficult, either due to a lack of roads or ongoing negotiations with surface rights holders. As a result, exploration work has been carried out over a relatively restricted area, largely restricted to the Ccalla and Azulccacca areas as access and infrastructure in these areas is reasonably good and results of the early drilling in these areas were encouraging.

More recently, exploration by Panoro in the vicinity of the Ccalla zone has located three additional areas, Buenavista, Ccochapata, and Maria Jose, to the north and northwest of the Ccalla zone (Figure 9-1). Panoro has also delineated four other target areas, Chuyllullo, Chaupec, Anarqui and Cullusayhua, which are skarn-hosted mineralization targets.

9.1 Geological Mapping

Reconnaissance-scale geological mapping has been carried out over the northern half of the Property from the town of Cotabambas to the Guaclle area in the west. More detailed 1:10:000 scale mapping has been carried out over the Azulccacca and Ccalla areas and work to extend the 1:10,000 scale mapping westward to Guaclle is under way. Detailed 1:2,000 scale mapping has been completed for the Azulccacca and Ccalla areas. Mapping is in progress to the south–southwest of Azulccacca. Prospect-scale 1:1,000 scale mapping is underway on the skarn and porphyry-style exploration targets.

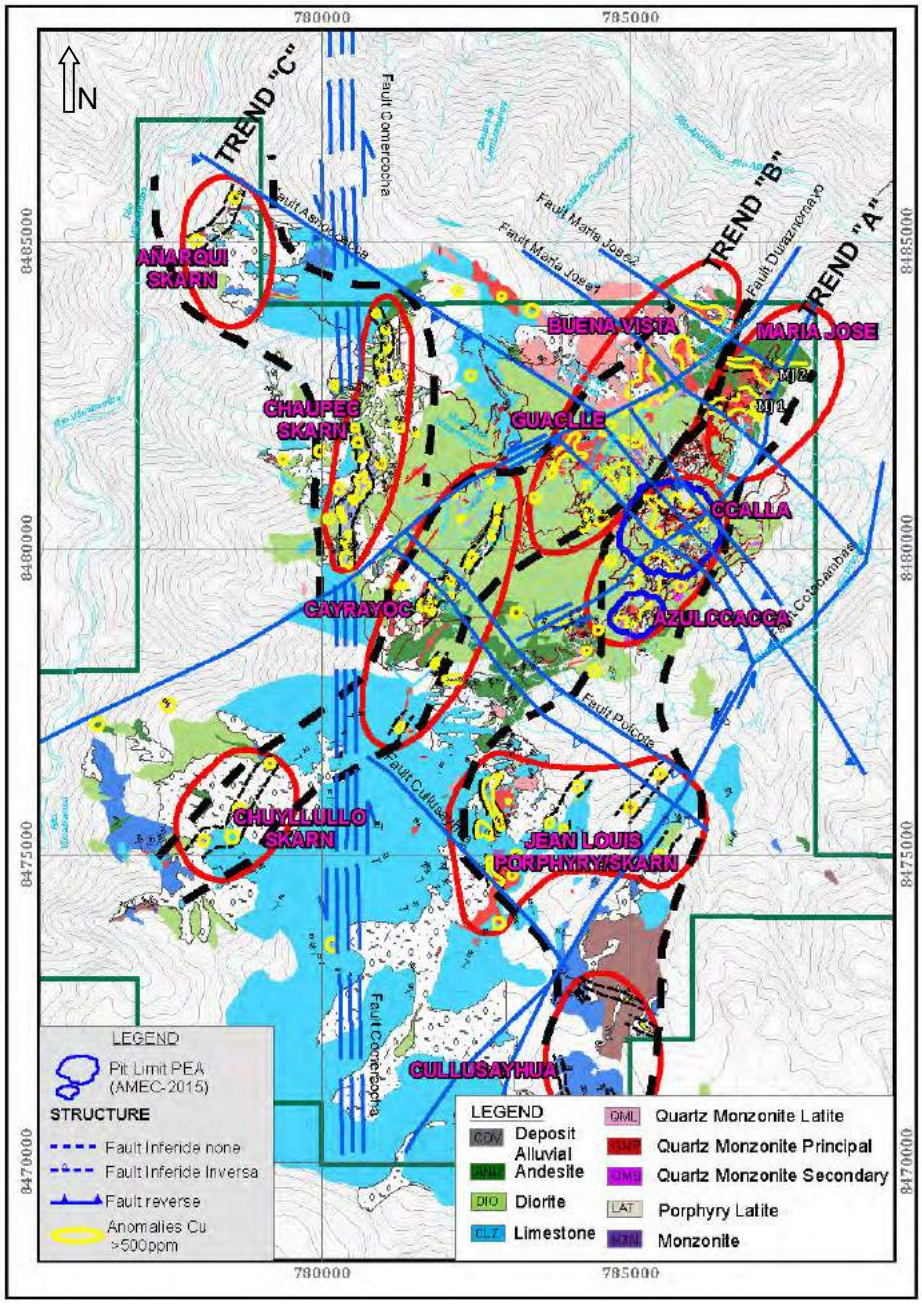
9.2 Geochemical Sampling

Soil and rock geochemical sampling has been carried out on a 100 m grid over the Azulccacca, Calla and Guaclle areas. Samples were collected by Panoro and previous workers and are used to define geochemical anomalies and refine areas for drill testing. Anomalous copper, gold and silver values correspond to known mineralization at Ccalla and Azulccacca. To the west, zinc and lead anomalies appear to be associated with skarn-type mineralization.

The systematic geochemical chip rock sampling conducted by Panoro has been focused on exploration targets, with 4,488 rock chip samples and 480 stream sediment samples collected as of April 2015. An additional 207 samples have been submitted for lithogeochemical determination.

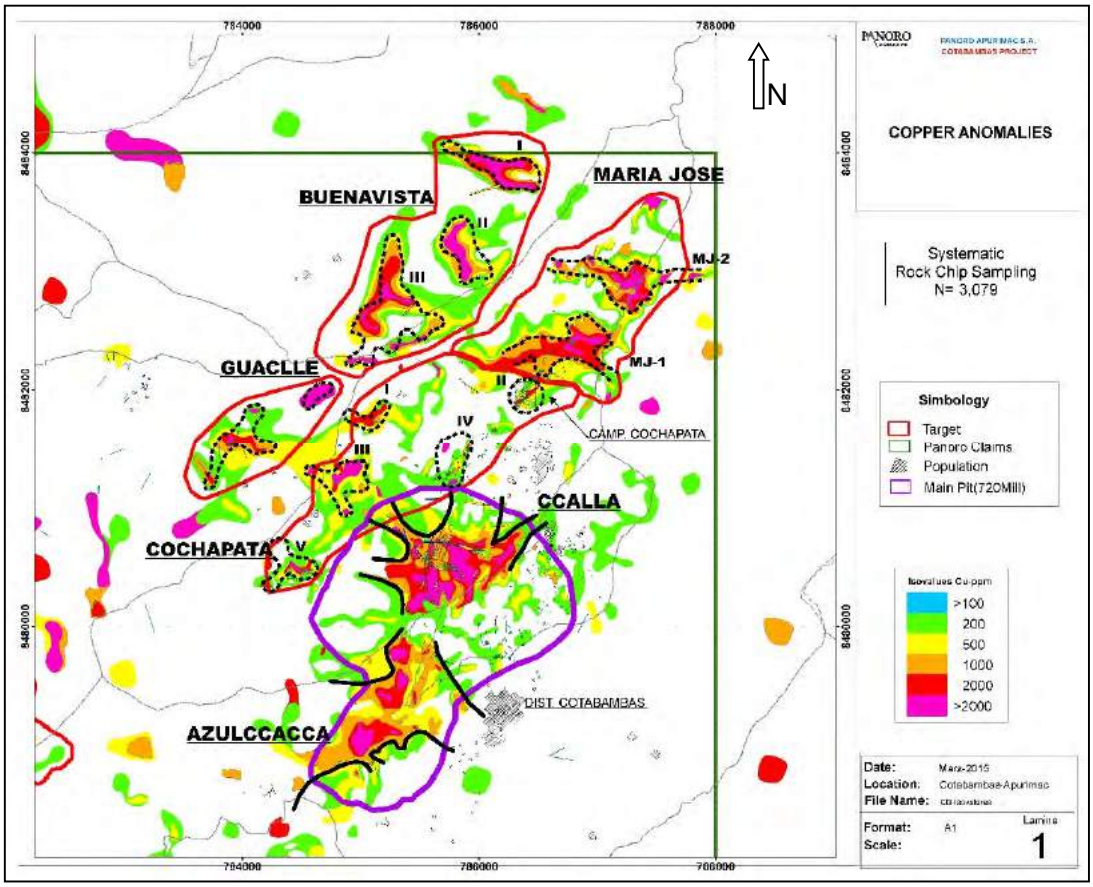
A number of anomalous copper values were returned, see Figure 9-2, with a copper cluster centred over the Ccochapata porphyry copper target area.

Figure 9-1: Exploration Targets and Mineralization Trends



Note: Figure courtesy Panoro, 2015. In the legend, "Fault Inferide none" = inferred fault (unknown sense of slip), and "Fault Inferide inversa" = inferred reverse fault

Figure 9-2: Copper Anomalies Generated by the Geochemical Sampling Program



Note: Figure courtesy Panoro, 2015.

9.3 Geophysics

Magnetometer and IP surveys have been carried out over the main exploration areas. In 1996, Antofagasta contracted Val d'Or Geophysics (Perú) to carry out IP and magnetometer surveys on the Ccalla and Azulccacca areas. In 2003, CDLM contracted Val d'Or Geophysics to carry out reconnaissance surveys in the Cayrayoc area. The surveys were carried out on lines spaced 200 m apart. A total of 42.8 km of magnetometer survey and 10.5 km of IP survey were completed. A chargeability anomaly was identified and tested in the 2003 CDLM drill campaign.

In August 2003, CDLM contracted Val d'Or Geophysics to extend the IP and magnetometer coverage at Ccalla and Azulccacca and westward towards the Guaclle area. A 162 line-km magnetic survey and 82 line-km of IP surveys were carried out. The surveys were centered on the Ccalla–Azulccacca area of the Project. The surveys confirmed that rather than being related to a single trend, the Ccalla and Azulccacca

zones are actually part of two separate, 2 to 3 km long, northeasterly-trending mineralized corridors defined by low chargeability values (Figure 9-3).

Two other northeasterly-trending chargeability lows, associated with the Ccochapata and Guaclle porphyry centres to the northwest of Ccalla, were identified.

Geophysics Consultants S.R.L was hired to complete a number of geophysical surveys. The first, in March 2011, consisted of an IP/resistivity survey. Fourteen lines, spaced at 200 m intervals, were completed over an area of 35.2 km^2. A second survey was completed by Geophysics Consultants in November 2011, over the Ccalla and Azulccacca deposits using self-potential (SP) methods. This survey covered 216.5 line km. The third survey, completed April 2015, consisted of IP, SP, magnetometer, and magnetic susceptibility readings, over the Chaupec, Ccaracyoc and Jean Louis exploration targets. It is planned to extend the coverage to the Ccochapata, Maria Jose and Buenavista target areas.

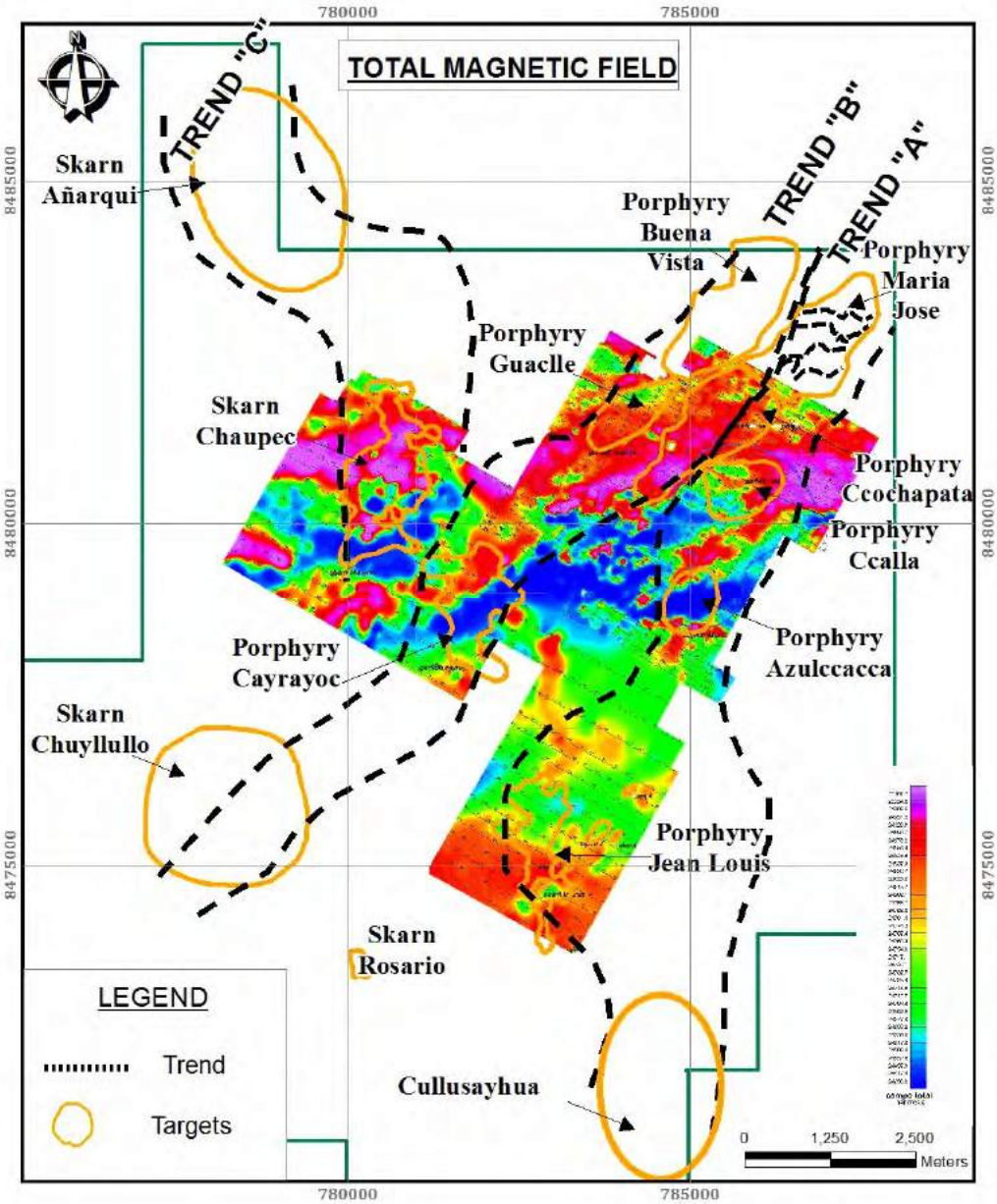
9.4 Exploration Potential

9.4.1 Principal Components Analysis

In October 2014, Amec Foster Wheeler proposed to Panoro that additional information could be gained by completing a principal component analysis (PCA) of the drill hole and rock sample data for the Cotabambas Project. PCA is a statistical method of reducing a set of many related variables to a few 'groupings' that, in turn, can be shown to be related to primary rock types, alteration types, or mineralization types based on their multi element signature and their spatial patterns. The study was conducted by Todd Wakefield and Angelica Torres of Amec Foster Wheeler.

- The PCA on the Cotabambas geochemical assay data has identified several groupings of elements that may be important for future exploration and development work. Based on this study, Amec Foster Wheeler reviewed the explorations targets, identifying 19 areas of interest that were subsequently highlighted and ranked as follows (Table 9-1 and Figure 9-4):
- Priority 1: Requires detailed mapping and geochemical sampling for definition of an initial drilling plan
- Priority 2: Requires extensive mapping and geochemical sampling for definition of the mineralization style concept
- Priority 3: Requires a suitable scale mapping and extensive geochemical sampling for a preliminary target evaluation.

Figure 9-3: Geophysical Plan showing Total Magnetic Field and Interpreted Anomalies

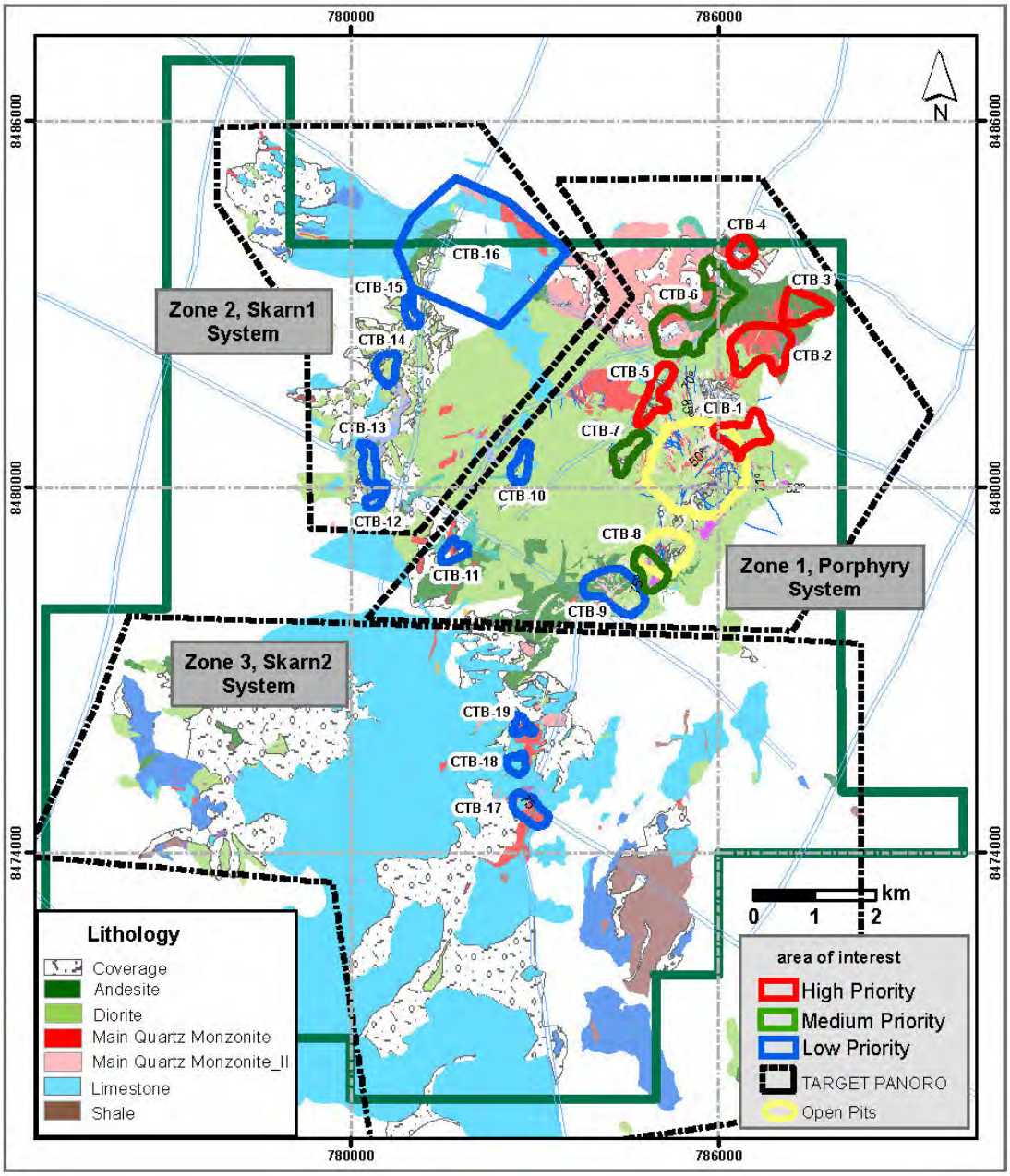


Note: Figure courtesy Panoro, 2015.

Table 9-1: PCA-based Exploration Target List

Zone	Areas of Interest	Rank	Comments
Zone 1	CTB-01	1	
	CTB-02	1	• High score factor two anomaly related to mineralization
	CTB-03	1	• Low score factor one anomaly related to intrusive unit (QMP?)
	CTB-04	1	• High score factor three anomaly related to alteration
	CTB-05	1	
	CTB-06	2	• Moderate score factor two anomaly related to mineralization
	CTB-07	2	• Low score factor one anomaly related to intrusive unit (QMP?) • Moderate score factor three anomaly related to alteration
	CTB-08	2	• High score factor five anomaly related distal and peripheral target (Zn, Pb, Mn)
	CTB-09	3	• Low score factor four (negative) anomaly related to As and Ag could be part of a epithermal environment
	CTB-10	3	
	CTB-11	3	• Isolated high score factor two anomaly related to mineralization
Zone 2	CTB-12	3	
	CTB-13	3	• High score factor one anomaly related to base metal mineralization
	CTB-14	3	• Strong contrast values for score factor two related to alteration and mineralization processes
	CTB-15	3	
	CTB-16	3	
Zone 3	CTB-17	3	
	CTB-18	3	• High score factor two anomaly related to base and precious metals mineralization
	CTB-19	3	

Figure 9-4: PCA-based Exploration Target Locations



Note: Figure prepared by Panoro and Amec Foster Wheeler, 2015.

9.4.2 Exploration Prospects/Targets

In addition to the PCA, Panoro has identified nine prospects/targets that the company considers warrant additional exploration. These were summarized in Figure 9-2 but are discussed in more detail in this section.

Figure 9-5 shows anomalous copper values generated from rock chip sampling, by area. Figure 9-6 shows the anomalous rock chip gold values, also by area. The resulting target areas are summarized in Table 9-2. Table 9-3 is the key to the abbreviations used in Table 9-2. Figure 9-7 illustrates trends along strike that may have exploration potential for copper.

9.4.3 Structural Targets

Panoro performed a structural study at the end of 2014 (Rodriguez, 2014) to determine the most favourable structural settings for porphyry emplacement and mineralization. Rodriguez noted:

Emplacement of the quartz monzonite porphyry and later latite dykes was controlled by a main system of high angle faults and shear zones that have a northeast–southwest orientation. Those same structures controlled emplacement of mineralization. A second set of structures, perpendicular to the northeast–southwest system and parallel to the regional thrust fault systems with northwest–southeast orientation runs between the Ccalla area and the Guaclle areas to the west.

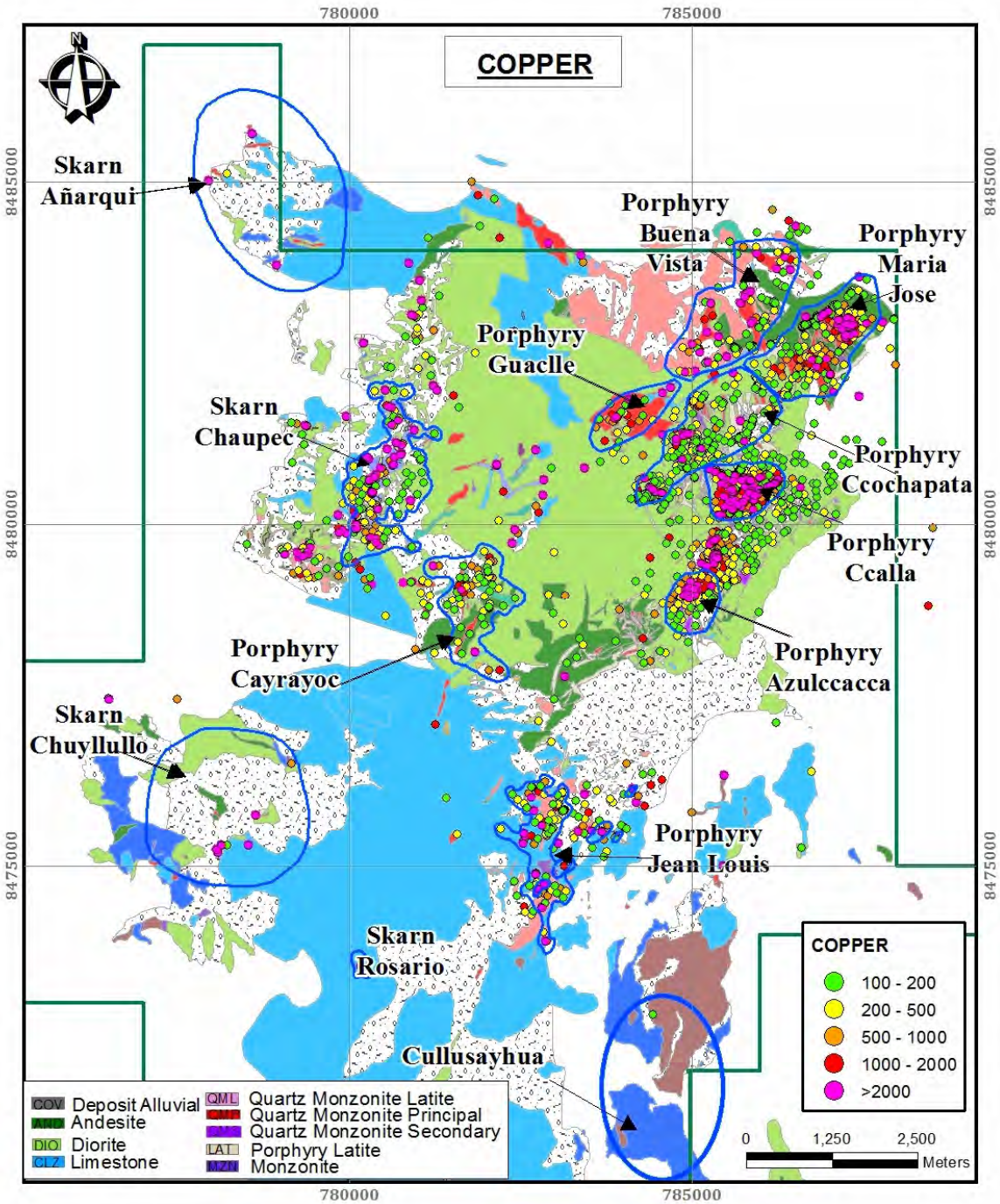
The northwest-orientated fault, located north of Azulccacca main fault, is responsible for the concealing mineralization at the north of the project. Additionally, a north–south fault with a sinistral sense of movement has displaced the crackle zone associated with the Azulccacca fault.

The key faults and potential structural trends that may be mineralization hosting or controlling are shown in Figure 9-7.

There are three main fault orientations, northeast–southwest, northwest–southeast, and north–south. Each set can show re-activation during later deformational stages.

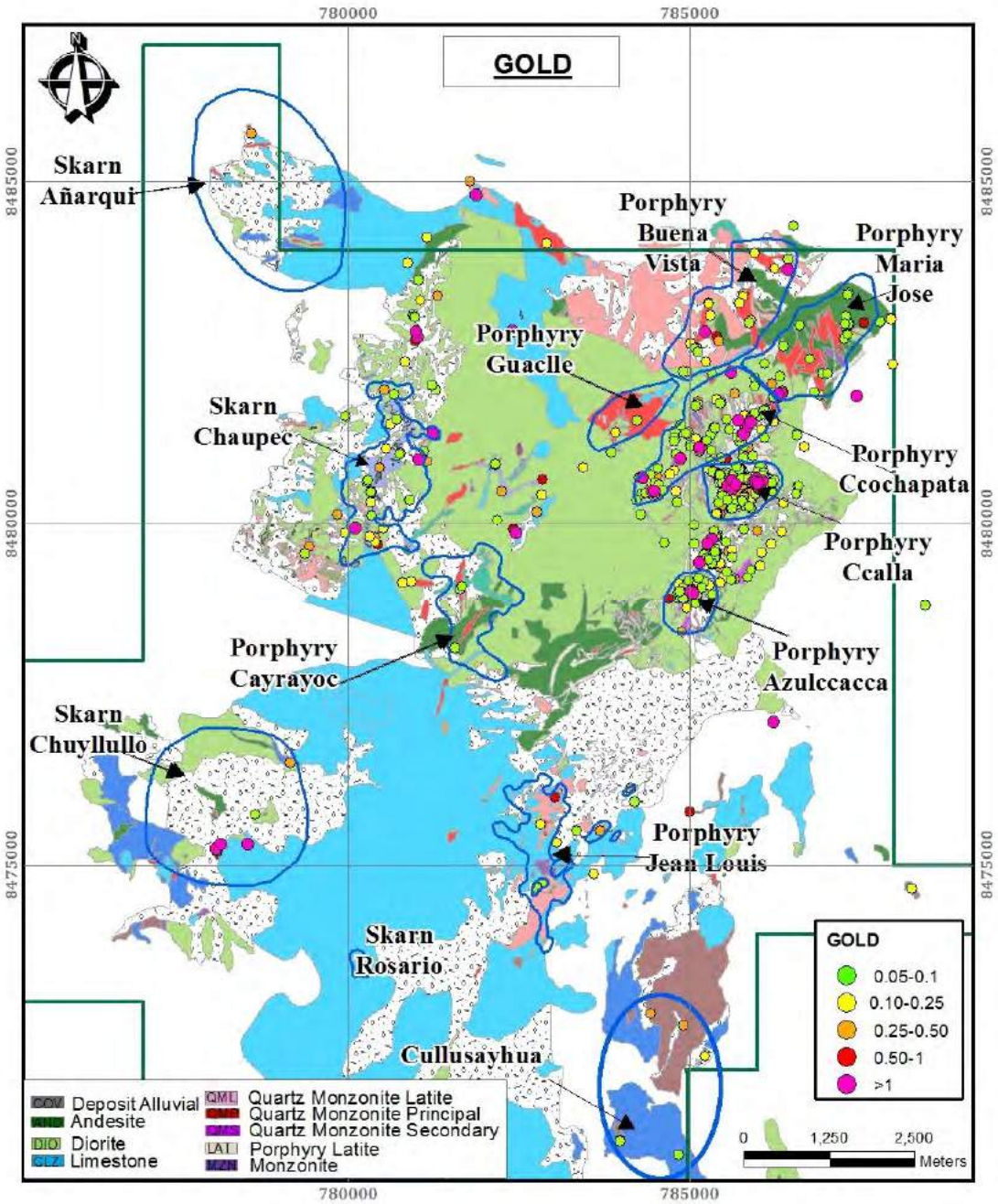
Figure 9-8 provides a history of the structural evolution of the Project area and will be used to further refine exploration targeting to areas where porphyry mineralization may have been uplifted toward surface by the fault actions.

Figure 9-5: Anomalous Copper Values, Rock Chip Sampling



Note: Figure courtesy Panoro, 2015. Assay values shown in ppm.

Figure 9-6: Anomalous Gold Values, Rock Chip Sampling



Note: Figure courtesy Panoro, 2015. Assay values shown in ppm.

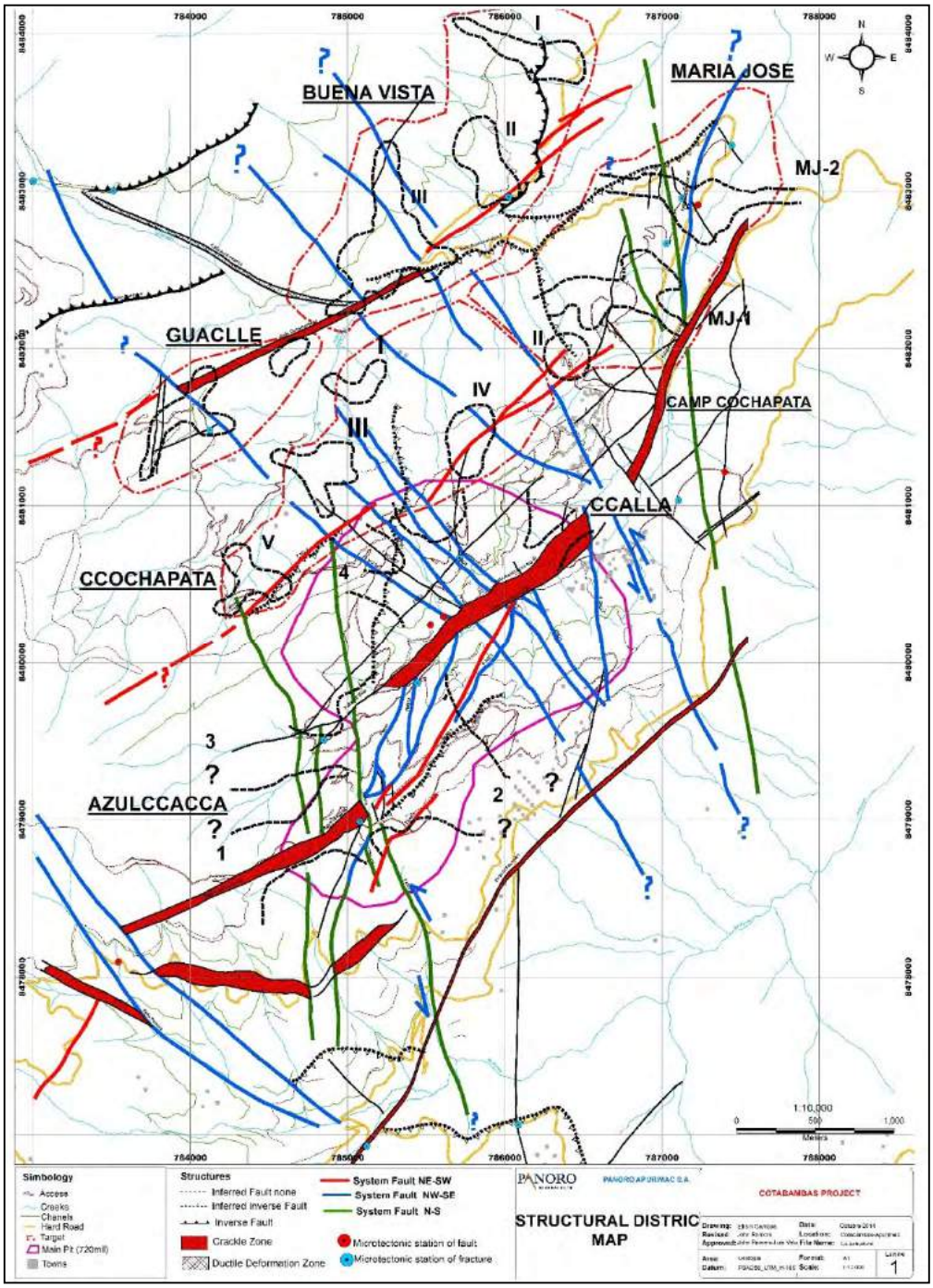
Table 9-2: Exploration Target Summary

Target	Type	Zone	Support	Support Quantity	Anomaly Dimension	Mineralogy	Average Grades						Other Studies Support	Exploration Priority
							Cu %	Au ppm	Ag ppm	Pb %	Zn %	Mo %		
Ccalla-Azulccacca	Porphyry	Ccalla	Drill Hole/Rock chip	Several holes	2500x1500 m	py, cpy, ten, chrys, mala, azu, sph, gn, mag, jar, goe, hem	0.13	0.06	1.18				Gphy, Gchy, PC	Priority 1
Guaclle	Porphyry	Zone I	Drill Hole/Rock chip	6 Holes/8 samples	400x100m	py, cpy, mala, mag; spec, hem, jar, goe	0.40	0.02	3.30	0.06		0.014	Gphy, Gchy, PC	Priority 1
		Zone II	Rock chip	4 samples	700x100m	py, cpy, mala, mag, spec, hem, jar, goe.	4.13	0.03	5.75	0.13			Gphy, Gchy, PC	Priority 1
Cayrayoc	Porphyry		Drill Hole/Rock chip	3 Holes/10 samples	900x700m	py, cpy, mala, mag, spec, hem, jar, goe.	0.16	0.03	0.84			0.002	Gphy, Gchy, PC	Priority 1
Ccochapata	Porphyry	Anomaly I	Rock chip	21 samples	300x200m	cpy. chrys, goe, jar	0.13	0.17	0.80				Gphy, Gchy, PC	Priority 1
		Anomaly II	Rock chip	20 samples	250x200m	mag, spec, py, ten, goe, hem.	0.09	2.98	2.85				Gphy, Gchy, PC	Priority 1
		Anomaly III	Rock chip	20 samples	250x250m	mala, ten, neo, goe, jar, hem.	0.33	0.11	4.25				Gphy, Gchy, PC	Priority 1
		Anomaly IV	Rock chip	117 samples	300x300m	py, mag, goe, hem	2.17	0.16	3.65				Gphy, Gchy, PC	Priority 1
		Anomaly V	Rock chip	48 samples	300x300m	qz, cpy, py, bio, mag, spec.	0.32	0.50	17.50	0.26	0.42		Gphy, Gchy, PC	Priority 1
Maria Jose	Porphyry	MJ-1	Rock chip	59 samples	400x1100m	chrys, cup, goe, hem, cpy, cc.	0.29	0.03	2.40				Gphy, Gchy, PC	Priority 1
		MJ-2	Rock chip	138 samples	250x800m	chrys, cup, goe, hem, cpy, cc.	1.27	0.35	4.95				Gphy, Gchy, PC	Priority 1
Buena Vista	Porphyry	Zone I	Rock chip	15 samples	850x200m	cpy, py, mal, neo, ten, jar, goe.	0.59	0.64	9.60	0.12	0.13		Gphy, Gchy, PC	Priority 1
		Zone II	Rock chip	15 samples	600x300m	cpy, py, mal, neo, ten, jar, goe.	3.52	0.18	38.55		0.16		Gphy, Gchy, PC	Priority 1
		Zone III	Rock chip	38 samples	1000x700m	cpy, py, mal, neo, ten, jar, goe.	1.28	0.77	15.00	0.14	0.41		Gphy, Gchy, PC	Priority 1
Chaupec	Porphyry/Skarn	Zone I	Rock chip	5 samples	500x300m	py, cpy, mala, mag, hem, jar, goe	0.44	0.08	1.10				Gphy, Gchy	Priority 2
		Zone II	Rock chip	70 samples	2000x500m	py, cpy, ten, chrys, mala, azu, sph, tt, gn, mag, jar, goe, hem	3.62	1.35	50.35	0.53	0.53		Gphy, Gchy	Priority 2
		Zone III	Rock chip	15 samples	5010x400m	py, cpy, mala, ten, neo, mag, hem, jar, goe	0.75	0.14	9.55				Gphy, Gchy	Priority 2
Jean Louis	Porphyry/Skarn	Zone I	Rock chip	26 samples	1500x1000m	py, cpy, ten, neo, mala, mag, hem, jar, goe.	1.20	0.16	5.85				Gphy, Gchy	Priority 2
		Zone II	Rock chip	27 samples	1500x1000m	py, cpy, ten, neo, mala, mag, hem, jar, goe.	1.20	0.16	5.85				Gphy, Gchy	Priority 2
		Zone III	Rock chip	13 samples	700x500m	cpy, mala, chrys, azu, sph, goe, hem, jar.	1.61	0.05	11.05	0.26			Gphy, Gchy	Priority 2
Añarqui	Skarn		Rock chip	4 samples	1000x500m	mag, cpy, mala, chrys, qz, ca	0.50	1.81	2.25				Gphy, Gchy	Priority 2
Chuyllullo	Skarn/Porphyry		Rock chip	21 samples	2000x400m	mag, cpy, py, mala, chrys, qz, ca.	1.46	1.54	8.80				Gphy, Gchy	Priority 2

Table 9-3: Abbreviations Key to Accompany Table 9-2

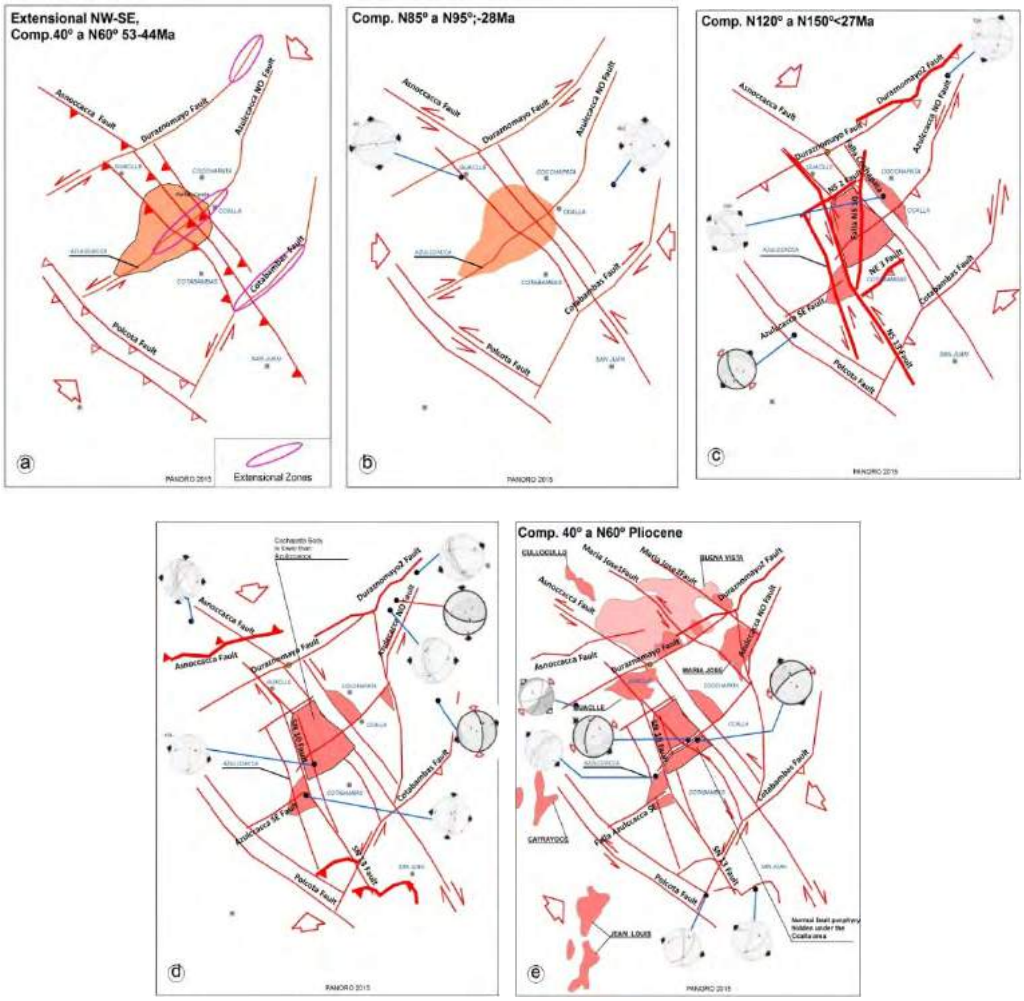
Abbreviation	Meaning
azu	Azurite
bio	Biotite
ca	Calcite
cc	Chalcocite
chrys	Chrysoscolla
cpy	Chalcopyrite
cup	Cuprite
Gchy	Geochemical Studies
gn	Galena
goe	Goethite
Gphy	Geophysical Studies
hem	Hematite
jar	Jarosite
mag	Magnetite
mala	Malachite
neo	Neotacite
PC	Principal Component Analysis
py	Pyrite
qz	Quartz
spec	Specularite
sph	Sphalerite
ten	Tenorite
tt	Tennantite

Figure 9-7: Key Structural Trends



Note: Figure courtesy Panoro and Rodriguez, 2014.

Figure 9-8: Project Structural History Interpretation



Note: (a) NW-SE extensional event, compression N40 to N60: The event was caused by a greater stress in the N40 to N60 direction; Azulccacca, Cotabambas and Durazno May faults are almost parallel to the direction of maximum stress. These faults originate tension zones with a NE-SW direction through which the Cotabambas and Buenavista porphyries were emplaced; (b) Compression event N85 to N95: The second event is a greater stress with N 85 to N 95 direction; NW-SE faults (Azulccacca, and Poicota) have a transcurrent sinistral movement and NE-SW faults (Duraznomayo, Azulccacca and Cotabambas) have a transcurrent dextral movement. This caused the crackle and subsequent displacement of Cotabambas porphyry; (c) NE-SW extensional event, compression N120 to N150: This event deformed the Cotabambas porphyry, resulting in NS faults with sinistral movement and reactivations with inverse movement in the NE-SW faults, resulting in the faults Duraznomayo 2 NS2 and FANE3. The Cotabambas porphyry was divided into three areas, the Azulccacca porphyry was displaced by the sinistral NS faults, the Ccochapata porphyry associated with normal movement on the Ccochapata fault, as was the Cotabambas porphyry; (d) Extensional event E-W, N-S compression: shows EW inverse -low angle fault affecting mineralization. At this time the NW-SW faults display a sinistral movement and NW-SE fault orientations have dextral movement, the mineralized porphyry remains shifted and faulted; (e) Event NE-SW compression, N40 to N60: This event has a typical NE-SW direction and is contemporary with the Andahuaylas batholith porphyry–Yauri mineralization event. The faults shows a younger episode associated with the normal Azulccacca fault and which allowed the emplacement of a new porphyry (Maria Jose). Figure courtesy Panoro and Rodriguez, 2014.

9.5 Comments on Section 9

In the opinion of the QP responsible for Section 9, the exploration concepts being applied to the exploration programs at Cotabambas are consistent with the Project setting and mineralization identified to date. The Project retains exploration potential, and Panoro have a number of geophysical, geochemical, structural and principal component analysis targets that could support further work.

10.0 DRILLING

Drilling completed on the Project to the cessation of the current drilling program in February 2014 is summarized in Table 10-1. Figure 10-1 and Figure 10-2 show the locations of the completed drilling for the Project as a whole, and the Ccalla and Azulccacca zones respectively.

The last hole in the current phase of drilling was completed on 17 February 2014, and no holes have been drilled since that date. Eleven holes were completed after the close-out date for the Mineral Resource estimate in Section 14.

10.1 Legacy Drill Programs

10.1.1 Antofagasta: 1996–2000

The Antofagasta drill campaign was carried out between July 1996 and April 2000. During the campaign, holes were drilled in the Ccalla, Azulccacca, and Guaclle areas. Drilling tested geochemical and geophysical anomalies. Significant copper–gold mineralization was intersected in the first hole drilled at Ccalla (CB-1), and there were anomalous intersections in the holes drilled at Azulccacca and Guaclle as well. A total of 24 HQ and NQ diameter diamond drill core holes totalling 8,538 m were drilled by Boart Longyear and Geotech using a combination of LF-38 and UDR-650 machines.

10.1.2 Cordillera de las Minas: 2002–2006

CDLM contracted Boart Longyear to drill diamond drill core holes of HQ and NQ diameter between June and November 2003. Hole CB-25, the first hole of the campaign was drilled at Ccalla and confirmed the mineralization delineated by the Antofagasta drilling. Drill hole CB-30 intersected three short (less than 1 m each) intervals of low-grade copper–gold mineralization at the Cayrayoc area. Drill hole CB-31 intersected elevated grades at the Guaclle area.

10.2 Panoro Drill Programs

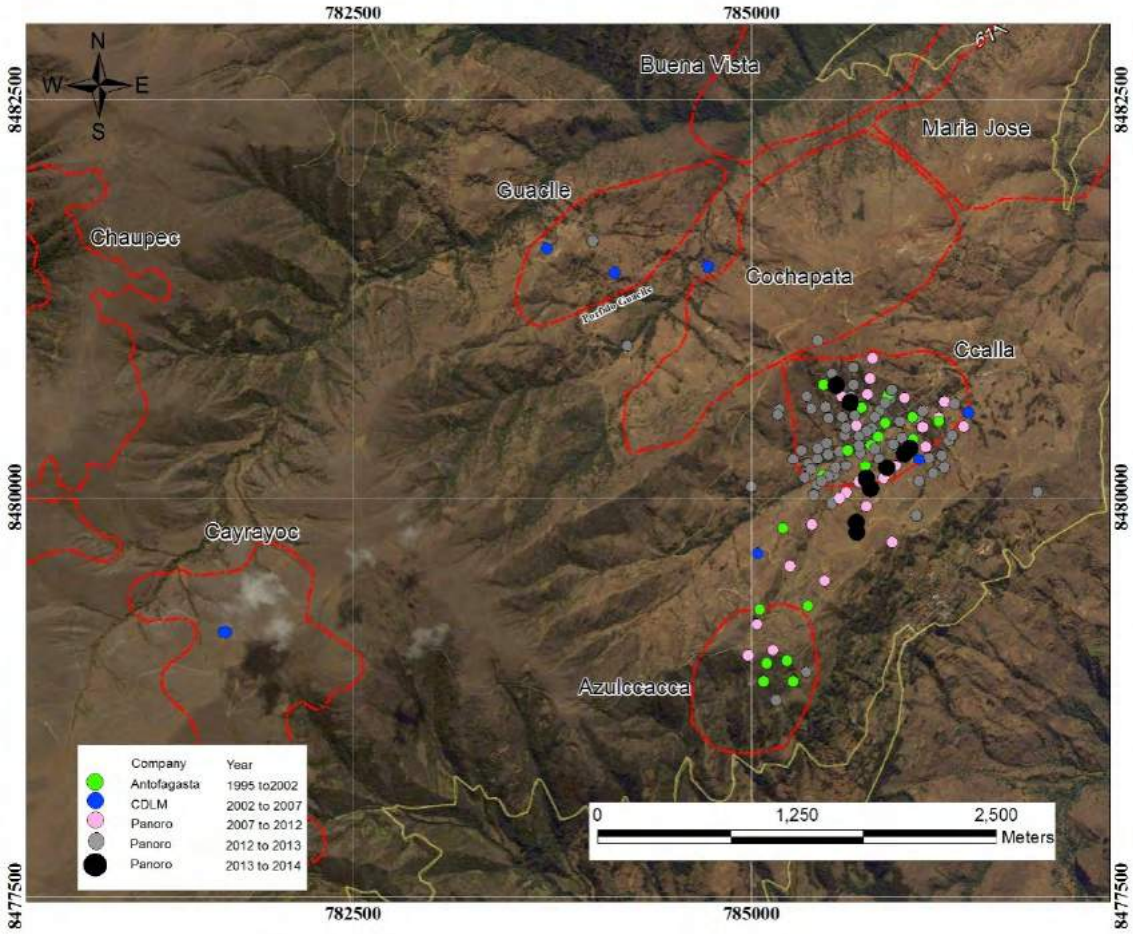
The first drill program carried out by Panoro in 2010–2011 confirmed the presence of copper–gold mineralization at Azulccacca and Ccalla. A total of five core holes totalling 2,809 m were drilled during the campaign. Panoro contracted Bradley Brothers for this drill campaign and drill holes were drilled using a Hydro-core machine drilling NQ diameter drill core.

Table 10-1: Drilling Completed to 17 February 2014

Company	Year	Drill Holes	Metres
Antofagasta	1995 to 2002 *	24	8,538
CDLM	2002 to 2007	10	3,252
Panoro	2007 to 2012	29	17,785
Panoro	2012 to 2013	81	40,467
Panoro	2013 to 2014	11	4,946
Total		**155**	**74,988**

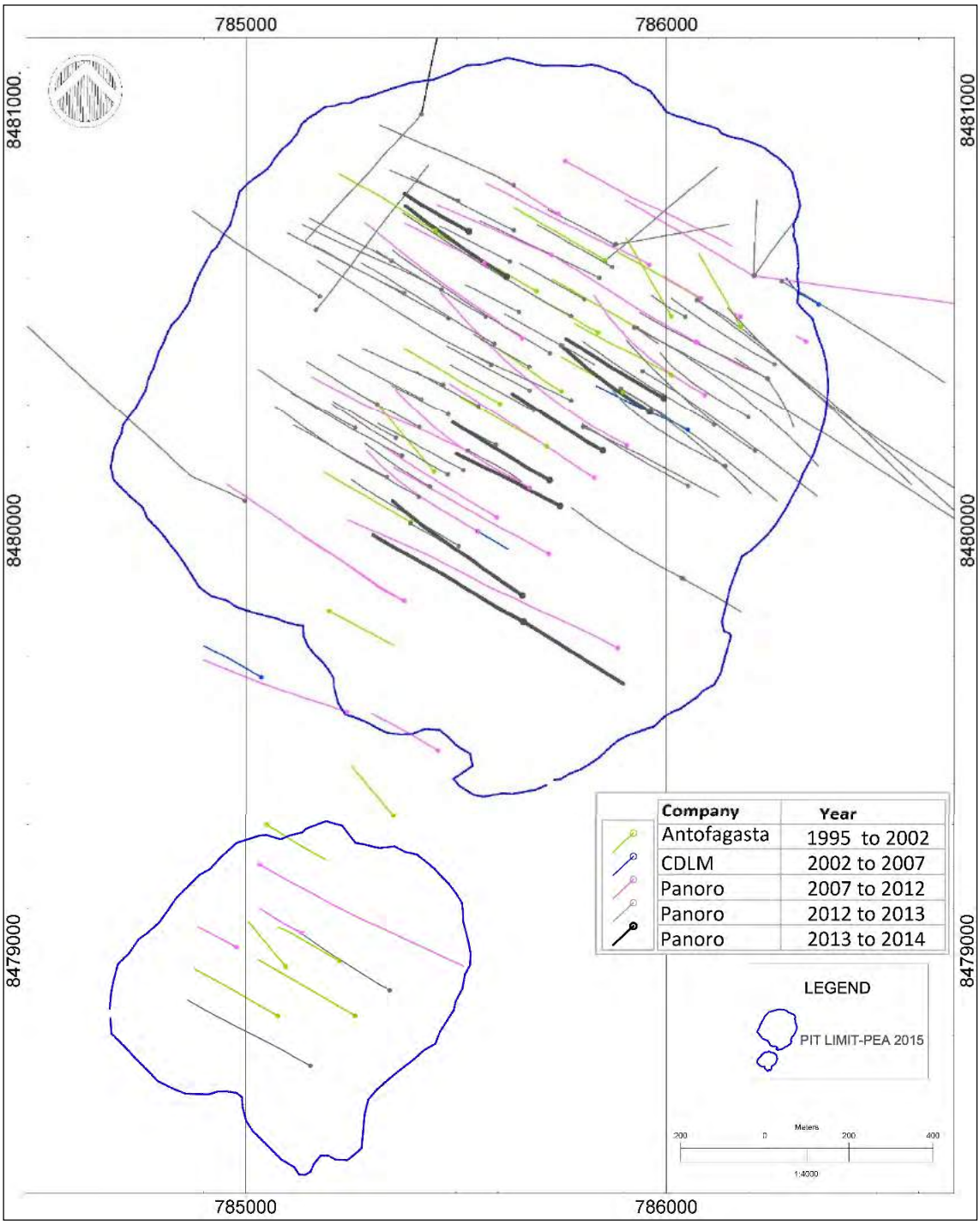
Note: no drilling occurred between 2000 and 2002.

Figure 10-1: Project Drill Location Plan



Note: Figure courtesy Panoro, 2015. Exploration targets indicated by the red outlines.

Figure 10-2: Drill Hole Location Plan, Ccalla and Azulccacca Zones



Note: Figure courtesy Panoro, 2015

In the second half of 2011, Panoro initiated a drill campaign with the objective of expanding the known limits of mineralization at Ccalla and Azulccacca. Panoro contracted Bradley Co. who used LD-250 and LF-38 machines to drill HQ (core diameter of 63.5 mm), NQ (47.6 mm) and BQ (36.5 mm) diameter core. In January 2012, Bradley brought a LF-70 machine to Cotabambas to drill 650 m to 1,100 m long holes.

The drilling during 2012 and 2013 was primarily step-out drilling to determine the lateral extents of the Ccalla and Azulccacca zones and infill/definition drilling within the known zones.

Detailed information on these campaigns is included in Wright and Colquhoun (2012) and Morrison et al., (2014).

10.3 Geological Logging

10.3.1 Legacy Drilling

All historic drilling has been re-logged by Panoro staff, using the same standardized log sheets for consistency. Panoro logging methodologies are described in Section 10.3.2.

10.3.2 Panoro Drilling

A Panoro geologist is assigned to each drill and supervises drilling on all shifts. The geologist supervises transfer of core from the core tube to core boxes, measurement of core recovery and insertion of core blocks marking the end of drill runs. Moulded plastic drill core boxes are used to store whole core. The moulded boxes are stacked and a cover is snap-fit onto the top box for transport and storage. Either the drill contractor or the Panoro geology team bring core boxes to the core storage area once per day.

Geotechnical and geological logging are carried out on whole core by the Panoro geology team. Standardized geological and geotechnical logs are filled out by hand and then entered into a Microsoft Excel® drill hole log template. The log sheets capture interval lengths, lithology code, alteration mineralogy and intensity, sulphide and oxide mineralogy, intensity and occurrence, and major structures.

10.4 Collar Surveys

In 2012, Panoro carried out a collar re-survey program visiting historic and current drill platforms to obtain high-precision total station GPS locations for all drill holes. All subsequent drill collars have been surveyed using the total station instrument.

10.5 Downhole Surveys

Down hole surveys were acquired using Eastman and Sperry Sun photographic tools at approximately 100 m intervals for drill holes drilled during the Antofagasta and CDLM drill campaigns.

For the Panoro 2010–2011 campaign, with the exception of CB-40-11, down hole surveys were acquired at roughly 3 m intervals using an electronic multi-shot magnetic survey tool. Drill hole CB-40-11 was survey with a single-shot magnetic tool at 50 m down-hole intervals.

For the Panoro 2011–2012 campaign, the first five drill holes, to hole CB-45-11, were surveyed at 3 m intervals with a multi-shot magnetic tool. Beginning with drill hole CB-46-11 and continuing to the end of the drill program (drill hole CB-153-14), drill holes were surveyed with a single shot magnetic tool at approximately 50 m down-hole intervals.

10.6 Recoveries

During reviews of pre-Panoro and Panoro drilling in 2012, Amec Foster Wheeler noted that drill core recovery is excellent at Cotabambas. In relatively competent and fractured rock, core recovery is greater than 95%. In intervals crossing strong faults of less than 5 m, generally intersected once or twice per drill hole, core recovery is poor, ranging from as low as 30 to 75% and loss of chalcopyrite from fractures resulting in a possible decrease in apparent grade for these zones.

10.7 Sample Length/True Thickness

Depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths during the Panoro programs are typically greater than true widths. Drill orientations from the Panoro drilling are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.

10.8 Summary of Drill Intercepts

Example drill intercepts for the Project are included in Figure 7-6 to Figure 7-11 in Section 7. These figures illustrate the relationship between the sample length and the true thickness of the mineralization, the mineralization orientation and schematically show areas of higher copper and gold grades.

10.9 Drilling Completed Since the Close-out Date for the Mineral Resource Estimate

Panoro provided Tetra Tech with information on the 11 holes (CB-143-13 to CB-153-14) that had been completed since the database was closed for resource estimation.

Tetra Tech made an assessment of these holes in relation to the Mineral Resource estimate. Tetra Tech considered that the inclusion of these holes in the Mineral Resource estimate would not have any impact on the result, either positively or negatively, and therefore these 11 drilled holes do not constitute a material change.

Tetra Tech assessed the existing sampling versus the new sampling with Q-Q plots for six main elements contributing to the resource. The plots of the samples track very closely indicating that the sample grades in the new samples would not change the estimate averages interpolated from the old samples. Minor deviations in the new sample traces are accounted for by the relatively small population of the new samples when compared to the current resource population. Figure 10-3 shows the Q-Q plot for the copper grades.

This observation is supported by box and whisker plots, and classic statistical assessment of the old samples, new samples, and the combined sample sets all behaving very similarly.

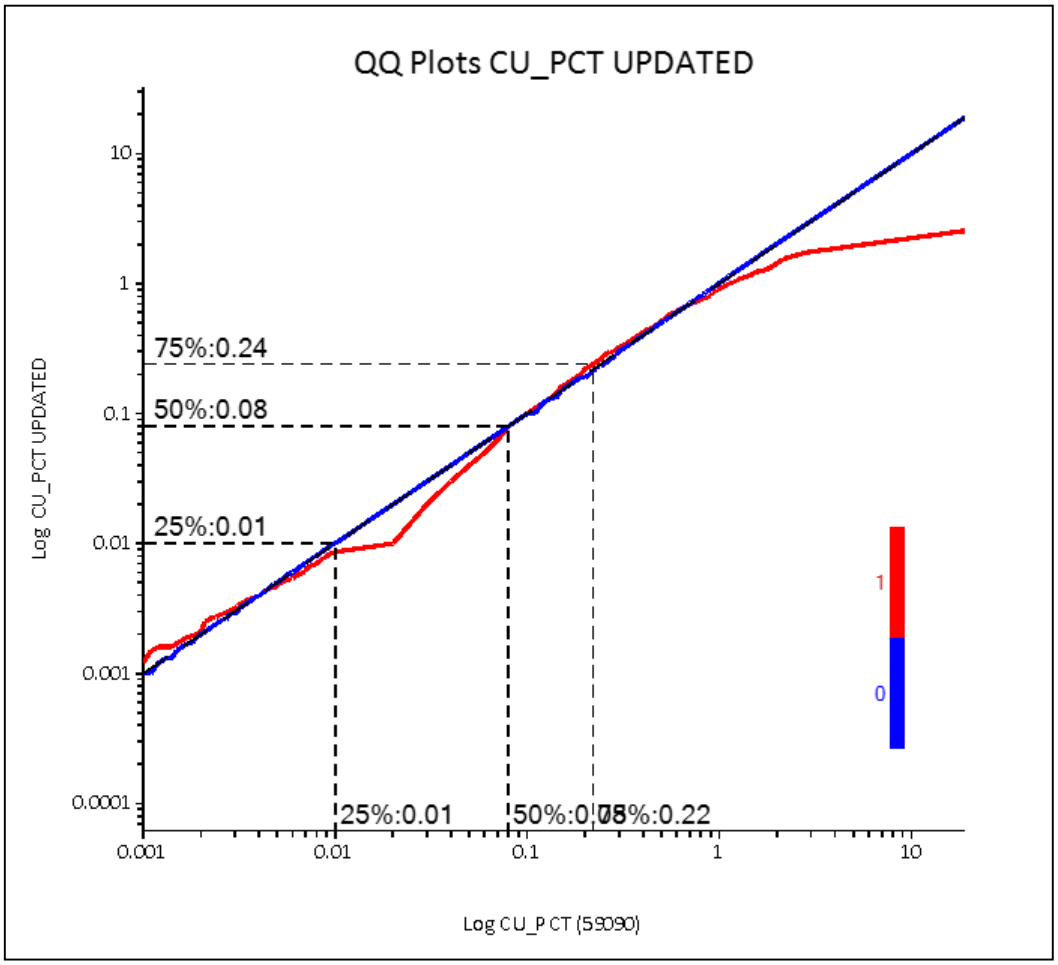
Tetra Tech also compared the model on section with the existing drilling and the new drilling. The correlation and grade distribution between the model informing samples is very closely reflected by the new drilling, as shown in Figure 10-4.

10.10 Comments on Section 10

The QP for Section 10 notes:

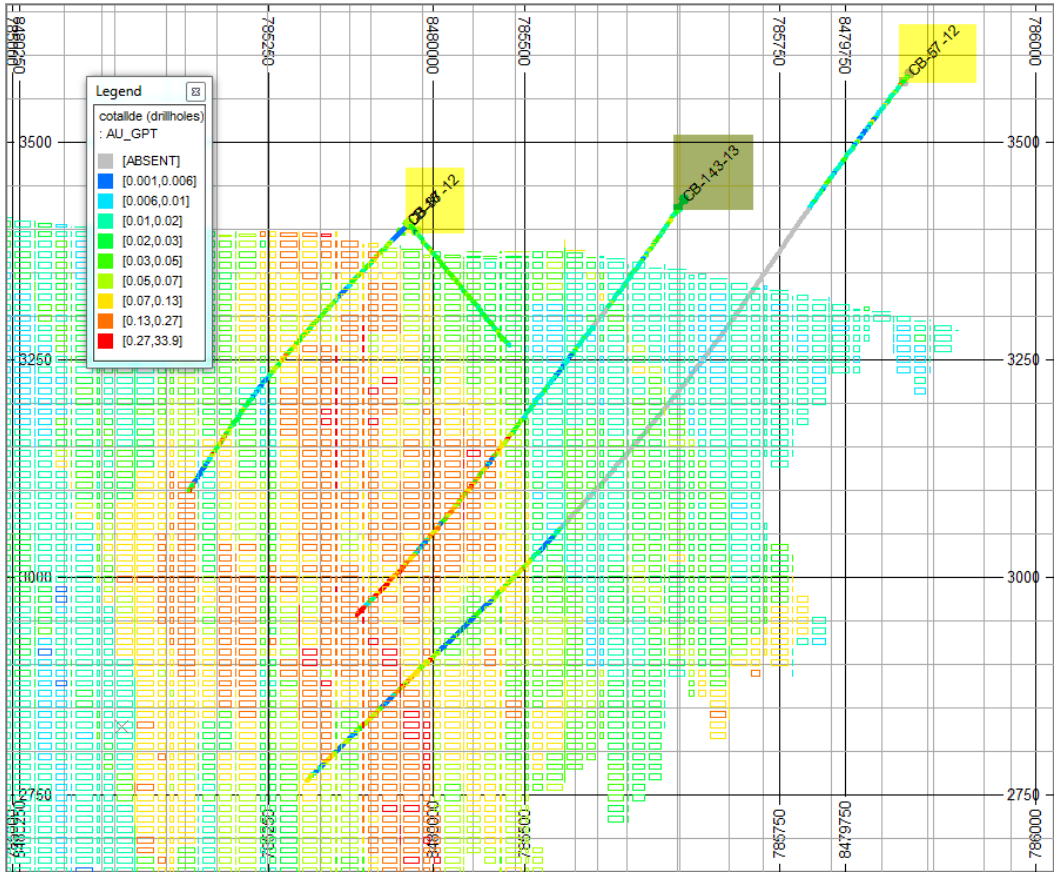
- Amec Foster Wheeler inspected the legacy data available in 2012 for support of Mineral Resource estimation. Legacy drill holes were deemed acceptable for use in resource estimation. Risks were noted (core recovery, surveys and less rigorous QA/QC protocol) relative to Panoro drill holes. Amec Foster Wheeler concluded that the legacy drilling comprises a relatively small portion of the total drill hole database and considers the associated risk with the data to be low.
- Core logging for the Panoro programs meets industry standards for copper and gold exploration in a porphyry setting
- Collar and down-hole surveys for the Panoro programs have been performed using industry-standard instrumentation
- Recovery data from the Panoro core drill programs were considered acceptable for the 2011–2012 drill programs that were reviewed by Amec Foster Wheeler

Figure 10-3: QQ Plot, Copper



Note: Blue trace: existing drilling, red trace: new drilling. Figure prepared by Tetra Tech, 2015.

Figure 10-4: Type Section of the Sample Similarities



Note: Yellow highlight: existing drilling, green highlight: new drilling. Figure prepared by Tetra Tech, 2015.

- The initial geotechnical logging of drill core meets industry standards for planned open pit operations at a PEA level of confidence
- Drill hole spacing is appropriate for porphyry-hosted copper–precious metals mineralization at a PEA level of confidence.
- During the Tetra Tech site visit, drill hole CB-13-114 was witnessed being drilled. Tetra Tech considered that Panoro's drill logging, surveying and sampling methods are adequate for this type of mineralization and deposit. Tetra Tech was of the opinion that the drill program met or exceeded industry norms.

As a result, the QP for Section 10 considers that the data collected in the drill programs is acceptable for support of Mineral Resource estimation.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Introduction

Core sampling methods can be split into two periods: historic sampling by Antofagasta and CDLM, and sampling by Panoro. All historic core drilled by Antofagasta and CDLM has been re-logged by Panoro and transferred to a core storage facility in Cusco.

11.2 Sampling Methods

11.2.1 Legacy Sampling

The details of drill core sampling methods for the Antofagasta and CDLM campaigns are not known; however, re-logging by Panoro and a review of the database has led to some conclusions regarding sampling practices.

Antofagasta took samples at continuous 2 m down hole intervals, splitting the 2 m samples at major geological contacts to produce two shorter samples, one on each side of the contact. Samples were split with a hydraulic press splitter. Half core samples were sent to the laboratory for preparation and the other half core was archived in corrugated plastic boxes with the hole name, box number and interval metreage marked on the box. Boxes were stacked in the core storage facility at Ccalla.

Samples were taken at continuous, un-broken 2 m lengths down hole during the CDLM drill program. Samples were not broken at rock type contacts. One half core was sent to the laboratory for sample preparation, the other half was archived in corrugated plastic boxes in a similar manner to core drilled by Antofagasta.

11.2.2 Panoro Sampling

Drill programs operated by Panoro followed the sample core sampling approach. The core storage facility in Ccalla was used during the first program, but during the second program, a new core storage and logging facility was built at Ccochapata and core logging and storage of new drill holes were moved to the new facility (Figure 11-1).

Figure 11-1: Panoro Ccochapata Core Logging Facility

Note: Photographs are (clockwise from top left): Sampling fracture drill core, core logging benches, cutting area, and container for storage of samples to be dispatched to the laboratory and control materials. Photographs taken by Amec Foster Wheeler, 2012.

11.2.3 Amec Foster Wheeler Sampling Observations (2012)

During a visit to site in April, 2012, Amec Foster Wheeler staff observed drill core handling, logging, sampling and density determination procedures as Panoro staff were processing drill holes CB-66-12, CB-68-12 and CB-69-12. The following text relates to the practices observed at that time.

During logging, the geologist assigned to the drill hole marks sample intervals on the core box. The sampling interval is nominally 2 m, but samples are broken at major contacts in lithology and mineralization type. Samples are divided so that the minimum sample length is approximately 0.5 m and the maximum sample length is 3.0 m. Drill core is washed in the core box prior and dried in open air prior to photography. Core is

photographed first dry, then wet, three boxes at a time with a graphic scale and a sign noting the drill hole number and metreage.

Density samples measuring 10 cm to 15 cm long are taken from the core boxes prior to sampling. The samples are marked with their drill hole number and metreage. Density samples are taken at roughly 10 m intervals or at least once per mineralized intersection as advised by the core logging geologist. Samples are dried for up to 30 minutes in an electric oven. Once dry, samples are weighed, then coated in clear, polyethylene film and weighed again with the film. Samples are weighted a third time, coated in film and suspended in water. The film is then removed and samples are weighed a fourth time, this time without film, suspended in water. Once the samples have been weighted, they are returned to the core boxes.

The core logging geologist marks a line down the length of competent drill core where continuous lengths and large pieces of core are cut using a rotary saw with a diamond carbide blade and returned to the core boxes.

Cut core is taken to the sampling area where core samplers put the half-sawn core in sample bags. Sample bags are pre-numbered with a felt tip pen and doubled to prevent bags from splitting and spilling sample. Broken core is sampled from the core tray using a small scoop. Once the nominally two meter sampling intervals have been taken, a sample tag with bar code is placed in the bag, the bag tops are rolled down and stapled shut then wound with clear packing tape.

A pre-defined sample dispatch sheet is filled out during sampling for lots of 70 samples. The dispatch sheet captures the sample number, sampling interval and has control samples pre-inserted into the sampling stream. Control samples consist of coarse blanks, commercially prepared certified reference materials (CRMs) and core twin samples. Core twin samples are sent as a quarter cut original and quarter cut twin sample. CRMs are of high, medium and low grade copper-gold standards prepared by WGM laboratories in Vancouver, Canada.

Samples are transferred to rice bags and stored in a 24-foot container at the core logging facility where they are stored until a truck load is ready for shipment. Panoro delivers samples sent to the ALS Chemex sample dispatch facility in Cusco, where ALS Chemex manages their transport to the sample preparation facility in Arequipa, and then the assay facility in Callao.

11.3 Density Determinations

There are three sources of density data for the Cotabambas Project:

- Pre-Panoro data for 3,125 samples for which individual weights are not recorded and density determination protocols are unknown

- Cellophane film-sealed water immersion density determinations on 1,443 samples with sealed and un-sealed weights in water and air carried out by Panoro
- 107 density validation determinations carried out on behalf of Panoro by ALS Chemex in Lima using a wax-sealed water immersion method.

All density samples were taken from 7 cm to 15 cm long pieces of un-cut drill core.

Historic, pre-Panoro data could not be validated and was rejected.

A systematic bias was observed with the cellophane sealed dry bulk insitu density values 10% lower on average than the corresponding ALS Chemex check samples. This bias was attributed to the inclusion of air bubbles in the cellophane used to seal the samples. Air bubbles increase the sample volume when immersed in water and decrease its apparent density.

Unsealed bulk insitu density values were plotted against the ALS Chemex density validation samples. There is excellent correlation between the ALS Chemex and unsealed Panoro determinations above a specific gravity (SG) value of 2.5, but below this value, the unsealed Panoro densities were systematically lower than the wax-sealed density (Figure 11-2). This conditional bias is due to the porosity of the lower density samples and the overstatement of the insitu dry bulk density of porous samples when determined by water immersion methods without sealant.

A least-squares linear regression equation was derived to relate unsealed bulk density to dry insitu bulk density and the full suite of Panoro density determinations were used to estimate dry insitu bulk density for each domain (refer to Figure 11-2).

11.4 Sample Preparation and Analysis

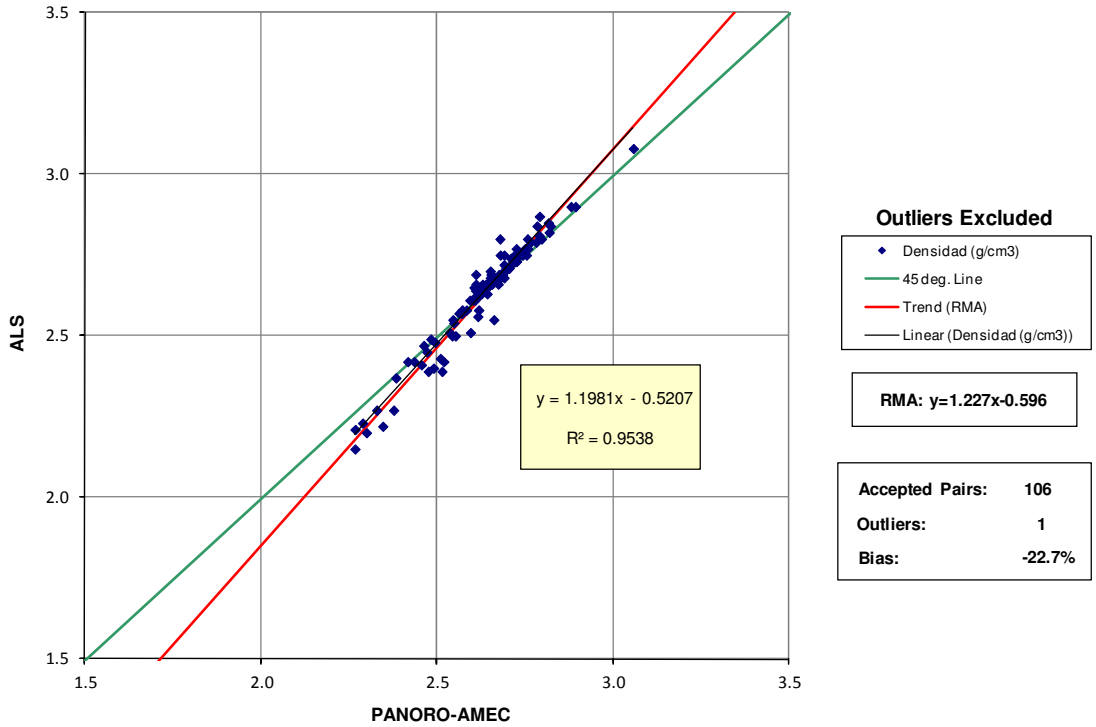
11.4.1 Legacy Campaigns

During the Antofagasta drill campaign, samples were prepared at ALS Geolab Perú in Arequipa and analyzed by ALS Geolab, an independent assay facility in Lima. ALS Geolab was the predecessor to ALS Chemex, now ALS Global, in Perú at the time. Near the end of the Antofagasta drill campaign, Antofagasta changed from ALS Geolab to CIMM Perú, another independent assay facility in Lima, for preparation and analysis. Results were reported for total copper by atomic absorption (AA) and gold by fire assay.

Preparation and analysis for the CDLM campaigns were carried out by CIMM Perú in Lima. Results were reported for total copper, sulphuric acid soluble copper (CuAS), silver by atomic absorption (AA) and gold by fire assay.

Accreditations for ALS Chemex and CIMM Perú are unknown for the time of the pre-Panoro campaigns.

Figure 11-2: Panoro Unsealed vs. ALS Chemex Wax Sealed Dry Insitu Bulk Density



11.4.2 Panoro 2011–2012 Campaigns

Panoro staff supervised drilling at drills on two shifts, transported core to the core handling facility, logged, and sampled all core. Bagged samples were stored in a locked container beside the core shed until a batch could be dispatched by pickup-truck to Cusco.

Samples were prepared by the ALS Minerals (formerly ALS Chemex) sample preparation facility in Arequipa. Samples were registered and assigned a laboratory information management system (LIMS) code upon reception. Samples were transferred from bags to steel pans and dried in racks in a large gas-fired oven for several hours at 100–105°C. Dry samples were crushed to better than 70% passing -2 mm. A 250 g sub-sample of the crushed sample was taken and pulverized to better than 85% passing 75 µm. The pulps were sent to the ALS Minerals chemical laboratory for analysis.

Samples were analyzed at the ALS Minerals chemical laboratory in Lima by AA with the AA62 package for total copper, molybdenum, lead, zinc, and silver, and fire assay for gold. A 2 g split of the prepared pulp was digested with a $HF–HNO_3–HClO_4$ solution, leached with HCl, and read by AA for each of the six elements. Gold was assayed using the Au-AA23 package where a 30 g sample aliquot is fused, cupilated, the bead digested

in aqua regia, and the final solution read by AA. Trace mercury was assayed using Hg-CV41 package and 33 elements were assayed by ME-ICP61 package.

ALS Global is now an independent laboratory with ISO 9001:2000 and ISO 17025 certifications at its facilities in Perú.

During the different sampling campaigns, assaying for copper has been done systematically for all samples; however assaying for gold, silver and other elements had not been done for all samples.

11.4.3 Panoro 2013–2014 Campaigns

No significant changes were made to the basic analytical protocols in 2013–2014, and ALS Global was maintained as the primary laboratory.

Leach packages were used for samples where supergene mineralogy was identified. Gold leached by cyanide was assayed by Au-AA13 package where 30 g sample aliquot was digested and the solution read by AA. Copper leached by sulphuric, cyanide, and residual were also assayed by Cu-AA06s, Cu-AA16s and Cu-AA62s packages, respectively.

11.5 Quality Assurance and Quality Control

11.5.1 Legacy Campaigns

During the Antofagasta and CDLM campaigns quality assurance practices relied on internal laboratory controls and do not meet current industry standard practices.

In early 2012 Panoro sent a total of 174 rejects to Inspectorate Services Perú S.A.C (Inspectorate) to evaluate the quality of the Legacy Panoro data. These rejects were selected from Antofagasta and CDLM drill campaigns to be re-analyzed as a verification program.

Inspectorate is certified under ISO 9001:2000 for assaying services.

Amec Foster Wheeler evaluated the results of the verification program, comparing the original Legacy Panoro results against to the Inspectorate results. The correlation coefficient of the original and check assays have a coefficient of correlation of 0.996 for copper and 0.983 for gold which demonstrates the high reproducibility of the historic data.

The check assays returned copper grades on average 4.2% lower than the original assays from ALS and Inspectorate. The check assay gold grades were on average 4.0% lower than the original gold grades. The reproducibility of silver grades were not evaluated because there were only six data pairs available.

The results of standard reference materials analyzed with the check samples indicate that the Inspectorate results are approximately -3% low for copper and -10% low for gold. The negative bias of the Inspectorate copper assays suggests that the original legacy Panoro assaying has a negligible negative bias for copper of approximately -1%. The negative bias of Inspectorate gold assays demonstrated by the reference standards, suggests that the original legacy Panoro gold assays have a negative bias of approximately -6%.

The check assay campaign indicates high reproducibility of the original copper and gold results, and a negligible -1% negative bias for the original copper assays and a negative bias of -6% for gold grades. The original legacy Panoro assaying was considered by Amec Foster Wheeler to be acceptable for Mineral Resource estimation.

11.5.2 2011–2012 Panoro Campaigns

The text which follows is applicable to the 2011–2012 program reviewed by Amec Foster Wheeler. Detailed information on the quality assurance and quality control (QA/QC) program reviews for this duration is included in Wright and Colquhoun (2012).

Panoro implemented a quality control (QC) procedure in the field in 2010. This procedure included the insertion of duplicates, coarse blanks and four certified standards. The certified standards were prepared by WCM Minerals of Burnaby, Canada. The frequency and control type inserted were modified and reduced for the 2011–2012 campaign.

Amec Foster Wheeler generally recommends that a maximum of 20% of the samples analyzed during an assaying campaign are control samples. The number of the control samples in the Panoro drill campaigns totals approximately 10%; however, the results of the quality assurance program implemented in the Panoro drill campaigns do demonstrate the lack of contamination, reasonable precision and accuracy of the assays by virtue of the reasonably high precision of the core twin analyses, which is the objective of a good QA/QC program.

11.5.3 2013–2014 Panoro Campaigns

Information on the QA/QC reviews by Tetra Tech that evaluated the results of the 2013–2014 Panoro campaigns are included in Section 12 of the Report.

Drilling completed after the close-out of the database used in Mineral Resource estimation has not been reviewed in detail, but a check of the results against the block model was performed, and is discussed in Section 10.9.

11.6 Sample Security

Sample security is performed in accordance with exploration best practices and industry standards. Core is taken from the core tube and placed in core boxes at the drill site to a locked sampling facility under the supervision of Panoro geologists.

Samples and reference materials are stored in a locked container until shipping in a truck to the ALS warehouse in Cusco from which point ALS Minerals takes responsibility for chain of custody. Drill core, coarse reject and pulps are archived at Panoro's core storage warehouse in Cusco.

Historic data have been validated via a check assaying program and the high reproducibility of the original results indicates to Amec Foster Wheeler that there was no tampering of the original results or the stored pre-Panoro core and coarse reject material.

11.7 Comment on Section 11

The QP for Section 11 notes:

Based on the reviews performed by Amec Foster Wheeler staff in 2012, Amec Foster Wheeler concluded that the legacy data and the 2011–2012 Panoro analytical, density and QA/QC data are acceptable to support Mineral Resource estimates.

Based on reviews performed by Tetra Tech, in 2014, throughout the 2013 drill program there has been no change in the sample preparation, methodology, and security. Tetra Tech considered that the sample preparation, methodology, and security were adequate for the purposes of Mineral Resource estimation, and subsequent mining studies.

The QP for Section 11 therefore concludes that the sample preparation, methodology, and security are adequate for the purposes of Mineral Resource estimation, and can support the PEA level mine plan and economic evaluation.

12.0 DATA VERIFICATION

12.1 Tetra Tech Data Verification (2014)

Upon receipt of Panoro's database for the Project, all relevant data underwent extensive data verification. Of the 144 drill holes included, 22 were chosen for validation, representing approximately 11% of the drill holes in the database.

12.1.1 Collar Data

Collar data was provided in "*csv*" (comma-separated values) spreadsheet format. No original survey information for the collar locations was provided, and therefore no verification was possible.

12.1.2 Lithology Data

Lithological data was provided in *.csv* spreadsheet format. The lithologies of the 22 chosen holes represent 20% of the lithological database. There were discrepancies between the originals and the database; however, all of these differences are attributable to recent updating, relogging, and consolidation of lithological data. When these changes are factored out, 100% of the data verified match the originals.

12.1.3 Assay Data

Assay data was provided in csv spreadsheet format. The assay results for the 22 chosen holes represent 18% of the assay database. These results were verified against the original laboratory certificates and laboratory-issued pdf spreadsheets. One hundred percent of the verified samples matched the original data.

12.1.4 Downhole Survey Data

Downhole survey data was provided in *.csv* spreadsheet format. No original downhole survey information was provided and therefore no verification was possible.

12.1.5 Quality Assurance/Quality Control Data

Blanks

Crushed quartz samples (external source) were used by Panoro as blank reference material. Performance of the blank samples was adequate, with a few anomalous samples returning higher than background values for copper and silver. Although these anomalies represent less than 1% of the samples, it is recommended that any failures be re-assayed by the laboratory. Figure 12-1 displays the blank performance.

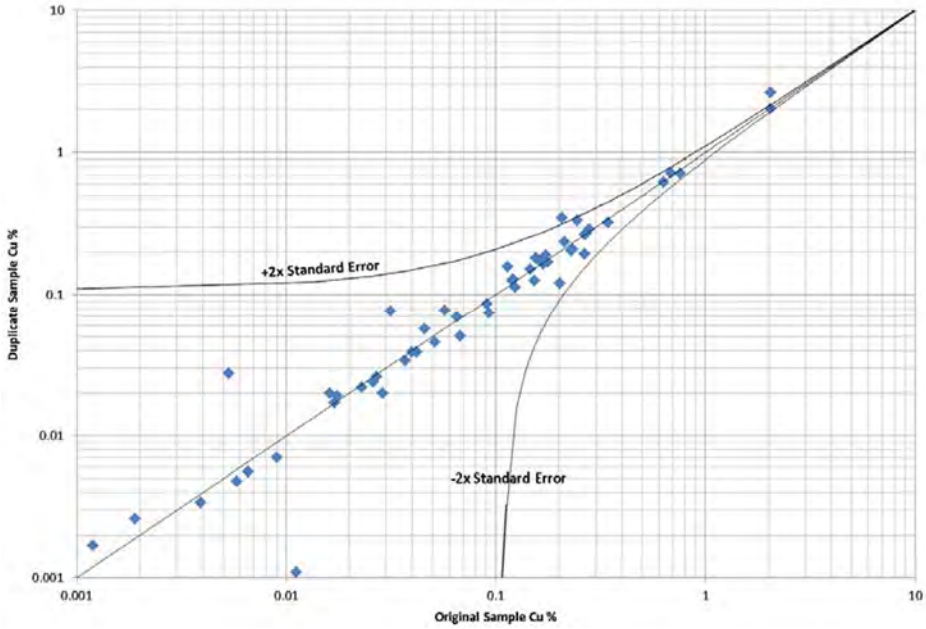
Figure 12-1: Blank Performance

Note: Figure prepared by Tetra Tech, 2014

Duplicates

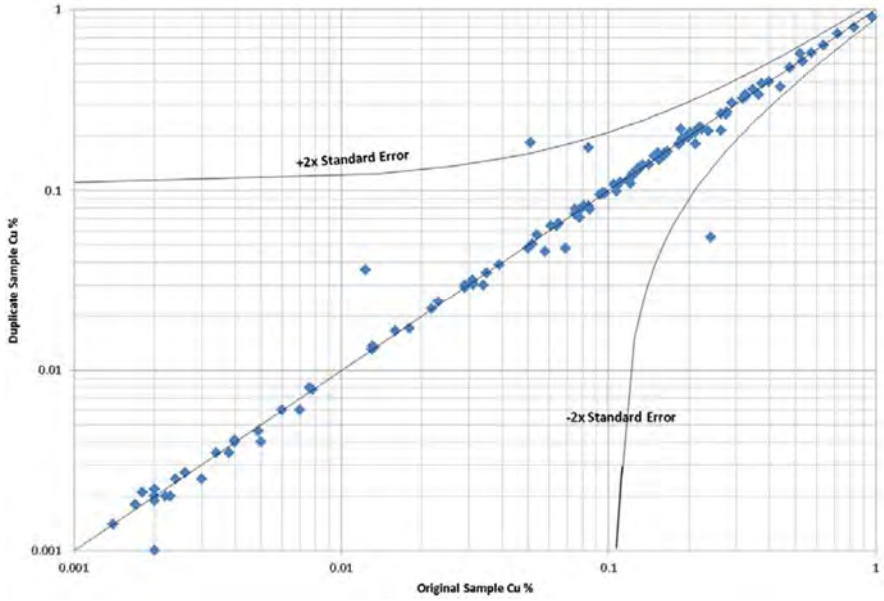
Three duplicate types were used by Panoro for reference material: split core duplicates, reject duplicates, and pulp duplicates. All three performed well, with less than 5% of the samples falling outside of the two times the standard error for each type. Figure 12-2, Figure 12-3 and Figure 12-4 display the duplicate control graphs.

Figure 12-2: Split Core Duplicate Control Graph



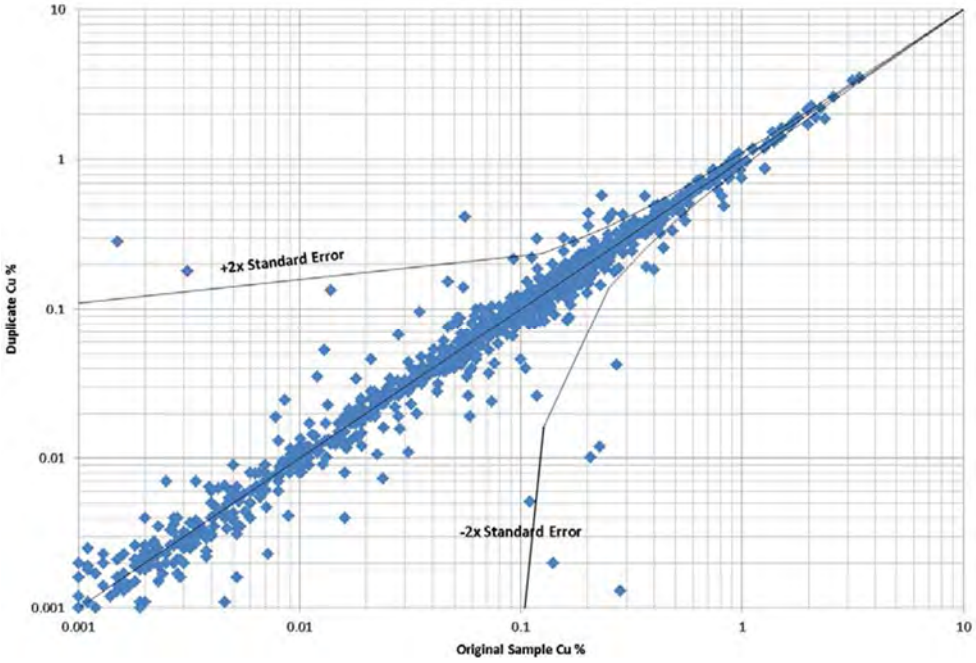
Note: Figure prepared by Tetra Tech, 2014

Figure 12-3: Reject Duplicate Control Graph



Note: Figure prepared by Tetra Tech, 2014

Figure 12-4: Pulp Duplicate Control Graph



Note: Figure prepared by Tetra Tech, 2014

12.1.6 Tetra Tech Site Visit

Mr Paul Daigle, P.Geo., Senior Geologist with Tetra Tech conducted a site visit to the Property from June 3 to 7, 2013, inclusive. One day was spent on the Property and one day at Panoro's core storage warehouse in Cusco. Mr Daigle was accompanied on the site visit by Mr Luis Vela Arellano, Vice President Exploration for Panoro; Mr John Romero Villanueva, Chief Project Geologist for Panoro; and Mr Edwin Mayta, Manager Technical Services, for Panoro.

12.1.7 Base Camps and Facilities

The Cotabambas base camps and project site were visited on June 6, 2013. The main base camp is located adjacent to the village of Ccochapata and is made up of several permanent bungalow-style buildings (offices), as shown in Figure 12-5, and semi-permanent wood and corrugated tin structures (drill logging, sampling), as illustrated in Figure 12-6 to Figure 12-9.

Figure 12-5: Cotabambas Base Camp

Note: Photograph by Tetra Tech, 2013

Figure 12-6: Cotabambas Base Camp – Core Logging and Sampling Facility

Note: Photograph by Tetra Tech, 2013

Figure 12-7: Cotabambas Base Camp – Core Sampling

Note: Photograph by Tetra Tech, 2013

Figure 12-8: Cotabambas Base Camp – Core Photography

Note: Photograph by Tetra Tech, 2013

Figure 12-9: Cotabambas Base Camp – Core Density Measurements

Note: Photograph by Tetra Tech, 2013

Storage of samples and drill core boxes are kept in a locked sea container at the camp before being transported to Cusco for shipping to the laboratory for analysis and for drill core storage. Drill core and core samples are temporarily stored at this base camp in a locked chain link fence enclosure until transport is arranged to the second base of operations in the village of Cotabambas, situated approximately 7 km away by road. Both bases are kept clean and are well-maintained.

The second base camp also has a permanent cement building for offices and includes a kitchen for personnel. It also has similar drill logging, sampling, and photographic facilities as the main base camp. Tetra Tech was able to review drill hole CB-141-13 that was currently being drilled by Panoro at the time of the site visit. Figure 12-10 and Figure 12-11 present the facilities at the second based camp.

The core logging and sampling facilities at both base camps are kept clean and orderly. When stored, the core boxes are stacked by drill hole. The plastic core boxes are sturdy and made to be stackable. The core boxes are marked in black text marker showing drill hole number, box number, and sample interval.

Figure 12-10: Second Base Camp – Drill Core Logging, Sampling and Storage Facility

Note: Photograph by Tetra Tech, 2013

Figure 12-11: Second Base Camp – Temporary Core Storage

Note: Photograph by Tetra Tech, 2013

12.1.8 Project Site

The Project site of the Ccalla deposit is located approximately 1.5 km by road to the main base camp. The Project site is situated on the western slope of the Ccalla Creek. The slope is relatively steep sided with a network of roads that allows passage for 4x4 vehicles to most of the drill hole locations as shown in Figure 12-12. The Project site was clear of drilling debris.

Sixteen drill hole collars were sited in by handheld global positioning system (GPS). All checked drill hole collars were consistent with the drill hole coordinates in the drill logs and in the database. Drill collars are clearly marked on the ground. The collar is fitted with polyvinyl chloride (PVC) pipe and cemented into place. The drill hole number is engraved in the cement and, at some drill hole locations, marked on a nearby boulder or outcrop. Figure 12-13 illustrates the collar for drill hole CB-80.

12.1.9 Core Storage Warehouse, Cusco

The Cotabambas drill core is stored temporarily at site or in one of three warehouses in Cusco. Tetra Tech visited one of these warehouses in Cusco prior to visiting the Property. The warehouse is secured under lock and has its own watchman. The warehouse contained most of the Cotabambas drill core and some drill core from Panoro's other projects.

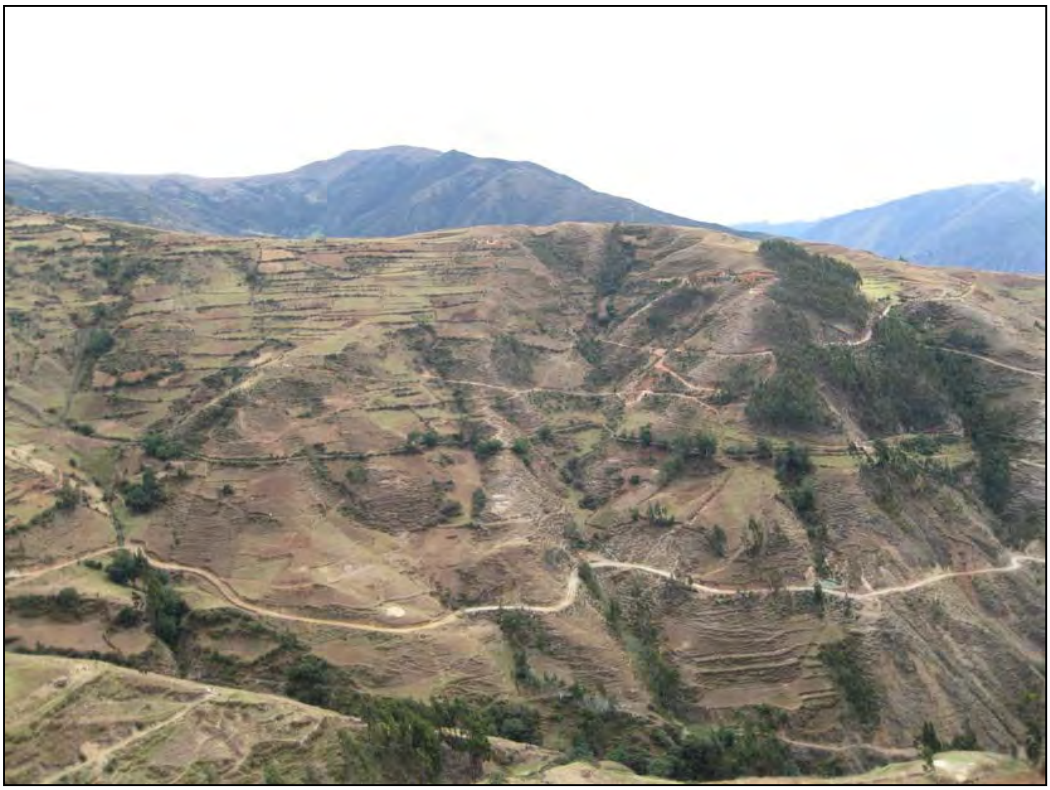
The warehouse also serves as a storage depot for exploration, field and camp supplies and equipment for the various projects. The warehouse is kept clean and has a wooden drill core tables along its length for viewing drill core (Figure 12-14).

12.1.10 Check Samples

Independent check samples were collected during Tetra Tech's site visit. Three samples were collected from the available drill core at the core storage site at Panoro's core storage warehouse in Cusco.

The check sample intervals were selected randomly within the mineralized lithologies and collected from the same sample intervals as Panoro. As no core saw was available, Tetra Tech selected alternating pieces of half core. The samples were collected by the author, placed in labelled sample bags and sealed. Sample tags were inserted in the core box and in the sample bag. The samples were kept with the author at all times during the site visit. Upon returning to Toronto, Tetra Tech shipped the samples to Activation Laboratories Ltd. (Actlabs) for analysis. Figure 12-15 shows a check sample taken from drill hole CB-68-12.

Figure 12-12: Access Roads on the Ccalla Deposit; Looking West



Note: Photograph by Tetra Tech, 2013

Figure 12-13: Drill Hole Collar for CB-80, -82, -74

Note: Photograph by Tetra Tech, 2013

Figure 12-14: Panoro's Drill Core Storage Facility, Cusco

Note: Photograph by Tetra Tech, 2013

Figure 12-15: Figure Drill hole CB-68-12

Note: Photograph by Tetra Tech, 2013

At Actlabs, the samples were prepared and analyzed as close to Panoro's method as possible. In sample preparation, the sample was crushed to up to 90% of the sample passing a 2 mm screen, split to 250 g and pulverized where 90% passed 105 µm screen (Actlabs Code RX-1). Analysis was conducted using four acid digestion (Actlabs Code 8 – Cu, Mo, Ag) and induced coupled plasma–optical emission spectroscopy (ICP-OES). For gold, fire assay and atomic absorption was employed (Actlabs Code 1A1).

The purpose of the check sample assays are to confirm indications of mineralization are not intended as duplicate or QA/QC samples. Tetra Tech check sample analysis correlates with Panoro's assay results, for the same sample intervals. Results of the check assay sample analysis and corresponding sample analysis by Panoro are shown in Table 12-1 and Table 12-2.

12.2 Comments on Section 12

Data verification was performed by third-party consultants independent of Panoro.

The QP for Section 12, who relys upon this work, has reviewed the reports and is of the opinion that the data verification programs completed on the data collected from the Project appropriately support the geologic interpretations and the database quality, and therefore support the use of the data in Mineral Resource estimation, and in mine planning at the PEA level.

Table 12-1: Summary of Check Samples Collected by Tetra Tech

Tetra Tech Sample No.	Panoro Sample No.	Drill Hole	Sample Interval (m)	Core Boxes	Deposit
626475	M166222	CB-68-12	336.8 to 338.8	88	Ccalla
626476	M166049	CB-68-12	43.0 to 45.0	7, 8	Ccalla
626487	M166297	CB-68-12	465.3 to 467.0	117	Ccalla Este

Table 12-2: Summary of Check Samples Results Collected by Tetra Tech

	Drill Hole	Cu%	Mo%	Au (g/t)	Ag (g/t)
Tetra Tech Sample No.					
626475	CB-68-12	0.342	0.008	0.076	<3
626476	CB-68-12	1.02	<0.003	0.064	<3
626487	CB-68-12	0.512	0.011	0.129	<3
Panoro Sample No.					
66457	CB-68-12	0.313	0.008	0.059	1
66492	CB-68-12	0.732	0.001	0.081	3
69580	CB-68-12	0.666	0.025	0.240	1
	—	0.029	0.000	0.017	—
Difference	—	0.288	—	-0.017	—
	—	-0.154	-0.014	-0.111	—

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testwork

No metallurgical testwork was conducted prior to 2012.

The results of preliminary comminution, hydrometallurgical and flotation test work carried out for Cotabambas since 2012 are discussed in this section. The objective of the test program was evaluation of amenability of the main mineralization types to conventional metallurgical flow sheets by carrying out preliminary leaching, flotation and comminution test work. A summary of the test work programs completed are listed in Table 13-1. The metallurgical samples tested were based on drill hole core composites generated from different mineralization zones.

The 2012 (Certimin) test program investigated the metallurgical responses of four mineralization zones to acid leach, cyanidation and flotation concentration.

The 2013 (Peacocke) test work, using reject samples from the 2012 test work, focused on investigating the metallurgical responses of the mineralization to gravity concentration, although preliminary scoping tests using acid leach, cyanidation and flotation procedures were also conducted on the tailings produced from the gravity concentration tests.

Certimin completed four flotation test programs on composite samples of hypogene sulphide, Cu oxide and Cu–Au oxide zones and blends of these.

Since the 2012 test program there has been a material increase in primary hypogene sulphide resources over previous estimates due to additional drilling. Objectives of the initial hypogene sulphide batch testing were to test flotation amenability of metallurgical samples from the most recent in-fill drilling program and select appropriate processing parameters to be used in final stage locked cycle flotation testing. An objective of the final stage locked cycle testing was to either reconfirm the 2012 hypogene sulphide global metallurgical recoveries using the new metallurgical composite or provide updated estimates for this study.

A significant increase in copper oxide Mineral Resources, some associated with moderate gold grades, in conjunction with the larger primary copper sulphide deposit currently estimated in relation to previous estimates, warranted an investigation of alternative processing strategies to recover copper and gold from the copper oxide zones. The traditional hydrometallurgical techniques for processing oxide mill feed by leaching–solvent extraction–electrowin (SX/EW) requires capital expenditure to construct a dedicated plant. Certimin also conducted two programs to assess options, conditions, and the amenability of copper oxide zones to flotation recovery either alone or as a blend with hypogene sulphide.

Table 13-1: Test Work Program Summary

Testing Program		Mineralization Type (Zone)	Metallurgical Test	Sample/Test Parameters
2012	Certimin	Au Oxide	• Comminution • Cyanide leach	Moderate gold grades, low copper grade, abundant iron oxides/hydroxides, from leached cap.
		Cu-Au Oxide	• Comminution • Cyanide leach • Acid leach	Moderate gold and copper grades, visible green copper oxides, abundant iron oxides/hydroxides, minor copper sulphides, from leached cap.
		Sec. Sulphide	• Comminution • Flotation	Sulphide zone with chalcocite and chalcopyrite. High copper grade, moderate gold grade.
		Hypogene Sulphide	• Comminution • LC Flotation	Main sulphide zone with predominantly chalcopyrite. Moderate to low Cu/Au grade.
2013	Peacocke	Au Oxide	• Gravity • Cyanide leach	As above.
		Cu-Au Oxide	• Batch Gravity • Cyanide Leach • Acid Leach	As above.
		Sec.Sulphide Hypogene Sulphide	• Batch Gravity • Batch Flotation	As above. As above.
2014	Certimin FEB4007	Hypogene Sulphide Hypogene Cu-Mo	• Comminution • Batch Flotation	Main sulphide zone with low gold grade Main sulphide sub-zone high Mo (trace Au).
	Certimin ABR4003	Cu-Au Oxide Cu Oxide	• Batch Gravity • Batch Flotation	Oxide moderate Cu and Au grades Oxide moderate Cu (trace Au)
	Certimin MAY4000	Cu-Au Oxide Blends Cu Oxide Blends	• Batch Gravity • Batch Flotation	7.5% & 15% Oxide with Hypogene Sulphide 7.5% & 15% Oxide with Hypogene Sulphide
	Certimin MAY4015	Hypogene Sulphide Cu-Au Oxide Blend	• Batch Gravity • LC Flotation	Main sulphide zone with low gold grade 7.5% Cu-Au Oxide with Hypogene Sulphide
2014	Aminpro	Cu-Au Oxide	• Acid Leach • Cyanide Leach	Oxide moderate Cu and Au grades. Batch bottle roll and column testing.
		Cu Oxide	• Acid Leach	Oxide moderate Cu (trace Au). Batch bottle roll and column testing.
2014	Outotec	Hypogene Sulphide Cu-Au Oxide Blend	• Settling • Filtration • Rheology	Flotation concentrate and tailings dewatering.

Aminpro conducted a hydrometallurgical program consisting of batch bottle roll and column testing to assess the amenability of the copper oxide zones to traditional acid leaching for copper recovery and cyanidation for gold recovery where relevant. Amec Foster Wheeler used the results of this test work to conduct a preliminary economic trade-off study on hydrometallurgical processing of copper oxide resources for the project resulting in the elimination of leaching as a processing option in this study.

Outotec conducted preliminary settling, filtration and rheological tests on concentrate and flotation tailings on hypogene sulphide and oxide blend products from the locked cycle testing of Certimin. Amec Foster Wheeler used the tailings dewatering data to conduct tailings disposal economic trade-off studies considering filtered stacked or pumped thickened tailings options resulting in the selection of the latter for the basis of this study.

Panoro selected the metallurgical samples based on the oxide and sulphide domain definition characterized at the time of sampling. Amec Foster Wheeler supervised the relevant sample collection and laboratory test work.

13.1.1 2012 Test Work – Certimin

The following is taken from Wright and Colquhoun (2012).

The results of preliminary comminution, hydrometallurgical and flotation test work carried out by Certimin are discussed in this section. The objective of the test program was evaluating the amenability of the main mineralization types at Cotabambas to conventional metallurgical flow sheets by carrying out preliminary of leaching, flotation and comminution test work. Details of sample locations and preparation and details of the tests are provided in Wright and Colquhoun (2012).

The head assays of the metallurgical composites are shown in Table 13-2.

Preliminary Comminution Test Work

Certimin determined the Bond ball mill work index (BWi) for the four mineralization types and the results are listed in Table 13-3.

Results of this BWi test work indicated that gold oxide, copper-gold oxide and secondary sulphide mineralization types are relatively soft and hypogene sulphide mineralization is of moderate hardness.

Table 13-2: Certimin 2012 Testwork Head Assay Results

Zone	Gold Oxide	Copper-Gold Oxide	Secondary Sulphide	Hypogene Sulphide
Determination by Atomic Absorption				
Au (g/t)	1.029	0.504	0.836	0.336
Ag (g/t)	4.5	3.8	7.2	6.0
Cu (%)	0.078	0.542	2.368	0.542
Cu H_2SO_4 (%)	0.014	0.346	0.282	0.012
CuCN (%)	0.011	0.027	1.395	0.037
Cu Residual (%)	0.047	0.152	0.654	0.501
Fe (%)	4.071	4.887	5.382	6.477
Multi-element ICP				
As (ppm)	18	<3	11	18
Ba (ppm)	530	680	607	556
Bi(ppm)	<5	8	<5	<5
Cd (ppm)	<1	<1	<1	<1
Mn (ppm)	104	307	569	1,634
Mo (ppm)	7	15	10	16
Pb (ppm)	61	110	148	142
Sb (ppm)	9	<5	<5	8
Sn (ppm)	<10	<10	16	<10
Zn (ppm)	56.3	93	260	231

Table 13-3: Bond Ball Mill Work Index Result

Mineralization Type	BWi (kWh/t)
Gold Oxide Zone	10.51
Copper-Gold Oxide Zone	10.22
Secondary Sulphide Zone	10.44
Hypogene Sulphide Zone	14.23

Cyanide Leaching Test Work

The gold oxide metallurgical sample was subjected to cyanide leach tests with varying grind sizes and cyanide concentrations. The result showed that on average it is possible to obtain a gold recovery of 79% and a silver recovery of 46% with a cyanide consumption of 0.70 kg/t from the gold oxide mineralization at a grind size of about P80 -200 mesh.

A similar battery of cyanide leach tests was carried out on the copper-gold oxide sample. The results indicated an 80% gold recovery and 23% silver recovery with a cyanide consumption of 2.10 kg/t at a grind size of about P80 -200 mesh.

Acid Leach Test Work

The copper–gold oxide sample was subjected to sulphuric acid bottle roll leach amenability testing at varying acid concentrations for samples crushed to P100 -10 mesh. The results indicated a copper recovery of 71% and an acid consumption of 29 kg/t.

Flotation Test Work

The flotation test work consisted of batch tests evaluating reagents, grind, pH, pyrite depressors, rougher kinetics, rougher concentrate regrind and cleaner kinetics, followed by a locked cycle flotation test. Work was carried out individually for the secondary sulphide composite and the hypogene sulphide sample.

The flotation flow sheet for the secondary sulphide mineralization consisted of grinding, conditioning, two-stage rougher flotation followed by re-grinding of the rougher concentrate product to produce a cleaner concentrate that was fed back to the conditioning circuit. The locked-cycle flotation test on the secondary sulphide sample resulted in a copper concentrate with 31% copper, 9.2 g/t gold and 92 g/t silver with recoveries of 83% for copper, 90% for gold and 92% for silver.

The flow sheet for the hypogene sulphide locked cycle test work consisted of milling, conditioning, and two-stage rougher flotation followed by three-stage cleaner flotation. The results indicated a copper concentrate with a grade of 27% copper, 11.9 g/t gold and 152 g/t silver with 87% copper, 62% gold and 60% silver recovery (Table 13-4). The combined lead and zinc grade was about 1.4% in the locked cycle test which was below the normal smelter penalty limit of about 3%.

Table 13-4: Hypogene Sulphide Zone – Locked Cycle Flotation Test Results

Product	Weight	RC	Au	Ag	Cu	Fe	Mo	Recovery %			
	%		g/t	g/t	%	%	g/t	Cu	Au	Ag	Fe
Copper Conc.	1.75	57.1	11.9	152	27.0	30.7	613	87.4	62.0	60.4	7.7
Tail	98.25		0.13	1.8	0.07	6.66		12.6	38.0	39.6	92.3
Calculated Head	100.00		0.34	4.4	0.54	7.08	16	100.0	100.0	100.0	100.0

Note: RC = Concentration Ratio

No other deleterious elements that could have a significant effect on potential economic extraction were noted. Mo is indicated in the concentrate but the head grade is too low to result in the production of a saleable by-product or credit.

Gold and silver credits can be expected and are likely to add material value to the net smelter revenue.

13.1.2 2013 Test Work – Peacocke and Simpson

The following is taken from Tetra Tech (2013).

The 2013 test work by Peacocke and Simpson mainly focused on the investigation of gravity-recoverable gold (GRG) for the samples generated from the gold oxides zone (labelled as Leached Zone), copper-gold oxides zone (labelled as Oxide Zone), secondary sulphides zone (labelled as Enrichment Zone) and hypogene sulphides (labelled Primary Zone).

Peacocke & Simpson indicated that at a primary grind size of 80% passing 75 µm, the GRG values for the samples are:

- 29.5% for the Leached Zone sample
- 14.9% for the Oxide Zone sample
- 17.4% for the Enrichment Zone sample
- 13.7% for the Primary Zone sample.

The grades of the centrifugal gravity concentrates produced were low, ranging from 4.8 g/t gold to 32.2 g/t gold.

The gold extractions by cyanidation on the tailings of the GRG tests from the Leached Zone and Oxide Zone samples were similar to the results produced by Certimin. The copper extraction by sulphuric acid leaching on the Oxide Zone sample (GRG tailings) was 83.3%. However, preliminary open batch flotation tests on the Enrichment Zone and Primary Zone samples (GRG tailings) were reported to produce low concentrate grades and metal recoveries.

13.1.3 2014 Test Work – Certimin Flotation

Hypogene Cu-Au and Cu-Mo Sample Sulphide Batch Flotation (Report FEB4007 R14)

Laboratory batch flotation amenability and comminution test work was initially conducted on Cu-Au and Cu-Mo sulphide mill feed type master composites in order to establish appropriate metallurgical processing parameters for flotation. Details of this test work are provided in Certimin report FEB4007 R14.

A total of 55 drill hole core intervals (398 kg) of Cu–Au sulphide mineralization were collected from six drill holes. A total of 22 samples were produced by combining two or three intervals for each hole by depth. A master composite representing Cu–Au sulphide mineralization was prepared and characterized by assay, assay size, Cu speciation and chemical analysis.

A total of 52 drill hole core intervals (312 kg) of Cu–Mo sulphide mineralization were collected from three drill holes. A total of 20 samples were produced by combining two or three intervals for each hole by depth. A master composite representing Cu–Mo sulphide mineralization was prepared and characterized by assay, assay size and chemical analysis.

Table 13-5 shows the chemical and sequential analysis of the samples.

Batch rougher and cleaner (three-stage) flotation tests were conducted on the Cu–Au and Cu–Mo master composites. This work had the objective of determining appropriate primary and regrind grind sizes, flotation pH, flotation time (kinetics) and reagent requirements for a subsequent locked cycle flotation recovery performance confirmation program in Stage 2. BWi was also determined.

The main results and conclusions of this program were:

- Primary grind and regrind size parameters of P80 106 and 26 µm were selected for subsequent locked cycle testing based on maximizing recovery and concentrate grade
- The BWi for the Cu–Au and Cu–Mo composites were reported to be 14.55 and 16.55 kwh/TM respectively, which are considered to be moderately competent to hard for grinding. The Cu–Au result is comparable to the Cu hypogene result reported in 2012 (Wright and Colquhoun, 2012). The Cu–Mo result is 15% higher than that noted in 2012, suggesting some inherent variability exists in hardness in the deposit for copper metallurgy

Table 13-5: Cu–Au and Cu–Mo Composite Sulphide Samples Assay Analysis

Sample	Chemical Analysis					Sequential Analysis			
	Cu, %	Au, g/t	Ag, g/t	Fe, %	Mo, ppm	Cu T, %	Cu CN-, %	Cu Res, %	Cu Sol H+ %
Primary Sulphide Cu–Au	0.45	0.34	3.80	5.06	11	0.44	0.04	0.39	0.01
Primary Sulphide Cu–Mo	0.23	0.05	1.80	5.00	125				<0.005

- Flotation rougher and final cleaner pH of 10.5 and 11.5 were selected with three cleaner stages. Cu recovery in each sample appeared to be insensitive to rougher pH in the range 9 to 10.5
- Alternative flotation reagent schemes were investigated. Lime consumption was about 1.0 kg/t. Flotation reagent consumptions of about Aeroflot 208 +Z11 collector of 20 g/t and MIBC frother (20 kg/t) were established. A minor consumption of cyanide of about 20 g/t was added to the cleaner circuit as an iron depressant
- The Cu–Au and Cu–Mo composites achieved relatively high Cu (Au) rougher recoveries of about 91.3% (64%) and 90.6% (63%) respectively. Mo rougher recovery from the Cu–Mo composite was high at about 87%. In general, the Cu rougher recovery is indicated to be relatively insensitive to copper feed grade as shown by the similar recoveries of the 0.45 Cu% Cu–Au sample relative to the lower grade 0.22 Cu% Cu–Mo sample close to the planned mining cut-off grade in Section 16
- It was noted that Cu recovery in both samples is relatively insensitive to rougher pH in the range 9–10.5, whereas gold recovery declined as expected by about 10%. There may be opportunity in future work to optimize Au recovery by reducing the rougher pH that should be investigated
- The Cu–Au and Cu–Mo concentrate Cu (Mo) cleaner grades reported were 28.7% and 23.5% (1.2%) respectively. There appears to be some loss in copper concentrate grade in the Cu–Mo sample due both to Mo dilution in the concentrate and possibly low feed grade to upgrade. It may be possible to optimize this with alternative reagent schemes but this option was not investigated

No deleterious elements that could have a significant effect on potential economic extraction were detected in the cleaner concentrates.

Cu–Ag and Cu–Au Oxide Sample Batch Flotation (Report ABR4003 R14)

Scoping batch flotation test work was conducted on Cu–Ag and Cu–Au oxide mill feed composites. Drill hole core interval samples were composited to produce 30 kg and 26 kg composite samples of Cu–Ag and Cu–Au oxide mineralization respectively, and characterized by assay, assay size, Cu speciation and chemical analysis.

This test work had the objective of assessing the amenability of these oxide mill feed types to gravity (Cu-Au mineralization type only) and flotation recovery. Various primary grind sizes and flotation reagent schemes were investigated to assess the optimum flotation conditions.

Batch rougher gravity and flotation tests were conducted on each mineralization type. The flotation tests were staged to recover Cu sulphides initially using a sulphide collector flotation reagent scheme. The samples were then chemically sulphidized to promote oxide flotation and refloated to assess Cu Oxide recovery.

The main results and conclusions of this program were:

- Cu–Au gravity Au recovery was about 28%. The inclusion of gravity did not appear to materially improve Au recovery over flotation alone
- Cu–Au total copper and gold recoveries with gravity and rougher flotation with sulphidation were low at about 45% and 44% respectively. With no sulphidation stage, copper and gold flotation recoveries were lower at about 29–34% and 39–40% respectively
- Cu–Ag total copper and silver flotation recoveries with sulphidation were very low at about 38% and 48% respectively. With no sulphidation stage, copper and silver flotation recoveries were lower at about 25–36% and 25–42% respectively\
- An extended 20 minute flotation time was required and resulted in high reagent consumptions
- Finer primary grind sizes did not materially improve recovery
- After cleaning total recovery could be expected to be lower than that the results indicated by the batch rougher tests conducted
- Cu rougher concentrate grades were low at about 1.0 % Cu
- The oxide samples tested alone did not demonstrate sufficiently acceptable amenability to economic flotation recovery and this testing was not taken further.

Cu–Au and Cu–Ag Oxide and Hypogene Sulphide Blend Batch Flotation (Report MAY4000 R14)

Given the poor amenability results indicated by the flotation of oxide mill feed alone in ABR4003 R14 testing the potential to blend oxide with sulphide mill feed in various proportions for recovery was evaluated in this program. Batch gravity and flotation test work was conducted on composite samples consisting of blends of sulphide and oxide.

The work focused on sulphide (both Cu–Au and Cu–Mo) blends with higher Cu–Au–Ag grade oxide mineralized material with a reasonable prospect of processing because of the economic potential for additional precious metal recovery and credits, while the copper recovery from the oxide could be expected to be low:

- 27 drill hole core sample intervals (398 kg) of Cu–Au oxide mineralization (Oxide 1) were collected from five drill holes
- 25 drill hole sample intervals (396 kg) of Cu–Ag oxide mineralization (Oxide 2) were collected from six drill holes.

The Cu–Au and Cu–Mo sulphide composite samples from the initial flotation testing were utilized to prepare four blended sulphide and oxide composites samples. Two blended composite samples of each Cu–(Au) (Composite 1&2) and Cu–(Mo–Ag) (Composite 3&4) mill feed types were prepared comprising of 7.5 wt% and 15 wt% oxide respectively, and were characterized by assay, assay size, Cu speciation and chemical analysis as shown in Table 13-6.

Batch rougher gravity and flotation test work was conducted on the four composite blend samples. This work had the objective of assessing the impact of oxide blending on recovery and determining appropriate grind size, flotation time (kinetics) and reagent scheme conditions.

The best test results achieved were:

- Composites 1 (& 2) Cu–Au rougher flotation Cu and Au recoveries were 88.9% (84.5%) and 77% (74%) respectively. Composite 2 was tested using gravity ahead of flotation and total gold recovery increased to 81%. By comparison the unblended Cu–Au source sample (no Oxide added) from FEB4007 R14 achieved Cu and Au rougher flotation recoveries of 91.3% and 64% respectively.
- Composites 3 (& 4) Cu–Mo rougher flotation Cu and Mo recoveries were 79.1% (67.5%) and 87% (85%) respectively. By comparison the unblended Cu–Mo source sample (no Oxide added) from FEB4007 R14 achieved Cu and Au rougher flotation recoveries of 90.6% and 87% respectively.

Table 13-6: Oxide & Hypogene Sulphide Blend Batch Flotation Sample Analysis

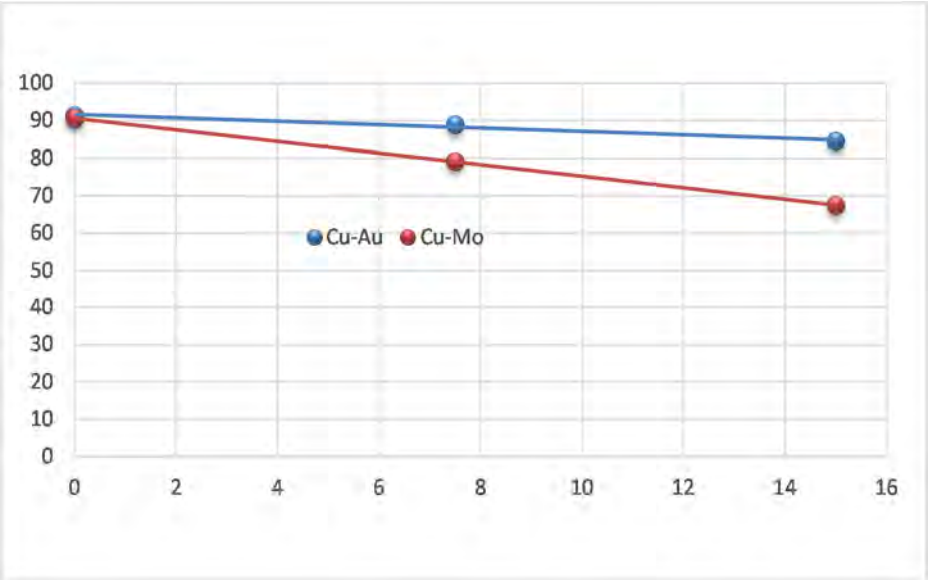
Sample	Chemical Analysis					Sequential Analysis			
	Cu	Au	Ag	Fe	Mo	Cu T	Cu CN-	Cu Res	Cu Sol H+
	%	g/t	g/t	%	ppm	%	%	%	%
Oxide Cu–Au	0.48	0.32	2.70	6.72	15	0.46	0.03	0.17	0.26
Oxide Cu–Ag	0.67	0.07	1.50	6.02	8	0.63	0.02	0.15	0.46
Blend I	0.46	0.34	3.20	5.62	14	0.44	0.05	0.35	0.04
Blend 2	0.46	0.32	3.20	5.71	15	0.43	0.05	0.32	0.06
Blend 3	0.27	0.06	1.50	5.19	97	0.26	0.02	0.20	0.04
Blend 4	0.29	0.05	1.60	5.22	91	0.27	0.01	0.19	0.07

Figure 13-1 shows the effect of oxide blending on rougher flotation recovery.

The conclusions of this program considering the best results achieved were:

- Cu, Au and Mo rougher recoveries were not very grind sensitive in the P80 80 to 125 µm size range investigated. Chemical sulphidation improved Cu recovery by about 2 to 4%. Sulphidizing results in higher operating costs and is probably only economic for higher-grade Cu oxide mill feed and Cu–Au oxide mill feed blends which may carry additional gold credits
- No material reduction in Au or Mo recovery was noted with blending
- Copper recovery reduced directly proportion to the amount of oxide material added. This indicates that there are no synergistic or observable material detrimental effects to the sulphide metallurgy with oxide blending in the range 7.5% to 15% oxide. The lower total Cu recovery is simply a result of the dilution of recoverable Cu sulphide with non-recoverable Cu oxide
- The impact on Cu recovery with oxide blending is more significant on the Cu–Mo sulphide mill feed type because of its lower copper sulphide grade and the effect of proportionally fewer Cu metal sulphide units relative to the fixed wt% proportion of Cu metal oxide units added.

Figure 13-1: Effect on Cu Rougher Flotation Recovery vs Oxide Blending (wt%)



Note: Figure from Certimin, 2014

Cu-Au Hypogene Sulphide and Oxide Blend LC Confirmation Flotation (Report MAY4015 R14)

Batch gravity and flotation test work was conducted on Cu–Au sulphide and a 7.5% oxide blend composite samples. Lock cycle flotation testing was also conducted. The main objective of this work was to confirm the sulphide flotation metallurgy and evaluate the metallurgical impact of the proposed mine plan strategy of blending higher-grade Cu–Au oxides with sulphides in limited proportion in order to recover primarily the oxide associated gold. Scoping cyanidation leach tests were also done on gravity concentrates and cleaner scavenger tailing products to assess their amenability for gold and silver recovery. Details of this test work are provided in the Certimin report MAY4015 R14.

The program utilized:

- 16 samples (405 kg) of Cu–Au sulphide mineralization were prepared from 55 samples of drill core intervals from six drill holes
- A sample of Cu–Au oxide mill feed material from the previous program MAY4000 R14 was utilized to prepare and test a 92.5% blend composite of Cu–Au primary sulphide with 7.5% Cu–Au oxide.

The composite Cu–Au sulphide, oxide and blend composite samples were characterized by assay, assay size, Cu speciation and chemical analysis. Table 13-7 summarizes the

chemical analysis of these. The Cu ASR (CuT/CuSolH+) for each of the samples was 50, 2 and 16 respectively.

Batch gravity and rougher-cleaner flotation tests were initially conducted on composites of the Cu–Au sulphide and blend using the grind and flotation conditions established in FEB4007 R14.

The main results and conclusions of the batch tests were:

- Cu–Au gravity Au recovery was about 21% for both the sulphide and blend samples
- Cu–Au batch rougher-cleaner tests of the sulphide (and blend) resulted in a 26% Cu concentrate for both samples and overall recoveries of Cu 82.0% (72.8%) and Au 39% (34%). Overall batch recovery results, unlike locked cycle testing, are not considered representative of potential flotation performance due to loss of intermediate stream products and are only provided for relative comparison here. The batch rougher Cu and Au recoveries were 92.5% (86.1%) and 68% respectively

Concentrate grade is not detrimentally affected by blending. The blend Cu recovery as expected is lower in proportion to the dilution of recoverable Cu sulphide with non-recoverable Cu oxide.

Locked cycle (LC) flotation tests were conducted on the Cu-Au sulphide and blend composites and the results are shown in Table 13-8 and Table 13-9. The concentrate products were also characterized by chemical analysis. The locked cycle test flotation conditions were based on those established by previous FEB4007 R14 testing and used in the process design criteria for this study considering a primary grind and regrind of P80 of 106 µm and 26 µm respectively with three cleaning stages. The blend LC test reagent scheme included sulphidation.

The main conclusions of the LC tests were:

- The Cu recovery of the blend 84% was as expected lower than the recovery in sulphide of 88%. This result is consistent with the proportion of high-grade Cu–Au oxide included in the blend and indicates that sulphide recovery is not detrimentally affected by blending.

Table 13-7: Cu-Au Sulphide and Oxide Blend LC Flotation Sample Chemical Analysis

Sample Code	Au	Ag	Cu	CuSolH+	CuCn-	CuRes	Fe
	g/t	g/t	%	%	%	%	%
Sulphide Cu–Au	0.37	3.40	0.49	0.01	0.03	0.44	5.31
Oxide Cu–Au	0.34	1.40	0.46	0.28	0.03	0.14	6.54
Blend Cu–Au	0.35	3.30	0.48	0.03	0.03	0.42	5.37

Table 13-8: Cu-Au Sulphide LC Test Result

Products	Weight	Grades				Recoveries			
	(%)	Au (g/t)	Ag (g/t)	Cu (%)	Fe (%)	Au (%)	Ag (%)	Cu (%)	Fe (%)
Copper Conc.	1.62	8.24	108.52	24.71	23.95	44.39	57.68	87.54	7.33
Cleaner Scavenger Tailings	7.80	0.89	8.65	0.25	18.27	23.16	22.17	4.27	26.96
Total Tailings	90.58	0.11	0.68	0.04	3.83	32.45	20.15	8.19	65.71
Head Calculated	**100.00**	**0.30**	**3.04**	**0.46**	**5.29**	**100.00**	**100.00**	**100.00**	**100.00**

Table 13-9: Cu-Au Sulphide and Oxide (7.5 wt%) Blend LC Test Result

Products	Weight	Grades				Recoveries			
	(%)	Au (g/t)	Ag (g/t)	Cu (%)	Fe (%)	Au (%)	Ag (%)	Cu (%)	Fe (%)
Copper Conc.	2.00	6.5	96.46	20.31	30.08	48.37	66.56	83.82	10.57
Cleaner Scavenger Tailings	6.17	0.66	7.27	0.21	18.85	15.08	15.47	2.67	20.42
Total Tailings	91.83	0.11	0.57	0.07	4.28	36.55	17.97	13.51	69.01
Head Calculated	**100.00**	**0.27**	**2.90**	**0.48**	**5.69**	**100.00**	**100.00**	**100.00**	**100.00**

- The copper concentrate grade of the sulphide was 25 Cu% which is reasonably consistent with the batch rougher-cleaner test results and previous testing. The copper concentrate grade of the blend CL test concentrate was lower than that expected from batch-cleaner testing and is probably not representative of the potential. It is reasonable to assume additional reagent optimization testing could likely improve this to a 25–26% concentrate similar to that indicated by batch cleaner testing

No deleterious elements that could have a significant effect on potential economic extraction were detected in the concentrates.

Scoping amenability cyanidation leaching tests were conducted over 24 hours on the batch gravity and LC cleaner scavenger tails for the Cu–Au sulphide and (oxide) blend samples at cyanide concentrations of 5 and 3 g/L respectively. Gold and silver extractions of about 95% (95%) and 73% (76%) were obtained for the gravity concentrate indicating its high amenability to cyanide leaching for recovery. Gold and silver extractions of about 48% (59%) and 59% (64%) were obtained on leaching the cleaner scavenger tailings indicating its moderately low amenability to cyanide leaching for recovery. The recovery of gold and silver from cleaner scavenger tailings by cyanidation is unlikely to be economic because of the relatively low grades, recoveries, and additional reagent consumption.

13.1.4 2014 Test Work – Aminpro Oxide Leaching

Laboratory leaching tests were conducted by Aminpro to assess the amenability of Cu–Au oxide (higher in gold) mineralization to acid leaching followed by cyanidation for gold recovery. The second aim of the testwork was to assess the amenability of Cu oxide mineralization to heap leaching, and define the relevant hydrometallurgical parameters for recovery. This test work was completed in support preliminary economic assessment trade-off studies of oxide leaching for this study.

Testing was conducted on splits of samples from 390 kg of Cu–Au (M1) and 385 kg of Cu (M2) oxide composites selected and shipped to Aminpro by Panoro. The composites were characterized by hardness, assay, assay size, mineralogy Cu speciation and chemical analysis.

The total copper and gold grades were:

- M1: 0.47% Cu, 0.4 g/t Au and 0.27% CuAS (acid-soluble copper)
- M2: 0.68% Cu, 0.1 g/t Au and 0.46% CuAS.

Bottle roll testing was conducted on both samples to characterize and assess their amenability to agitated acid leaching for copper recovery at various acid concentrations. After initial acid bottle leaching and rinsing sample M1 was subjected to cyanidation for Au recovery. Inverse leaching was also investigated for sample M1.

Column testing was conducted on each composite M1 (M2) to assess the amenability of the various size fractions (12, 18, 25 and 50 mm) to heap leaching. M1 was tested over 77 days and M2 over 65 days at various sizes and acid concentrations.

The main results and conclusions of this leach test work were:

- Sample M1 contained more primary sulphide copper chalcopyrite mineralization than expected for an oxide sample which is evident by its relatively low acid and cyanide copper solubility. Consequently recovery by acid leaching alone could be expected to be lower
- Sample M2 contained more secondary copper mineralization than expected for an oxide sample which was evident by its higher cyanide solubility. Consequently recovery by acid leaching alone could be expected to be lower but greater than M1
- Both samples contained significant limonite and magnetite which could be expected to result in relatively high acid consumptions

Overall copper extractions achieved were considered to be low within the industry typical 65–75 day oxide leach cycle tested, were unlikely to be economic, and leach testing was discontinued. This was confirmed in a subsequent trade-off study.

13.1.5 2014 Test Work – Dewatering

Outotec were contracted by Panoro to conduct preliminary laboratory high rate thickener settling and filtration tests on sulphide and blend final concentrate samples produced by flotation testing. This established the relevant thickening settling rate and filtration parameters and reagent scheme and consumption for the basis of this study.

Outotec also conducted preliminary laboratory thickening settling, slurry rheology and filtration tests on flotation tailings samples. The results were used as the basis for trade-off studies considering either dewatering and stacking tailings or pumping thickened tailings. The results also form the basis of the design criteria used for the thickened tailings pumping disposal option selected for the basis of this study. The results were utilized by Patterson and Cooke in the development of the study tailings pumping and pipeline design.

No significant problems were noted dewatering either concentrate product or tailings.

13.2 Recovery Estimates

Mineralized material will be processed in a conventional copper porphyry flotation concentrator plant to produce a copper, gold and silver concentrate, with copper as the main payable metal in the concentrate and gold and silver providing important economic by-product credits.

The process considers a primary crusher, a semi-autogenous grind (SAG) mill and a ball mill prior to the flotation plant consisting of rougher flotation, regrinding and three cleaner stages. Flotation tailings will be thickened and pumped to a tailings storage

facility for disposal and final product concentrate is dewatered for storage and transport to port by trucks.

A summary of the flotation metallurgical recoveries estimated for given mineralized zones processed relevant to the mine plan schedule in this study are shown in Table 13-10.

Hypogene recoveries are based in the results of the 2012 and the most recent 2014 in-fill testing, which are in reasonable agreement.

The copper enriched supergene zone accounts for about 4% of the total combined sulphide plant feed tonnage in this PEA. No testing was conducted on this zone in the latest 2014 Certimin program. Previous preliminary batch testing indicated similar recoveries can be expected to hypogene sulphide with similar or higher final concentrate grade up to 30% Cu. In this PEA the same metallurgical recovery parameters have been assigned to supergene material as were assigned to hypogene mill feed.

Hypogene and supergene mill feed materials are shown combined as sulphide mineralization in the mine production and processing schedule presented in the PEA financials. This is because of the relatively small quantity of supergene mineralization relative to hypogene and the assignment of similar metallurgical recovery parameters to both. Therefore, the mining and process schedule presented in the project financials indicates sulphide, mixed and High Au Oxide mineralization scheduled as plant feed.

Testwork to date on sulphide composites with grades ranging between 0.22–0.54% Cu indicates that copper recovery is relatively insensitive to the copper grade in these grade ranges. This grade range approximately corresponds to the average annual grades used in the LOM mine plan prior to treating low grade stockpiled sulphide at the end of the mine life, and a constant metallurgical recovery is estimated for this material. However, there is a risk that the grade ranges may not be reached during mining, and low-grade material that is near the copper equivalent cut-off grade used for conceptual mine planning purposes may not return some or all of the projected economics assumed in the PEA. Variability testing is planned in the next phase of work to establish or confirm the constant grade–recovery relationship assumed in the PEA.

In this updated PEA, a low-grade stockpiling strategy has been implemented in the mine plan by MMTS to improve early copper grades. About 118 Mt of low-grade sulphide ore (0.13%Cu) is stockpiled over years 1–12 and is reclaimed to the plant during years 13–17 after primary mining stops.

Table 13-10: Projected Recoveries by Mineralization Type (Zone)

Mineralization Type	Recovery			Concentrate Grade		
	Cu %	Au %	Ag %	Cu %	Au g/t	Ag g/t
Hypogene Sulphide	87.5	62.0	60.4			
Supergene Sulphide	87.5	62.0	60.4			
Mixed Oxide Cu-Au	60.0	55.0	—			
Oxide High Au	—	65.0	—			
Overall Life-of-Mine Average *	**80.4**	**61.3**	**52.2**	**27.0**	**11.5**	**130**

Note: Average life-of-mine figures from PEA overall annual mine plan schedule

Testing has been conducted to date on sulphide metallurgical samples ranging in grade from Cu 0.21-0.4% which corresponded to the range of annual average grades in the previous PEA study mine plan, including lower-grade material that is now stockpiled. Samples tested in this grade range did not exhibit strong grade-recovery sensitivity. Therefore for the purposes of the original PEA level assessment a constant recovery was assumed by Amec Foster Wheeler over the mine life for each ore type, until more detailed metallurgy can establish grade-recovery relationships.

The grade of the now segregated low-grade sulphide stockpile with an average grade of 0.13% lies below the range of grades tested to date, and there is a risk the constant average LOM sulphide recoveries of Cu 87.5%, Au 62.0% and Ag 60.4% assumed in the cash flow model in this study will not be achieved on this stockpiled low grade sulphide material processed at the end of the mine life. There is also a risk recoveries from stockpiled material will be lower than expected, and reagent consumption and costs will be higher because of potential in-situ sulphide mineral oxidation effects during storage. Overall however, because this low-grade material is processed at the end of the mine life, if lower recoveries than assumed in this study are experienced this is not expected to impact materially on the overall project net present value (NPV) indicated by this PEA study. Further work is recommended in future testwork to support a recovery basis for processing low-grade stockpiled material at the end of the mine life.

The relatively fine sulphide primary grind of P80 106 µm selected should be reviewed as test work indicates that recovery is relatively insensitive to primary grind size in the range up to 130 to 150 µm. Additional baseline/variability work and economic grind recovery trade-off is recommended in future work to confirm an opportunity to specify a coarser primary grind than that considered in this study thereby reducing grinding related costs.

Gravity-flotation did not appear to offer a material advantage over flotation only in any testing phase on the composites tested and has not been included in this studies

flowsheet. The inclusion of gravity should however be re-evaluated in future variability testing and associated with oxide blend mill feed high in gold produced early in the mine plan as an economic trade-off for producing doré on site versus smelter concentrate credit.

Oxide leaching was investigated in a trade-off study and eliminated as an option for the project in this preliminary assessment study. However, as sample characterization and testing progressed it was noted that the oxide samples tested were biased towards a mixed mineralization containing relatively higher proportions of chalcopyrite than that indicated by the global characterization of the oxide zone by copper sequential analysis in the resource model. One of the main contributing factors to this was a change in oxide domaining based on geological logging implemented during the last resource estimate update (Morrison et al., 2013) on which oxide metallurgical sampling was based versus previous estimates that were based on copper speciation (Wright and Colquhoun, 2012).

For the purposes of the initial PEA, Tetra Tech restated the oxide resource model tabulations to include mixed Cu–Au and oxide Cu–Au zones that were defined by copper speciation parameters provided by Amec Foster Wheeler. As flotation testing of the global samples of oxide tested in the most recent program (2014) indicated relatively low copper recoveries (attributable mainly to the contained sulphide mineralization with little or no copper oxide recovery) this was done with the objective of estimating suitable mixed sulphide and oxide Cu–low Au mineralization within the oxide resource (Mixed Cu–Au) that would have a reasonable prospect of economic copper and gold recovery by blending with hypogene sulphide in the mine plan. Some oxide (oxide–Au) mineralization, either with no recoverable copper or grade, with a high gold grade (where the net smelter return was higher than the cut-off grade) would also be blended for gold recovery by flotation.

As a result of the division in oxide Cu zoning into Mixed Cu–Au oxide and oxide implemented during the initial PEA, the global oxide copper recoveries indicated by the 2014 Certimin testwork are not directly applicable to estimating copper recovery from the mixed mineralization. The Certimin results understate sulphide copper recovery potential from the Mixed oxide zone, as it contains a higher proportion of recoverable sulphides over the global oxide sample tested.

The Certimin oxide test work indicated about 90% of the sulphide copper in oxides is recoverable, and using sequential analysis characterization of the Mixed oxide zone Amec Foster Wheeler has inferred a copper recovery for this zone (Table 13-10) for the basis of the mine plan. No direct testwork has been conducted on samples of the Mixed oxide zone to date.

Gold recoveries were also estimated from the Certimin test work, considering the higher grade of the Au oxide zone. Testing is required on representative samples of the Mixed

zone and Oxide Au in future work, and as blends with hypogene sulphide to confirm the recoveries inferred by Amec Foster Wheeler.

Most of the oxide testing in this study evaluated sulphidation prior to flotation. This indicated that little or no oxide recovery was achieved over the sulphides and is a result quite typical of the dominant chrysoscolla mineralization associated with the oxide mineralization. The use of high levels of sulphidizing reagent required to improve chrysoscolla recovery can depress primary sulphides. As a result, only sulphide flotation was considered in the reagent scheme, and operating costs to recover copper sulphide and gold associated with Mixed oxides and gold associated with Oxide Au were incorporated in the PEA.

The use of alternative hydroxamate collectors to enhance copper oxide recovery in blends should be investigated in future programs. Hydroxamate flotation collectors are generally more compatible with copper sulphide and oxide blends containing chrysoscolla, and this represents an opportunity to enhance oxide copper recovery.

Although a specific sample of a sulphide copper zone high in Mo (120 ppm) was collected and tested (Certimin 2014 - FEB4007 R14) the global Mo grade in hypogene sulphide is too low globally (about 15 ppm) to consider the production of a separate Mo by-product for PEA purposes, and this was not considered in the PEA study flowsheet or costs. Testing of the high Mo sample did indicate good Mo recovery potential, and concentrate grades with the sample tested indicating potential to produce a separate Mo concentrate with this material that might be revaluated in future.

13.3 Metallurgical Variability

Test work to date has focused on the global metallurgical characterization of composites of the main mineralization types, and no variability testing has been conducted at this stage.

13.4 Deleterious Elements

No deleterious elements that could have a significant effect on potential economic extraction were noted.

Molybdenum is indicated in the concentrate but the head grade is too low globally to result in the production of a saleable by-product or credit.

Sulphur levels of annual oxide/sulphide concentrate blends should be assessed in future work to confirm they meet minimum sulphur contract specifications.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Mineral Resource Estimation (Tetra Tech, 2014)

The text in this sub-section is reproduced from Morrison et al., 2014.

For the purposes of this mineral resource estimate, Datamine® Studio 3 (version 3.21.7164.0) resource software was employed to analyze data, create associated wireframes of mineralization, and subsequent block modelling and grade interpolation.

14.1.1 Introduction

The following sections outline the NI 43-101 compliant resource estimate for the Property. The effective date of this resource estimate is August 12, 2013. The following sections describe and discuss the Cotabambas deposit resource estimate. The resource estimate includes, but is not restricted to:

- Review of geological and assay data provided by Panoro
- Geological interpretation and domaining of the mineralization
- Application of the interpretation in the form of designed domain wireframes
- Assessment of the data with respect to the different geological domains
- Construction and configuration of a suitable block model
- Interpolations of attributes (grade, density, etc.) Into cells of the block model
- Verification and validation of the interpolations
- Application if resource classification (Inferred and Indicated)
- Reporting on the respective mineral inventory
- Recommendations for further work.

14.1.2 Geological Interpretation

The geology of the Cotabambas deposit has been described by Wright & Colquhoun (2012) as part of the AMEC (Perú) S.A. NI 43-101 technical report on the Property. The Property has been recognized as porphyry copper-gold mineralization, coincident with the intrusion of Eocene to Oligocene Period quartz monzonites. Wright & Colquhoun (2012) described “emplacement of the quartz monzonite porphyry and later latite dykes are controlled by a system of strong sub-vertical fault and shear zones that have an azimuth of approximately 030°. A second set of structures, perpendicular to the 030° system and parallel to the regional thrust fault systems with azimuth 120° runs between the Ccalla area and the Guaclle area to the west.”

Galley et al. (2007) noted that the Cotabambas deposit represents an example of a large tonnage volcanogenic massive sulphide (VMS) deposit at 25.06 Mt at 1.79% copper and 10.63 g/t silver.

Geological Domains and Wireframes

Four broad domains are recognized from drill core logging at the Cotabambas deposit. These domains have been flagged in the drill hole data with the attribute "zone1" and are listed as follows:

- Fresh rock where primary sulphides (chalcopyrite, pyrite) are noted, also referred to as the hypogene zone (zone1 = 1).
- A zone of secondary enrichment, noted for the development of secondary sulphides (chalcocite and lesser covellite), also known as the supergene zone (zone1 = 4).
- A copper-gold oxide zone overlying the zones of primary and secondary sulphide mineralization (zone1 = 2).
- A leached cap zone characterized by abundant iron-magnesium oxide and oxy-hydroxide minerals (zone1 = 3), which occasionally contains copper-gold oxides.
- Sub-vertical and NE-trending latite dykes (zone2 = 1).

Planar wireframes forming the base and top of these zones were designed and constructed in Datamine® using the drill core logs. These were confirmed by comparing with geological wireframes and sections prepared by Panoro geologists. The topographic surface wireframes were provided by Panoro geologists. These were compiled in Datamine® to create a single digital terrain model (DTM) wireframe which was used in the resource model.

The sub-vertical and northeast trending latite dykes formed a sub-domain of these four geological domains. Wireframes modelling these dykes were also designed and constructed in Datamine® using the drill core logs. These were confirmed by geological wireframes prepared by Panoro geologists for the latite dykes.

The final resource model is comprised of a total of 8 domains as delineated in the attribute "ZONE"; ZONEs 1-3 represent unmineralized rock (fresh rock, oxide rock and leached cap respectively. ZONEs 4-7 represent mineralized rock (hypogene, oxide, leached cap respectively and supergene). Hypogene mineralization in latite dykes (zone2 = 1) were estimated separately from the rest of the hypogene mineralization, while latite dykes in other domains (supergene, oxide and leached cap) were estimated together.

14.1.3 Data

Data was provided by Panoro as both a Microsoft Access® database (.mdb file) and as a series of csv text files to replicate diamond drill hole collar, survey, geology and assay

data. After separate verification and validation, all data were imported into Datamine® software and resurveyed to create an appropriate drill hole file for geological interpretation and grade estimation. The desurveyed drill hole data were flagged with the appropriate domain attribute. A representative northwest–southeast cross-section is depicted in Figure 14-1.

The final mineralization model employed in the Cotabambas resource model was designed, developed, and verified by Tetra Tech.

14.1.4 Exploratory Data Analysis

The following discussion describes the data used in the Panoro resource estimate. It outlines the data statistics for respective domains; methodology used to identify and control the influence of outlier data and compositing data to maintain consistency in the estimation process.

Drill hole Statistics

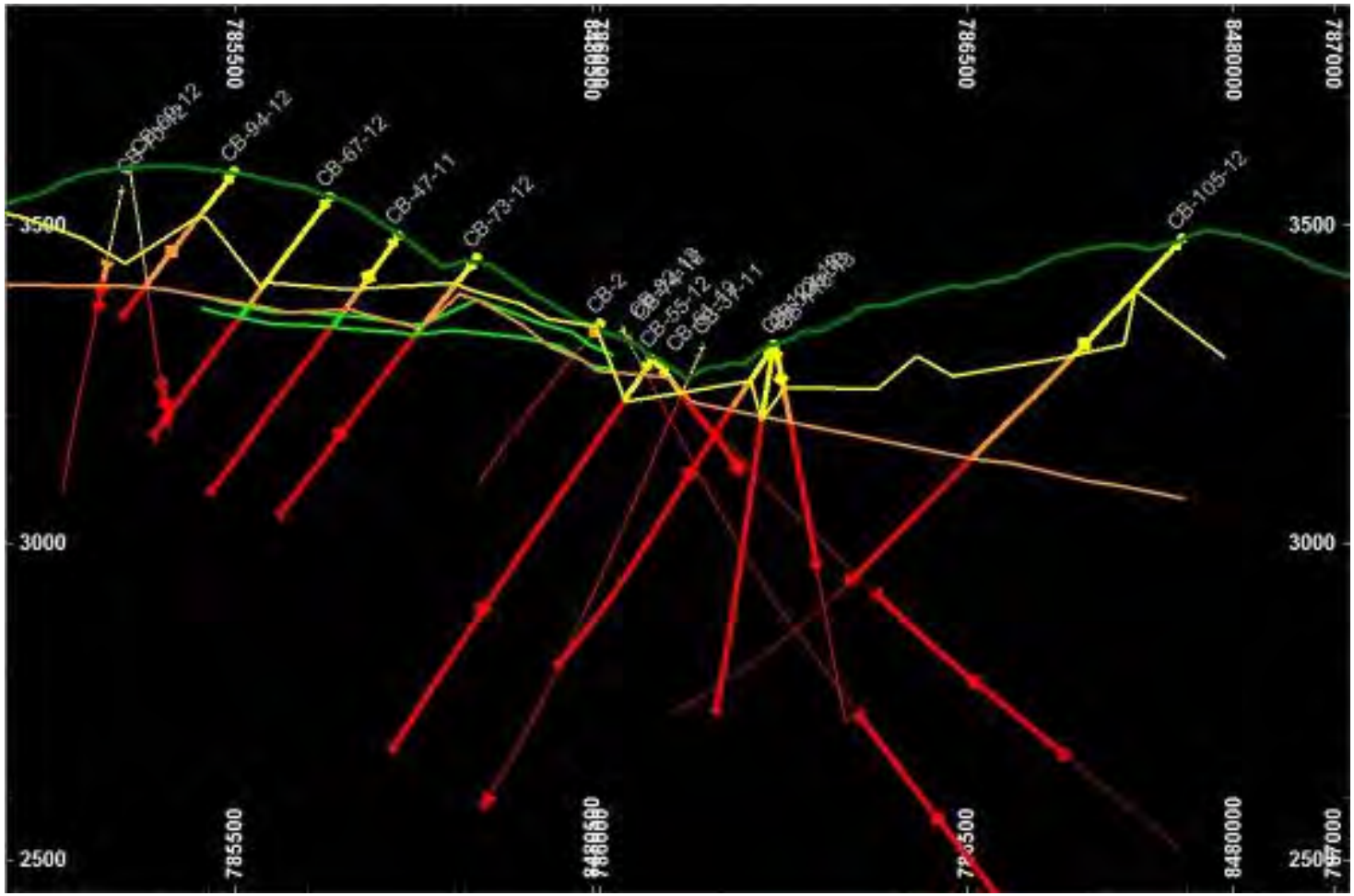
Metals – Cu, Au, Ag, Mo

Table 14-1 displays the drill hole statistics for the Project. It includes drill holes only within the broad zone of mineralization which was manually delineated by Datamine® wireframes. One wireframe was used to identify the copper-gold-silver volume of mineralization, and another smaller separate wireframe identified a volume of molybdenum mineralization. These volumes, with the drill holes, are depicted as a west-east section in Figure 14-2.

Drill hole data were also coded relative to the geological unit in which they were logged and located. These geological units were designated as follows:

- (zone1=1) “hypogene” or sulphide mineralization which occurs at depth in fresh rock
- (zone1=2) “oxide” mineralization which occurs at shallower levels
- (zone1=3) “leached cap” mineralization, which occurs near surface
- (zone1=4) “supergene” or secondary enrichment mineralization which occurs proximal to the oxide-sulphide boundary.

Figure 14-1: Northwest–Southeast Cross-section of Domain Wireframes with Drill Hole Traces Coloured by Domain



Note: Dark green line represents the topographic wireframe surface. Domain colours: yellow – leached cap, orange – oxide, bright green – supergene, red – hypogene. Figure prepared by Tetra Tech, 2014

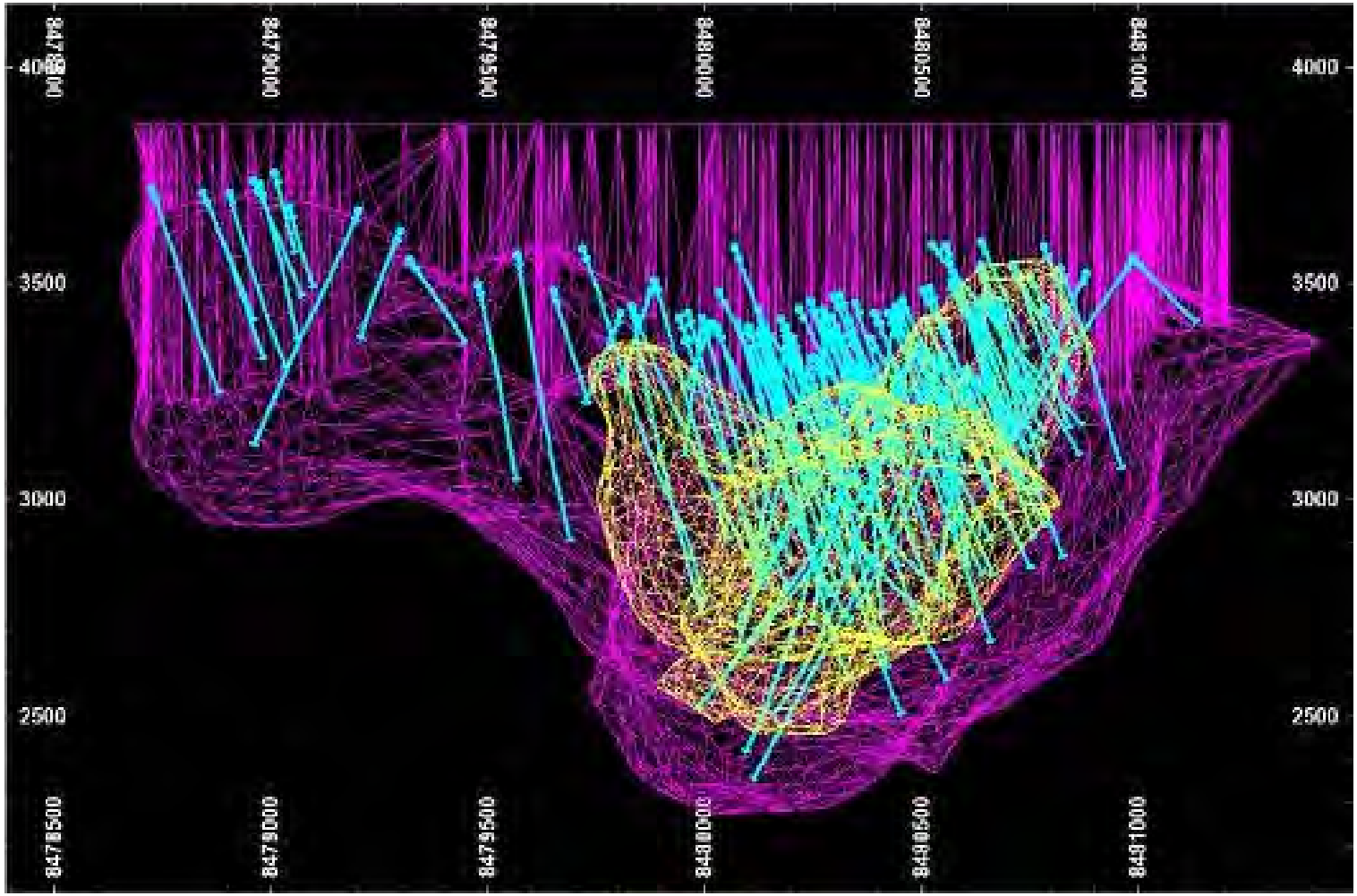
Table 14-1: Cotabambas Deposit Raw Drill Hole Statistics (Cu, Au, Ag, length)

Type	Zone1	Field	Records	Samples	Minimum	Maximum	Mean	Variance	Std Dev	CV
Raw	1	Cu_pct	52,655	41,127	0.0001	19	0.1843	0.1026	0.3203	1.738
		Au_gt	52,655	36,839	0.001	33.8	0.1058	0.0912	0.3019	2.855
		Ag_gt	52,655	36,921	0.19	112	1.9226	6.7957	2.6069	1.356
		LENGTH	52,655	41,448	0.002	3.4	1.2387	0.4879	0.6985	0.564
	2	Cu_pct	52,655	4,047	0.0008	8.95	0.2740	0.2461	0.4961	1.811
		Au_gt	52,655	3,713	0.001	4.49	0.1183	0.0649	0.2547	2.153
		Ag_gt	52,655	3,592	0.5	41	1.6891	3.4428	1.8555	1.099
		LENGTH	52,655	4,119	0.002	10	1.2904	0.5279	0.7265	0.563
	3	Cu_pct	52,655	5,847	0.0004	3.14	0.1217	0.0600	0.2449	2.012
		Au_gt	52,655	5,421	0.001	2.72	0.1024	0.0467	0.2161	2.112
		Ag_gt	52,655	5,215	0.5	33	1.4726	2.1143	1.4541	0.987
		LENGTH	52,655	5,937	0.001	7.6	1.2947	0.4951	0.7036	0.543
	4	Cu_pct	52,655	1,149	0.0006	8.14	0.8270	1.1062	1.0518	1.272
		Au_gt	52,655	1,138	0.003	5.12	0.2903	0.1577	0.3971	1.368
		Ag_gt	52,655	981	0.5	34	3.0238	11.7396	3.4263	1.133
		LENGTH	52,655	1,151	0.005	2	1.2502	0.4456	0.6675	0.534
Capped	1	Cu_pct	52,655	41,127	0.0001	3.5	0.1831	0.0775	0.2784	1.520
		Au_gt	52,655	36,839	0.001	2.6	0.1033	0.0405	0.2013	1.949
		Ag_gt	52,655	36,921	0.19	30	1.9065	4.9643	2.2281	1.169
		LENGTH	52,655	41,448	0.002	3.4	1.2387	0.4879	0.6985	0.564
	2	Cu_pct	52,655	4,047	0.0008	3.5	0.2682	0.1856	0.4308	1.606
		Au_gt	52,655	3,713	0.001	2.6	0.1162	0.0514	0.2268	1.951
		Ag_gt	52,655	3,592	0.5	30	1.6843	3.1291	1.7689	1.050
		LENGTH	52,655	4,119	0.002	10	1.2904	0.5279	0.7265	0.563
	3	Cu_pct	52,655	5,847	0.0004	3.14	0.1217	0.0600	0.2449	2.012
		Au_gt	52,655	5,421	0.001	2.6	0.1023	0.0466	0.2158	2.109
		Ag_gt	52,655	5,215	0.5	30	1.4720	2.0798	1.4422	0.980
		LENGTH	52,655	5,937	0.001	7.6	1.2947	0.4951	0.7036	0.543
	4	Cu_pct	52,655	1,149	0.0006	3.5	0.7837	0.7560	0.8695	1.109
		Au_gt	52,655	1,138	0.003	2.6	0.2881	0.1419	0.3767	1.308
		Ag_gt	52,655	981	0.5	30	3.0197	11.5033	3.3917	1.123
		LENGTH	52,655	1,151	0.005	2	1.2502	0.4456	0.6675	0.534
Capped and Composited (4 m)	1	Cu_pct	16,463	12,728	0.0001	3.0329	0.1760	0.0639	0.2528	1.436
		Au_gt	16,463	11,487	0.0020	2.5782	0.0973	0.0321	0.1792	1.840
		Ag_gt	16,463	11,832	0.1959	30.0000	1.8623	3.6504	1.9106	1.026
		LENGTH	16,463	12,829	2.0000	4.7500	3.9998	0.0008	0.0276	0.007
		Cu_pct	16,463	1,307	0.0016	3.4466	0.2610	0.1546	0.3932	1.507
		Au_gt	16,463	1,194	0.0040	2.4022	0.1112	0.0384	0.1960	1.763

Type	Zone1	Field	Records	Samples	Minimum	Maximum	Mean	Variance	Std Dev	CV
	2	Ag_gt	16,463	1,205	0.5000	16.7402	1.6414	2.2441	1.4980	0.913
		LENGTH	16,463	1,337	2.0000	5.0000	3.9731	0.0218	0.1477	0.037
	3	Cu_pct	16,463	1,908	0.0007	2.4124	0.1252	0.0552	0.2349	1.877
		Au_gt	16,463	1,799	0.0017	1.8350	0.1049	0.0451	0.2123	2.024
		Ag_gt	16,463	1,760	0.5000	17.4286	1.4635	1.6296	1.2766	0.872
		LENGTH	16,463	1,934	2.0000	5.5000	3.9737	0.0223	0.1493	0.038
	4	Cu_pct	16,463	363	0.0006	3.5000	0.7680	0.6888	0.8299	1.081
		Au_gt	16,463	358	0.0045	2.1573	0.2793	0.1207	0.3474	1.244
		Ag_gt	16,463	325	0.5000	24.0000	2.9103	10.4897	3.2388	1.113
		LENGTH	16,463	363	3.0000	4.9000	3.9642	0.0500	0.2236	0.056

Note: Std Dev = Standard Deviation; CV – Coefficient of Variance

Figure 14-2: West–East Cross-Section of Data Wireframes



Note:Wireframes were also used to restrain grade extrapolation. Magenta used for copper, gold and silver; yellow used for molybdenum. Figure prepared by Tetra Tech, 2014.

The drill holes statistics are also sub-divided into raw data, raw data with the "cap" or "top-cut" for each of the metals applied, and capped and composited data (to 4 m intervals). A discussion relating to the capping strategy (i.e. "outlier management") and compositing strategy can be found in the immediately following sections of this Report.

Table 14-2 shows the statistics for molybdenum. Note that the CV is relatively low; generally less than two for most all scenarios. This indicates that defined domains by the four zones are valid for grade interpolation.

Sequential Leaching

Data pertaining to sequential leaching characteristics, including cyanide leaching (CuCN), acid leaching (CuAS), and sulphide flotation (CuR) was also collected in a more limited number of drill hole samples in the immediate vicinity of the supergene mineralization (Zone1 = 4). Table 14-3 summarizes the statistics for this data in each of the respective domains (Zone1 1-4). This is illustrated in Figure 14-3.

Note the order of magnitude difference in the number of standard assays in comparison with the number of leach assays. To maximize the number of leach assays available for interpolation, no capping or compositing was applied to the dataset.

Outlier Management and Capping Strategy

For sample outlier population management, the entire dataset was considered for copper, gold, and silver. Although the paragenesis of the deposit differed between domains, it was considered that the entire dataset provided sufficient samples to adequately interrogate the statistics for outlier capping (or "top-cutting"). Histograms and log normal plots were used to identify outlier sample populations. These populations were subsequently confirmed not to form independent volumetrically discrete high-grade domains. The following discussion provides a synopsis of this management strategy.

Of the 47 samples with greater than 3.5% copper, 28 correspond to Zone 4 (supergene mineralization). However, none are sufficiently spatially related to warrant the generation of a separate high grade copper domain.

Gold

The histogram for gold raw assay data for in the deposit shows several outliers (Figure 14-4). In order to manage these abnormally high grades, a cap was set at 2.6 g/t where the continuity of the distribution begins to break down. This represents application of a cap to 37 samples. Of these, 26 samples belong to Zone 1 (hypogene mineralization). Like copper, these high grade samples are insufficiently clustered to warrant the generation of a separate high grade domain.

Table 14-2: Cotabambas Deposit Raw Drill Hole Statistics (Mo)

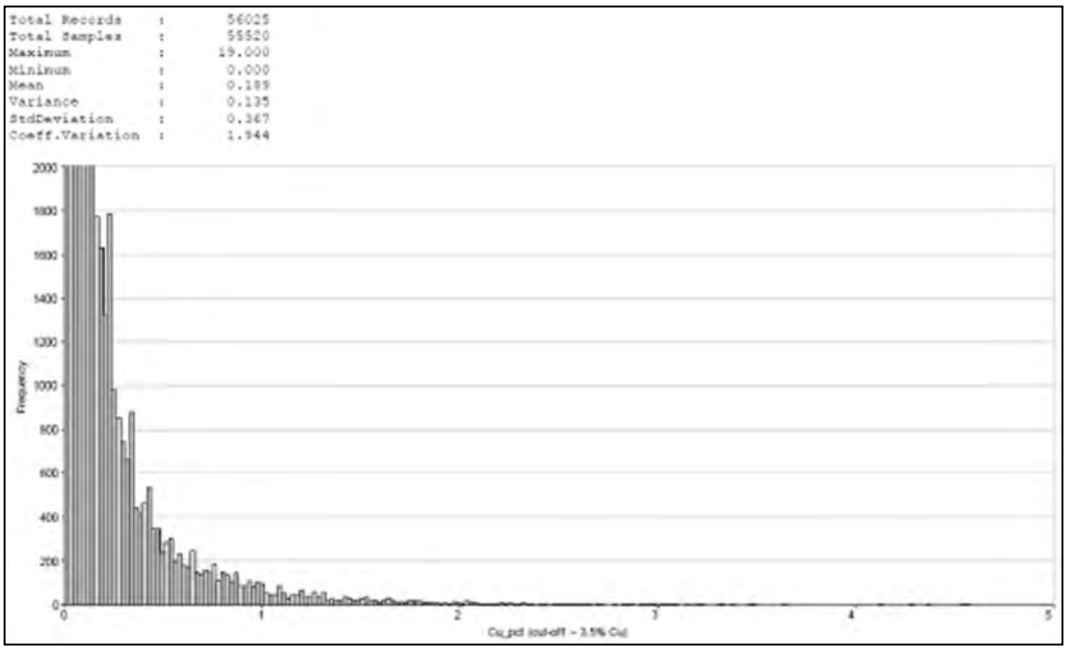
Type	Zone1	Field	Records	Samples	Minimum	Maximum	Mean	Variance	Std Dev	CV
Raw	1	Mo_pct	20,880	17,517	0.0001	0.1900	0.0041	0.0001	0.0086	2.0842
	2		20,880	402	0.0001	0.0200	0.0025	0.0000	0.0024	0.9621
	3		20,880	337	0.0001	0.0100	0.0022	0.0000	0.0018	0.8326
	4		20,880	95	0.0006	0.0100	0.0024	0.0000	0.0022	0.9267
Capped	1	Mo_pct	20,880	17,517	0.0001	0.1900	0.0041	0.0001	0.0086	2.0842
	2		20,880	402	0.0001	0.0200	0.0025	0.0000	0.0024	0.9621
	3		20,880	337	0.0001	0.0100	0.0022	0.0000	0.0018	0.8326
	4		20,880	95	0.0006	0.0100	0.0024	0.0000	0.0022	0.9267
C & C (4 m)	1	Mo_pct	6,623	5,763	0.0001	0.1125	0.0042	0.0000	0.0067	1.5855
	2		6,623	141	0.0001	0.0120	0.0026	0.0000	0.0021	0.8150
	3		6,623	111	0.0001	0.0082	0.0021	0.0000	0.0016	0.7357
	4		6,623	31	0.0007	0.0069	0.0022	0.0000	0.0016	0.7603

Note: "C & C" refers to capped and composited data.

Table 14-3: Cotabambas Deposit Raw Drill Hole Statistics (Cu leaching)

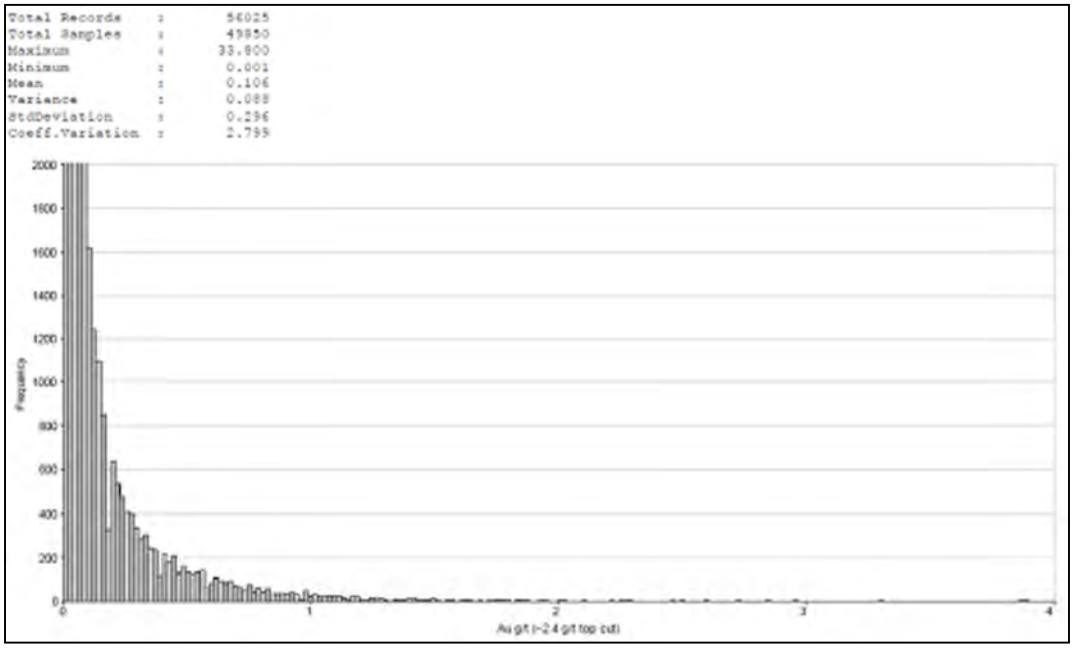
Zone1	Field	Records	Samples	Minimum	Maximum	Mean	Variance	Std Dev	CV
1	CuCN_pct	39,683	697	0.022	0.84	0.1280	0.0099	0.0995	0.7772
	CuAS_pct	39,683	697	0.013	0.911	0.1387	0.0226	0.1504	1.0845
	CuR_pct	39,683	697	0.039	0.943	0.7333	0.0391	0.1978	0.2698
2	CuCN_pct	39,683	1,374	0.003	0.705	0.0833	0.0069	0.0828	0.9940
	CuAS_pct	39,683	1,374	0.03	0.982	0.4763	0.0545	0.2334	0.4901
	CuR_pct	39,683	1,374	0.012	0.909	0.4405	0.0462	0.2150	0.4881
3	CuCN_pct	39,683	1,426	0.004	0.63	0.1126	0.0095	0.0972	0.8632
	CuAS_pct	39,683	1,426	0.053	0.951	0.3609	0.0345	0.1858	0.5148
	CuR_pct	39,683	1,426	0.037	0.895	0.5265	0.0325	0.1802	0.3423
4	CuCN_pct	39,683	337	0.008	0.866	0.2881	0.0613	0.2476	0.8596
	CuAS_pct	39,683	337	0.029	0.933	0.3601	0.0811	0.2848	0.7909
	CuR_pct	39,683	337	0.011	0.9	0.3518	0.0707	0.2659	0.7559

Figure 14-3: Copper Log Histogram of Raw Drill hole Data



Note: Figure prepared by Tetra Tech, 2014.

Figure 14-4: Gold Log Histogram and Raw Drill hole Data



Note: Figure prepared by Tetra Tech, 2014.

Silver

The raw data histogram for silver in the deposit shows several outliers (Figure 14-5).

In order to manage these abnormally high grades, a cap was set at 30 g/t where the continuity of the distribution begins to break down. This 30 g/t cap effects 22 samples, of which 18 are from Zone 1 (hypogene mineralization). None are spatially related to warrant the generation of a separate high grade silver domain. Although Zone 1 also hosts the higher grade gold assays, there is little immediate correlation between the high grade silver assays and the high grade gold assays.

Summary

The raw assay data was examined for outlier populations and a “cap” (or “top-cut”) as applied to copper, gold, and silver; these are 3.5%, 2.6 g/t and 30 g/t respectively. Table 14-4 summarised the statistics for the capped drill hole assay data.

Copper in Zone 4 experienced the greatest change in the average value, reduced by 7%, followed by gold in Zone 3. Most metals and zones experienced less than 2% change in mean grade with the application of a cap. The CV was reduced by up to 31% in gold (Zone 1 and 3), and up to 15% in copper (Zone 4).

Drill hole Compositing

The histogram of raw drill hole sample lengths is depicted in Figure 14-6.

The majority of raw sample lengths used for assaying measure 2.0 m in length, with a significant number of 1.0 m samples.

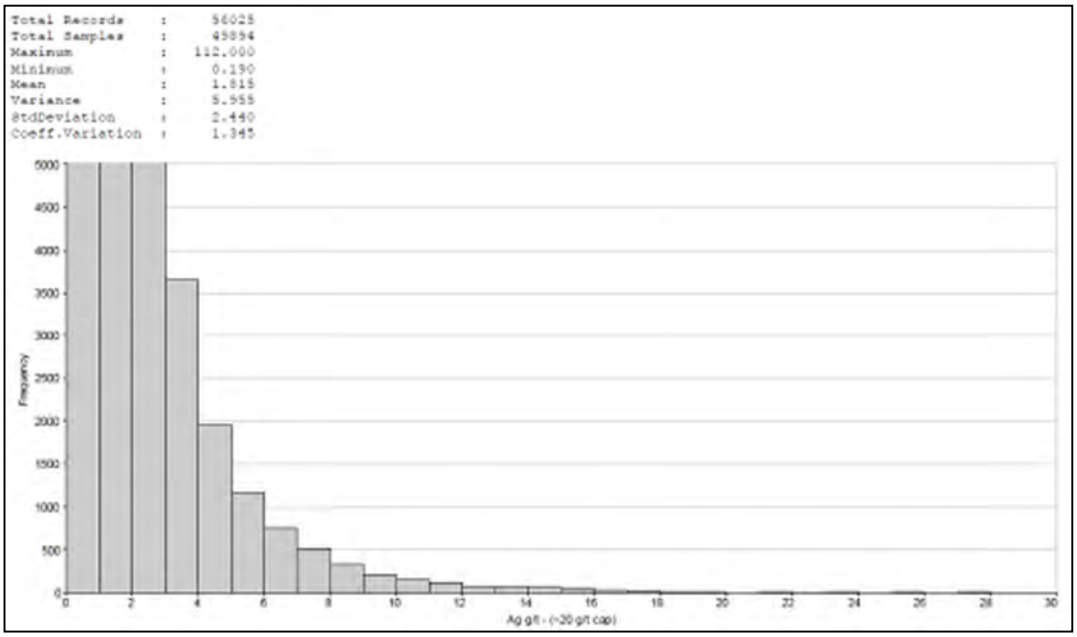
To minimize estimation bias through differing sample lengths, the drill hole data was composited to a number of lengths and the resultant statistics assessed. Compositing honoured the zone boundaries, and composite lengths for individual samples adjusted to minimize excessively long or short samples at the end of individual drill holes.

The statistics for different sample lengths is tabulated in Table 14-4.

The statistics for the different composite lengths show that the mean grade for copper, gold, and silver begins to fall at lengths above 4 m.

In order to capture most of the sample lengths into a consistent composite while maintaining sufficient resolution and maximizing the number of composited samples available for grade interpolation, a composite length of 4 m was used in this resource estimate.

Figure 14-5: Silver Log Histogram of Raw Drill hole Data

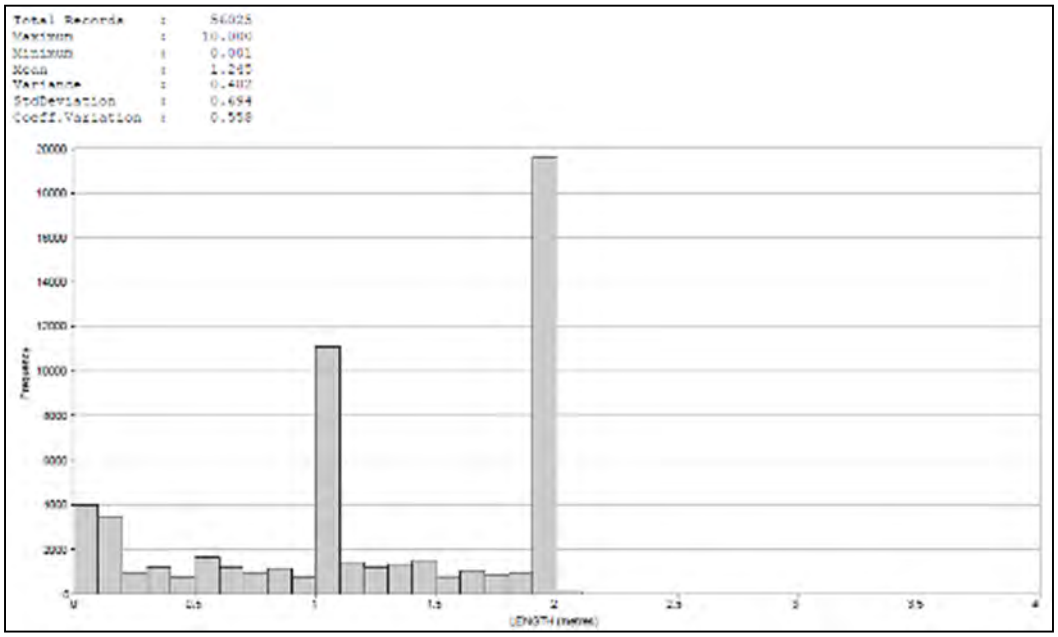


Note: Figure prepared by Tetra Tech, 2014.

Table 14-4: Cotabambas Deposit – Composite Length Statistics

Length (m)	Field	Records	Samples	Minimum	Maximum	Mean
2	Cu_pct	34,906	34,569	0.000	3.50	0.181
	Au_gt	34,906	31,113	0.001	17.81	0.102
	Ag_gt	34,906	31,994	0.190	109.09	1.775
3	Cu_pct	23,219	23,000	0.000	3.50	0.181
	Au_gt	23,219	20,748	0.001	22.49	0.101
	Ag_gt	23,219	21,362	0.199	93.46	1.774
4	Cu_pct	17,434	17,273	0.000	3.50	0.181
	Au_gt	17,434	15,626	0.002	17.06	0.101
	Ag_gt	17,434	16,056	0.196	92.14	1.775
5	Cu_pct	13,930	13,803	0.000	3.50	0.181
	Au_gt	13,930	12,512	0.002	2.41	0.099
	Ag_gt	13,930	12,854	0.215	30.00	1.765
6	Cu_pct	11,609	11,504	0.000	3.50	0.181
	Au_gt	11,609	10,442	0.003	2.38	0.099
	Ag_gt	11,609	10,725	0.207	27.05	1.766

Figure 14-6: Sample Length Histogram of Raw Drill hole Data



Note: Figure prepared by Tetra Tech, 2014.

Wireframing

The domain zones were modelled by wireframes in Datamine®. These wireframes were used to define geological zones in the resource model (base of oxidation, base of leached cap and zone of supergene mineralization), and are based on interpreted geological contacts as logged in drill holes by Panoro geologists. These wireframes were extrapolated from the drill holes on which they are based, to provide reasonable coverage for the resource model. To the north, where there is insufficient drill data to extrapolate the wireframes, a nominated elevation was used to define the base of leached cap and the base of oxidation.

The topographical wireframes were supplied by Panoro and combined in Datamine® to create a single surface.

Wireframes were also constructed to provide reasonable grade extrapolation limits to the block model drill hole to contain mineralization for, (1) copper, gold, and silver mineralization and (2) molybdenum mineralization. These wireframes are based on spatial positions of drill hole data, and were not based on any grade shells.

Contact Profiles

Mineralization contact profiles were generated between zones for all of the contiguous mineralized domains. Copper, gold and silver were examined. Due to the relatively limited extent and low grades, molybdenum was treated as one domain.

These contact profiles were used to establish if a mineralization domain boundary represented a "hard" (i.e. domains cannot share assays for grade interpolation) or "soft" (i.e. domains can share assays for grade interpolation) boundaries. A "semi-soft" boundary is such that external lower grade samples can be used to interpolate internal cells, whereas higher grade external samples are prevented from interpolating internal cells. This only occurred with respect to Zones 3 and 4. The oxide assays (Zone 3) were used to help estimate the supergene (Zone 4) cells, but the high-grade supergene assays were not used to estimate the contiguous oxide (Zone 3) cells (Table 14-5).

Variography

Variography was completed using Datamine® software. Variography was completed of all interpolated grades (copper, gold, and silver) based on mineralized geological domains (Zones 1 to 4) and the latite dykes. Variography was also performed on molybdenum, but data was not domain-restricted as for copper, gold, and silver. Density was also interpolated, and density variography was completed utilizing all spatial density data available.

Downhole variography was first performed to calculate the intrinsic sample variance (or "nugget") for each respective estimation domain. If necessary, the experimental variogram generated from the downhole analysis was used to model the third (shortest) axis. Downhole variography used capped raw data with a 2 m lag distance.

Spatial variography was performed on a series of orientations. This variography used 4 m composited and capped data, usually with 40 m lag distances. For the hypogene domain (zone1 = 1), orientations around both the full vertical and horizontal axes were evaluated. For the more planar domains (zone1 = 2, 3 and 4), the dip of the plane was estimated and variograms constructed around the horizontal axis. These multiple experimental variograms were inspected, and the optimal variogram used in the estimate was based on ranges, sample pair numbers and nugget-to-sill characteristics. The optimal variogram was modeled, and the variogram parameters were recorded in the appropriate file.

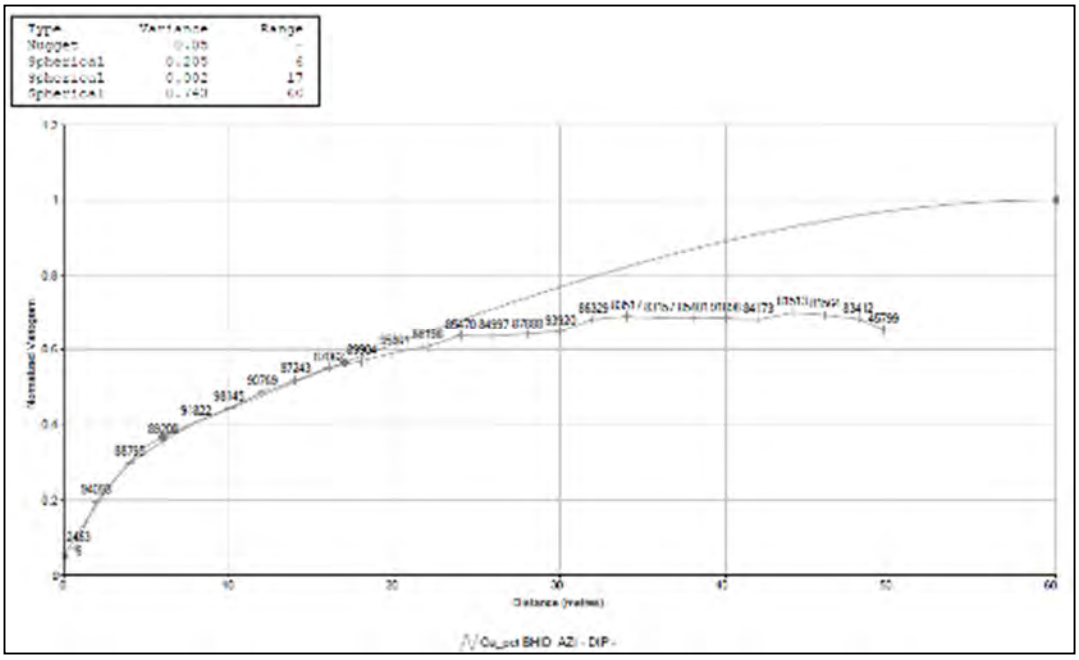
Figure 14-7 to Figure 14-10 show representative variograms, both experimental and modelled, generated from Datamine®.

In contrast with the oriented anisotropic variography performed for each of the metals by domain (or Zone), density was estimated using a single omnidirectional isotropic variogram for all domains (Figure 14-11).

Table 14-5: Boundary Relations as Interpreted from Domain Contact Profiles for Resource Estimation

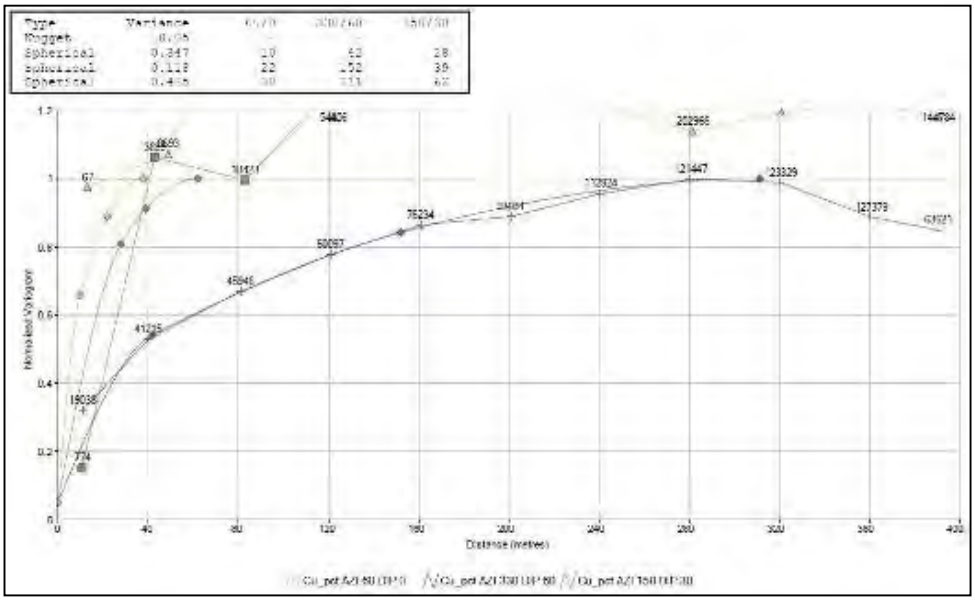
	Zone 1	Zone 2	Zone 3	Zone 4
Zone 1	—	Soft	Hard	Soft
Zone 2	Soft	—	Soft	Soft
Zone 3	Hard	Soft	—	Semi-soft
Zone 4	Soft	Soft	Semi-soft	—

Figure 14-7: Copper Downhole Variograms for Zone 1 (Hypogene mineralization)



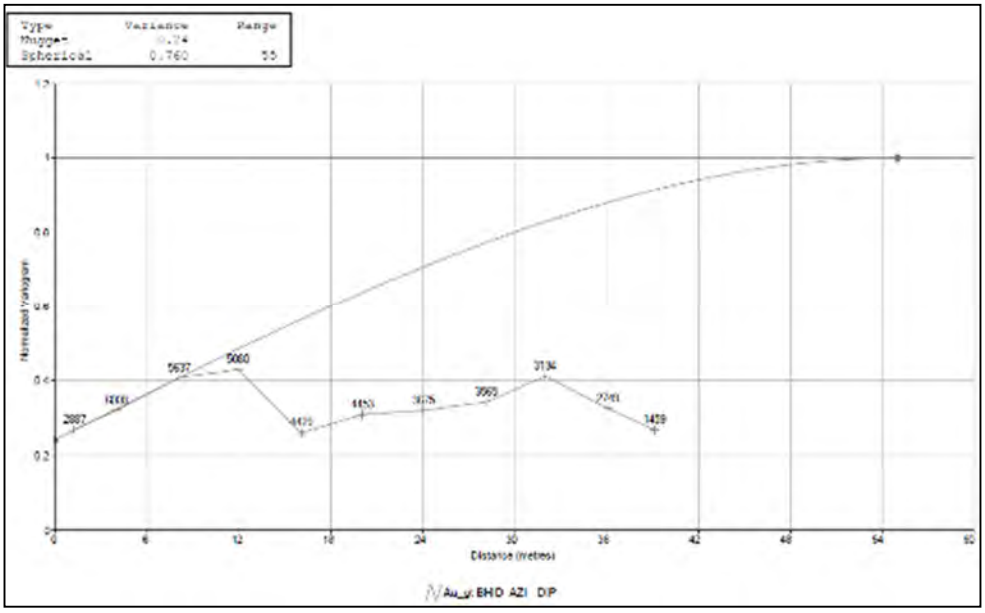
Note: Figure prepared by Tetra Tech, 2014.

Figure 14-8: Copper Downhole Variograms for Zone 1 (Hypogene mineralization)



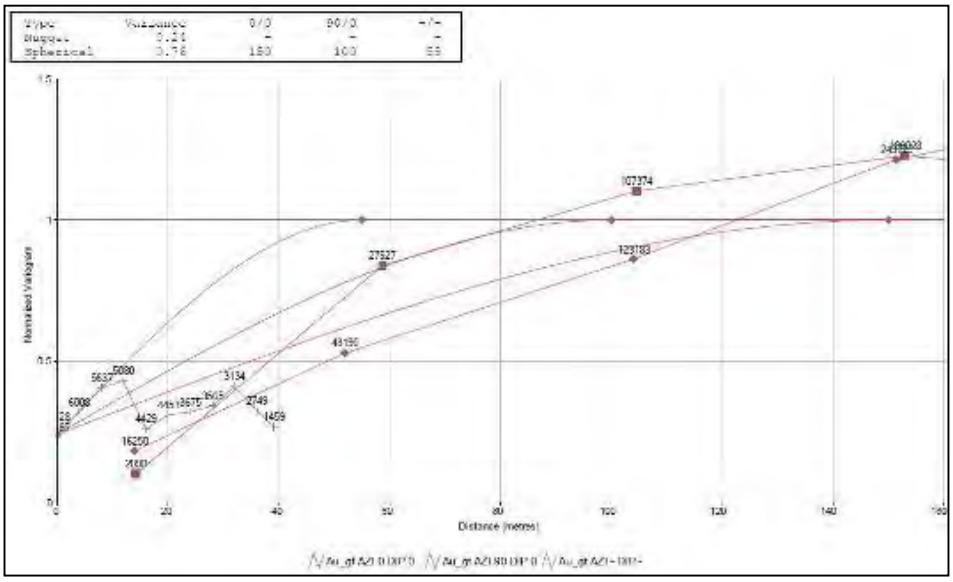
Note: Figure prepared by Tetra Tech, 2014.

Figure 14-9: Gold Downhole Variograms for Zone 2 (Oxide mineralization)



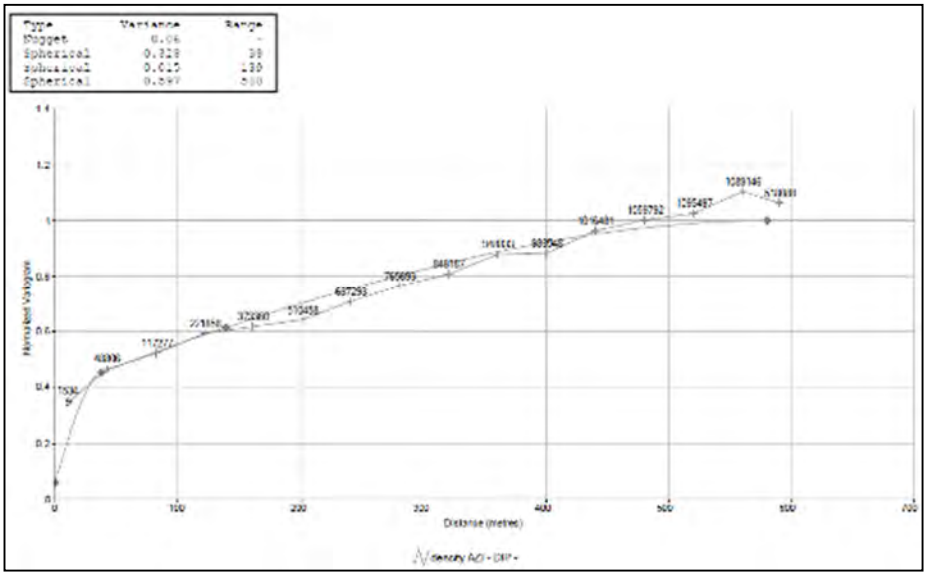
Note: Figure prepared by Tetra Tech, 2014.

Figure 14-10: Gold Variograms for Zone 2 (Oxide mineralization)



Note: Figure prepared by Tetra Tech, 2014.

Figure 14-11: Density Variograms



Note: Figure prepared by Tetra Tech, 2014.

14.1.5 Block Model

Block Model Configuration

The Cotabambas resource model was designed and constructed using Datamine® software. Table 14-6 summarizes the fundamental model design. The block model was not rotated.

The parent cell size (20 m by 20 m by 12 m) is significantly larger than the 2012 model completed by Amec Foster Wheeler which employed 10 m by 10 m by 10 m cells. This is in order to provide more robust sample support for the estimation of grade into individual cells, given the current level of drill hole spacing. Sub-cells (no smaller than 5 m by 5 m by 3 m) were used to improve the model resolution along the topographic surface and domain boundaries. The estimated model can be re-blocked to a smaller parent cell size if mine planning requires smaller cells.

Model Domains

The same domains as identified in the drill hole data were applied to the block model. These include the leached cap (zone1 = 3), the oxide zone (zone1 =2), hypogene mineralization (zone1=1) and supergene mineralization (zone1=4). The wireframes and the drill hole data used to design these wireframes are depicted in Figure 14-12 along with the block model, coloured by domain.

14.1.6 Grade and Density Estimation

Interpolation of metal grades (copper percent, gold grams per tonne, silver grams per tonne and molybdenum percent) and interpolation of density (specific gravity) was performed in Datamine®. Datamine® requires the use of three parameter files (estimation, samples and variography) to complete estimation. These parameters are listed and described below.

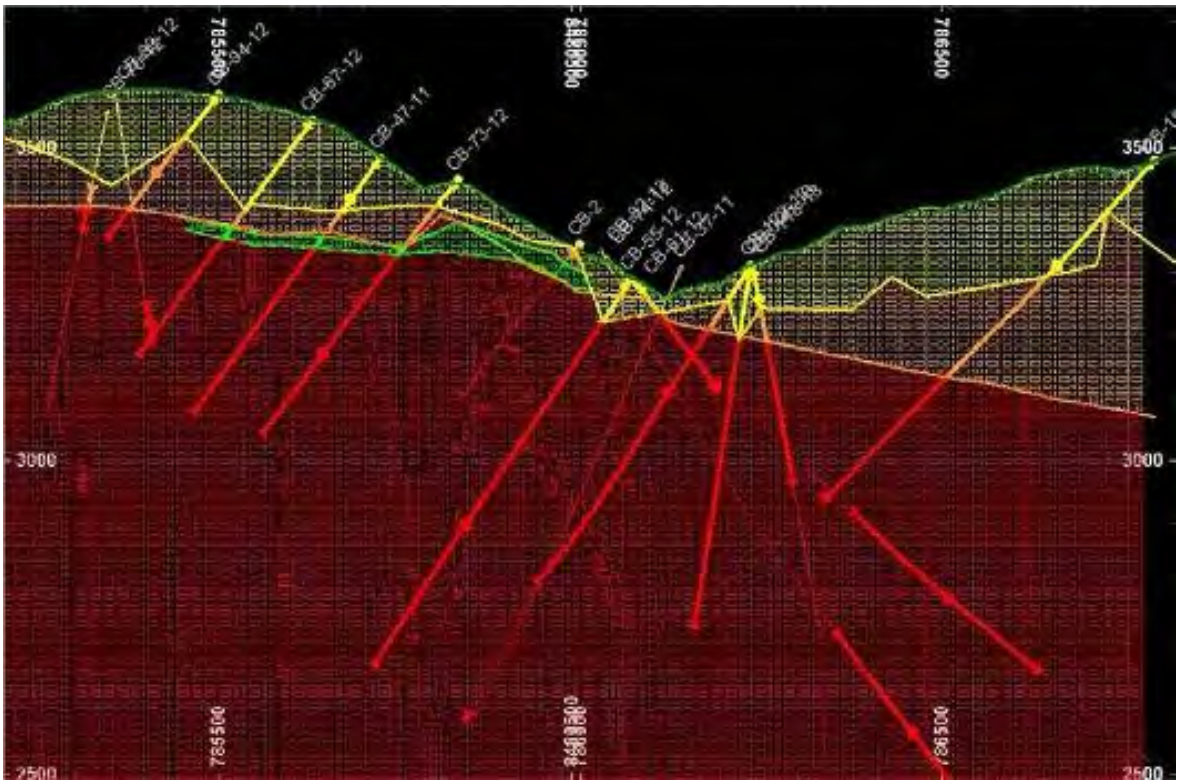
Estimation Parameters

The estimation parameter files lists what variables are estimated and what types of estimation methods are used. In addition to metals and density, the Lagrange Multiplier and F-Function, based on copper grades, were also interpolated. These values (LG and F) facilitated the calculation of the theoretical slope of regression (ZZ*) and Kriging Efficiency (KE). These are variables which assist in evaluating the quality of the ordinary kriged estimation, and are discussed in subsequent sections.

Table 14-6: Resource Block Model Configuration

	Easting (m)	Northing (m)	Elevation (m)
Model origin	783,700	847,760	2,200
Cell size (m)	20	20	12
Number of cells	190	200	158
Sub-cells per cell	4	4	4

Figure 14-12: Northwest–Southeast Cross-Section of Domain Wireframes with Drill Hole Traces and Block Model, Coloured by Domain



Note: Domain colours: yellow – leached cap, orange – oxide, bright green – supergene, red – hypogene. Figure prepared by Tetra Tech, 2014.

Most estimated variables used three different interpolation methods: OK, inverse distance squared (ID2 ["power" is set to 2]), and nearest neighbour (NN). Of these methods, only OK was reported on in the final resource tabulation. The other methods are only used for model estimation validation.

Density

Three estimation methods were used to interpolate density throughout the entire block model; OK, ID2, and NN. These methods are listed in Table 14-7 as "I Method" 3, 2 and 1 respectively. The "srefnum" and "vrefnum" refers to the sample search and variogram parameter files used, and "krignegw" flags a request to retain negative kriging weights. As density estimation was not restricted to geological units (i.e. zone1=1, zone1=2, zone1=4 and zone1=4), no further refinement of the estimation parameter file was required.

Metals

Like density, molybdenum estimation was based on three methods and was not restricted by and domains or zones, apart from the designated molybdenum wireframe shell.

Copper, gold, and silver were interpolated by domain (i.e. zone1=1, zone1=2, zone1=4 and zone1=4; hypogene, oxide, leached-cap and supergene respectively). Each domain interpolation required a specific drill hole dataset as identified thought the contact profile analyses (i.e. a specific zone would use that zone's drill hole data plus contiguous zones data if the contact profile has identified "soft" contact boundaries). Table 14-8 uses copper for an example of the metals estimation parameter file.

Note IMETHOD 101 and 102 are used to interpolate the F-Function and Lagrange Multiplier based on Cu variography and copper values only. Also recorded are the number of samples ("NUMSAM") and which search volume ("SVOL" - out of three) are used for a successful interpolation. These attributes assist in evaluating the quality of the estimate, and in particular reference to the search volume, help configure the Indicated Resource classification.

Similar estimation parameter files were compiled and completed for density, latite dykes (hypogene zone only), molybdenum, gold and silver.

Table 14-7: Density Estimation Parameter File

Description	Value In	Value Out	srefnum	imethod	power	vrefnum	krignegw
Density	Density	dens_ok	1	3	-	1	0
Density	Density	dens_id	1	2	2	1	0
Density	Density	dens_nn	1	1	-	1	0

Table 14-8: Copper Estimation Parameter File

Description	EREFNUM	Value In	Value Out	SREFNUM	NUMSAM	SVOL	VAR	IMETHOD	POWER	VREFNUM	KRIGNEGW	zone1
Cu_ok	1	Cu_pct	Cu_ok	1	numsam	svol	kv	3	-	1	0	1
Cu_id	2	Cu_pct	Cu_id	1				2	2	1	0	
Cu_nn	3	Cu_pct	Cu_nn	1				1	-	1	0	
Cu_f	4	Cu_pct	Cu_f	1				101	-	1	0	
Cu_lg	5	Cu_pct	Cu_lg	1				102	-	1	0	
Cu_ok	6	Cu_pct	Cu_ok	2	numsam	svol	kv	3	-	2	0	2
Cu_id	7	Cu_pct	Cu_id	2				2	2	2	0	
Cu_nn	8	Cu_pct	Cu_nn	2				1	-	2	0	
Cu_f	4	Cu_pct	Cu_f	1				101	-	1	0	
Cu_lg	5	Cu_pct	Cu_lg	1				102	-	1	0	
Cu_ok	9	Cu_pct	Cu_ok	3	numsam	svol	kv	3	-	3	0	3
Cu_id	10	Cu_pct	Cu_id	3				2	2	3	0	
Cu_nn	11	Cu_pct	Cu_nn	3				1	-	3	0	
Cu_f	4	Cu_pct	Cu_f	1				101	-	1	0	
Cu_lg	5	Cu_pct	Cu_lg	1				102	-	1	0	
Cu_ok	12	Cu_pct	Cu_ok	4	numsam	svol	kv	3	-	4	0	4
Cu_id	13	Cu_pct	Cu_id	4				2	2	4	0	
Cu_nn	14	Cu_pct	Cu_nn	4				1	-	4	0	
Cu_f	4	Cu_pct	Cu_f	1				101	-	1	0	
Cu_lg	5	Cu_pct	Cu_lg	1				102	-	1	0	

Variography Parameters

Variography was completed using Datamine® software for each zone (VREFNUM =1, 2, 4 and 4 equate to Zone1 =1, 2, 3, and 4 respectively). Modelled variograms were anisotropic and spherical (ST1, 2, 3 = 1) with the exception of density (isotropic) and sequential copper leaching (ID2 only).

Table 14-9 summarizes the variography parameter file used in copper resource estimation. Rotation methodology used is positive degrees are rotated clockwise with the long axis represented by the X-axis (west-east - VAXIS1), and the short axis represented by the Z-axis (elevation - VAXIS3).

Similar variography parameter files were completed for density, latite dykes (hypogene zone only), molybdenum, gold and silver. As sequential leaching data was interpolated by ID2 only, no variography was required.

Search and Sample Parameters

The sample search used for estimation sample selection is based on the geological strike and dip orientation of mineralization as interpreted by Panoro site geologists. The total sill ranges and orientations derived from variography for each respective zone, such that the search ellipse (SMETHOD=2) for the first estimation pass matched the total sill ranges and rotations. However, for copper, the search ellipse size was designed to represent only 66% of the total sill range. This was purposely completed to assist in defining an Indicated resource classification in the estimated cells (i.e. SVOL = 1). An example of the search and sample parameter file, with restricted search ranges (SDIST1, 2 and 3), is tabulated in Table 14-10.

This parameter file also defines the minimum and maximum number of samples to satisfy cell interpolation requirements. For sample selection, only three samples could be allocated to any single drill hole through the definition of “MAXKEY” (3). This was to ensure optimal special representation for interpolation.

The minimum number of samples required for successful interpolation in all passes was set to six. This was designed to maximize the number of cells which could be interpolated while minimizing excessive averaging to preserve local variability in the estimate. The maximum number of samples was capped at 14. This was also to prevent over-averaging of the interpolation.

In Table 14-10, the search volume was expanded to 1.5 times the initial volume in the second pass, then twice the volume in the third pass (i.e. SVOLFAC2, 3). This was to accommodate the 66% ranges for the copper estimate. In other parameter files, the second and third search passes at two and three times the volume respectively.

Table 14-9: Cotabambas Copper Variography Parameter File

Description	VREFNUM	VANGLE1	VANGLE2	VANGLE3	VAXIS1	VAXIS2	VAXIS 3	NUGGET
Cu1	1	-30	0	60	3	2	1	0.05
Cu2	2	-32.5	-15.94	-8.39	3	2	1	0.08
Cu3	3	-49.78	-4.83	-18.4	3	2	1	0.03
Cu4	4	-60.85	-7.44	-13.06	3	2	1	0.06

Description	ST1	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4
Cu1	1	10	42	28	0.347
Cu2	1	22	30	30	0.444
Cu3	1	18	14	2	0.061
Cu4	1	109	16	26	0.64

Description	ST2	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4
Cu1	1	22	152	39	0.118
Cu2	1	76	127	30	0.476
Cu3	1	59	24	11	0.209
Cu4	1	124	162	26	0.3

Description	ST3	ST3PAR1	ST3PAR2	ST3PAR3	ST3PAR4
Cu1	1	38	311	62	0.485
Cu2	-	-	-	-	-
Cu3	1	80	155	140	0.7
Cu4	-	-	-	-	-

Table 14-10: Cotabambas Copper Search and Sample Parameter File

SDESC	SREFNUM	SMETHOD	SDIST1	SDIST2	SDIST3	MAXKEY		
Cu1	1	2	250	150	80	3		
Cu2	2	2	76	127	30	3		
Cu3	3	2	80	155	140	3		
Cu4	4	2	124	162	26	3		
SDESC	**SANGLE1**	**SANGLE2**	**SANGLE3**	**SAXIS1**	**SAXIS2**	**SAXIS3**		
Cu1	-50	-80	0	3	1	2		
Cu2	-32.5	-15.94	-8.39	3	2	1		
Cu3	-49.78	-4.83	-18.4	3	2	1		
Cu4	-60.85	-7.44	-13.06	3	2	1		
SDESC	**MINNUM1**	**MAXNUM1**	**SVOLFAC2**	**MINNUM2**	**MAXNUM2**	**SVOLFAC3**	**MINNUM3**	**MAXNUM3**
Cu1	6	14	1.5	6	14	2	6	14
Cu2	6	14	1.5	6	14	2	6	14
Cu3	6	14	1.5	6	14	2	6	14
Cu4	6	14	1.5	6	14	2	6	14

14.1.7 Resource Block Model

Metal Estimation

Copper, gold, silver and molybdenum were estimated into the resource model cells by means of OK, ID2 and NN. Both ID2 and NN were performed for model validation purposes only. Copper, gold and silver were estimated into their domains using samples as defined by the contact profiles for each domain. Molybdenum was estimated without any domain restraints due to the relatively low grades of the metal.

Density Estimation

Density was estimated into cells of the block model without domain restraints. Both ID2 and NN were also estimated for density, but were only used for validation purposes only. If the OK estimation failed to interpolate into any cell, it was replaced by the ID2 value. If ID2 value failed to interpolate, it was replaced by the NN value. If all three estimation methods failed to estimate a cell, the absent value was assigned the average estimated cell value for that specific domain.

Cell Attributes

Table 14-11 lists the cell attributes in the Cotabambas resource model. These attributes are either estimated, or assigned to calculate.

Table 14-11: Cotabambas Model Cell Attributes

Attribute	Description	Type	
IJK	Unique cell identifier	Integer	Assigned
XC	Easting cell centroid	Integer	Assigned
YC	Northing cell centroid	Integer	Assigned
ZC	Elevation cell centroid	Integer	Assigned
XINC	Cell size - easting	Integer	Assigned
YINC	Cell size - northing	Integer	Assigned
ZINC	Cell size - elevation	Integer	Assigned
Density	Density	Variable	Estimated
zone1	Domains for estimation	Integer	Assigned
zone2	Latite dykes domain	Integer	Assigned
dens_ok	Density estimated by OK	Variable	Estimated
dens_id	Density estimated by ID	Variable	Estimated
dens_nn	Density estimated by NN	Variable	Estimated
Cu_ok	Copper estimated by OK	Variable	Estimated
Cu_id	Copper estimated by ID2	Variable	Estimated
Cu_nn	Copper estimated by NN	Variable	Estimated
Cu_f	F-Function estimated with copper	Variable	Estimated
Cu_lg	Lagrange Multiplier estimated with copper	Variable	Estimated
Au_ok	Gold estimated by OK	Variable	Estimated
Au_id	Gold estimated by ID2	Variable	Estimated
Au_nn	Gold estimated by NN	Variable	Estimated
Ag_ok	Silver estimated by OK	Variable	Estimated
Ag_id	Silver estimated by ID2	Variable	Estimated
Ag_nn	Silver estimated by NN	Variable	Estimated
Mo_ok	Molybdenum estimated by OK	Variable	Estimated
Mo_id	Molybdenum estimated by ID2	Variable	Estimated
Mo_nn	Molybdenum estimated by NN	Variable	Estimated
numsam	Number of samples used in cell estimation	Integer	Assigned
Svol	Search volume used in cell estimation	Integer	Assigned
Kv	Kriging variance for copper	Variable	calculated
Bv	Block variance	Variable	calculated
KE	Kriging Efficiency	Variable	calculated
ZZ	Slope of Regression (ZZ*)	Variable	calculated
Cu_eq	Copper equivalent grade	Variable	calculated
rescat	Resource classification	Integer	Assigned
Pit	Optimized pit shell number	Integer	Assigned
CuAS	Acid leach recovery	Variable	Estimated
CuCN	Cyanide leach grade	Variable	Estimated
CuR	Flotation recovery grade	Variable	Estimated
CuL	Copper sequential leaching grade	Variable	calculated
AuL	Gold sequential leaching grade	Variable	calculated
AgL	Silver sequential leaching grade	Variable	calculated
CuL_eq	Copper sequential leaching equivalent grade	Variable	calculated

14.1.8 Model Validation

This section describes the methods and results in validating the OK block model. It essentially involves comparing statistics, visual sectional valuation of estimated cell grades and proximal drill hole data, and construction of swath plots which show the OK results in comparison with ID2 and NN interpolations.

Statistics

Table 14-12 compares the statistics, per domain, of the different interpolation methods, as well as with the drill hole data (capped and composited) used for the estimation. The model statistics were generated from a regularized block model (no sub-cells) and confined to cells within pit number 36.

Table 14-12 confirms the close affinity of the estimated grades with the actual (drill hole) grades. Note that the resource model was regularized to the parent cell configuration, and that the cells with “zero” Cu_ok were not deleted from the calculation. This is to accommodate absent values, as absent values cannot be regularized and must retain a “zero” value.

Sections

The block model was visually validated by comparing the OK estimated grades and density with the immediate drill hole data. In general, there is good correlation, although localized averaging or “smoothing” will have, (a) lower estimated grades than immediate drill hole assays, and (b) in some cases have higher estimated grades than immediate drill hole assays. As there are many lower drill hole grade assays than higher drill hole grade assays, (a) will be more common than (b). Figure 14-13 to Figure 14-23 are representative northwest–southeast cross-sections of the block model with the drill hole traces.

Swath Plots

Swath plots compare the different interpolations for each of the estimated attributes based on equivalent northings, eastings and elevations. Representative images of these plots (copper and gold) are depicted in Figure 14-24 to Figure 14-29.

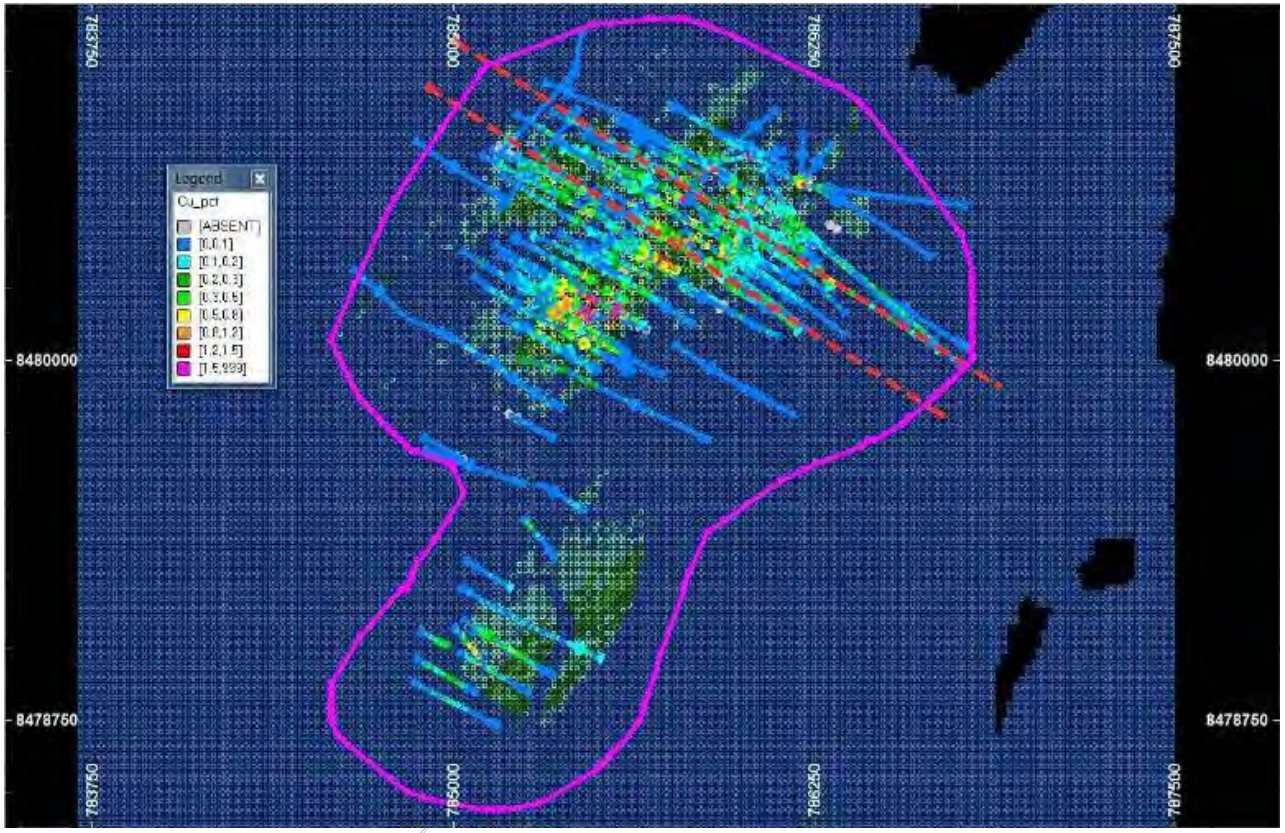
With respect to all metals, there are good correlations between the OK estimation and the ID2 estimation. As expected, the NN estimation is more erratic in comparison, and as there are lower grade samples than higher grade samples, the NN curve is commonly slightly below that of either ID2 or OK.

Density shows very close correlation between all interpolations methods. This is in part due that domains were not used to restrict sample selection.

Table 14-12: Cotabambas Model and Drill Hole Statistics by Domain

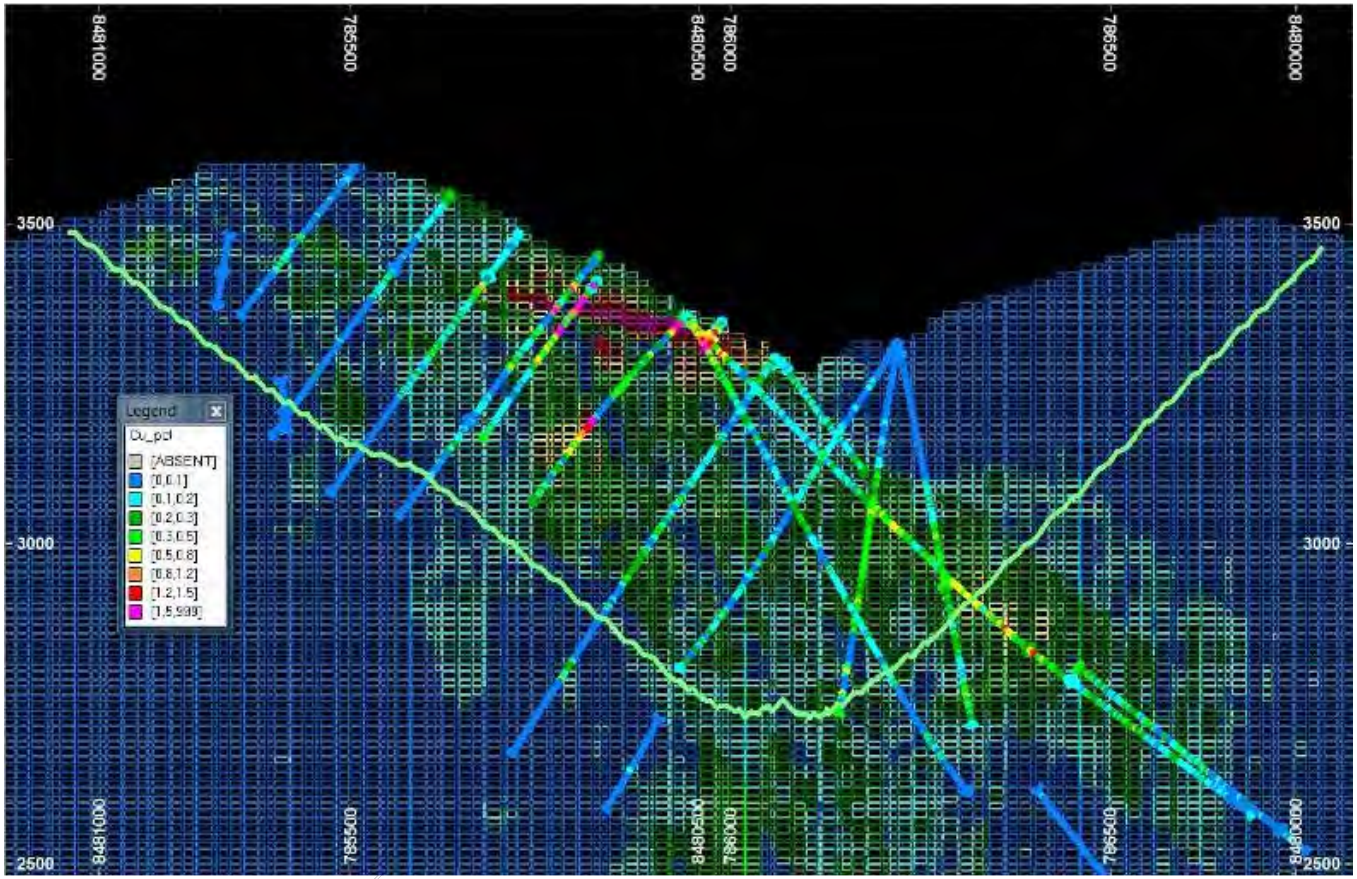
Zone	Type	Field	Records	Samples	Minimum	Maximum	Mean	Variance	Std Dev	CV
4	Model	Cu_ok	219,281	121,410	0.000	2.132	0.161	0.024	0.156	219,281
		Au_ok	219,281	121,410	0.000	1.647	0.095	0.012	0.111	1.3489
		Ag_ok	219,281	121,410	0.000	14.556	1.794	0.958	0.979	219,281
	ddh	Cu_pct	16,463	12,728	0.0001	3.0329	0.1760	0.0639	0.2528	1.4358
		Au_gt	16,463	11,487	0.0020	2.5782	0.0973	0.0321	0.1792	1.8405
		Ag_gt	16,463	11,832	0.1959	30.0000	1.8623	3.6504	1.9106	1.0260
5	Model	Cu_ok	219,281	26,036	0.000	2.266	0.161	0.052	0.229	0.9828
		Au_ok	219,281	26,036	0.000	1.200	0.076	0.012	0.111	1.2424
		Ag_ok	219,281	26,036	0.000	10.032	1.153	1.165	1.079	0.7876
	ddh	Cu_pct	16,463	1,307	0.0016	3.4466	0.2610	0.1546	0.3932	1.5065
		Au_gt	16,463	1,194	0.0040	2.4022	0.1112	0.0384	0.1960	1.7631
		Ag_gt	16,463	1,205	0.5000	16.7402	1.6414	2.2441	1.4980	0.9127
6	Model	Cu_ok	219,281	29,517	0.000	1.063	0.052	0.005	0.071	1.2217
		Au_ok	219,281	29,517	0.000	1.033	0.034	0.004	0.067	219,281
		Ag_ok	219,281	29,517	0.000	6.646	1.218	0.430	0.656	0.9610
	ddh	Cu_pct	16,463	1,908	0.0007	2.4124	0.1252	0.0552	0.2349	1.8766
		Au_gt	16,463	1,799	0.0017	1.8350	0.1049	0.0451	0.2123	2.0243
		Ag_gt	16,463	1,760	0.5000	17.4286	1.4635	1.6296	1.2766	0.8722
7	Model	Cu_ok	219,281	2,893	0.036	2.707	0.646	0.224	0.474	0.6973
		Au_ok	219,281	2,893	0.000	1.524	0.235	0.041	0.203	0.8146
		Ag_ok	219,281	2,893	0.500	12.605	2.451	2.411	1.553	0.5745
	ddh	Cu_pct	16,463	363	0.0006	3.5000	0.7680	0.6888	0.8299	1.0807
		Au_gt	16,463	358	0.0045	2.1573	0.2793	0.1207	0.3474	1.2436
		Ag_gt	16,463	325	0.5000	24.0000	2.9103	10.4897	3.2388	1.1129

Figure 14-13: Plan View (3,200 m elevation)



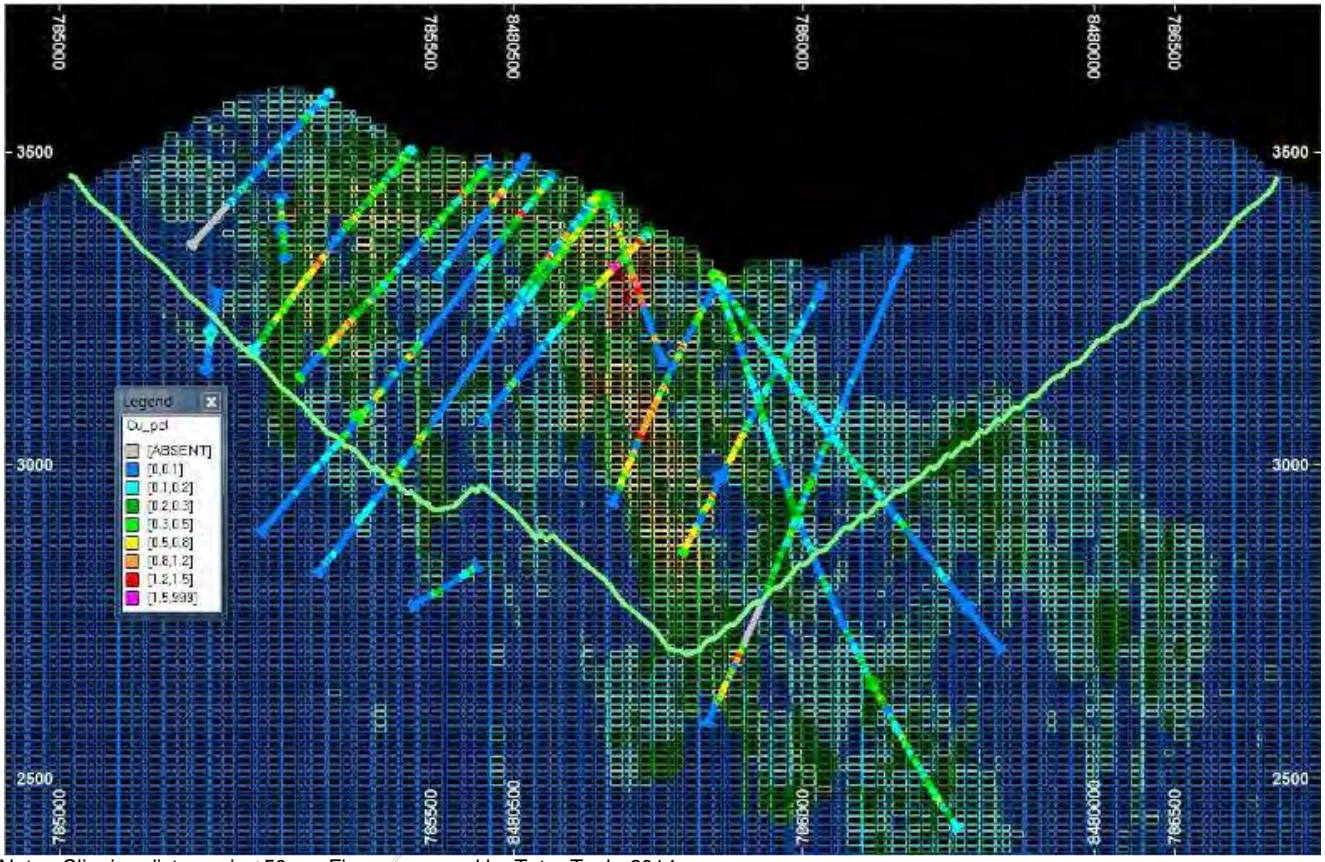
Note: Colours show Cu% grades. Figure shows conceptual pit outline (magenta) and locations of northwest–southeast cross-sections (red dashed lines). No clipping distance applied to drill holes. Figure prepared by Tetra Tech, 2014

Figure 14-14: Copper Northwest–Southeast Section Furthest to Northeast (Cu %)



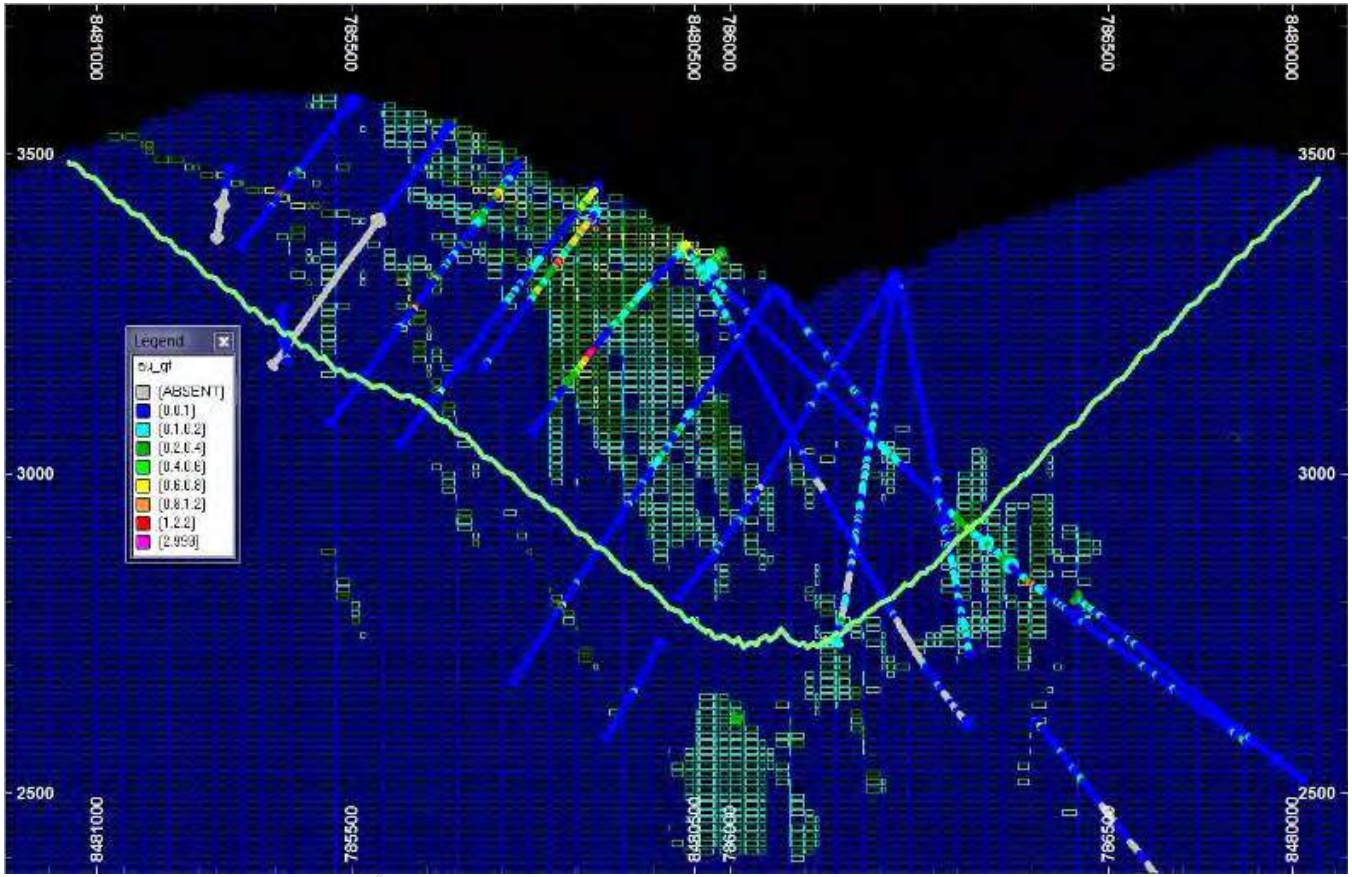
Note: Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-15: Copper Northwest–Southeast Section Furthest to Southeast (Cu %)



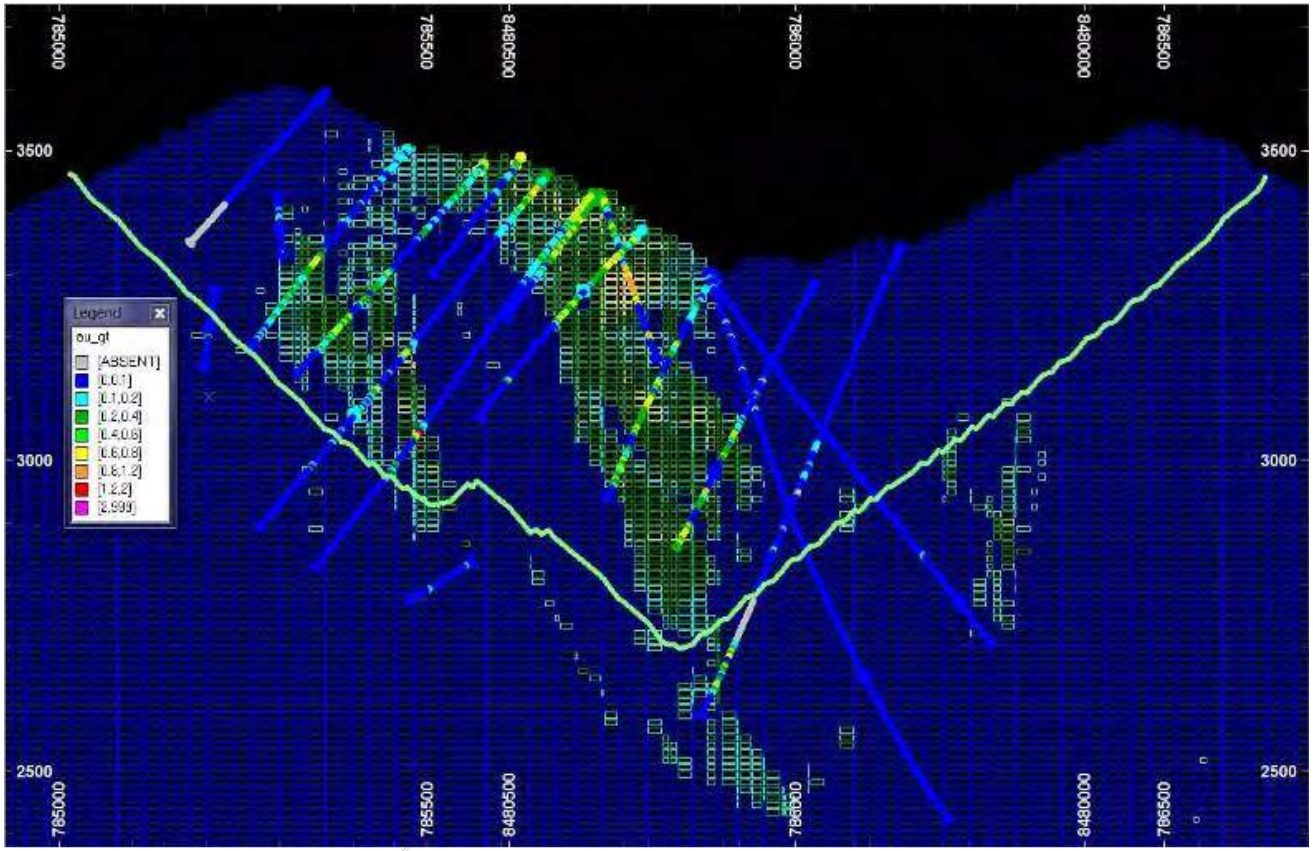
Note: Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-16: Gold Northwest–Southeast Section Furthest to Northeast (Au g/t)



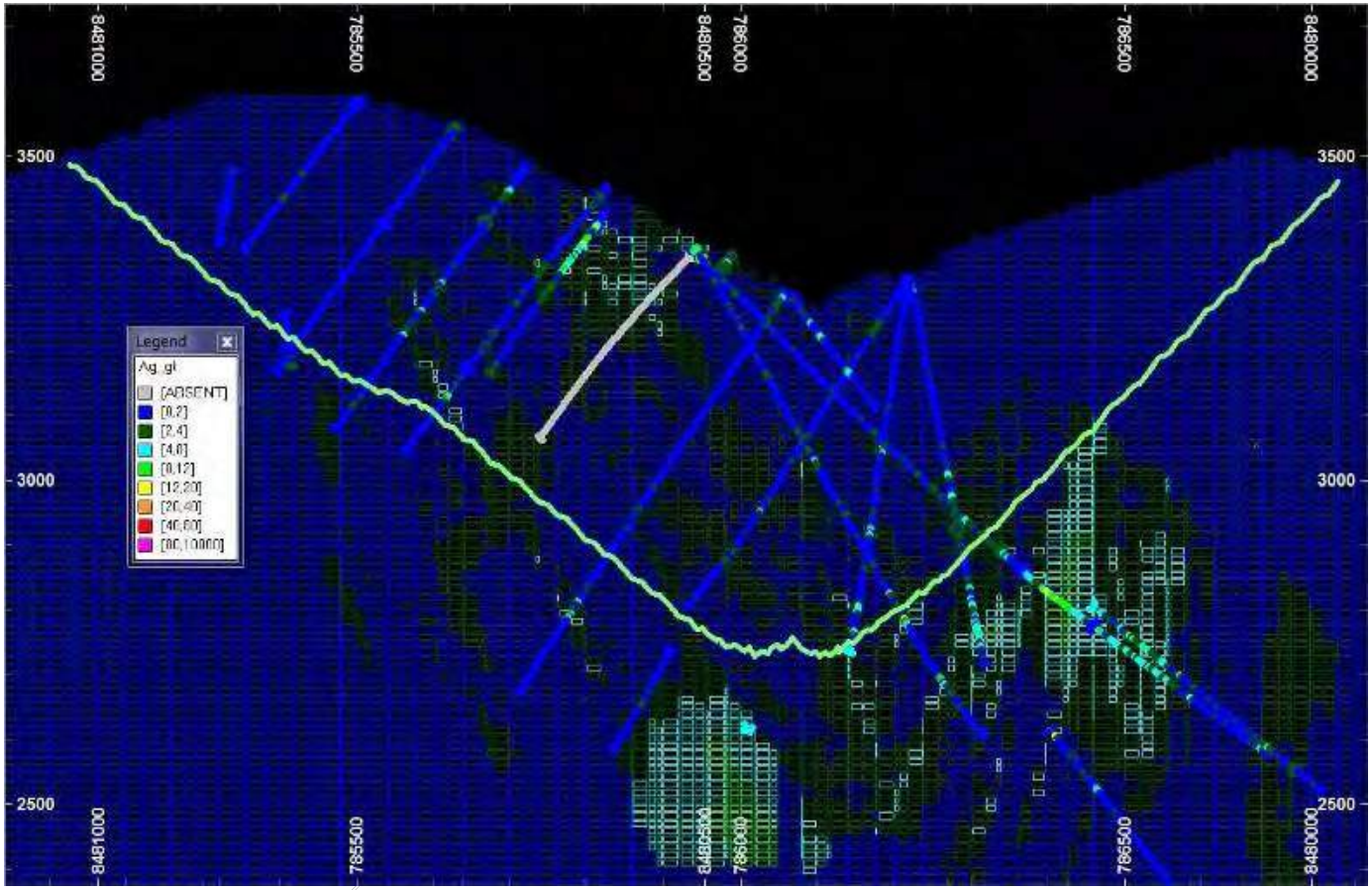
Note: Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-17: Gold Northwest–Southeast Section Furthest to Southwest (Au g/t)



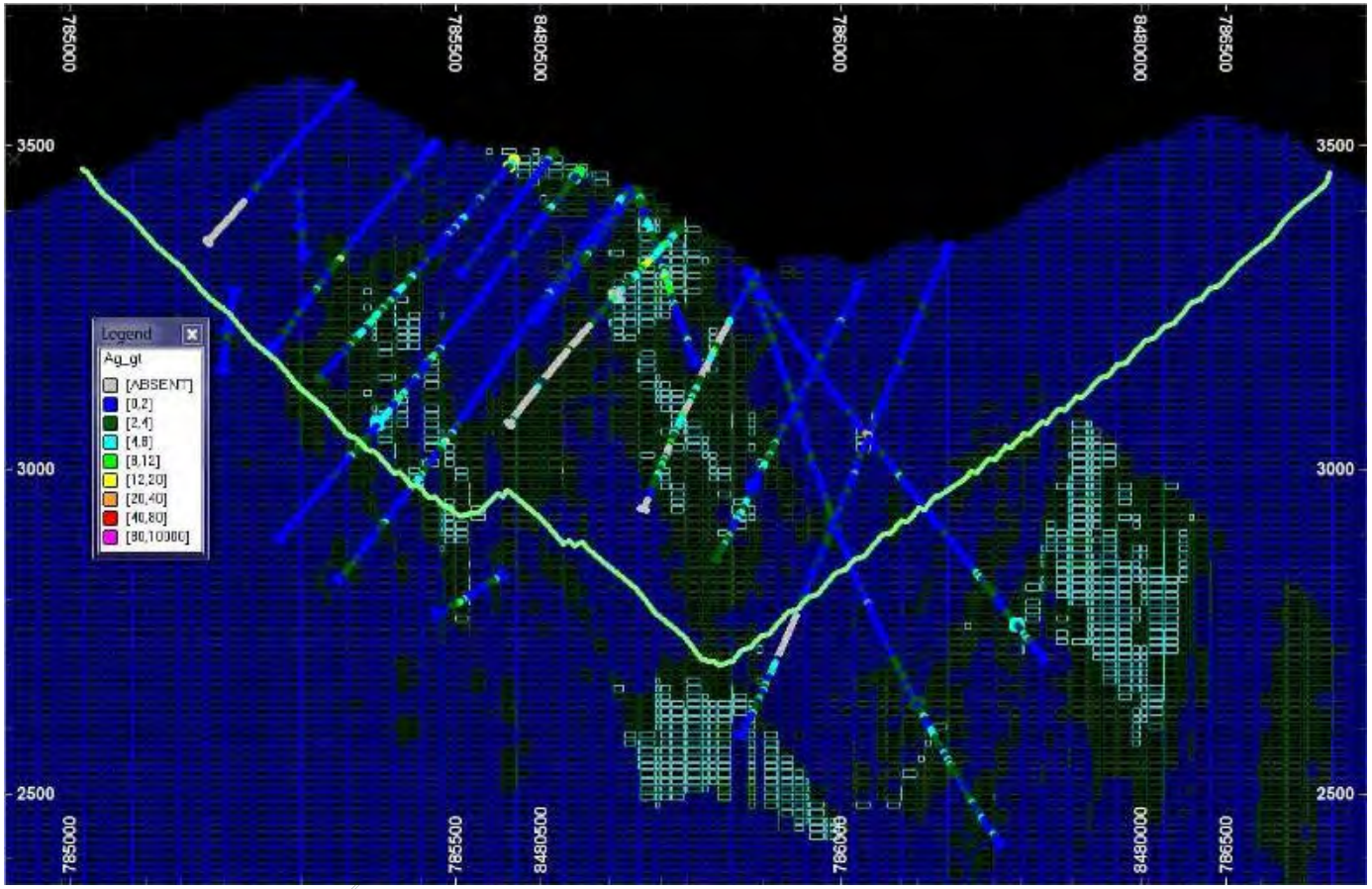
Note: Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-18: Silver Northwest–Southeast Section furthest to Northeast (Ag g/t)



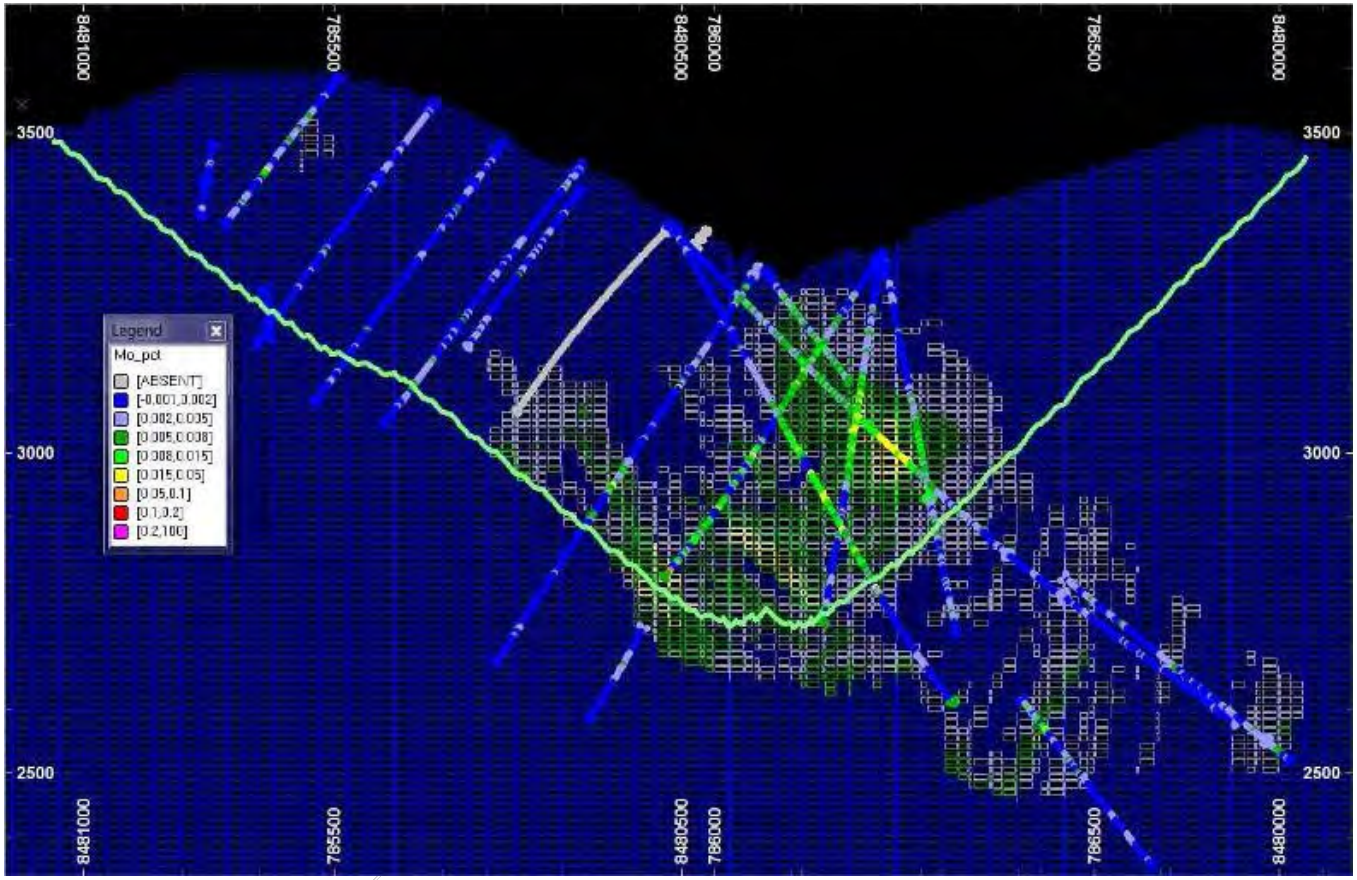
Note: Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-19: Silver Northwest–Southeast Section Furthest to Southwest (Ag g/t)



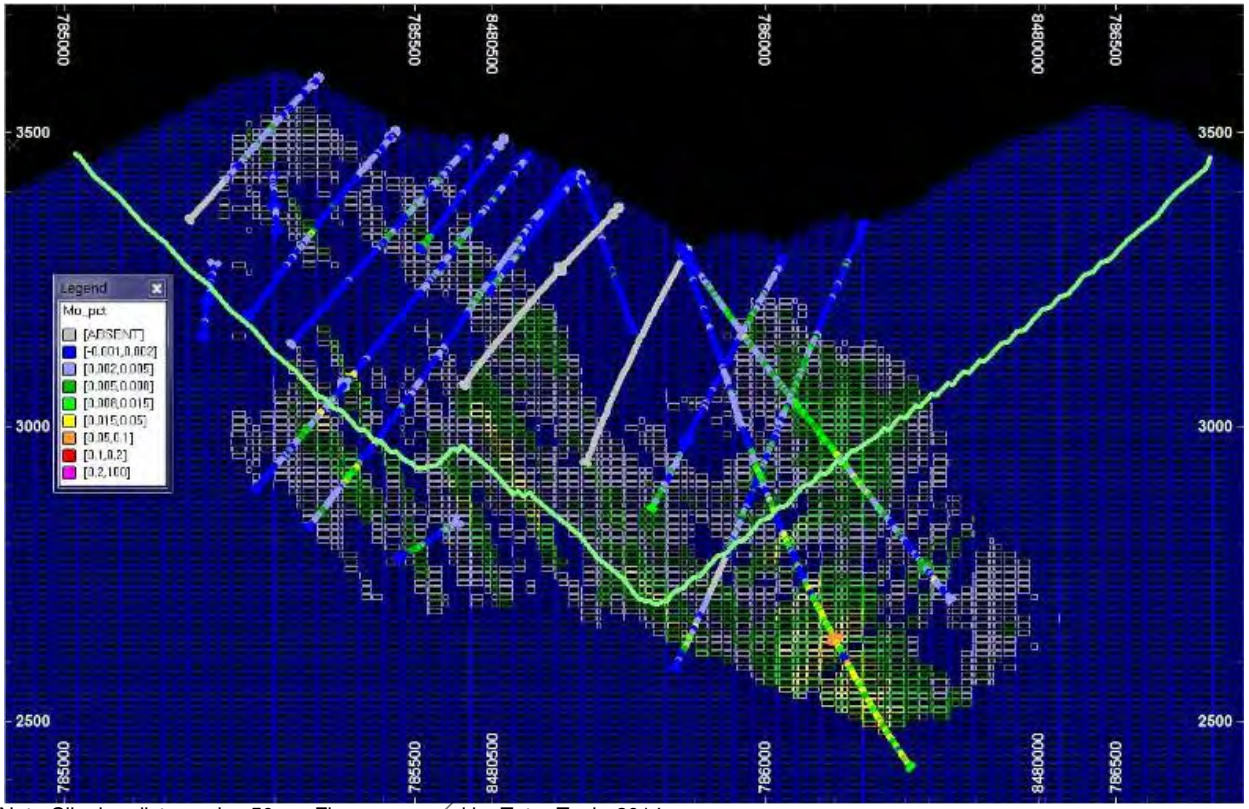
Note: Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-20: Molybdenum Northwest–Southeast Section Furthest to Northeast (Mo %)



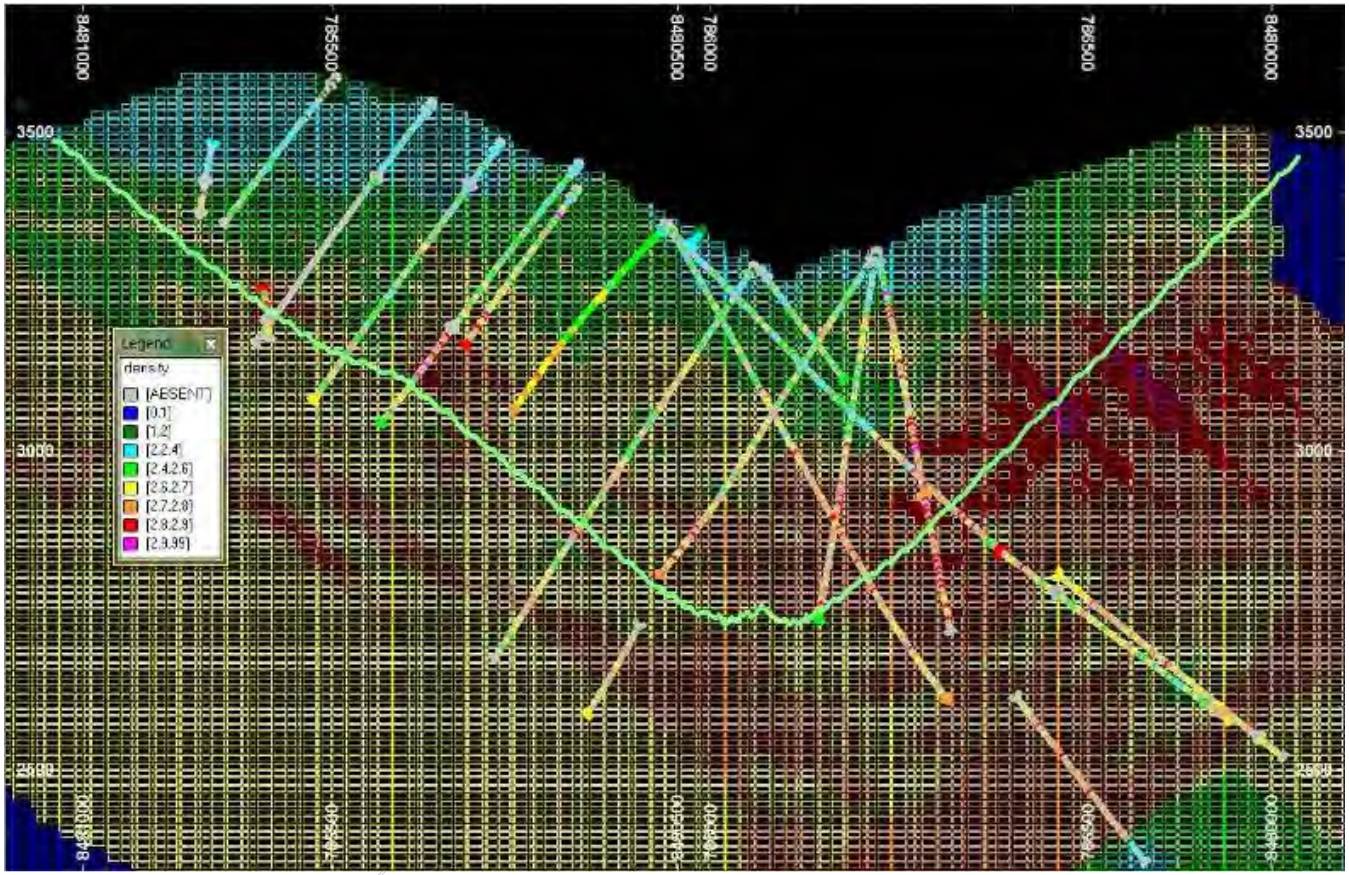
Note:Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-21: Molybdenum Northwest–Southeast Section Furthest to Southwest (Mo %)



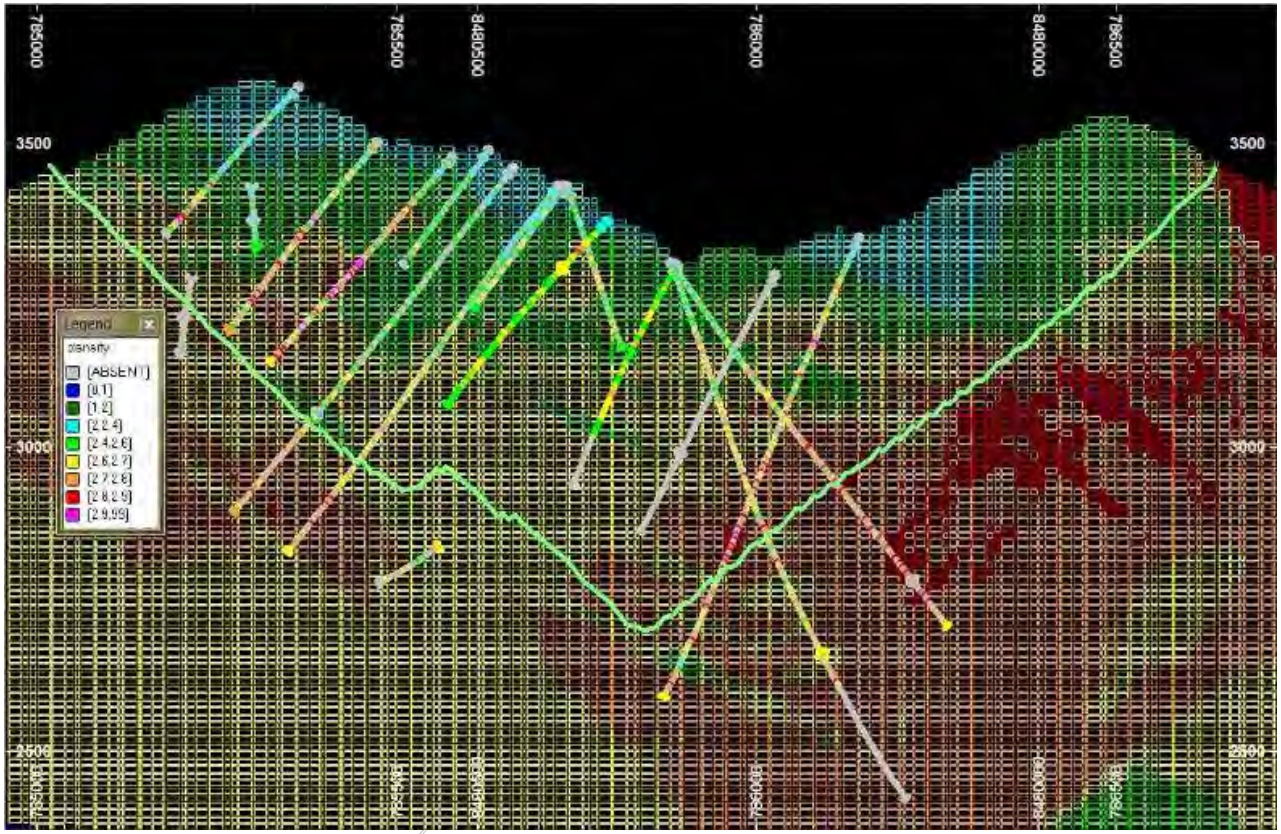
Note:Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-22: Density Northwest–Southeast Section Furthest to Northeast (SG)



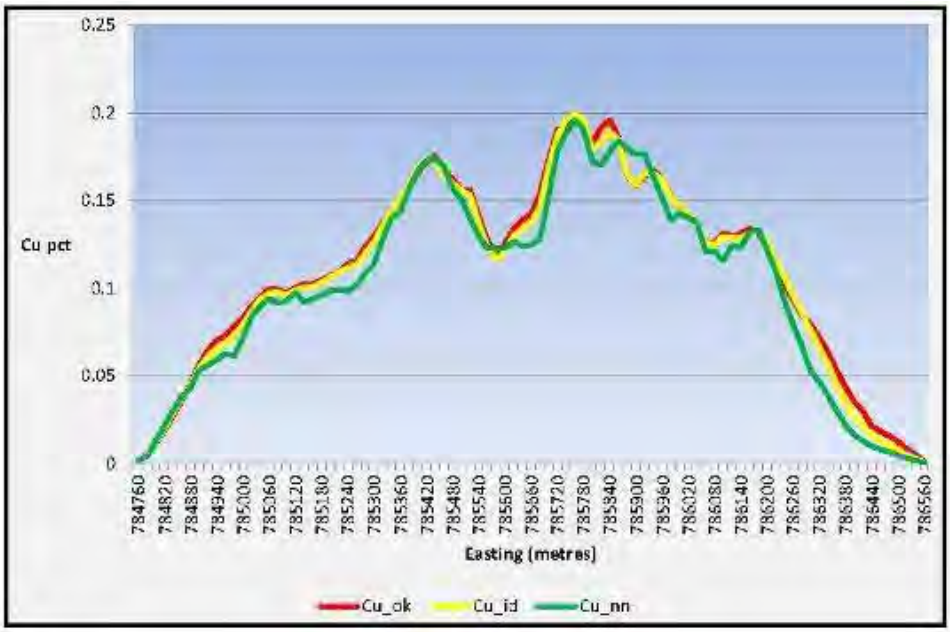
Note:Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-23: Density Northwest–Southeast Section Furthest to Northeast (SG)



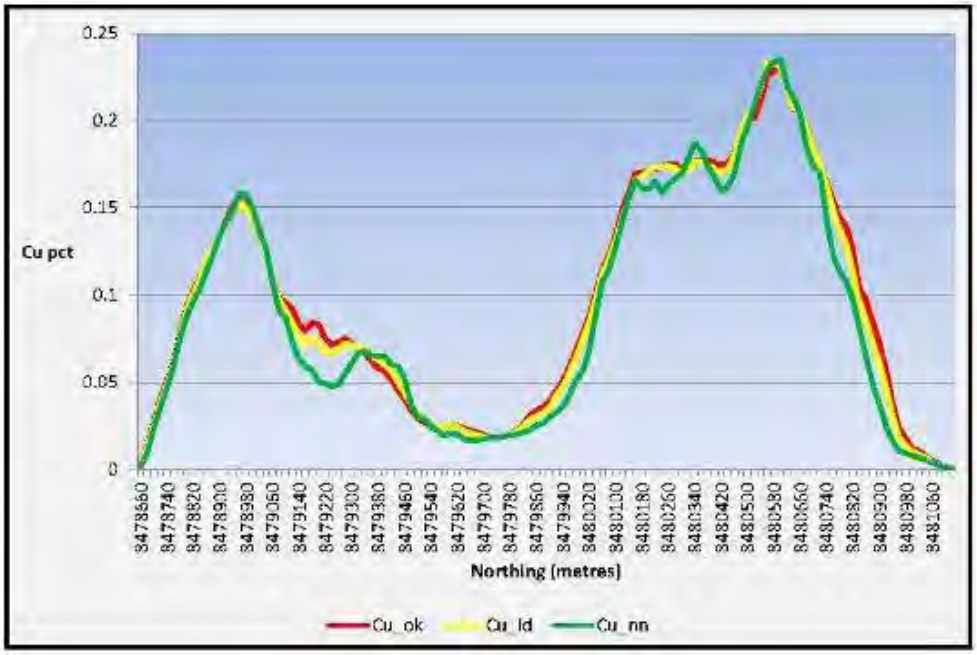
Note:Clipping distance is ±50 m. Figure prepared by Tetra Tech, 2014

Figure 14-24: Copper Swath Plot by Easting (metres)



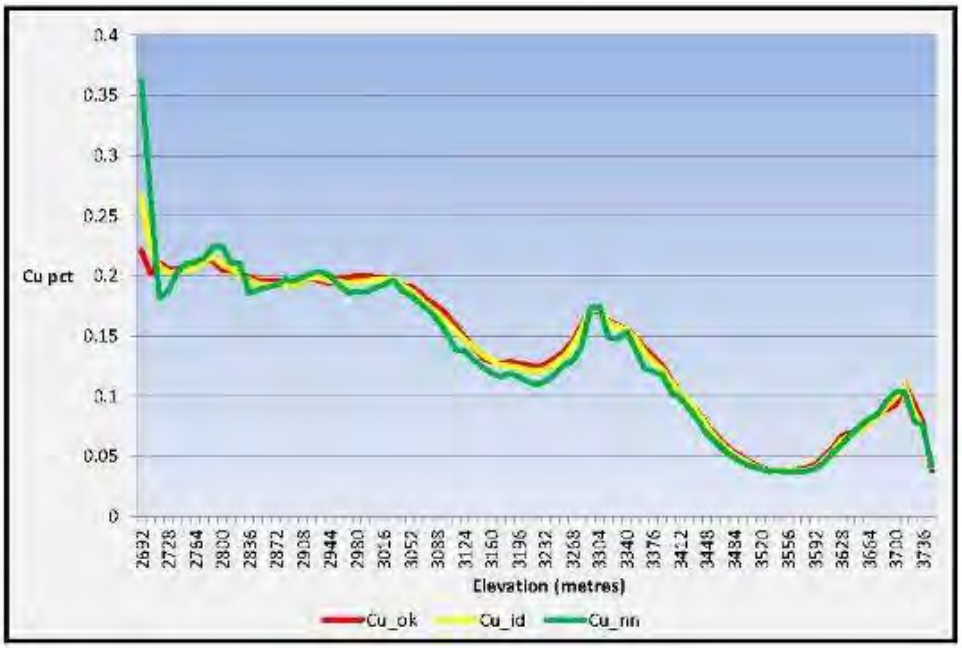
Note: Figure prepared by Tetra Tech, 2014

Figure 14-25: Copper Swath Plot by Northing (metres)



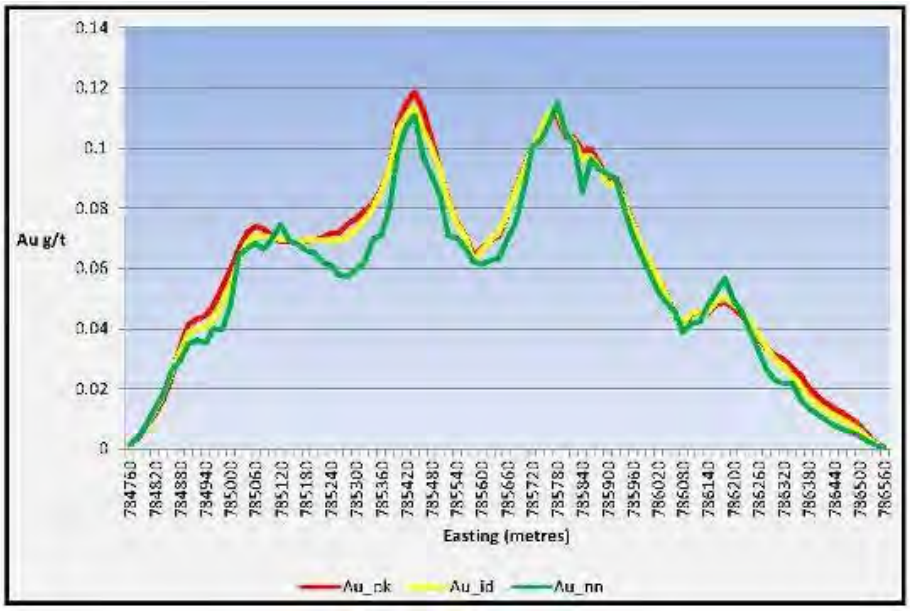
Note: Figure prepared by Tetra Tech, 2014

Figure 14-26: Copper Swath Plot by Elevation (metres)



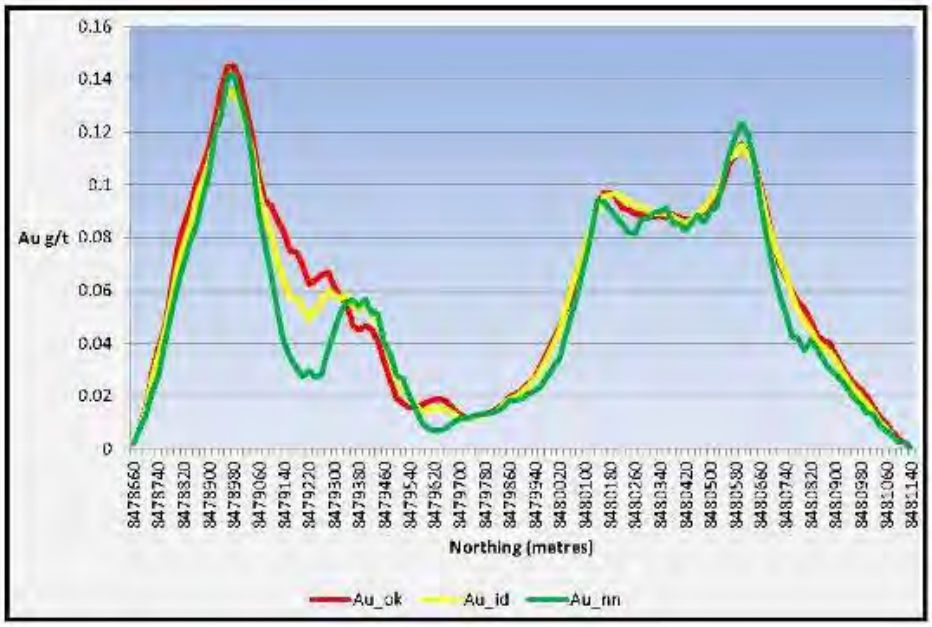
Note: Figure prepared by Tetra Tech, 2014

Figure 14-27: Gold Swath Plot by Easting (metres)



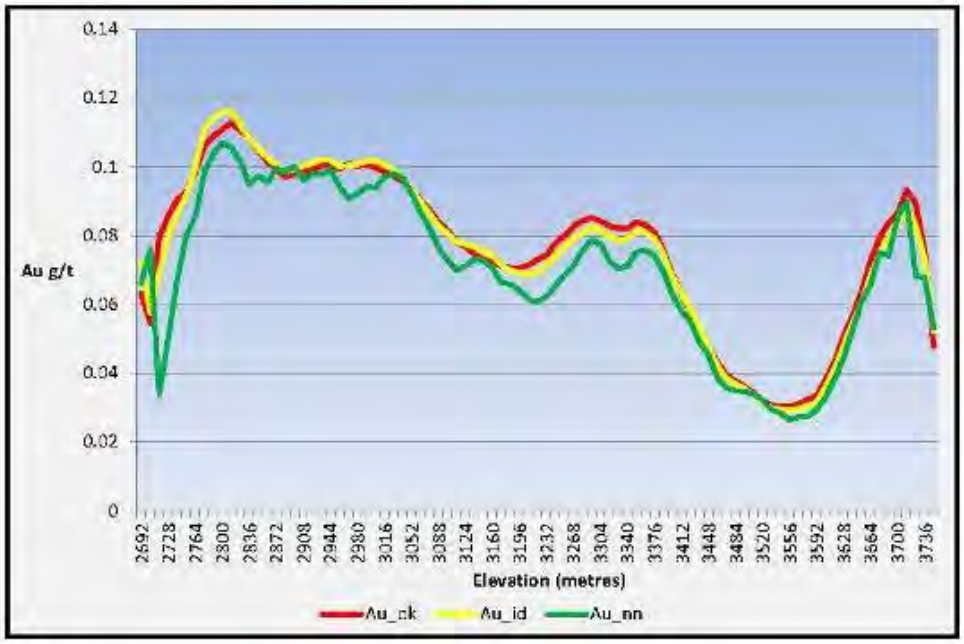
Note: Figure prepared by Tetra Tech, 2014

Figure 14-28: Gold Swath Plot by Northing (metres)



Note: Figure prepared by Tetra Tech, 2014

Figure 14-29: Gold Swath Plot by Elevation (metres)



Note: Figure prepared by Tetra Tech, 2014

14.1.9 Mineral Resource Classification

Introduction

Mineral resource classification is the application of Measured, Indicated and Inferred categories, in order of decreasing geological confidence, to the resource block model.

These categories are applied in consideration of, but not limited to, drill and sample spacing, QA/QC, deposit-type and mineralization continuity, surface and/or underground mineralization exposure, variography, KE, ZZ* and/or prior mining experience. With respect to resource classification of the Cotabambas deposit, Wright and Colquhoun (2012) previously classified the entire resource as Inferred.

In this resource model, the KE and ZZ* are examined in conjunction with variography ranges in the first search pass to assign an Indicated resource class. The remaining estimated resource is assigned an Inferred status.

Kriging Efficiency and Theoretical Slope of Regression

Conditional bias is the systematic under- and over-valuation of block estimates in different grade categories (Figure 14-30). Krige (1996) presented a practical analysis of the effects of spatial continuity and the available data within the search ellipse as it affects measures of conditional bias.

The two parameters Krige suggested using to investigate whether the block size used for grade estimation is appropriate are KE (KE as a percentage) and ZZ* which can also be used to calibrate the confidence in block estimates and are given as follows:

- KE = (BV-KV)/BV
- ZZ* = BV-KV+ |μ|

Where:

BV = theoretical variance of blocks within domain

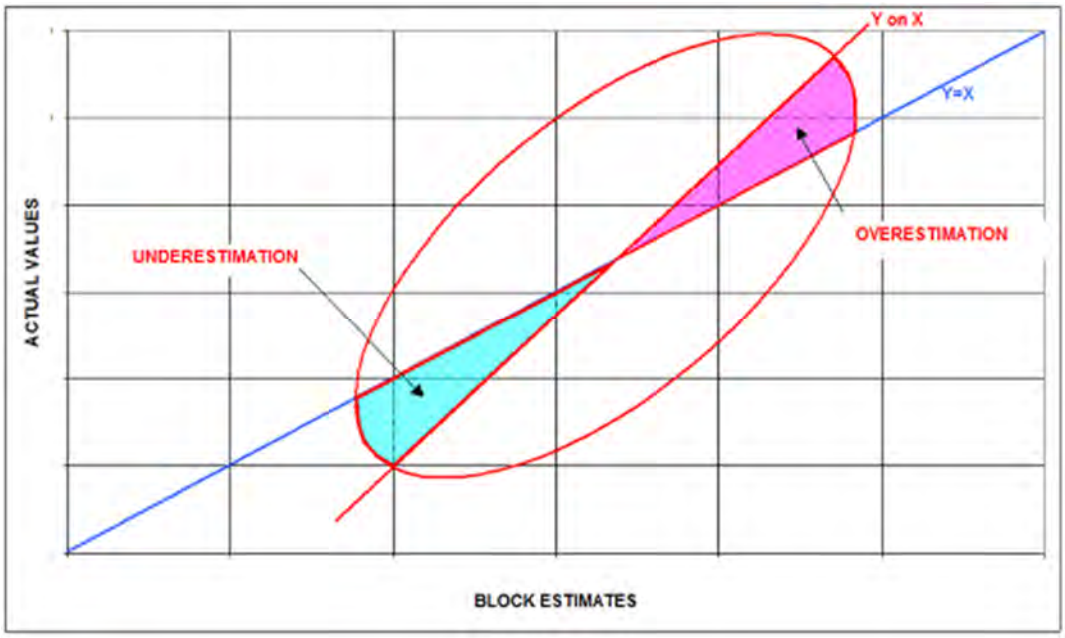
KV = variance between Kriged grade and true (unknown) grade, i.e. kriging variance

|μ| = LaGrange multiplier

Perfect estimation would give values of KV = 0, KE = 100% and ZZ*=1.

Confidence in the geological framework is all important and generally takes precedence over any mathematical indicator of confidence. However, KE and ZZ* can be used to identify “challenged” estimated areas within a specified resource classification which require further investigation. Ultimately, KE and ZZ* are both tools to be used in conjunction with block size, drill spacing, mineralization continuity and geological confidence.

Figure 14-30: Actual Value (Z) versus Estimated Value (Z*)



Note:Slope of regression is expressed as ZZ*. Figure prepared by Tetra Tech, 2014.

With respect to the Cotabambas resource model, the LaGrange multiplier and F-Function were estimated into each cell as a function of the copper drill hole data and associated variography. Thus drill spacing, sample support and variography all were intrinsically involved in the assigning of resource classification through the application of KE and ZZ*.

A solid wireframe was manually generated to encompass fairly contiguous cells which were successfully interpolated in the first copper search pass (svol=1) with KE greater than 30% (refer to Figure 14-25). A cross-section of this relation is provided in Figure 14-31.

Table 14-13 summarizes the KE and ZZ* for the block model based on Inferred and Indicated cells. There are significantly higher KE and ZZ* cells for the Indicated resource than the Inferred resource confirming a higher quality of estimate.

Mineral Resources within a Conceptual Pit Shell

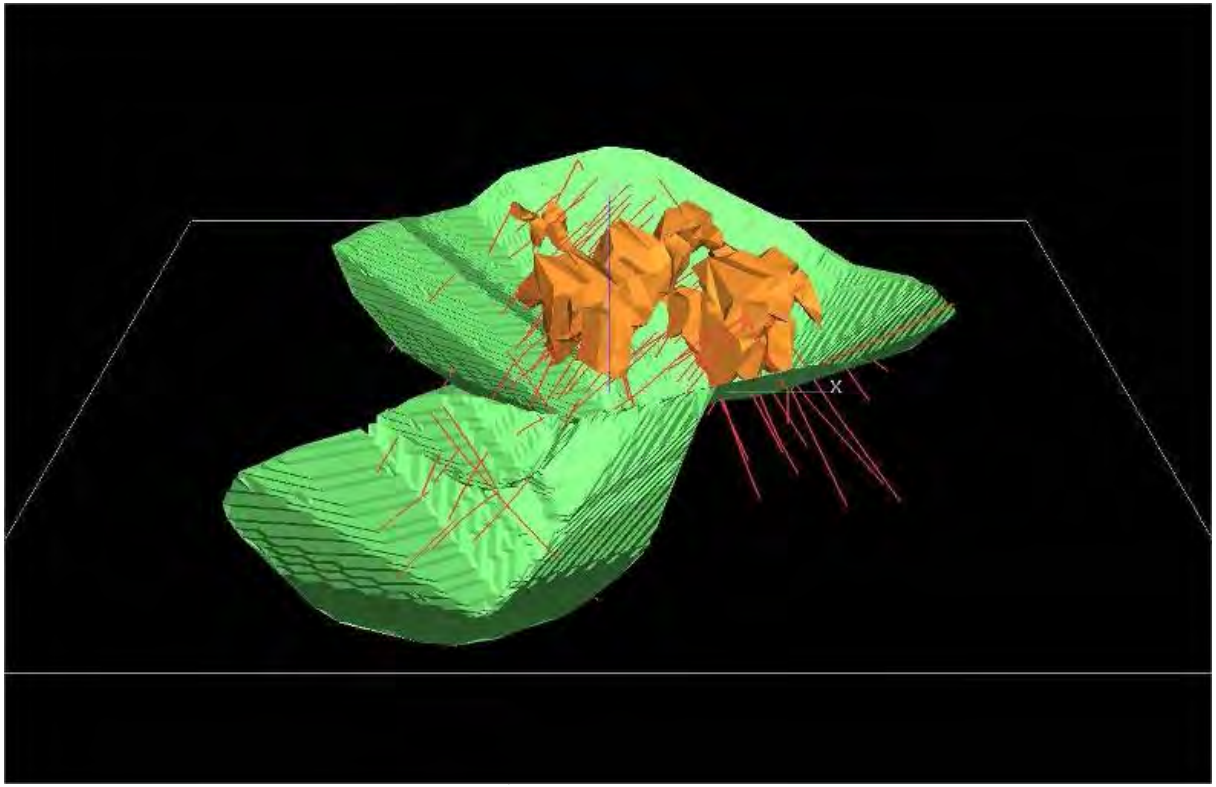
Mineral resources of Cotabambas were constrained by a conceptual pit shell. This pit shell was generated using Gemcom Whittle® software. Input parameters for the Gemcom Whittle® pit optimization are tabulated in Table 14-14.

Table 14-13: Cotabambas Model Statistics for KE and ZZ* by Resource Category

Resource Category	Field	Records	Samples	Minimum	Maximum	Mean	Variance	SD	CV
2	KE	192,886.00	18,489.00	-58.81	94.52	43.27	612.20	24.74	0.572
	ZZ*	192,886.00	18,489.00	-182.36	17.00	0.82	2.25	1.50	1.829
3	KE	192,886	174,397	-64.55	92.66	-5.96	927.38	30.45	-1.8187
	ZZ*	192,886	174,397	-48175	723.04	0.04	13322	115.42	5.0308

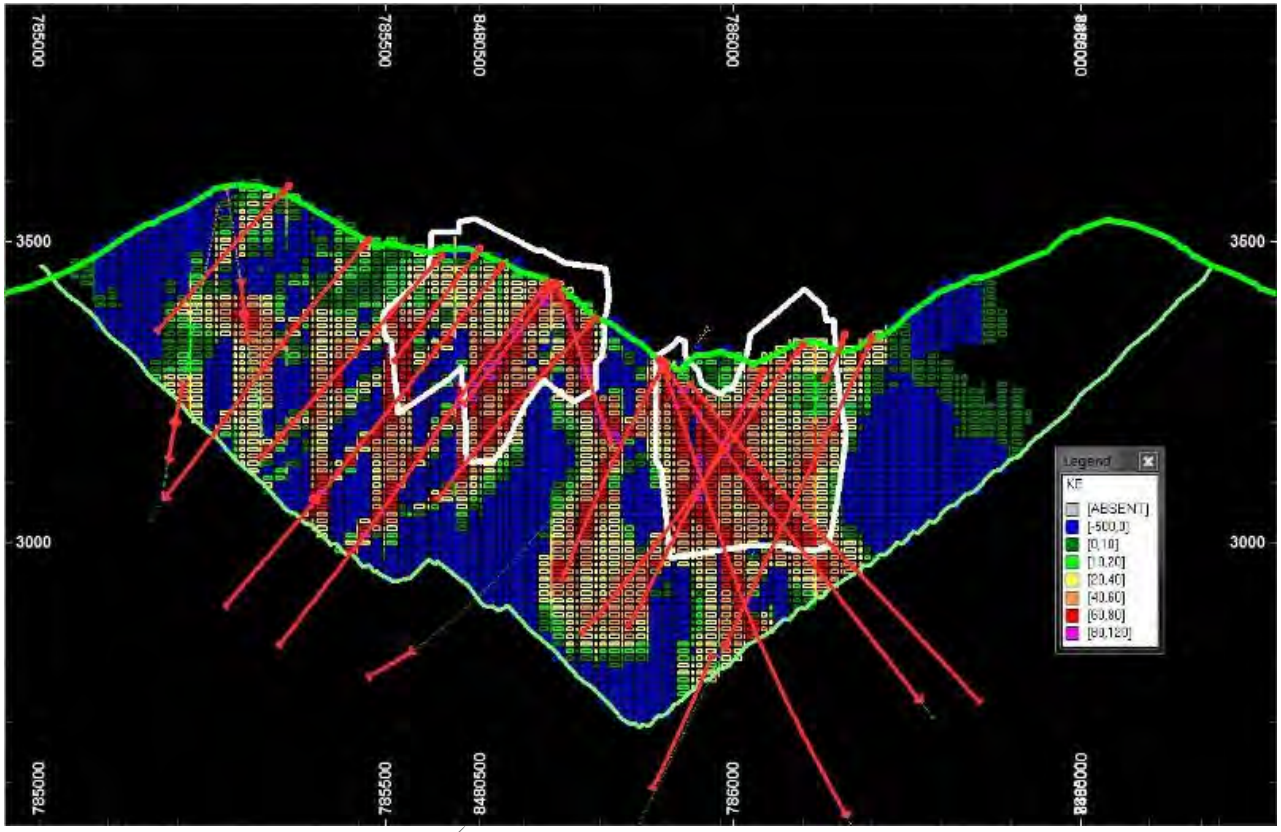
Note:Resource Category 3 = Inferred, Resource Category 2 = Indicated

Figure 14-31: Resource Indicated Classification Wireframe within Conceptual Pit Shell



Note:Isometric view looking north. Figure prepared by Tetra Tech, 2014

Figure 14-32: Resource Indicated Classification Wireframe (White) within Conceptual Pit Shell



Note:Northwest–southeast section within ±50 m clipping distance. Regularized block model coloured by KE and drill hole traces coloured red. Figure prepared by Tetra Tech, 2014

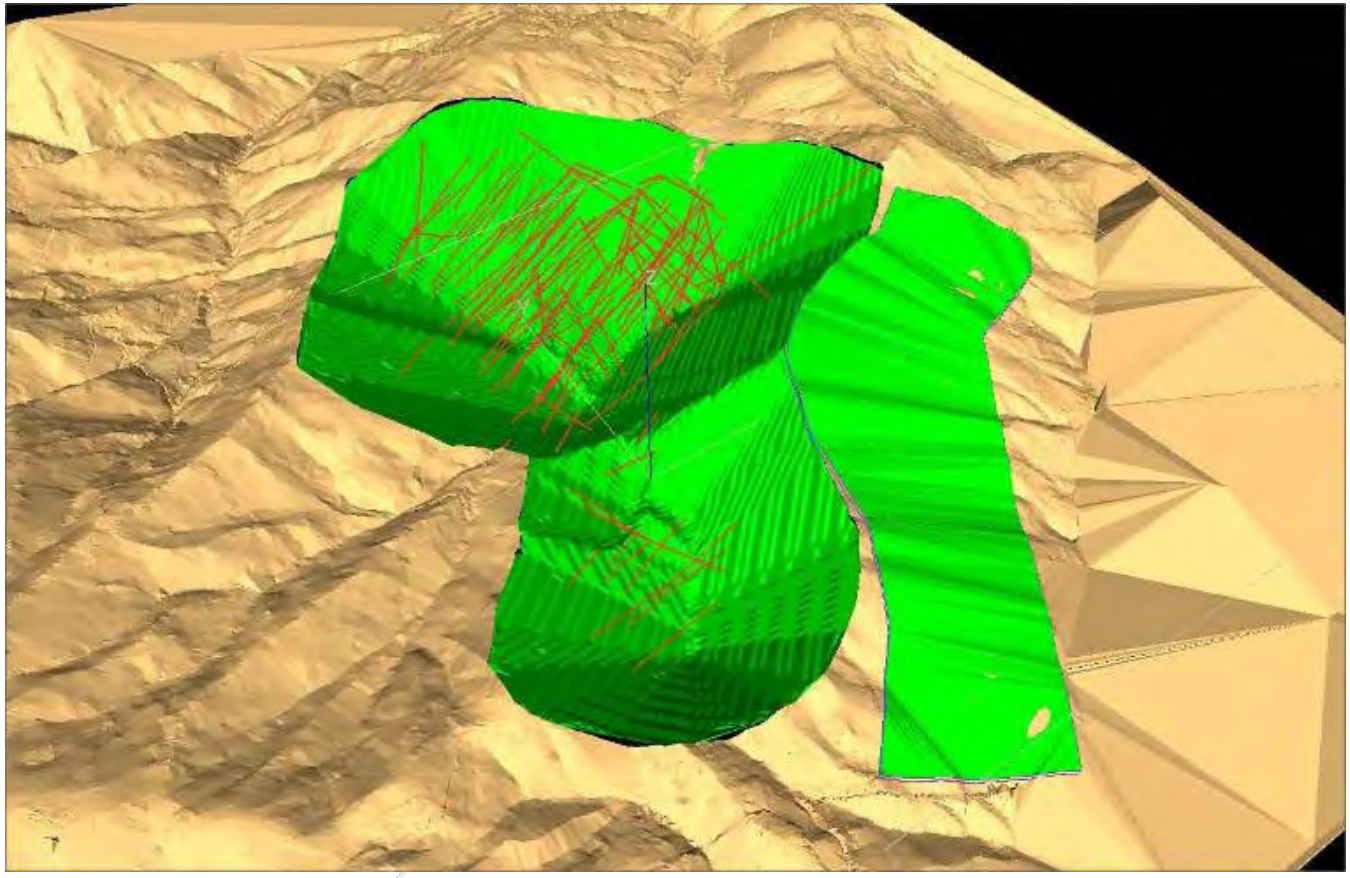
Table 14-14: Pit Optimization Input Parameters

Item	Unit	Parameters
Discount Rate	%	8.00
Selling Price	-	-
Au	US$/troy oz	1,350.00
Cu	US$/lb	3.20
Mo	US$/lb	12.50
Ag	US$/troy oz	23.00
Metal Recovery Rate	-	-
Au	%	62.00
Cu	%	90.00
Mo	%	40.00
Ag	%	64.00
Selling Cost	%	5
Processing Throughput	t/d	80,000
Mining Recovery Rate	%	97.00
Mining Dilution Rate	%	3.00
Mining Cost	-	-
Rock	US$/t mined	1.90
Processing Cost	-	-
Mill cost	US$/t milled	4.72
Additional Cost for Mineral Resource	US$/t milled	0.15
G&A Cost	US$/t milled	1.11
Mill Feed Material Handling Cost	US$/t milled	0.32
Environmental Cost	US$/t milled	0.50
Total Processing Cost	US$/t milled	6.80
Overall Pit Slope Angles	-	-
Rock	degree	45

Pit shell selection is based on a conceptual break-even scenario where the total costs equal that of the metal value gained. For this scenario, the pit was also constrained by the perimeter of the town of Cotabambas such that the pit would not encroach upon the town. This scenario is depicted in graphical form in Figure 14-33.

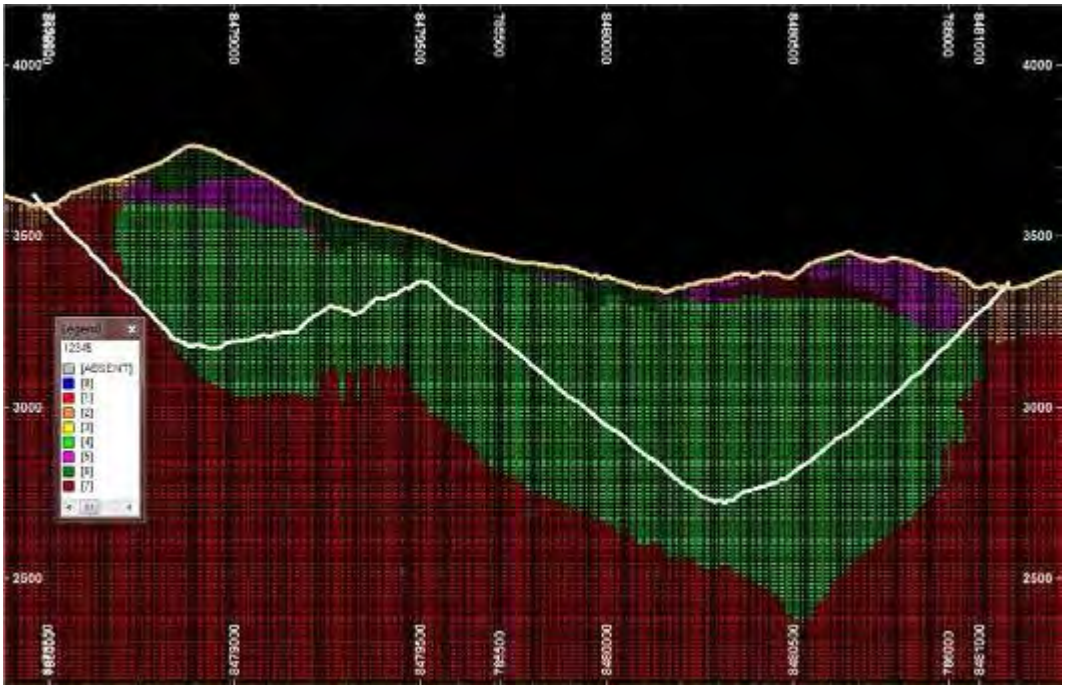
Figure 14-34 depicts the conceptual pit shell used in the resource estimate. The regularized block model shown is coloured by ZONE.

Figure 14-33: Isometric View of "Break-Even" Pit Looking to the Northeast with Exclusion Area



Note: Exclusion area of the Town of Cotabambas shown in green to the southeast. Figure prepared by Tetra Tech, 2014

Figure 14-34: Conceptual Pit Shell Cross-section (south-southwest to north-northeast)



Note: Figure prepared by Tetra Tech, 2014

14.1.10 Mineral Resource Tabulation

The Cotabambas mineral resources are entirely within the conceptual pit shell, as described immediately above. The mineral resource does not include interpolated cell outside the conceptual pit shell. This mineral resource is tabulated using all the OK interpolated grades in the resource model.

The estimate uses gold, silver and molybdenum recovery to report a CuEq grade.

CuEq cut-offs were used. These cut-offs are a function of metal price and recoveries. In the resource, estimated gold, silver and molybdenum are then converted to US dollars and combined. The combined figure is re-converted to copper and added to the in situ copper values.

The following metal prices are used:

- Copper – US$3.20/lb
- Gold – US1,350/troy oz
- Silver – US23.00/troy oz
- Molybdenum – US$12.50/lb.

Recoveries

The following metal recoveries were applied to the resource:

- Molybdenum – 40%
- Gold – 64%
- Silver – 63%
- No recovery is applied to copper.

Resource Tables

At a 0.2% CuEq cut-off, Tetra Tech's 2013 resource model (this report) estimates an Indicated Resource of 117 Mt at 0.42% copper, 0.23 g/t gold and 2.74 g/t silver, and an Inferred Resource of 605 Mt at 0.31% copper, 0.17 g/t gold and 2.33 g/t silver. The Mineral Resource estimate is summarized in Table 14-15.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Qualified Person for who prepared the estimate in Table 14-15 is Dr Robert Morrison, P.Geo., an employee of Tetra Tech at the time the estimates were completed. The estimate has an effective date of 20 June, 2013. The Qualified Person for the estimate for the purposes of this Report is Mr Luis Vela, CMC, an employee of Panoro.

14.2 Model Re-coding

Subsequent to the completion of the Mineral Resource estimate, Tetra Tech was requested to recompile the tabulation to accommodate sub-categorisation of transitional mineralisation to allow different metallurgical recovery estimates to be assigned to the sub-categories.

Amec Foster Wheeler reviewed the available sequential copper solubility data in the oxide zone to assess the opportunity to include some mixed material (MIX2) that might be considered as a suitable feed to the concentrator as a blend with sulphide mill feed. As a result of the review, Tetra Tech applied an equation that was derived by Amec Foster Wheeler to the existing block model in order to report the oxide mineralisation in greater detail.

Table 14-15: Mineral Resource Table

Resources Category	Zone	Cut-Off Grade% Cueq	Million Tonnes	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu (Blb)	Au (Moz)	Ag (Moz)	Mo (Mlb)
Indicated	Hypogene Sulphide	0.2	84.2	0.37	0.21	2.73	0.0018	0.69	0.58	7.39	3.43
	Supergene Sulphide	0.2	8.9	0.73	0.31	3.07	-	0.14	0.09	0.88	0.01
	Oxide Copper-Gold	0.2	23.8	0.49	0.24	2.63	-	0.26	0.18	2.01	0.01
	Oxide Gold	Na	0.2	-	0.66	3.74	-	-	0	0.02	-
	Total		**117.1**	**0.42**	**0.23**	**2.74**	**0.0013**	**1.09**	**0.86**	**10.3**	**3.45**
Inferred	Hypogene Sulphide	0.2	521	0.29	0.18	2.41	0.0021	3.36	2.94	40.35	24.22
	Supergene Sulphide	0.2	7.4	0.73	0.18	1.93	0.0007	0.12	0.04	0.46	0.11
	Oxide Copper-Gold	0.2	75.8	0.41	0.15	1.82	0.0003	0.68	0.37	4.44	0.5
	Oxide Gold	Na	1.2	-	0.61	3.27	-	-	0.02	0.12	-
	Total	0.2	**605.3**	**0.31**	**0.17**	**2.33**	**0.0019**	**4.16**	**3.38**	**45.37**	**24.83**

Note: Mineral Resources have an effective date of June 20, 2013 and were prepared by Qualified Person Robert Morrison, P.Geo. (APGO, 1839). The Qualified Person for the estimate is Mr Luis Vela, CMC, a Panoro employee. The estimate is based on 56,813 meters of drilling by Panoro and 9,923 meters of drilling from legacy campaigns. Copper equivalent (CuEq) is calculated using the equation: CuEq = Cu + 0.4422 Au + 0.0065*Ag, based on the differentials of long range metal prices net of selling costs and metallurgical recoveries for gold and copper and silver. Mineralization would be mined from open pit and treated using conventional flotation and hydrometallurgical flow sheets. Rounding in accordance with reporting guidelines may result in summation differences. CuEQ cut-offs were used to report almost all of the resource. These cut-offs are a function of metal price and recoveries. In the in situ resource, estimated gold, silver and molybdenum are then converted to US dollars and combined. The combined funds are re-converted to copper and added to the in situ copper values. The following metal prices are used: copper – US$3.20/lb; gold – US1,350/troy oz; silver – US23.00/troy oz; molybdenum – US$12.50/lb. The following metal recoveries were applied to the in situ resource: molybdenum – 40%; gold – 64%; silver – 63%. As the resource is reported as in situ, no recovery is applied to copper.

The equation was:

- (CuCN+CuR) / (CuAS+CuCN+CuR) >0.5

Where:

CuCN: Cyanide leach grade Variable Estimated

CuR: Flotation recovery grade Variable Estimated

CuAS: Acid /each recovery Variable Estimated

The coding was applied in three passes:

- Pass 1-Category 1: (CuCN+CuR) / (CuAS+CuCN+CuR)>0.5
- Pass 2 - Any block not in category 1, and where Au>0.25 g/t
- Pass 3 - Any remaining blocks in the zone.

The Indicated and Inferred Mineral Resource re-tabulation is presented in Table 14-16 and Table 14-27. The re-tabulation was prepared by Mr Joe Hirst, an employee of Tetra Tech. The re-tabulation has an effective date of 7 July 2014.

The Qualified Person for the retabulation is Mr Luis Vela, CMC, a Panoro employee.

Readers are cautioned that these tables are re-tabulations of the estimate in Table 14-15 and are not additive to that table. Key parameters and assumptions that apply to Table 14-15 also apply to the re-tabulations. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.3 Factors That May Affect the Mineral Resource Estimate

Factors which may affect the Mineral Resource estimate include:

- Commodity price and exchange rate assumptions
- Pit slope angles and other geotechnical and hydrogeological factors
- Assumptions used in generating conceptual pit shell used to constrain the estimate, including metal recoveries, and mining and process cost assumptions
- Ability to permit and operate the Project
- Ability to obtain and maintain the social licence to construct and operate the Project.

Table 14-16: Re-tabulated Indicated Mineral Resource Estimate

Zone	Subzone	Cut-off Grade (% CuEQ)	Tonnes (Million)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
Hypogene Sulphide		0.2	54.2	0.37	0.21	2.73	0.0018
Supergene Sulphide		0.2	8.9	0.73	0.31	3.07	—
Oxide Copper–Gold	Oxide Copper	0.2	5.8	0.61	0.12	2.16	0.0001
	Mixed	0.2	14.1	0.38	0.24	2.63	—
	Oxide Copper–Gold	0.2	3.9	0.70	0.41	3.38	—
Oxide Gold		n/a	0.22	—	0.64	3.80	—
Total		0.2	117.1	0.42	0.23	2.74	0.0013

Table 14-17: Re-tabulated Inferred Mineral Resource Estimate

Zone	Subzone	Cut-off Grade (% CuEQ)	Tonnes (Million)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
Hypogene Sulphide		0.2	521.0	0.29	0.108	2.41	0.0021
Supergene Sulphide		0.2	7.4	0.73	0.18	1.93	0.0007
Oxide Copper–Gold	Oxide Copper	0.2	25.8	0.51	0.10	1.47	0.005
	Mixed	0.2	44.6	0.35	0.16	1.92	0.0002
	Oxide Copper–Gold	0.2	3.5	0.64	0.42	2.78	0.0001
Oxide Gold		n/a	1.2	—	0.61	3.27	—
Total		0.2	603.5	0.31	0.18	2.33	0.0019

Note: Readers are cautioned that these tables are re-tabulations of the estimate in Table 14-15 and are not additive to that table. Key parameters and assumptions that apply to Table 14-15 also apply to the re-tabulations. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The Qualified Person for the re-tabulation is Mr Luis Vela, CMC, a Panoro employee.

15.0 MINERAL RESERVE ESTIMATES

This section is not relevant to this Report.

16.0 MINING METHODS

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary economic assessment results will be realized.

The updated PEA is based on a subset of the Indicated and Inferred Mineral Resources that have been considered in the mining method selection, mine design, and production scheduling. In this updated PEA, the waste footprint, pit designs, throughput and mining equipment remain the same. Updates done since the initial PEA report include an optimized mine plan, variable processing/stockpiling cut-off grade strategy and eliminating the waste conveyor and tunnel.

The current work is based on the resource model from the "Technical Report and Resource Estimate of the Cotabambas Copper-Gold Project, Peru", dated 7th July 2014.

This section outlines the parameters and procedures used to calculate an economic pit limit shape, design the open pit mine and waste rock storage facility (WRF) for Cotabambas, calculate the mill feed tonnes and establish a mining schedule to feed the process plant at a rate of 29.3 Mt/a (80,000 t/d).

Table 16-1 shows the breakdown of the subset of Mineral Resources included in the updated PEA mine plan by category.

Total waste is 604,193 ktkt for an average LOM strip ratio of 1.25. The mine plan uses a variable cut-off grade strategy to increase the mill feed grades in the earlier parts of the schedule by stockpiling marginal economic material. The stockpiled material is processed towards the end of the mine life. This strategy helps maximize the NPV and increase the internal rate of return (IRR) of the Project. The mine plan supports a mine production life of 17 years.

16.1 Topography

The current surface topography of the pit area was constructed by Seggistem SRL using the PSAD 1956 system. The surface topographies have an overall accuracy of ±1 m. This study was done during 2012–2013.

Table 16-1: Subset of Mineral Resources Included in Updated PEA Mine Plan

Class	Mill Feed ktkt	NSR $/t	Cu %	Au g/t	Ag g/t
Indicated	127,285	$21.14	0.37	0.21	2.58
Inferred	355,794	$17.79	0.30	0.17	2.30

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

16.2 Block Model and Dilution

The block model used has block dimensions of 20 m x 20 m in plan view; with each block having a height of 12 m. It is a whole block model with a single zone and grade value for each block. The present study does not consider mining dilution into the block model as a certain amount of dilution along the waste/mill feed boundary is already included in the whole block grades.

16.3 Throughput Rate

In order to determine an optimum plant capacity of the project, Amec Foster Wheeler has conducted a plant capacity study during the initial stages of the PEA. Costs for mining, processing and general administration are factored from Amec Foster Wheeler's project database. Recoveries are estimated based on historical test work. Economic parameters and metal prices are assumed based on data from similar recent projects. Four processing capacity options are investigated: 60,000, 80,000, 100,000 and 120,000 t/d. The study concluded that given the current resource estimate for Cotabambas, and agreed-upon parameters, 80,000 t/d is the economically optimum option. This capacity has been selected as the basis of the PEA design.

16.4 Pit Optimization

Pit optimization is developed using Whittle® software and is based on Indicated and Inferred Mineral Resources. The mineralization considered in the optimization is limited to the hypogene, supergene, mixed, and high Au grade materials. The other mineralization types are considered to be waste for PEA purposes.

In the pit optimization process, the area beneath the town of Cotabambas is not allowed to be mined. As a result, blocks below the town are not considered in the nested pit process.

16.4.1 Economic Pit Limit Parameters

Economic pit limit analysis considered metal prices which are shown in Table 16-2.

Table 16-2: Optimization Parameters

Item	Value	Units
Copper Price	3.25	US$/lb
Gold Price	1,300	US$/oz
Silver Price	20.5	US$/oz
Pay factor Cu	96.5	%
Pay factor Au	Variable	
Pay factor Ag	Variable	
Land freight	58.00	$/wmt-conc
Port Charges: Storage & handling	5.00	$/wmt-conc
Ocean freight	55.00	$/wmt-conc
Marketing & Other	0.60	$/dmt-conc
Insurance premium	0.40	$/wmt-conc
Treatment charge	85.00	$/dmt-conc
Refining Charges Cu	0.085	$/lbCu
Refining Charges Au	5	$/oz
Refining Charges Ag	2.1	$/oz
G&A Cost	0.4	US$/t
Discount Rate for NPV	7.5	%
Average Mining Cost	1.723	US$/t
Mining Cost Mill Feed	2.02	US$/t
Mining Cost Waste	1.5	US$/t

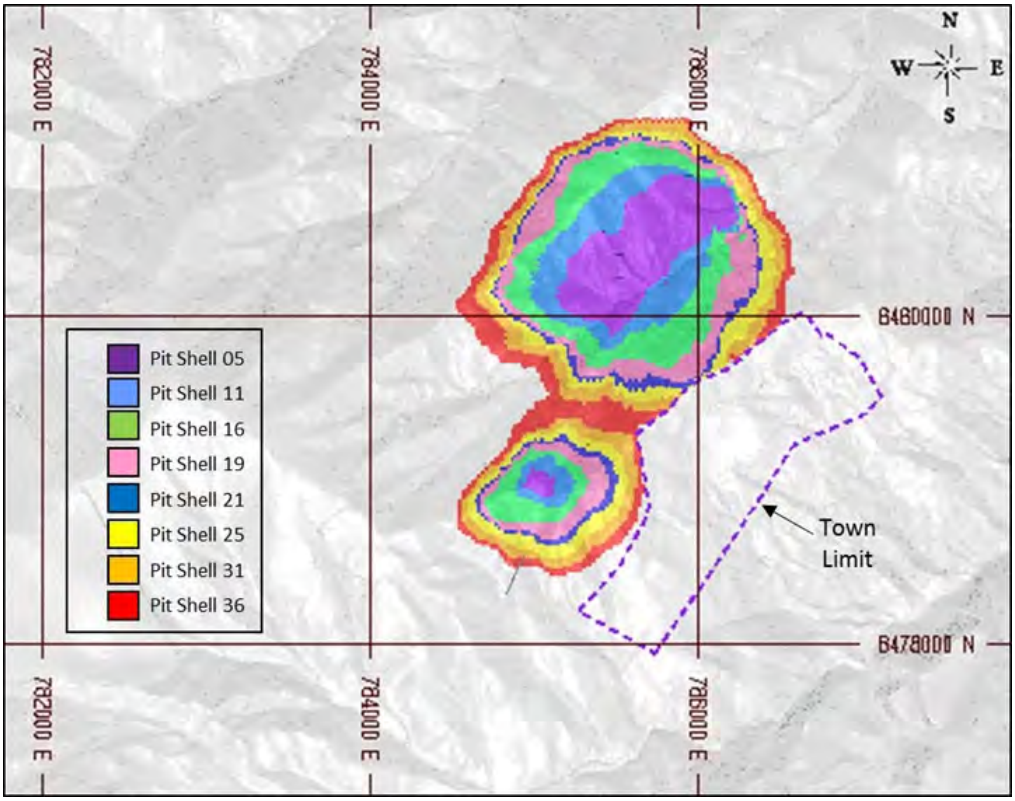
Note: metal prices used in the optimization differ from those used in the financial analysis.

The overall slope angle (OSA) used in the pit optimization is 42º. This angle is based on a benchmarking analysis of mines with similar characteristics and pit depths. It is recommended that a more detailed study should be conducted in the next stage of the Project when more information becomes available.

16.4.2 Pit Optimization Results

A total of 36 nested pit shells are generated in the pit optimization process for different revenue factors (RF), which varies from 0.3 to 1.0 using intervals of 0.02. Figure 16-1 shows a plan view of the nested pits generated in Whittle.

Figure 16-1: Nested Pit Shells



Note: Figure prepared by Amec Foster Wheeler, 2015

16.5 Ultimate Economic Pit Shell Selection

To select the ultimate economic pit shell, a skin analysis is performed. This methodology quantifies the impact of adding neighboring layers to the pit shell selected in terms of net present value (NPV). Based on this approach, a sequence of phases (based on the pit shells) is selected. The size of the last phase varies by producing a mine plan and a NPV for several sets of variations. The phase with the best economic benefit is selected as the ultimate pit shell.

Initially, pits 11, 16, and 19 are selected as mining phases. Therefore, pit 19 is designated as a preliminary ultimate pit shell. Subsequently, pit shells 20 to 36 are evaluated as potential ultimate pits after observing a significant NPV drop in pit shell 26, hence reducing the search range from pit shell 20 to pit shell 25 (RF=0.78).

However, between pit shells 20 and pit 25 a minimum mining width is not achieved to allow a new phase. Therefore, a second skin analysis is performed, this time starting from pit 19 to pit 25. This analysis includes the additional capital cost estimation for the tailings dam.

The pit final selection criterion considers two objectives: (1) achieve the best economic outcome and (2) maximize the mineable mineral resources. Therefore, the final pit selected is pit 21.

A sensitivity analysis of the main parameters that influence the definition of the final pit is performed, including: overall slope angles, mining cost, process cost and metallurgical recovery. The overall slope angle varies by ± 3° and the other parameters by ± 10%. The parameters that have the most impact on the Mineral Resources are, from highest to lowest, the metallurgical recovery, processing cost, and the overall slope angle.

16.6 Detailed Pit Design

The final pit design and the phase designs are developed with MineSight software. There has been no change in the PEA designs between the initial PEA and the update PEA.

16.6.1 Pit Design Parameters

Parameters for the pit design such as minimum mining width, ramp grade, bench height, inter-ramp slope angle and others are based on the mine equipment selected. Table 16-3 shows the design parameters used in the final pit and pit phase designs.

16.6.2 Final Pit Design

The final North and South pits are shown in Figure 16-2.

The bottom of North pit is at 2932 masl, with an overall wall height of 760 m at the west side, 715 m at the east wall and a minimum height of 316 m at the northeast wall. The main pit includes two accesses and two ramp sets developed over the north and south walls in order to decrease the risk of production stoppages if one ramp is blocked.

The bottom of the South pit is at 3412 masl, with a maximum wall height of 335 m at the west side and a minimum wall height of 170 m at northeast side of the pit. The South pit includes only one access and one ramp set due to its smaller size.

16.6.3 Phase Design

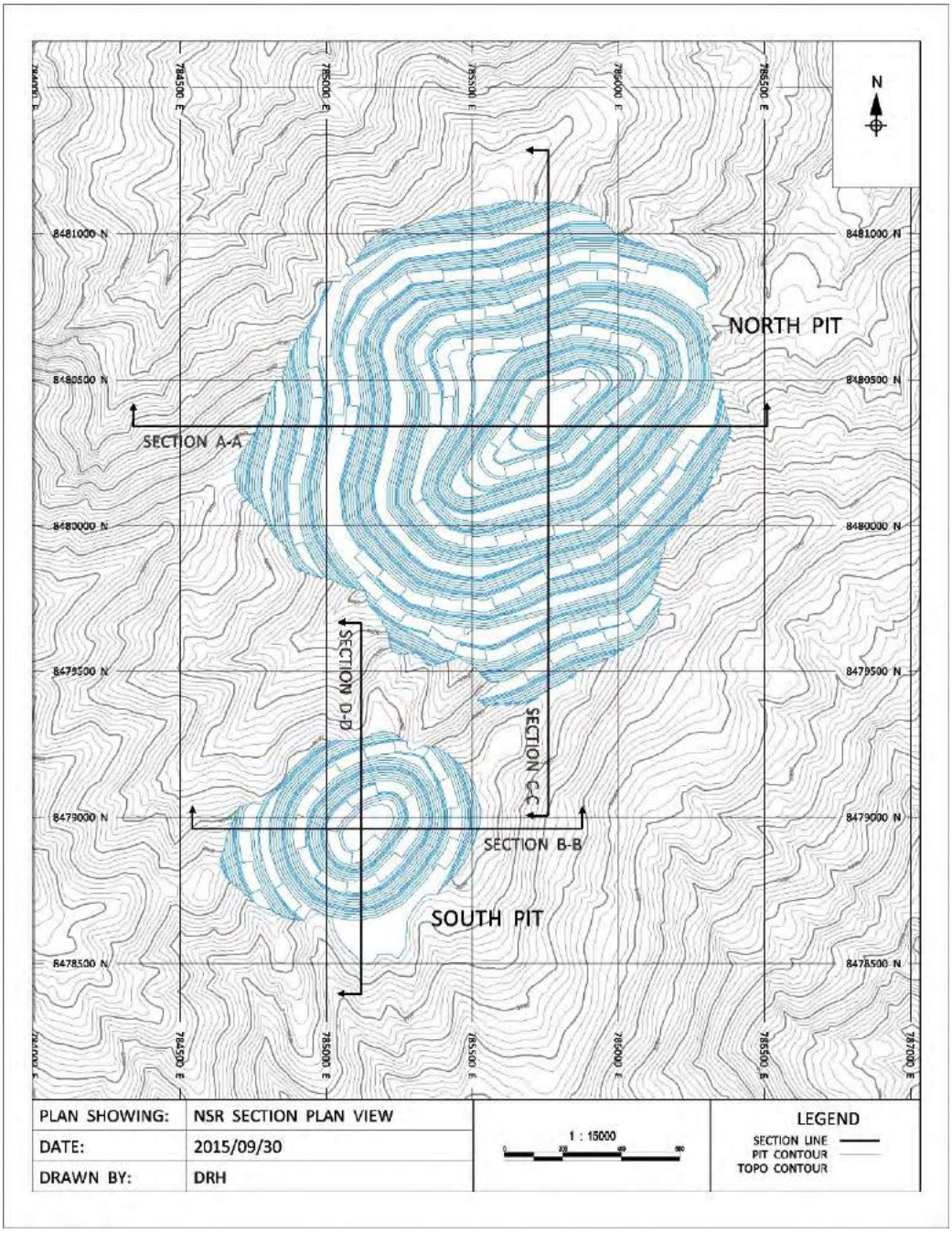
The selection and the design of the mining phases are based on the pit shells obtained in the optimization process and on the operational criteria exposed previously. Four and three phases are defined for the North pit and South pit, respectively, making a total of seven phases. Breaking the North and South pits into smaller phases allows a smoother progression of waste stripping and mill feed tonnes mined throughout the schedule.

Small, initial phases allow the mining schedule to target areas of highest economic return more quickly and increase the economics of the Project.

Table 16-3: Pit Design Parameters

Parameter	Unit	Value
Bench Height	m	12
Minimum Operational Mining Width	m	50
Minimum Production Mining Width	m	90
Ramp Width	m	33
Geotechnical Berm	m	33
Max Vertical Distance b/w geotechnical berms	m	120
Ramp Grade	%	10
Bench Face Angle	º	65
Inter-ramp Slope Angle	º	50
Minimum Berm width	m	4.5
Vertical Distance between berms	m	12

Figure 16-2: North and South Ultimate Pits



Note: Figure prepared by MMTS, 2015

16.6.4 Mill Feed and Waste Selection

The net smelter return (NSR value is used to determine whether blocks are waste or mill feed.

NSR (units are $/t of material) is calculated for each mineralized block in the resource model using the net smelter price (NSP), mill process recoveries and grade of each metal in the block. The NSP for each metal is the market price from Table 16-2, net of the smelting, refining and other offsite charges such as concentrate transportation. NSP represents the "metal price at the mine gate" after off-site costs are accounted for. Material inside the economic pit shell has mining and process costs accounted for, therefore any material with NSR value greater than the cost to process the material is economic and should be sent to the mill.

Table 16-4 shows the NSP prices used for each metal after smelter terms and costs, freight and distribution costs have been accounted for. The sum of all these costs is considered as off-site costs.

The process recoveries used vary by zone and are shown in Table 16-5.

An NSR value for each block is calculated using the following formula:

- NSR ($/t) = [Cu grade (%) * Cu Recovery (%) * 22.046 * $2.48/lb] + [Au grade (g/t) * Au Recovery (%) * $36/g] + [Ag grade (g/t) * Ag Recovery (%) * $0.50/g]

Material inside the pit limit with NSR>=$6/t is considered as mill feed while material with NSR<$6/t is considered as waste. The cut-off grade used is based on a small premium on top of the process + G&A cost of $4.80/t. This provides some conservatism in the mill feed tonnes used in the production schedule.

A summary of the waste/mill feed by pit phase is shown in Table 16-6.

Cross-sections (the locations of which were illustrated in Figure 16-2) showing distribution of NSR values are included as Figure 16-3 to Figure 16-6.

16.7 Mining Schedule

16.7.1 Proposed Mining Configuration

The proposed mining configuration consists of two open pits (North and South), to be mined using 12 m benches and a total of seven pit phases.

A waste rock facility with a maximum capacity of 369 Mm^3 (667 Mt) will be located in Guaclle Creek. Waste will be transported to the waste facility via trucks.

Mine infrastructure will include shops, warehouse, washing and welding bays, fuel stations, explosives magazine and administrative offices.

A layout plan showing the ultimate pits and waste dump location is included as Figure 16-7.

Table 16-4: Net Smelter Prices

Metal	Off-site Costs	NSP
Copper	$0.52/lb	$2.48/lb
Gold	$5.80/gram	$36/gram
Silver	$0.14/gram	$0.50/gram

Table 16-5: Process Recoveries by Zone

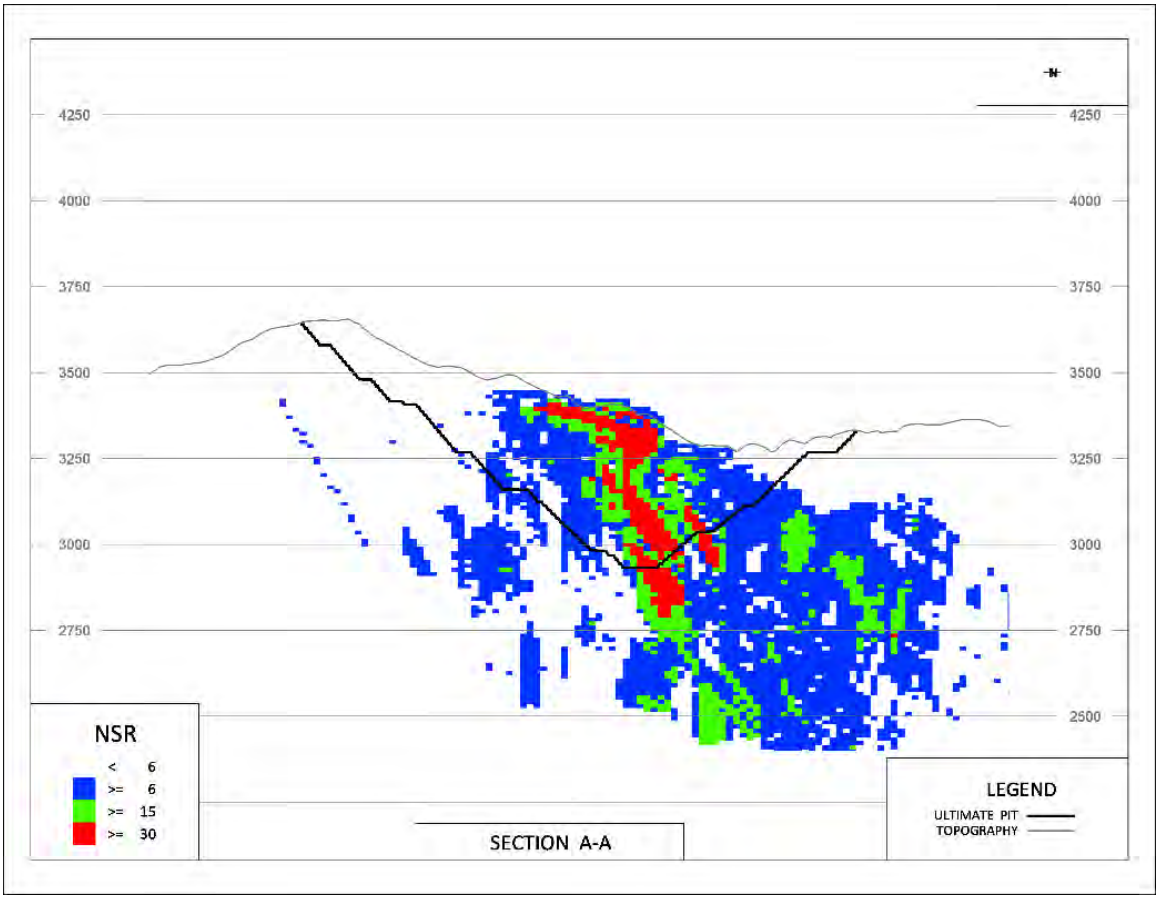
	Process Recoveries		
Zone	**Cu**	**Au**	**Ag**
Primary (hypogene)	87.5%	62%	60.4%
Enriched (supergene)	87.5%	62%	60.4%
Mixed	60%	55%	0%
Oxide	0%	65%	0%

Table 16-6: Summary of Waste/Mill Feed by Pit Phase

	Mill Feed - kt (NSR>=$6)	Waste - kt	S/R	In-situ grades (undiluted) Cu (%)	Au (g/t)	Ag (g/t)	NSR
Phase 1 North	57,765	47,104	0.82	0.533	0.283	2.775	26.01
Phase 2 North	82,922	121,587	1.47	0.336	0.168	2.174	19.00
Phase 3 North	41,094	155,702	3.79	0.213	0.109	1.924	12.80
Phase 4 North	240,114	192,610	0.80	0.297	0.156	2.417	18.27
North sub-total	**421,895**	**517,003**	**1.23**	**0.329**	**0.171**	**2.370**	**18.94**
Phase 1 South	22,486	25,983	1.16	0.305	0.273	3.003	17.84
Phase 2 South	18,390	23,027	1.25	0.229	0.188	2.164	15.40
Phase 3 South	20,308	38,179	1.88	0.243	0.214	1.957	17.00
South sub-total	**61,184**	**87,189**	**1.43**	**0.262**	**0.228**	**2.404**	**16.83**
TOTAL	***483,079***	***604,192***	***1.25***	***0.321***	***0.178***	***2.374***	***18.67***

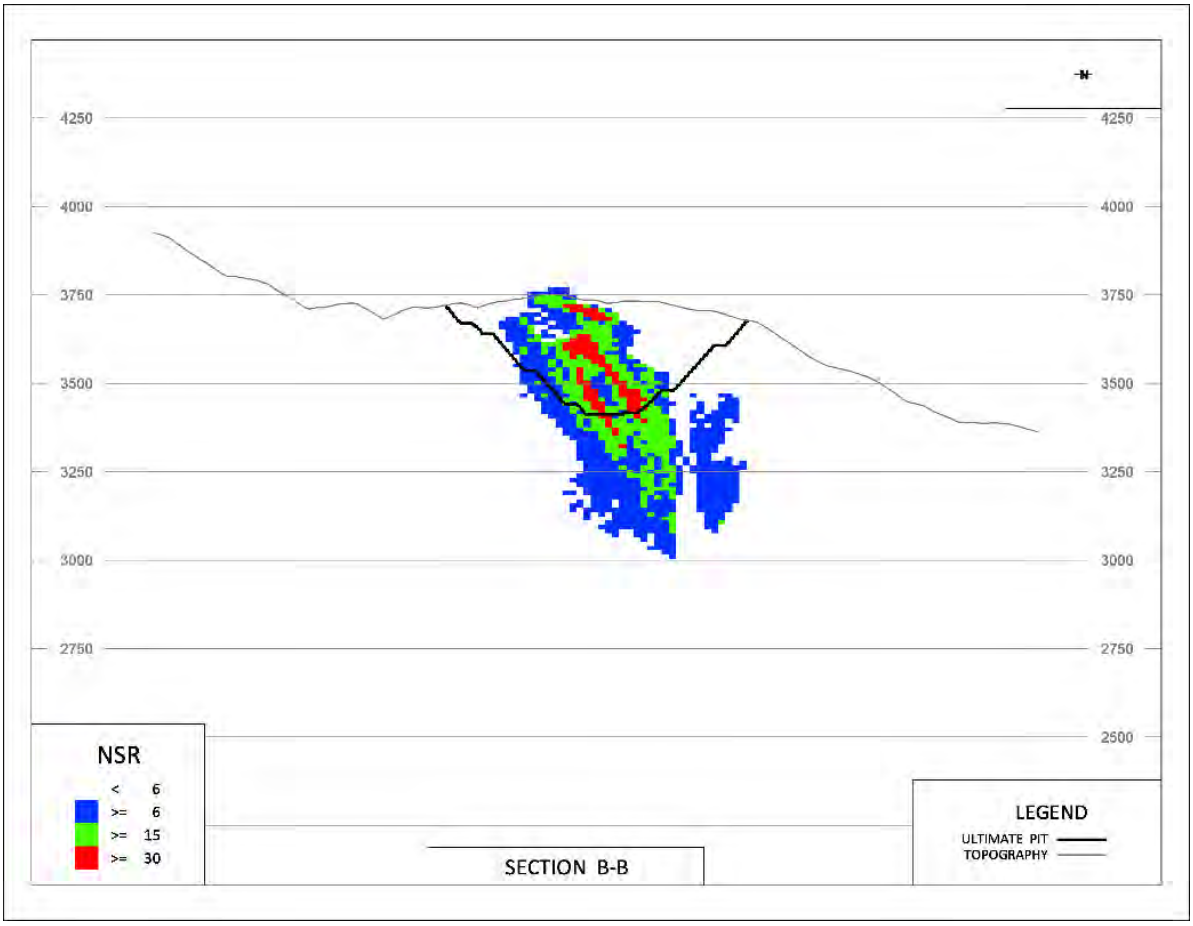
The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Figure 16-3: North Pit East–West Section A-A showing NSR Values



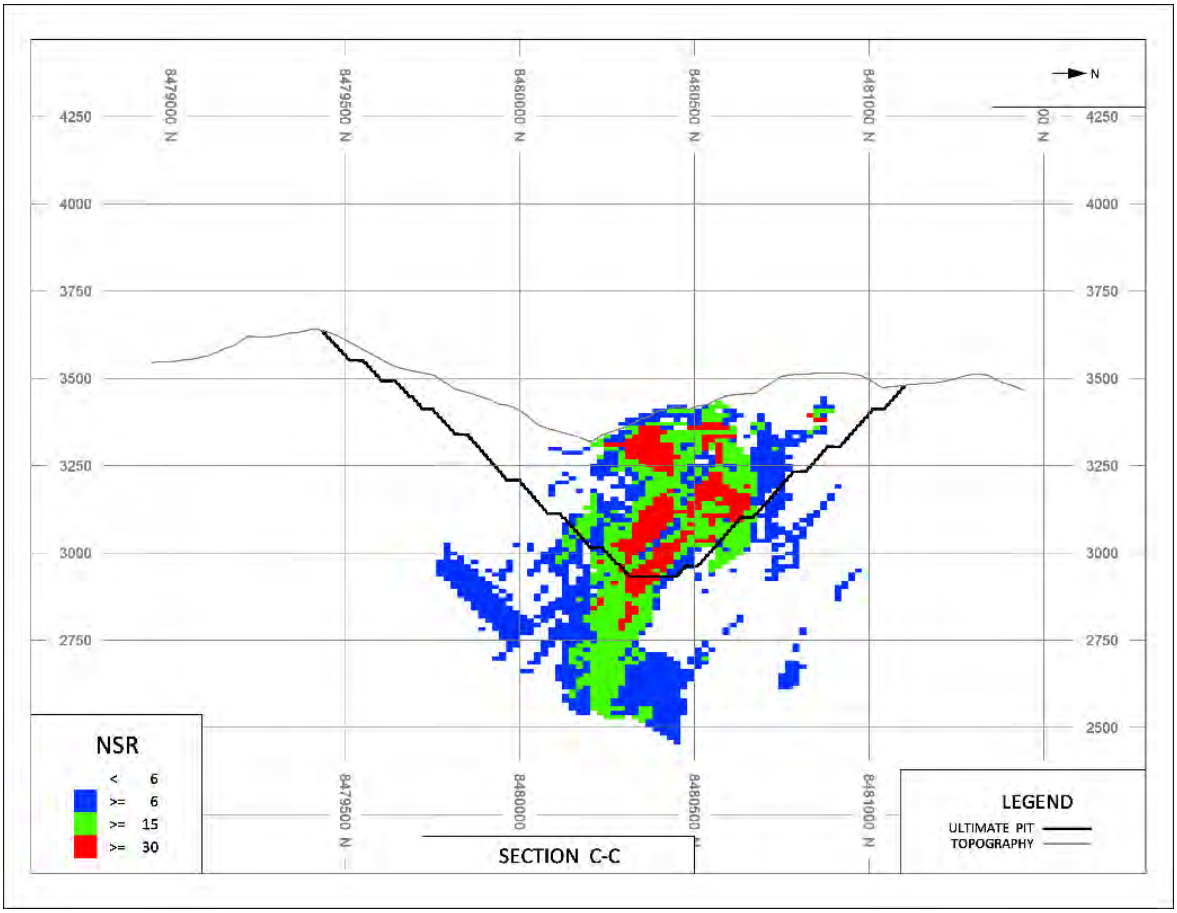
Note: Figure prepared by MMTS, 2015

Figure 16-4: South Pit East–West Section B-B showing NSR Values



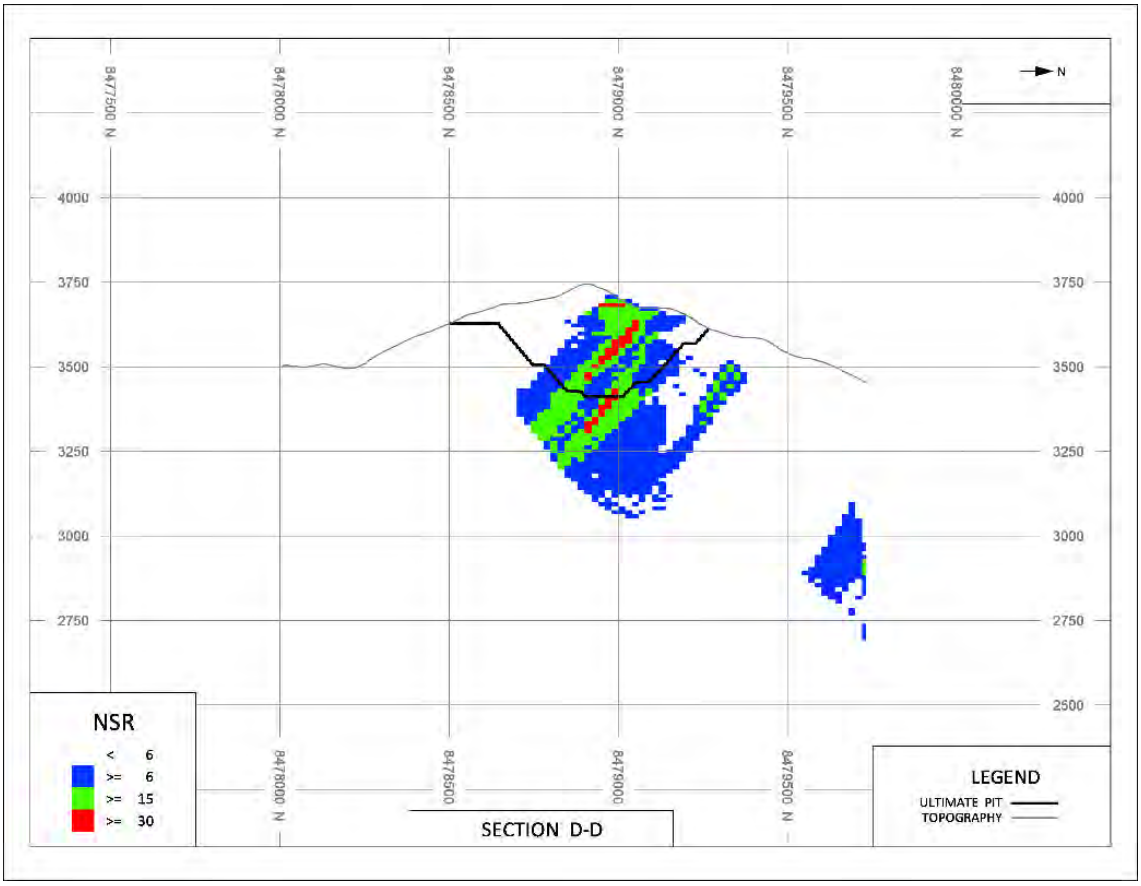
Note: Figure prepared by MMTS, 2015

Figure 16-5: North Pit North–South Section C-C showing NSR Values



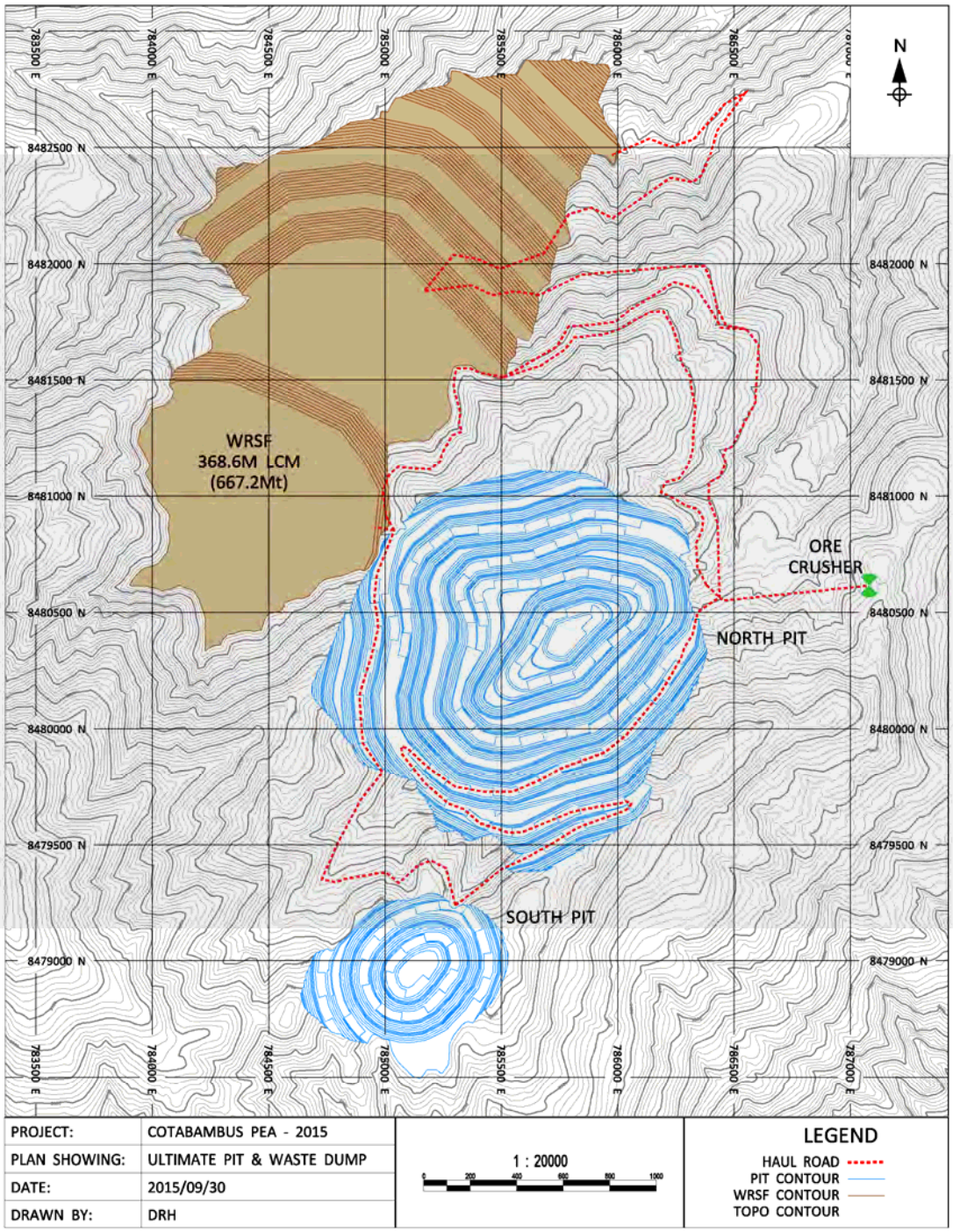
Note: Figure prepared by MMTS, 2015

Figure 16-6: South Pit North–South Section D-D showing NSR Values



Note: Figure prepared by MMTS, 2015

Figure 16-7: Ultimate Pits and Waste Rock Facility



Note: Figure prepared by Amec Foster Wheeler, 2015

16.7.2 Mining Sequence

The proposed Cotabambas mine schedule has a mine life of 18 years, comprised of 1 year of pre-stripping followed by 17 years of mill feed. Table 16-7 provides a summary of the proposed mine schedule. Mining strip ratio is calculated as tonnes of waste mined divided by tonnes of mill feed mined. Mining strip ratio does not include stockpile reclaim tonnes.

The production schedule is developed with the following criteria:

Annual mill feed of 80,000 t/d (29.3 Mt/a) with increased mill feed grades during the first years of production by stockpiling marginally economic material (results in higher material movement in the first years but an improved cash flow). Maximum bench advance (sinking rate) of twelve benches per year during production. During pre-stripping the sinking rate nominally reaches 15 but this includes the topmost 3-4 benches which are mined with dozers pushing the material downslope.

Figure 16-8 shows the proposed mine extraction plan by material movement category and Figure 16-9 shows the planned feed to plant by zone.

Mine production is planned to start in the North pit and the first phase of the South pit to target the best grades most quickly. The commencement of mining in each phase is as follows:

- Phase 1 North – during pre-stripping period
- Phase 2 North – during pre-stripping period
- Phase 3 North – Year 1
- Phase 4 North – Year 4
- Phase 1 South – during pre-stripping period
- Phase 2 South – Year 3
- Phase 3 South – Year 10

Mining in the North and South pits is completed at the end of Year 13.

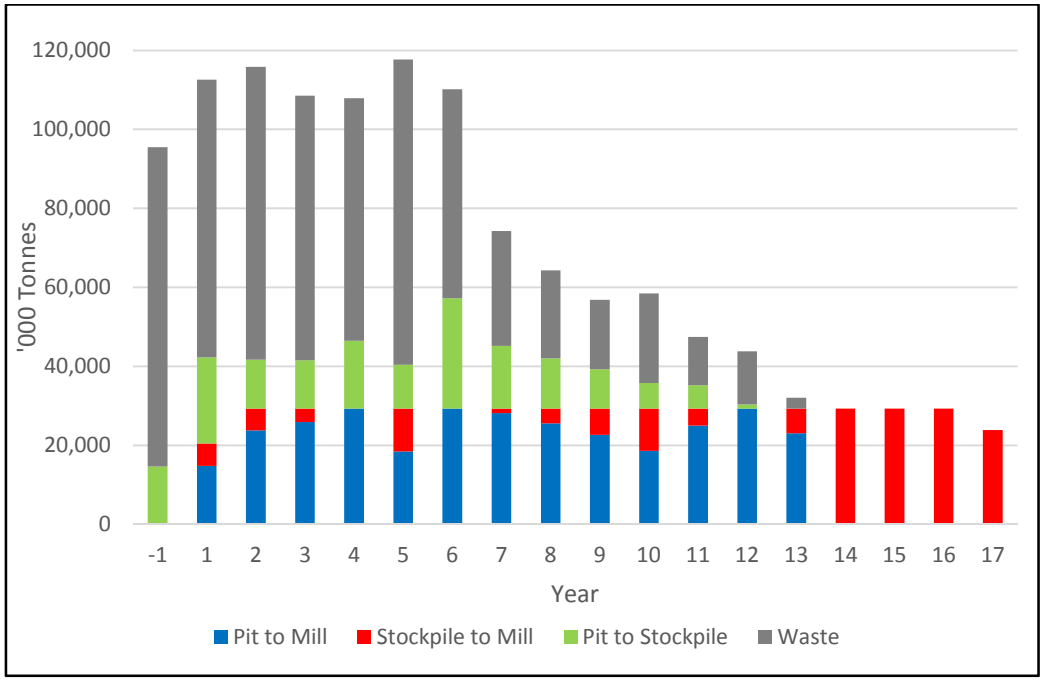
16.8 Waste Rock Storage Facility (WRF) and Waste Management

The design parameters for the WRF construction are shown in Table 16-8.

Table 16-7: Mining Schedule Summary

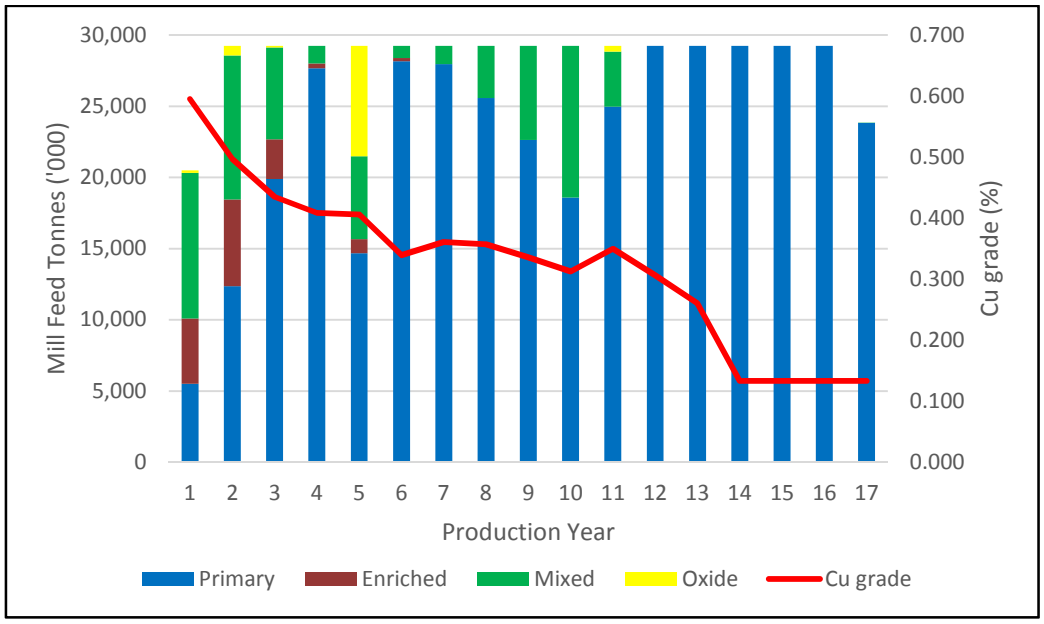
Year	kt Pit to Mill	Pit to Stockpile	Stockpile to Mill	Total Mill Feed	In-Situ NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mining S/R	kt Waste	Total Material	Stockpile Inventory
-1	11	14,624	—	—	—	—	—	—	5.53	80,880	95,515	14,624
1	14,760	21,755	5,730	20,490	33.15	0.60	0.38	3.33	1.93	70,366	112,611	30,650
2	23,740	12,443	5,510	29,250	26.43	0.50	0.24	2.54	2.05	74,158	115,851	37,583
3	25,860	12,286	3,390	29,250	24.56	0.43	0.22	2.50	1.76	66,981	108,517	46,478
4	29,250	17,181	—	29,250	25.19	0.41	0.24	2.63	1.32	61,502	107,934	63,660
5	18,398	11,151	10,852	29,250	16.21	0.41	0.22	2.42	2.62	77,277	117,678	63,958
6	29,250	27,971	—	29,250	20.14	0.34	0.16	2.34	0.93	52,946	110,166	91,929
7	28,142	15,886	1,108	29,250	22.99	0.36	0.23	2.89	0.66	29,118	74,254	106,707
8	25,576	12,774	3,674	29,250	22.40	0.36	0.22	2.94	0.58	22,236	64,261	115,807
9	22,623	9,993	6,627	29,250	20.24	0.34	0.19	2.72	0.54	17,573	56,816	119,172
10	18,597	6,502	10,653	29,250	17.79	0.31	0.16	2.40	0.91	22,747	58,500	115,022
11	24,959	5,987	4,291	29,250	20.49	0.35	0.18	2.52	0.39	12,215	47,452	116,719
12	29,250	1,100	—	29,250	19.29	0.31	0.17	2.45	0.44	13,425	43,775	117,819
13	23,008	—	6,242	29,250	17.25	0.26	0.19	2.08	0.12	2,769	32,019	111,577
14	—	—	29,250	29,250	8.42	0.13	0.07	1.69	—	—	29,250	82,327
15	—	—	29,250	29,250	8.42	0.13	0.07	1.69	—	—	29,250	53,077
16	—	—	29,250	29,250	8.42	0.13	0.07	1.69	—	—	29,250	23,827
17	—	—	23,827	23,827	8.42	0.13	0.07	1.69	—	—	23,827	0
TOTALS				***483,067***	***18.67***	***0.32***	***0.18***	***2.37***	***1.25***	***604,193***	***1,161,410***	

Figure 16-8: Material Movement by Year



Note: Figure prepared by MMTS, 2015.

Figure 16-9: Mill Feed by Zone



Note: Figure prepared by MMTS, 2015.

Table 16-8: Waste Rock Facility Design Parameters

	Units	Value
Swell Factor	%	34
Lift Height	m	96
Dump Face Angle	degrees	37
Wrap-Around/Berm Width	m	70
Final Slope Angle		2H:1V

Specific equipments for the foundation and site preparations for the WRF are considered. This is a precaution to clean and control the surface where the material will be placed, and thus prevent accumulation of material remaining on the hillsides that may be unstable. However, it must be noted that there is no geotechnical information available for the WRF foundation. Some criteria used in the design are based on and more detailed work in future studies may be able to design a lower cost solution.

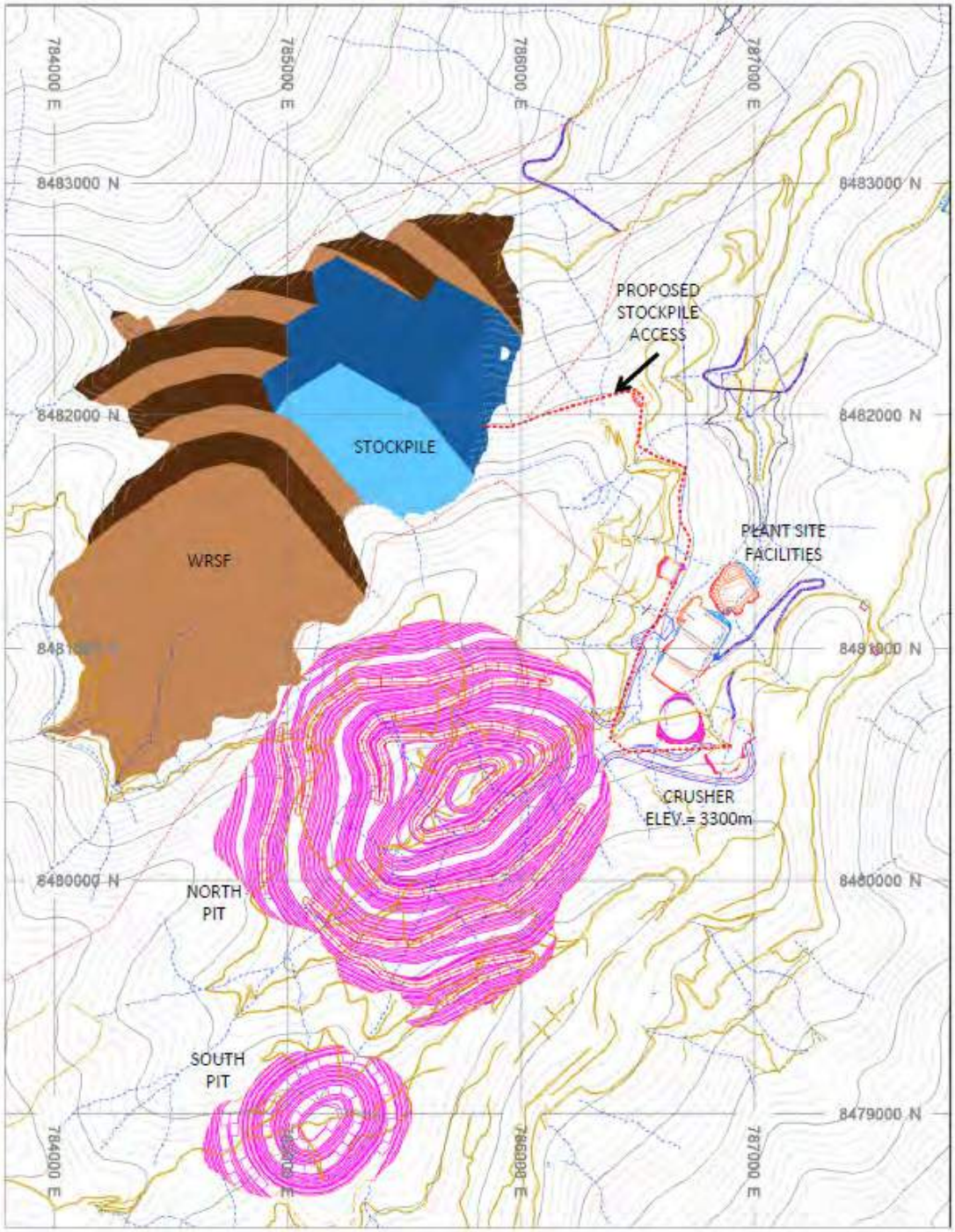
The construction of the WRF will be done generally utilizing top-down dumping methods. The first ~12 Mt of waste rock mined will be hauled to the bottom of the ultimate WRF to establish a toe berm approximately 100 m high. The toe berm will provide catchment for rolling rocks off of lifts dumped from above. All other waste will be dumped from the 3472 elevation (upper platform), the 3376 elevation (lower platform) or as wrap-around lifts which establish the final dump slope angle for closure purposes.

An area of 225,000 m^2 is considered for the site preparation of this facility. Contact water below the facility will be collected and returned to the process plant as no water treatment is assumed. Non-contact water above the facility will be diverted to the fresh water reservoir below the process plant.

16.9 Stockpiled Material

Material that is stockpiled will temporarily be placed on the WRF from where it will be reclaimed during the last years of production. The maximum stockpile size is approximately 119M tonnes in Year 9 of production. At the end of the mine life, the stockpile is fully reclaimed. The location and maximum stockpile is shown in Figure 16-10.

Figure 16-10: Stockpile Location



Note: Figure prepared by MMTS, 2015.

16.10 Mine Operating Units

The Cotabambas mine fleet consists of a conventional diesel fleet with a maximum capacity of approximately 116 Mt of total material per year operating on 12 m benches.

Equipment requirements are estimated using first principle calculations. Equipment sizing and numbers are based on operational factors, including fleet life, average availabilities, and efficiency percentages. A schedule of 365 days per year, with two shifts of 12 hours per day, and seven days per work week is used.

Equipment availabilities are provided by vendors with declining values as the fleets age, in order to better estimate trucks, shovel and drill equipment requirements throughout the life of the mine. Availability is presented in 6,000 hour increments. Primary equipment availabilities are shown in Table 16-9:

16.10.1 Drilling and Blasting

Because rock strength test and compression test work have not been done for Cotabambas deposit, estimated powder factor values of approximately 0.30 kg/t for mill feed and 0.28 kg/t for waste rock are used. The average drill penetration rate for mineralized material and waste is estimated to be 28 m/h. Typical pattern designs assume Atlas Copco PV271 drill equipment with a 251 mm of drill diameter size and average blast hole spacing between 7.5–8 m.

For the pre-strip mining, six drills are required and for the peak years of production, seven Atlas Copco PV271 drills are required.

No hydrogeological studies are currently available and since there is no evidence of ground water during exploration drilling (according to client information), pit dewatering requirements are not considered at this stage. However, a cost allowance is included to account for some pit dewatering. To determine the blasting requirements, a water resistant emulsion is assumed to account for some wet loading conditions. A 50/50 split between emulsion and ANFO is assumed.

16.10.2 Loading

The primary production loading fleet for mineralized material and waste will consist of five 34 m^3 hydraulic shovels. For the purposes of the PEA, the shovels were assumed to be CAT 6060B models.

Table 16-9: Primary Equipment Availabilities

Hours From	To	Truck Availability - %	Shovel Availability - %	Drill Availability - %
-	6,000	86%	87%	87%
6,000	12,000	86%	87%	85%
12,000	18,000	86%	86%	84%
18,000	24,000	85%	86%	84%
24,000	30,000	85%	86%	84%
30,000	36,000	85%	85%	82%
36,000	42,000	85%	85%	80%
42,000	48,000	84%	84%	80%
48,000	54,000	84%	84%	79%
54,000	60,000	84%	83%	79%

16.10.3 Hauling

Haulage requirements are based on measured annual haulage profiles from the two pits. All waste is delivered to the WRF by trucks. The average haulage distance for waste by trucks is between 3 and 4 km. Haul trucks have a capacity of 227 t and are used to haul both mill feed and waste. The PEA assumes the use of CAT 793F models. The pre-stripping stage will require 35 trucks, and during the first two years of production the truck fleet will reach a peak of 40 units. The truck fleet will be comprised of a mix of owner-operated and rental units as shown in Figure 16-11.

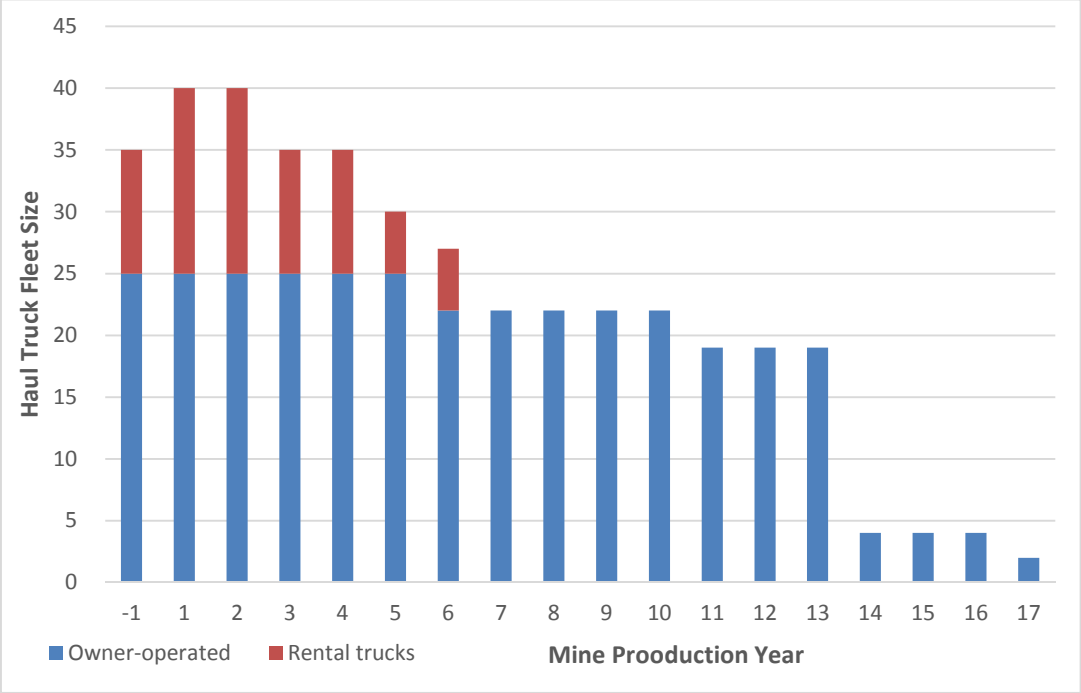
16.10.4 Support

The mine support fleet is shown in Table 16-10. The highest levels of annual material movement occur in Years 1–6. Material movement in Years 14–17 is comprised strictly of stockpile reclaim material and as such, the required support fleet drops considerably.

16.10.5 Manpower

The mine plan assumes that Cotabambas will operate seven days a week, 24 hours per day with four crews rotating to fill the mine roster of 12 hours per shift. Owner manpower rises to 454 mine employees in Year 2. This figure includes technical staff, operators, mine maintenance and mine operations general and administrative (G&A) costs. Rental trucks will be operated using Owner manpower.

Figure 16-11: Truck Fleet Size



Note: Figure prepared by MMTS, 2015.

Table 16-10: Mine Support Fleet

Description	Unit and Size	Years 1–6	Years 7–13	Years 14–17
Dozers	Cat D10 - 433 kW	4	4	2
Rubber tire dozers	Cat 834H - 372 kW	4	3	1
Water truck	Cat 777G - 20,000 gallons	3	3	1
Graders	Cat 16M - 221 kW	3	3	1
Fuel/Lube Truck	Cat 740 - 4,000 gallons	2	2	1
Front-end Loader	Cat 994F - 19m^3	1	1	0
Front-end Loader	Cat 988H - 6.5m^3	2	2	1
Excavator	Cat 374DL - 355 kW	1	1	0
Excavator	Cat 349EL - 295 Kw	1	1	1
Crane	LTM1250 - 250 tonnes	1	1	1
Service truck	F550 - 4x4	2	2	2
Welding truck	F550 - 4x4	2	2	2
Warehouse truck	F550 - 4x4	1	1	1
Forklift	Hyster H210HD - 10 tonnes	1	1	1
Forklift	Hyster H620HD - 30 tonnes	1	1	1
Light plants	Terex AL8000 - 20 kW	13	10	4
Crusher	Sandvik QJ430	1	1	1
Screening Plant	Sandvik QA440	1	1	1
Crew Van	15 passenger	4	4	2
Crew-cab Pickup	1/2 tonne	12	12	4

16.10.6 Main Consumables

Main consumables for mine operations include diesel fuel, ANFO, emulsion, and tires. Yearly average consumable requirements are detailed in Table 16-11.

Table 16-11: Yearly Average Consumable Requirements

	Diesel Million Litres	ANFO & emulsion kt	Tires
Years 1–6	48	19	210
Years 7–13	33	5	160
Years 14–17	7.5	N/A	22

16.10.7 Grade Control

The Cotabambas technical staff will conduct in-pit grade control. This effort requires the ability to accurately predict the contact between mill feed and waste with the aim of separating higher grade mill feed from lower grade mill feed and to control dilution.

The grade control group will be responsible for:

- Sampling, and geological mapping of blast holes
- Merging assay data with blast hole coordinates
- Generating short range planning block models
- Generating dig plans with ore grade polygons
- Mine-to-plant reconciliations and quality control.

17.0 RECOVERY METHODS

17.1 Process Flowsheet

A simplified process flowsheet is shown in Figure 17-1. The plant design consists of a plant with a nominal processing capacity of 80,000 t/d and includes crushing, grinding, flotation, concentrate dewatering and tailings disposal.

17.1.1 Crushing

The primary crushing station will be a fixed 60 x 113 inch gyratory crusher. Mine haul trucks will dump run-of-mine (ROM) material directly into the dump pocket of the crusher.

Crushed mill feed will be discharged to two apron feeders that will discharge to a sacrificial coarse ore conveyor, feeding a stockpile conveyor.

The coarse ore stockpile will be conical with a live capacity of 80,000 t, providing an equivalent of 24 hours of plant feed. The stockpile will provide a buffer between the mining and crushing operations and the process plant, so that the plant can continue to be fed during intermittent stoppages of feed from the crusher or the mine.

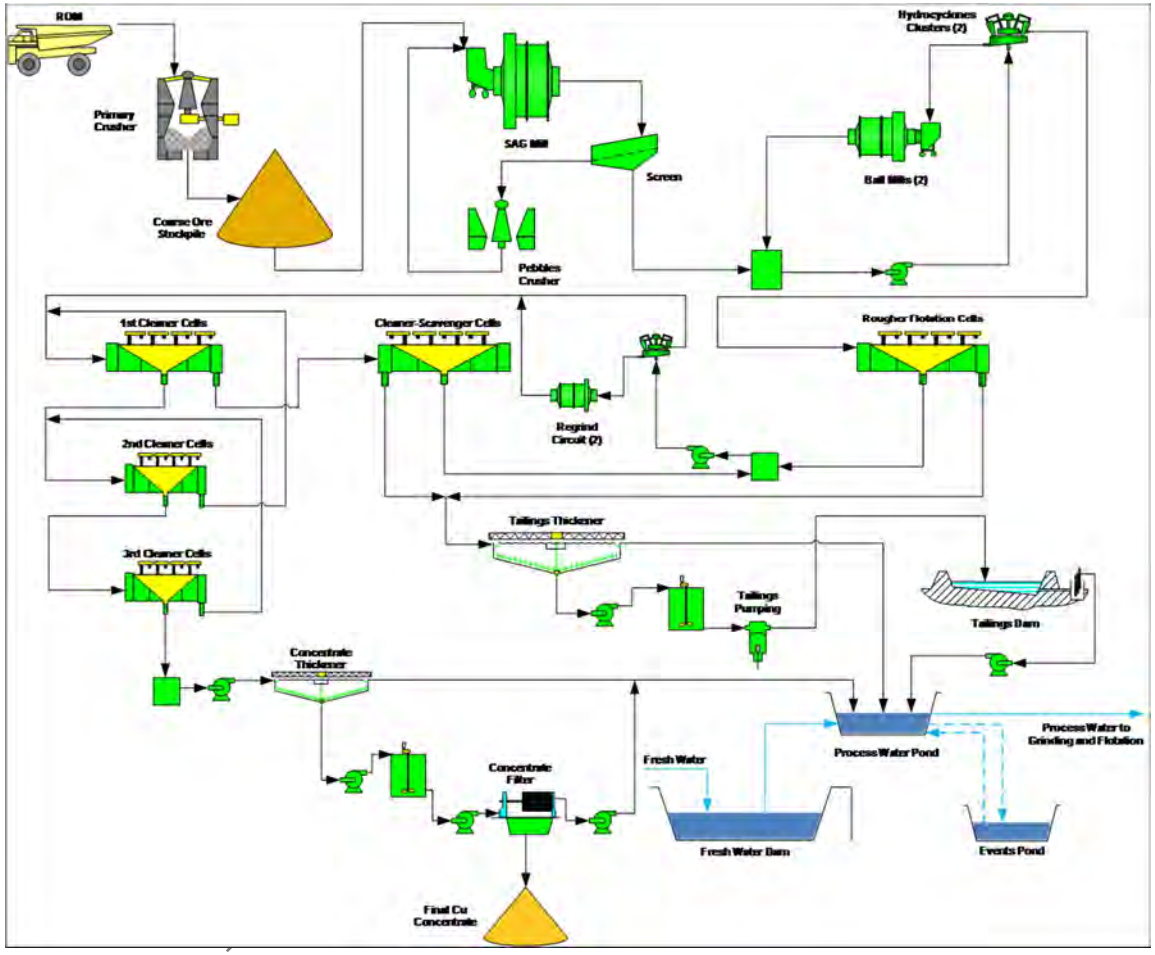
Mill feed will be reclaimed from the stockpile with four apron feeders installed in a concrete tunnel below the stockpile. The feeders will discharge material onto the SAG mill feed conveyor. Solid lime and grinding media will be added to the SAG mill feed as required to maintain grinding charge and to condition the mill circuit slurry pH for subsequent flotation.

17.1.2 Grinding

The grinding circuit configuration consists of an open-circuit SAG mill followed by two ball mills in closed circuit with cyclones.

SAG mill discharge will be screened and pebble oversize material (>15 mm) will be crushed by means of two cone crushers, and then recycled back to the SAG mill feed conveyor. The SAG mill discharge screen undersize slurry will be transferred to a ball mill discharge cyclone feed pumpbox from a SAG mill discharge transfer pumpbox. The SAG mill is sized at 40 ft diameter x 26 ft length, powered by a 28 MW gearless motor drive. The two ball mills have been sized at 28 ft diameter x 40 ft length, each driven by a 22 MW gearless motor drive, fixed-speed, operating in parallel. The ball mill discharge will be combined with the SAG mill product in the ball mill discharge cyclone feed pumpbox.

Figure 17-1: Simplified Process Flow Diagram



Note: Figure prepared by Amec Foster Wheeler, 2015

The ball mill pumpbox with two cyclone feed pumps operating in parallel, will feed four clusters of 12 D26 cyclones (10 operating, 2 stand by). Two clusters will operate in closed circuit with each ball mill. Cyclone overflow at about 30% solids will be transferred to flotation, and the underflow at about 70% solids will discharge to the ball mill feed hoppers. The average final product size from the grinding circuit will be P_{80} 106 µm.

17.1.3 Flotation

The flotation circuit is designed to recover the minerals containing copper and gold into a final concentrate. It consists of a rougher flotation stage, regrinding, a first cleaner stage followed by a cleaner-scavenger stage and two stages of re-cleaner (second and third cleaner) using conventional flotation tank cells.

The ball mill circuit product will feed a bank of rougher cells (seven total, 600 m^3 each). The rougher tailings will be pumped to the flotation tailings thickener. The rougher and the cleaner-scavenger concentrates will feed two regrind hydrocyclone clusters, of 12 D10 hydrocyclones per cluster (10 operating, two stand by). Each cluster will be coupled to one regrind mill. The underflow from the hydrocyclones will be fed to the regrinding circuit, consisting of two regrind mill units. The regrind circuit product will have a P_{80} of 26 µm and will discharge to a transfer pumpbox that feeds the first cleaner flotation.

The regrind hydrocyclone overflow will feed the first cleaner cells (four total, 160 m^3 each). The first cleaner concentrate will feed the second cleaner cells and the tailings will feed the cleaner-scavenger cells (four total, 100 m^3 each). The cleaner-scavenger cell tailings will be pumped to the flotation tailings thickener for dewatering with the rougher flotation tailings. Cleaner scavenger concentrate will be recycled to regrinding.

The concentrate from the second cleaner stage cells (four total, 70 m^3 each) will feed the third cleaner stage cells (two total, 70 m^3 each). The tailings from the second and third cleaner stages will be recycled back to the first and second cleaner cells, respectively, in closed circuit. The third, final cleaner concentrate will be transferred to concentrate dewatering to prepare it for storage and subsequent transport.

Frother (MIBC) and collectors will be added in the flotation circuit as required. Lime will be used as the primary pH modifier throughout and will be added as required to depress pyrite.

17.1.4 Concentrate Dewatering

The final cleaner concentrate slurry will contain copper and gold, and will be dewatered initially in a concentrate thickener (16 m diameter) where flocculant will be added to assist settling. The thickener overflow will be recycled to the process water system, and the thickened underflow, at approximately 60% solids, stored in a concentrate storage tank. Concentrate slurry from the storage tank will be fed in batches to pressure filters

for dewatering to about 10%. Thereafter, the final concentrate will be stored for transportation to the export port in trucks.

17.1.5 Tailings Disposal

The flotation tailings will be fed to four thickeners (40 m diameter), where flocculant will be added to assist material settling. The thickener overflow will be recycled back as process water, whilst the thickened tailings at 62% solids will be pumped to the tailings storage facility (TSF) via a pumping system station, consisting of positive displacement pumps (TDPM 2500 type, 10 in operation, one stand by).

Water reclaimed from the TSF has been estimated to be about 30% of the water in the tailings stream.

17.2 Plant Design

17.2.1 Process Design Criteria

The current PEA conceptual process engineering criteria are based on currently-available supporting metallurgical studies, as well as recent comparable project benchmarks in Amec Foster Wheeler's database. Fresh water will be supplied from a nearby reservoir for process water make-up and service requirements.

Open pit mining methods are planned. Sulphide, Mixed and Oxide Au mineralization will be processed in a conventional copper porphyry flotation concentrator plant as a blend to produce a copper, gold and silver concentrate, with copper as the main payable metal in the concentrate and gold and silver providing important economic by-product credits.

The estimated recoveries by metal and zone are shown in Table 17-1. No Cu or Ag recoveries are applied to Oxide Au mineralization. Constant recoveries for each mineralized zone and final product grade were applied (no relationship with grade) in the mine plan in proportion to the quantity of each zone processed to estimate annual production and recoveries.

Testwork on two hypogene sulphide zone composite samples of lower-grade material (Cu grade of 0.27%) and higher-grade material (Cu grade of 0.46%) did not indicate any material grade-recovery sensitivity in this range, which also corresponds approximately to the range of annual grades expected to be processed in the PEA mine plan. There is a risk however that recoveries for lower-grade stock stockpiled sulphides planned to be processed in years 14-17 will be lower than expected.

Table 17-1: Recoveries Applied by Zone

Mineralization Type and Metal	Recovery (%)
Oxide Au	
Cu	-
Au	65.0
Ag	-
Mixed	
Cu	60.0
Au	55.0
Ag	-
Sulphides	
Cu	87.5
Au	62.0
Ag	60.4

Approximately 127 Mt of Indicated Mineral Resources and 356 Mt of Inferred Mineral Resources, comprising Sulphide, Mixed and Oxide-Au mineralization will be treated through the concentrator. Figure 16-7 in Section 16 and Figure 17-2 summarizes the key processing production and performance parameters expected over the LOM.

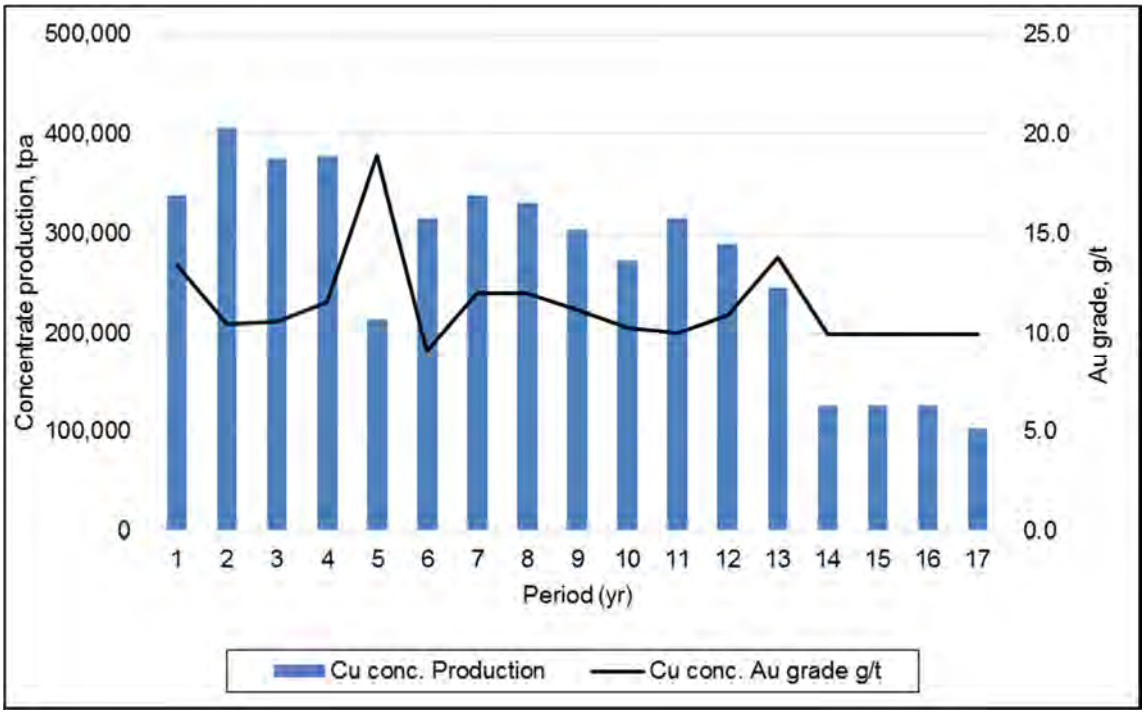
Cotabambas copper sulphide is generally amenable to conventional flotation concentration with good recoveries. Chalcopyrite is the dominant copper mineral in the sulphide, with some associated pyrite. Copper mixed material recoveries by flotation are relatively lower, representing its lower proportion of recoverable sulphide and higher oxide content estimated by sequential analysis. The mineralization gradually changes and sulphide material will be dominant in later years. Since the metallurgical testwork was done on few samples, the concentrate grade for this PEA has been assumed at a constant value of 27.0% Cu in the financial model.

A summary of the process design criteria is presented in Table 17-2.

It is planned that 80 kt/d of mill feed will be crushed in a primary crushing circuit, and milled through a primary grind SAG mill and secondary grind ball mills to produce fine material at P_{80} 106 µm.

Only four ball BWI tests have been done to date on composites of the main hypogene and this indicates some variability in hardness within the deposit. During the next stage of PFS testing, SAG, crushing, abrasion and additional ball mill comminution baseline testing should be conducted as well as a preliminary variability program.

Figure 17-2: Key Processing Production and Performance Parameters – Concentrate Produced



Note: Figure prepared by Amec Foster Wheeler, 2015

Table 17-2: Process Design Criteria Summary

Process Facility	Criteria Concentrator	
Key Production Metrics		
LOM Mill Feed (Mt)	371 (years 1 to 13)	112 (years 14 to 17)
Plant throughput (Mt/a)	29.2	
Plant Throughput (t/d)	80,000	
Mineralization Types	Oxide Mixed Sulphides	
CuT Grade (%)	0.38 (years 1 to 13)	0.13 (years 14 to 17)
Au Grade (g/t)	0.21 (years 1 to 13)	0.07 (years 14 to 17)
Ag Grade (g/t)	2.58 (years 1 to 13)	1.69 (years 14 to 17)
CuT Recovery [average LOM] (%)	80.4	
Au Recovery [average LOM] (%)	61.3	
Ag Recovery [average LOM] (%)	52.2	
Concentrate Production [average LOM] (kt/a)	270	
Concentrate Grade [average LOM] (%)	27.0	
Crushing		
Configuration	Open circuit primary crushing	
Crush Size	P_{80} 100 - 150 mm	
Grinding		
Mill Type	Open circuit SAG mill, followed by two Ball Mills, coupled with two hydrocyclones clusters	
Grind Size	P_{80} 106 µm	
Bond Work Index (kWh/t)	16.6	
SAG Mill Size (ft)	40’ x 26’	
Ball Mill Size (ft)	28’ x 40’	
Flotation		
Configuration	Fresh water flotation: Rougher, regrinding, three cleaning stages, cleaner-scavenger	
pH	10.5 (lime addition)	
Regrind Size	P_{80} 26 µm	
Rougher Cells	Cell Tank, 7 × 600 m^3	
1st Cleaner Cells	Cell Tank, 4 × 160 m^3	

Process Facility	Criteria Concentrator
2nd Cleaner Cells	Cell, 4 × 70 m³
3rd Cleaner Cells	Cell, 2 × 70 m³
Cleaner Scavenger Cells	Cell, 4 × 100 m³
Concentrate Handling	
Dewatering Method	Thickening and filtering
Concentrate Flowrate [Design] (t/h)	37 (41)
Thickener Type	High rate
No. of Thickeners	1 × 16 m D
Underflow Solids Percentage (%)	60
Filter Type	Press Filter
No. of Filters	1
Concentrate Moisture (%)	8–10
Tailings Disposal	
Concept	Thickened Tailings
Water Recovery	30% water reclaim from TSF
Thickener Type	High Rate
Underflow Solids Percentage (%)	62
No. of Thickeners	4 × 40 m D

The selection of the conventional SAG/ball grinding circuit and equipment is based on a typical process flowsheet for this type of mineralization and scale of the Project. However a comminution trade-off study and further testing needs to be done during more detailed studies to support the basis of these selections.

Milled feed will be processed in a conventional rougher, regrind and cleaner copper flotation plant to produce about 270,000 t/a of copper concentrate with a life-of-mine average concentrate grade of 27.0% Cu, and 11.5 g/t Au and 130 g/t Ag. Concentrate will be dewatered to about 8% moisture, and tailings will be pumped to a TSF at 62 wt% solids for disposal.

17.2.2 Process Plant Major Equipment

Table 17-3 presents a summary of the sulphide plant major process equipment. The selection of positive displacement pumps to transport the tailings to the TSF is based on a trade-off study conducted by Patterson and Cooke.

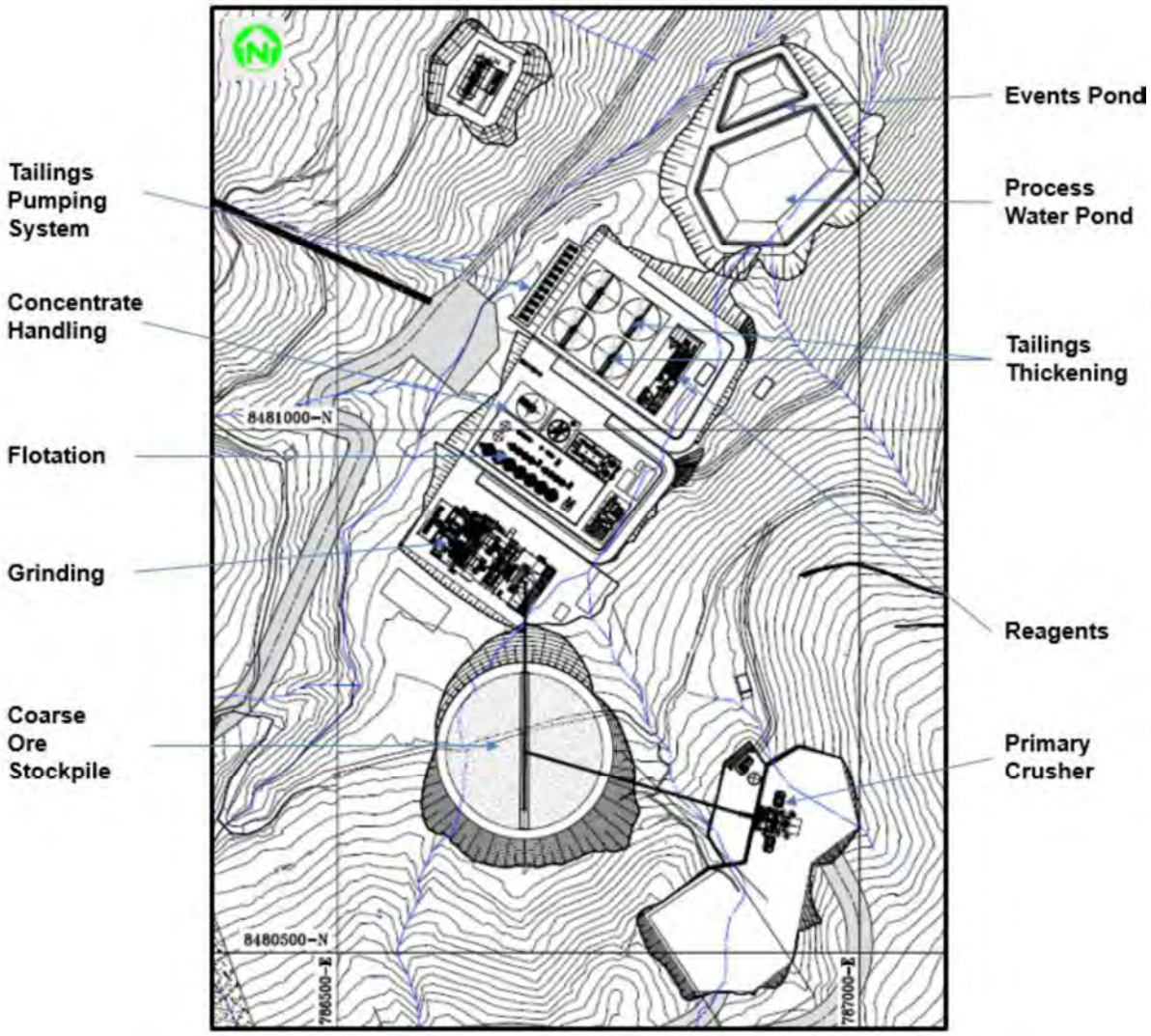
Table 17-3: Plant Major Process Equipment Summary

Equipment	Qty	Type	Size / Capacity / Flow (Indicative)
Primary Crusher	1	Gyratory	1600 mmx 2900 mm 113' Heavy Duty x 750 kW
Primary Crusher Discharge Feeder	2	Apron	2750 t/h x 224 kW
Coarse Ore Stockpile Reclaim Feeders	4	Apron	965 t/h x 112 kW
SAG Mill Feed Conveyor	1	Belt	3509 t/h
SAG Mill	1	SAG	40' 0" x 26' 0" / 1 x 28 MW Gearless
Ball Mill	1	Ball	28' 0" x 40' 0" / 1 x 22 MW Gearless
Hydrocyclones Cluster (grinding circuit)	4	Cluster	10 op. + 2 standby
Rougher Flotation Cells	7	Tank	600 m^3
1st Cleaner Flotation Cells	4	Tank	160 m^3
Cleaner Scavenger Cells	4	Tank	100 m^3
2nd Cleaner Flotation Cells	4	Tank	70 m^3
3rd Cleaner Flotation Cells	2	Tank	70 m^3
Regrinding Mill	2	Isamill	710 m3/hr
Regrinding Hydrocyclones Cluster	2	Cluster	10 op. + 2 standby
Concentrate Thickener	1	HRT	16m D
Concentrate Filter	1	Press Filter	500 Kg/h.m^2 (TBC)
Tailings Thickener	4	High Rate	40m D
Tailings Pumps System	1	Positive displacement	TDPM 2500 Type, 10 op. + 1 standby
Flocculant Preparation System	1		TBD
Blowers	4		33000 CFM

17.2.3 Process Plant Layout

Figure 17-3 presents an overview of the plant layout for the PEA. This shows the ROM feed to primary crushing and coarse ore stockpiles to the south, the central concentrator with tailings disposal directing to the west via positive displacement pumps to the TSF, and the process water and event ponds to the north. No trade-off studies or geotechnical investigations have been completed to support the plant site selection and these should be completed in the next study stage.

Figure 17-3: Overall Conceptual Plant Layout



Note: Figure prepared by Amec Foster Wheeler, 2015

17.3 Product/Materials Handling

Final concentrate product containing copper, gold and silver will be dewatered to about 8% moisture and stored on site for subsequent trucking to port for shipping.

The concentrate road transport option selected for the basis of this PEA considers a Cotabambas–Challhuahuacho–Espinar–Imata–Arequipa–port of Matarani route, with a total distance of 598 km.

The road section from Cotabambas to Challhuahuacho should be reviewed during PFS to verify its condition to support 35 t truck loads. The Challhuahuacho–Espinar–Imata road section can handle truck traffic as it is connected with the Las Bambas, Constancia and Antapaccay mine access roads. The Imata–Arequipa–Matarani road section corresponding to about 50% of the total distance is completely paved.

Matarani is the main seaport in the Arequipa region and it is accessed by road and the railroad that runs through Mollendo–Matarani–Arequipa–Juliaca–Puno–Cusco–Macchu Picchu. This port has the capacity and readiness to handle copper concentrates, sulphuric acid and containers. As such it has been considered as the base case concentrate handling port for the PEA.

17.4 Energy, Water, and Process Materials Requirements

17.4.1 Mass and Water Balance

The mass balance nominal flows are based on a 365 day operation, and the relevant plant area availability. Figure 17-4 presents a simplified plant water balance. Table 17-4 presents the water requirements for operating the concentrator plant.

Evaporation and precipitation data were not available since no hydrological studies have been undertaken on the Project at this time. Therefore, these have not been considered in the balance, and the water make-up is based on the moisture losses in both concentrate and tailings streams. About 80% of process water in the plant flotation tailings stream will be recovered and recycled to the process from tailings thickeners (75%) located at the plant site and as reclaim from the thickened tailings storage facility (5%). Water make-up requirements for the plant are estimated to be about 364 L/s.

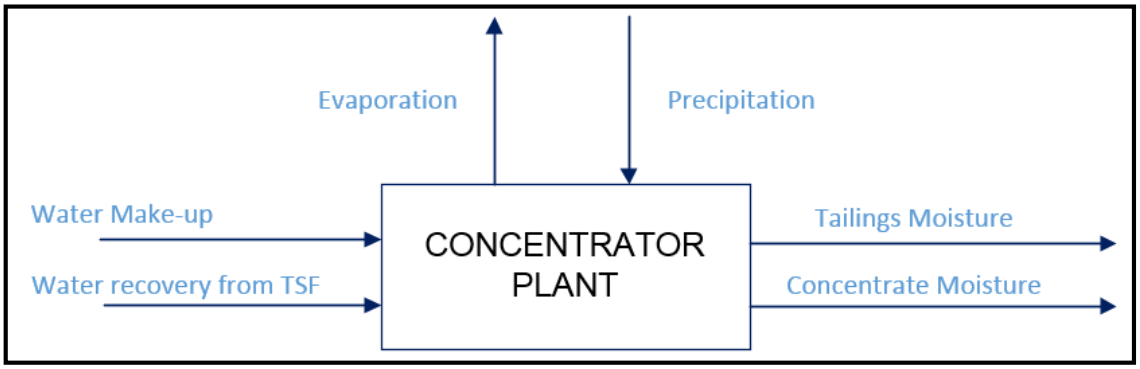
17.4.2 Power

A summary of the estimated power consumption by area is presented in Table 17-5. The power requirement has been factored from the installed power indicated in the major equipment list and forms the basis of the operating cost power estimate.

17.4.3 Process Consumables

A summary of the relevant consumables are shown in Table 17-6 and Table 17-7. These are based on the mass balances and metallurgical testwork, as well as comparable industry benchmarks in Amec Foster Wheeler's project database where no Project-specific data were available.

Figure 17-4: Water Balance Simplified Diagram



Note: Figure prepared by Amec Foster Wheeler, 2015

Table 17-4: Water Balance Summary

Water Requirements	Unit	Concentrator Plant
Total Water Make-up	m³/h	1,310
	L/s	364

Table 17-5: Power Consumption Summary

Area	Charge of Operation (kW)	Factor for Electrical Charge	Average Demand (kW)	kWh/t Milled
Crushing	1,257	0.95	1,191	0.33
Milling	73,500	0.95	69,632	19.22
Flotation	9,430	0.95	8,934	2.47
Concentrate Thickening and Filtration	791	0.95	749	0.21
Tailings Thickening	26,174	0.95	24,796	6.84
Services (3%)	3,335	0.95	3,159	0.87
Minor Equipment (5%)	5,558	0.95	5,265	1.45
Total Annual Power Cost Estimate	**120,045**		**113,727**	**31.4**

Table 17-6: Reagents Summary

Reagent	Consumption (g/t)	Consumption (t/year)	Source
Aerofloat 208	14.00	376.10	Metallurgical Testwork, Certimin 2014
MIBC (Flot Cu Bulk)	19.80	531.91	Metallurgical Testwork, Certimin 2014
Sodium Cyanide (Flot Cu Bulk)	20.00	537.28	Metallurgical Testwork, Certimin 2014
Lime	950.00	25,520.80	Metallurgical Testwork, Certimin 2014
Z-11 (Flot Cu Bulk)	6.00	161.18	Metallurgical Testwork, Certimin 2014
Flocculant 1120	15.00	402.96	Outotec Testwork 2014

Table 17-7: Steel Consumables Summary

Equipment	Consumption	Source
Primary Crushing		
Liner	2 set/year	Typical for the scale of the project
Milling		
SAG		
Liners	2.0 set/year	Typical for the scale of the project
Balls	320 g/t of mill feed	Typical for the scale of the project
Ball Mill		
Liners	2 set/year	Typical for the scale of the project
Balls	450 g/t of mill feed	Typical for the scale of the project
Screen Panels	2 set/year	Typical for the scale of the project
Hydrocyclones		
Vortex	12 unit/year	Typical for the scale of the project
Apex	180 unit/year	Typical for the scale of the project
Regrinding		
Ball Mill		
Balls	30 g/t of mill feed	Typical for the scale of the project
Hydrocyclones Regrinding		
Vortex	8.3 unit/year	Typical for the scale of the project
Apex	50 unit/year	Typical for the scale of the project

18.0 PROJECT INFRASTRUCTURE

18.1 Access and Logistics

Two routes are proposed to be used to transport goods to and from the Project. The route from the Project site to the port of Callao (Lima) is illustrated in Figure 18-1. A second route, from the project site to the port of Matarani is included in Figure 18-2. Distances from the ports to the Project site are provided in Table 18-1 and Table 18-2 respectively.

The route from the port of Callao will be utilized to transport personnel and minor equipment during construction and through-out operations. The Matarani port route will be utilized to transport major equipment during construction. Copper concentrate will also be transported by truck, utilizing this route to Matarani port, where it will be offloaded and shipped to the end user.

The last 66 km from Pamputa to Cotabambas (using the Matarani port route) is a gravel road in poor condition. Upgrade of 11 points consisting of approximately 23 km is assumed as part of the Project scope. Upgrades encompass widening the road from 4 m to 6.6m and widening of curves up to 55 m.

18.2 Waste Storage Facilities

The proposed waste rock storage facilities are discussed in Section 16.

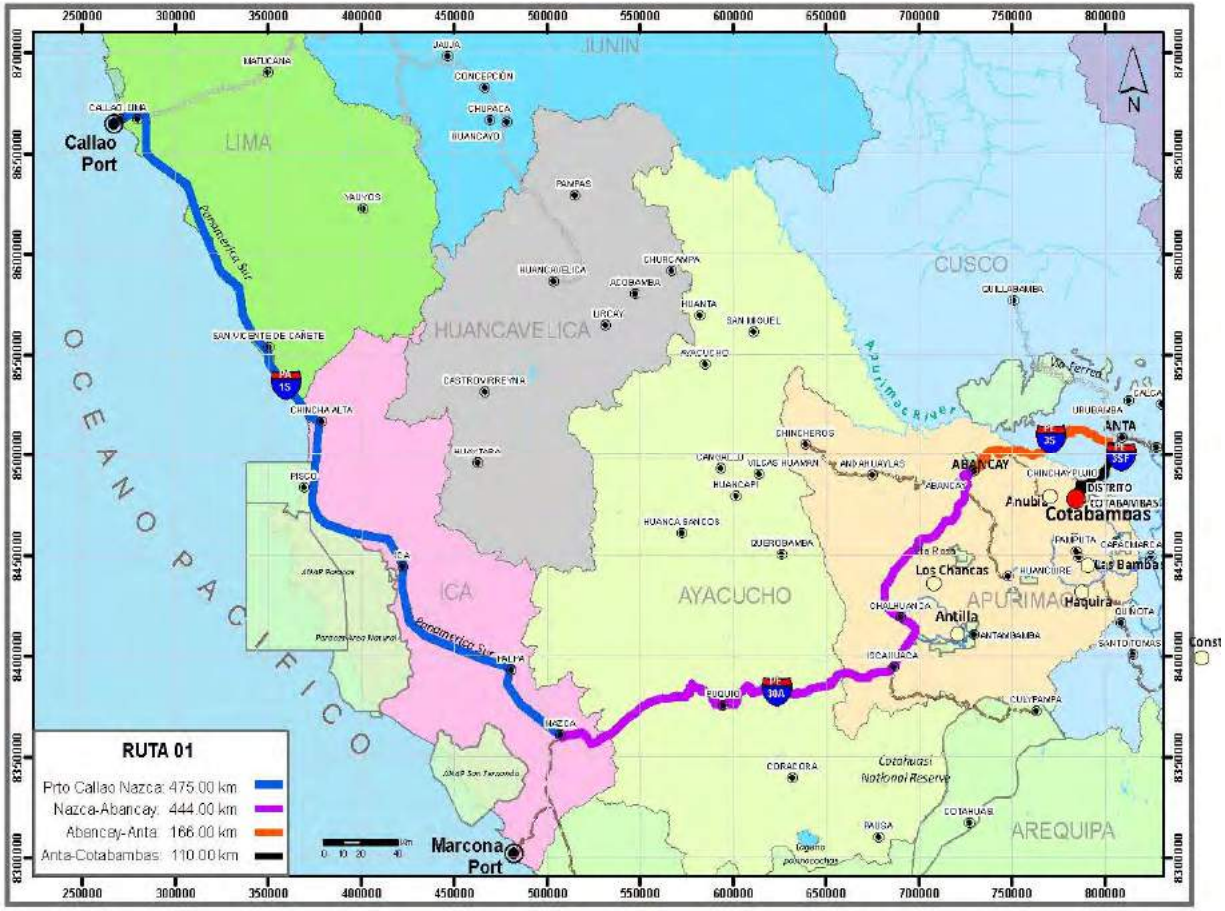
18.3 Tailings Storage Facilities

During the development of the PEA, several alternatives for tailings location and disposal for the Cotabambas project were analyzed. Initial development focused on locations for a TSF with a storage capacity of 450 Mt using high density thickened tailings and filtered tailings in the Duraznomayo, Culluchaca and Futajanov drainages.

The final concept, which forms the basis of the PEA, is the disposal of thickened high density tailings to a facility with a capacity of 483 Mt. This would be located within the adjacent basin of the Ccayarayoc hill, which is located 6.2 km from the process plant, at an elevation of 1,000 m above the process plant (4,255 m).

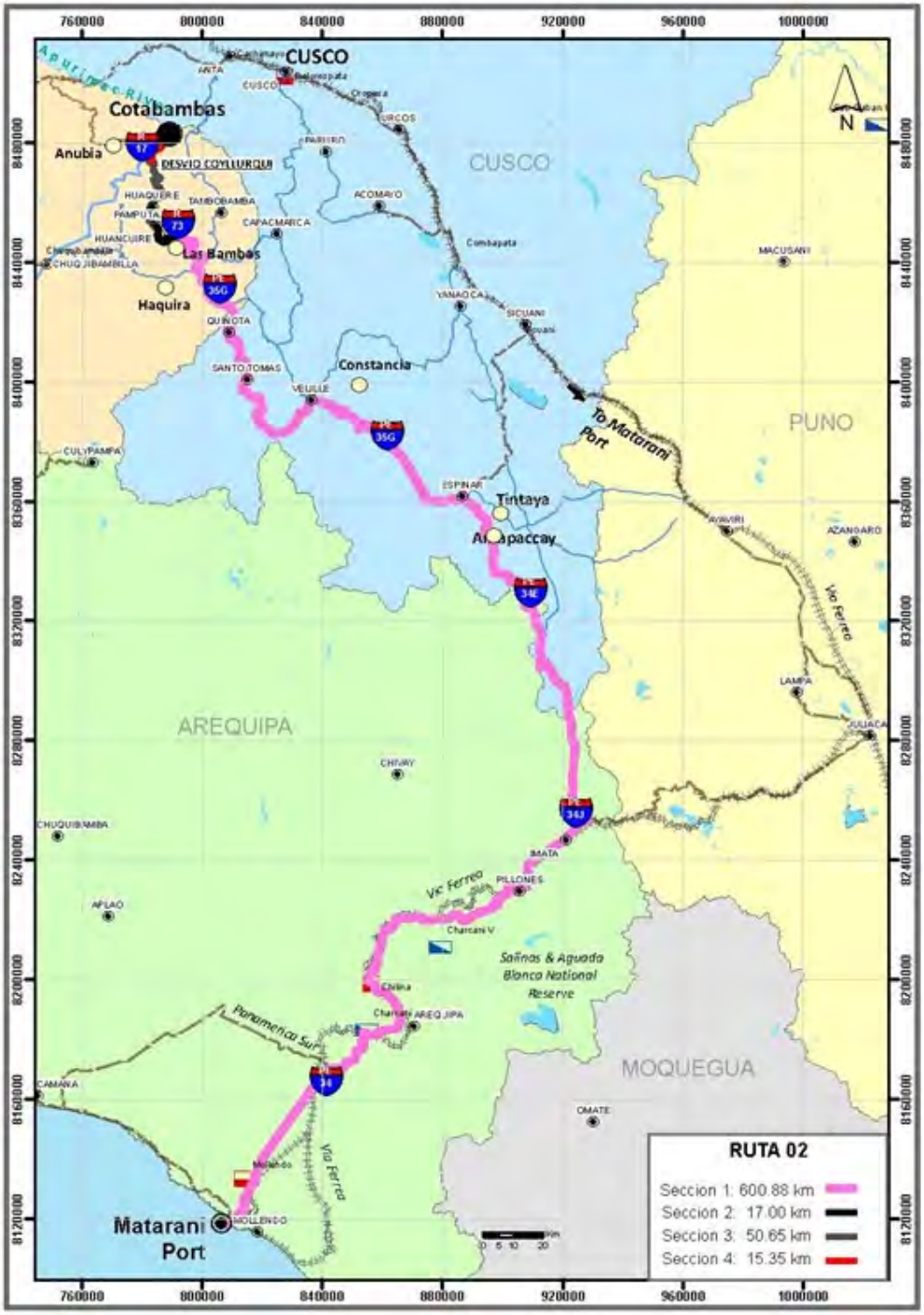
This configuration of the TSF has three retention dams with a dam volume of 38.23 Mm^3 obtained at the minimum crest elevation of the retention dams at 4,302 m, and a dam: tailings volume ratio of 0.10.

Figure 18-1: Road Route from the Project Site to the Port of Callao



Note: Figure prepared by Amec Foster Wheeler, 2015

Figure 18-2: Road Route from the Project Site to the Port of Matarani



Note: Figure prepared by Amec Foster Wheeler, 2015

Table 18-1: Road Distances, Callao Port to Cotabambas Site

Route 1	Distance	Detail
Callao Port–Nazca	475 km	Paved road
Nazca–Abancay	444 km	Paved road
Abancay–Anta	166 km	Paved road
Anta–Cotabambas	110 km	Gravel road, 4 to 6 m width, in good condition
Total	***1,195 km***	

Table 18-2: Road Distances, Matarani Port to Cotabambas Site

Route 2	Distance	Detail
Matarani Port–Imata	251 km	Paved road
Imata–Espinar	139 km	Gravel road – good condition
Espinar–Quiñacota	114 km	Paved road
Quiñacota–Las Bambas	97 km	Paved road
Las Bambas–Pamputa	17 km	Gravel road – good condition
Pamputa–Coyllurqui	51 km	Gravel road – bad condition
Coyllurqui–Cotabambas	15 km	Gravel road – bad condition
Total	***684 km***	

Thickened tailings will be transported into the reservoir area through a 12.7 km steel pipe and 0.6 km HDPE pipe from the process plant, with 11 positive displacement pumps (210 bar) located in the plant area. Due to the proposed TSF location lying within a different basin, a constant operating lake with a volume of 1 Mm^3, a conservative pond volume, was assumed. The tailings surface will have a discharge slope at 1.2%. Tailings will discharge from two opposite points to the upstream face of the retention dams. Key components of the design criteria are presented in Table 18.3.

Tailings deposition will require further evaluation during more detailed studies.

Table 18-3: Tailings Facility Design Criteria "High Density"

Description	Unit	Criteria
Dry Density (Thickened Tails)	t/m^3	1,5
Deposit Slope	%	1,2 (towards the upstream face of the dam)
Slope	H:V	2H:1V
Crest width	m	10
Maximum height	m	250
Required Capacity	Mt/Mm^3	483/322

18.4 Water Management

The conceptual water management was developed based on water balance use of the non-contact water pond (fresh water pond).

The pond is located at the northeast from the open pit and downstream from the process plant. The pond capacity will be about 3 Mm^3 and the minimum volume will be 0.1 Mm^3. The main purpose will be to complement the make-up water supply for the plant, as the main water source will be from the tailings pond recycle water. The non-contact water pond will be supplied by surface runoff water and direct precipitation.

Contact water will be collected below the waste rock facility or within an event pond within the plant footprint. No allowance for a water treatment plant has been considered at this time.

In order to support this preliminary water balance, the following criteria were considered:

- Surface runoff values (N2) are from the drainage area upstream from the pond
- Open pit (FON) runoff
- Tailings pond modification brings a reduction in the diversion channel drainage area (FCN), therefore runoff decreases. The water balance includes this modification
- The water balance considers the tailings pond recycled water (FDP) as a main supplier to the process plant therefore the process plant fresh water requirement will be 0.96 Mm^3/month
- The outflows will be the lake evaporation and the fresh water supply
- No compensation water for areas downstream of the water dam was considered at this time.

A flow diagram is included in Figure 18-3 for the water balance.

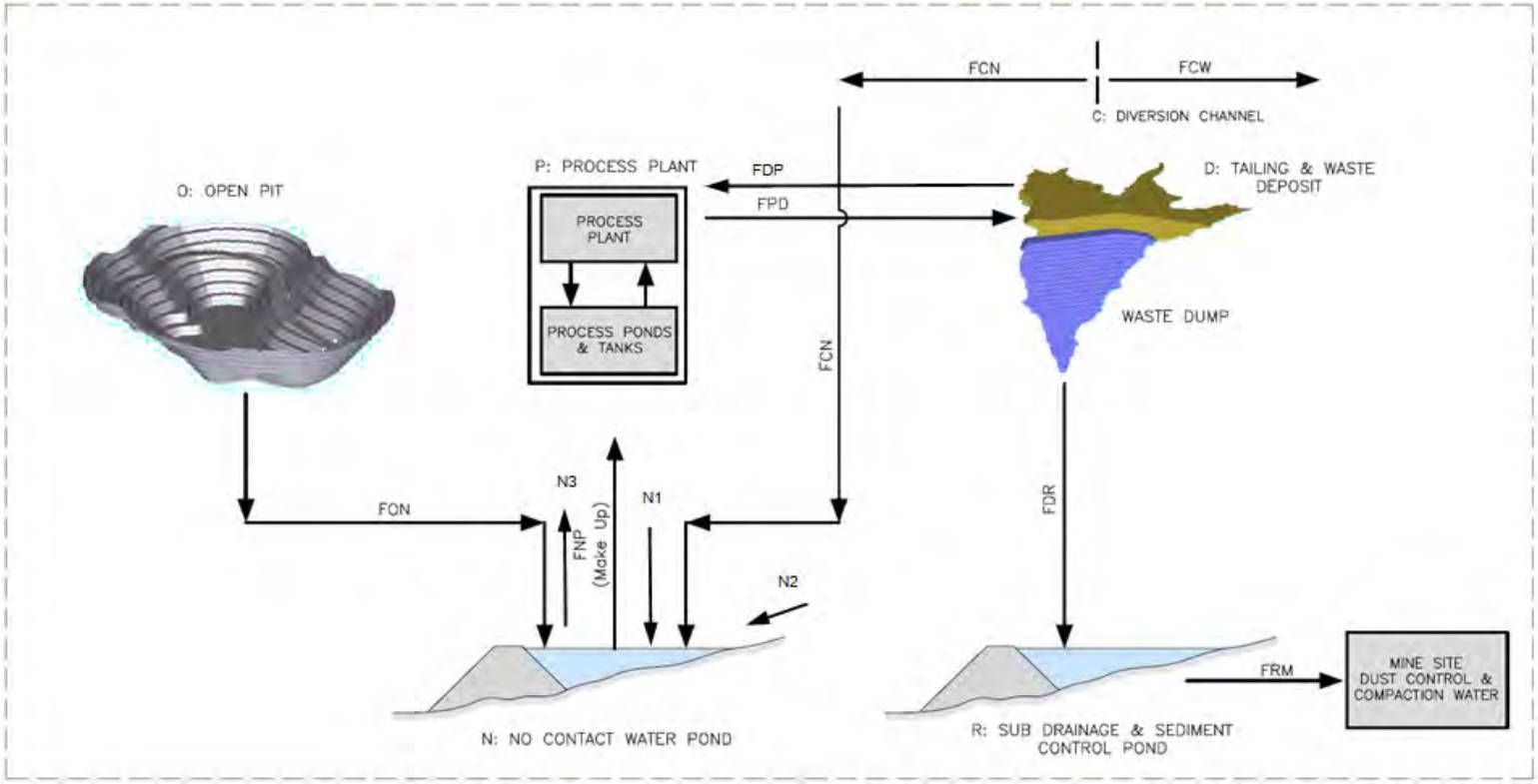
18.5 Site Infrastructure

The following infrastructure has been considered in order to support mine and plant operations:

- Security office: 20 m^2
- Training building for 15 persons: 40 m^2
- Medical centre: 100 m^2
- Camp: 10,000 m^2
- Diner: 1,200 m^2
- Fuel station: 110 m^2
- Administrative plant offices: 200 m^2
- Plant warehouse: 600 m^2
- Maintenance workshop: 400 m^2
- Process plant sample warehouse: 150 m^2
- Laboratories: 1,250 m^2
- Mine administrative offices: 300 m^2
- Mine operation offices: 250 m^2
- Mine control booth: 20 m^2
- Explosive warehouse: 150 m^2
- Six-bay truck shop (including spares warehouse): 13,500 m^2
- Mine and geology sample warehouse: 2,500 m^2
- Provisional laydown area: 1,950 m^2

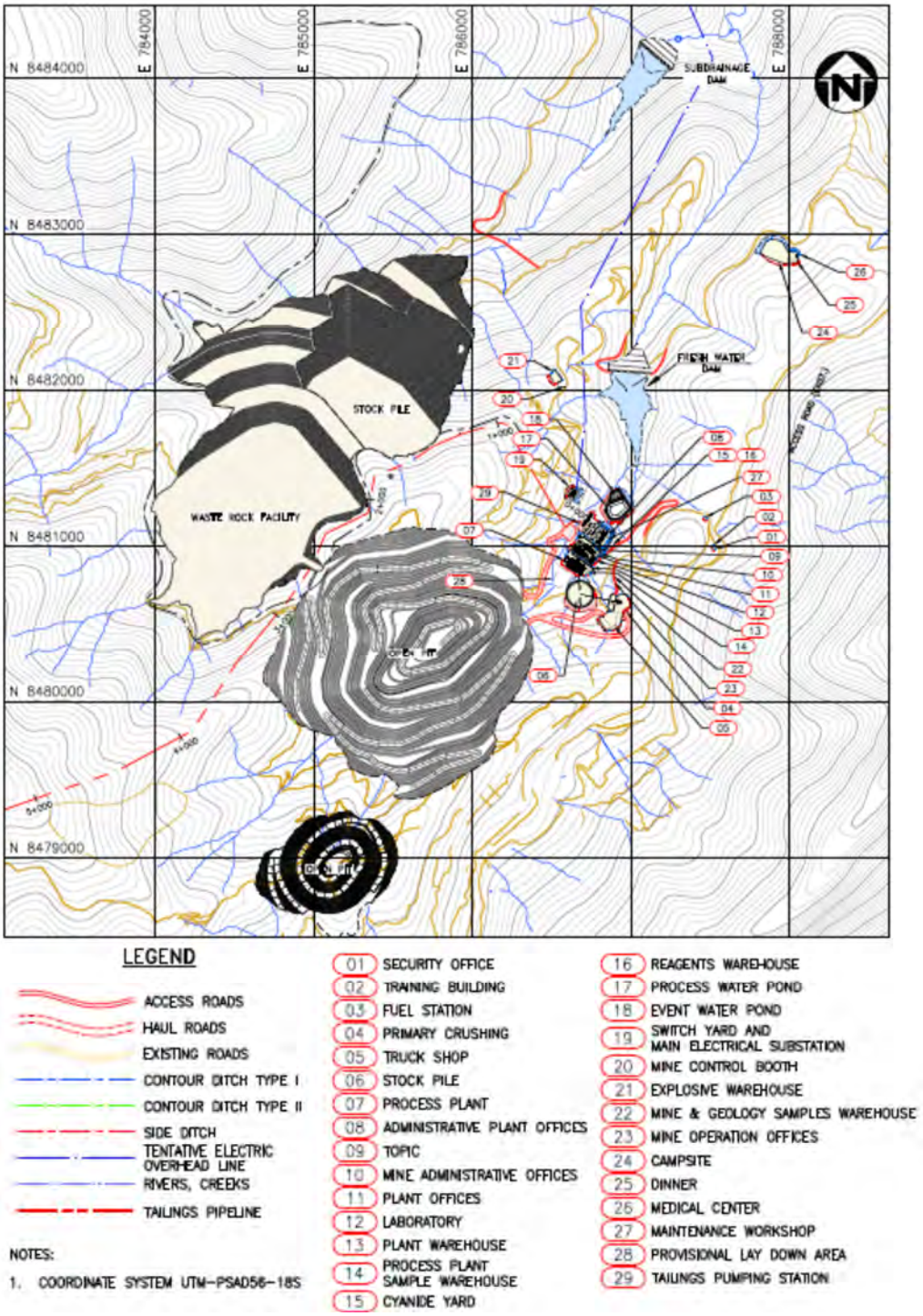
The location of each of these components is illustrated in Figure 18-4.

Figure 18-3: Water Balance Schematic



Note: Figure prepared by Amec Foster Wheeler, 2015

Figure 18-4: Project Infrastructure Layout Plan



Note: Figure prepared by Amec Foster Wheeler, 2015

18.6 Camps and Accommodation

A permanent operations camp facility has been designed and will be located northeast of the process plant area, approximately 3 km down the existing access road to the Huallpachaca Bridge over the Apurimac River. The 10,000 m^2 camp (and associated construction laydown area) will have catering and accommodation capacity for approximately 400 persons. A temporary camp will provide accommodation during construction and be located within the permanent operations camp area.

18.7 Power and Electrical

An annual power consumption of 120 MW has been estimated. The Project will be connected to the national electrical network through the extension of an existing substation located in Abancay (Panapex 2014). The Project considers this extension and a 61 km 220 kV transmission line from Abancay to Cotabambas.

The starting elevation for the powerline is at 3,229 m and it will terminate at 3,850 m after passing through a low elevation of 2,037 m. This line will feed a substation (three winding transformers 220/22.9/13.8 kV 150MVA) that will distribute power to the plant site, mine, camp site and other auxiliary buildings.

No permanent electrical power supply is anticipated for the TSF.

18.8 Solid Waste Management

The solution for disposal of solid waste will be through construction and implementation of a landfill near to the campsite platform area. It is recommended that in future studies, an evaluation of a solution for disposal of solid waste off site through a contract with a solid waste management company is undertaken

18.9 Comments on Section 18

The electrical design assumes that space will be available for expansion of the existing Abancay substation.

At the time of the study no geotechnical investigation program, surface geological mapping, specific hydrological study of the sites or seismic hazard studies had been completed. Conceptual work was developed using survey information including demarcation of the watersheds.

For the next stage of the study Amec Foster Wheeler recommends that acid water tests are completed to determine if a water treatment plant will be required.

Tailings design and construction represents a significant opportunity to the Project if it can be effectively optimized. Future work should consider revaluation of alternative locations and deposition strategies.

The road upgrades should be completed prior to the initiation of construction activities. This will facilitate transport of personnel and equipment. Delays in road construction could impact construction productivity.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No market studies have specifically been conducted on the Cotabambas Project. The final sulfide copper concentrate produced is expected to be shipped to smelters located in China, Japan or India. The copper concentrate is also expected to realize important payable silver and gold credits at various times in the mine life. No credit for Mo has been considered.

Testwork results indicate the copper concentrate will be relatively clean and can reasonably be expected to be marketable with a Cu grade in the range of 25 to 28%.

No deleterious elements that could have a significant effect on potential economic extraction have been detected in the concentrates produced to date by the various test programs.

The sulfide concentrate sales are expected to be subject to minor concentrate losses during transport, land freight, storage and handling, ocean freight, transport insurance, smelting and refining charges, and results in a pay factor deduction. These costs are presented in Section 22 of this Report.

A marketing plan and sales terms for the copper concentrate are recommended to be determined during the next study phase.

19.2 Commodity Pricing

Project economics were estimated on the basis of Amec Foster Wheeler long-term guidelines rates for copper at $3.25/lb, gold at $1,300/oz and silver at $20.50/oz, as of January 2015, derived from prices periodically published by a number of large banking and financial institutions.

These prices were updated in the updated PEA and changed to copper at $3.00/lb, gold at $1,250/oz and silver at $18.50/oz.

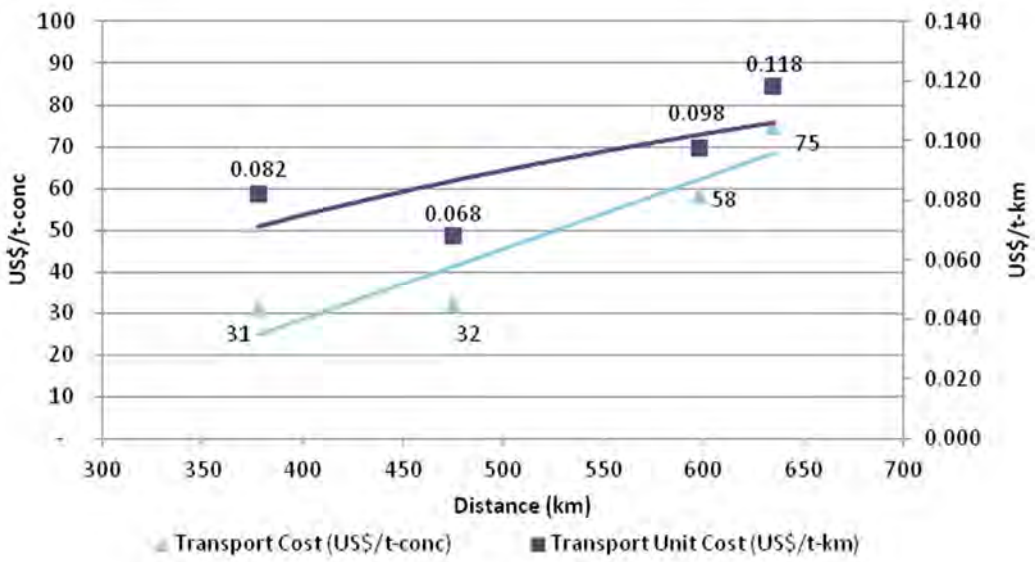
19.3 Freight

For the purpose of this study, Amec Foster Wheeler estimated the land freight cost to transport the concentrate from Cotabambas to Matarani utilizing costs sourced from benchmarks. This cost was estimated at $58/t of concentrate assuming that the 598km route from Cotabambas to Matarani is 45% asphalted. Table 19-1 and Figure 19-1 present the benchmark utilized to establish land freight costs for the copper concentrate.

Table 19-1: Benchmark Description for Land Freight Cost

Description	Benchmark 1	Benchmark 2	Benchmark 3
Route Characteristics	65% asphalted	40% asphalted	55% asphalted
Transport Distance to Matarani (km)	378	635	475

Figure 19-1: Benchmark Costs for Land Freight



Note: Figure prepared by Amec Foster Wheeler, 2015

Ocean freight was estimated at $55/t and port storage and handling costs were assumed to be $5/t. A marketing and umpiring allocation of $0.60/t was also included.

19.4 Smelter Terms

For the copper concentrate, expected to have a grade of 27%, the pay factor was assumed to be 96.3% which corresponds to the unit deduction. A treatment charge of $85/t was assumed along with copper refining charge of $0.085/lb payable.

Gold is assumed to be payable based on the assumptions in Table 19-2. A gold refining charge of $5/oz payable was assumed.

Silver is assumed payable based on the information in Table 19-3. A silver refining charge of $0.50/oz payable was assumed.

Table 19-2: Gold (grade in concentrate, g/t-conc)

L Min	L Max	Unit	Au_Payable
0.000	0.999	%	0.0%
0.999	2.999	%	90.0%
2.999	4.999	%	94.0%
4.999	9.999	%	96.0%

Table 19-3: Silver (grade in concentrate, g/t-conc)

L Min	L Max	Unit	Ag_Payable
0.000	0.999	%	0.0%
0.999	2.999	%	86.0%
2.999	4.999	%	88.0%
4.999	9.999	%	90.0%

19.5 Contracts

No contracts have been signed at the reporting date of this study.

At this stage of Project development, it is expected that future sales contracts would be negotiated such that the sales contracts would be typical of, and consistent with, standard industry practice, and be similar to contracts for the supply of copper–gold concentrate elsewhere in the world.

19.6 Comments on Market Studies and Contracts

Information presented in this section can be used to support an economic analysis of the Project at the PEA level.

As the Project progresses into the next study stages, Amec Foster Wheeler recommends that ports, potential transport and shipping companies and smelters are contacted to obtain firmer estimates for shipping and smelter treatment terms and refining charges.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Conditions

The baseline information presented in this sub-section is summarized from information collected by Schlumberger Water Services (SWS, 2012). SWS was contracted by Panoro in 2012 to prepare a Semi-detailed Environmental Impact Assessment (EIAsd) that was approved by the Ministry of Energy and Mining in 2012.

The main economic activities conducted by the communities located in the vicinity of the Project consist of small-scale agriculture and animal grazing. There are no industrial activities present in the Project area.

Figure 20-1 and Figure 20-2 illustrate the current physiography in the area of the proposed South Pit and the North Pit, respectively.

Air quality and noise measurements conducted in the Project area show levels are typical of rural areas in Perú and comply with environmental quality standards. The closest meteorological station with 15 years of historic records is located in the Town of Tambobamba, capital of the District of Cotabambas. Average annual precipitation is 937 mm with most of the precipitation occurring during the wet season between December and March.

The Project components included in the PEA are situated in catchments that discharge towards the Apurimac River. The planned open pits would occupy areas of Ccalla Creek while the WRF is proposed to be located in Duraznomayo Creek. Both drainages join to discharge into the Apurimac River approximately 3 km downstream of the Project.

The planned TSF location sits in the headwater of two basins, which also drain towards the Apurimac River. Since the two basins are located at a higher elevation than the proposed plant site, pumping of tailings will be required during the operations phase of the Project.

Figure 20-1: Proposed Location of South Pit

Figure 20-2: Proposed Location of North Pit

Note: Photographs taken by Amec Foster Wheeler, 2014

A water quality program covering surface water (i.e., creeks) and springs present in the Ccalla and Duraznomayo catchments potentially affected by the Project has established that the water is suitable for agricultural and human consumption. However, exceedances in surface water were identified for parameters such as iron, aluminium and manganese, potentially associated with the presence of the mineralized zones located within the catchments. The presence of coliforms potentially linked to the raising of animals has also been identified in some of the surface water bodies in the Project area. Biological water resources identified in the local catchments are limited, and no fish were identified in the Ccalla or Duraznomayo streams.

The presence of undisturbed habitats such as forest or wetlands is very limited, which also limits the presence of wildlife. However, during the baseline investigations conducted in 2012, one wildcat (*leopardus jocobitus*) and one deer (*hippocamelus antisensis*) were observed. These two species are listed as endangered and vulnerable, respectively, under Peruvian regulations.

20.2 Environmental and Social Issues

A formal environmental and social impact assessment will be required to be completed for the Project. This section presents an overview of the key issues associated with the Project at an early-stage, PEA-level of evaluation.

Potential environmental and social effects are associated with activities that will be undertaken during construction, operations, and closure of the Project, and include:

- Loss of land currently used by the communities of Ccalla and Ccochapata for subsistence agricultural activities
- Loss of biological aquatic resources in the Duraznomayo and Ccalla Creeks to accommodate mining facilities and provide water during operations
- Potential effects on water quantity downstream of the Project site due to dewatering activities and contact water uses within the Project layout
- Potential effects on water quality downstream of the Project site due to discharge of mining effluents
- Permanent changes in natural landforms for the development of the open pit, construction of the TSF, and development of the waste rock dumps
- Temporary effects on air quality and temporary increases in noise emissions during the construction and operation phases
- Cumulative impacts due to the presence of other mining operations in the Apurimac basin upstream of the Project site.

- Potential for relocation of some stakeholders from the communities of Ccalla and Ccochapata, and the town of Cotabambas and surrounding district.

20.2.1 Mine Waste Management

Preliminary geochemical testing has been conducted for waste rock, indicating that the rock does not have the potential to generate acid (SWS, 2012). Leaching tests show some potential for the release of manganese.

Segregation of waste rock is currently not considered; its placement is proposed in one facility in the WRF located to the north of the main open pit in the Duraznomayo Creek.

A comprehensive geochemical characterization will be required as part of the environmental impact assessment (EIA).

20.2.2 Mine Water Management

Site run-off will constitute the main source of water for the Project. Mine water will be recycled as much as possible and evaporation and seepage losses will be minimized in order to reduce fresh water requirements for the Project and avoid potential effects on other surface and groundwater users in the vicinity of the Project.

The Project will require a water licence for use of fresh water and an authorization to discharge liquid effluents, if applicable.

A comprehensive assessment of effects on water quality and quantity will be required as part of the EIA. Potential effects on water quantity downstream of the Project site due to dewatering activities and the use of contact water in mining activities needs to be quantified during future studies. In addition, the assumption in the PEA that water quality downstream of the Project will remain unaffected by the discharge of mining effluents needs to be verified. An assessment of the likely effects on stakeholders in the Project area of influence will need to be conducted, together with an appropriate assessment of permittability, social impacts, financial impacts, and mitigation measures that may be applicable.

20.3 Closure Plan

The EIA for the Project will include a conceptual closure plan to obtain the environmental approval. A detailed Mine Closure Plan must be submitted within one year after EIA approval. Posting of the corresponding financial assurance for closure must also be completed before the start of production and within the first 12 business days of the following year during which the closure plan is approved, as stated by the Regulaciones para el Cierre de Minas or Mine Closure Regulations. The final closure plan will be submitted two years prior to closure of operations.

A provision of $50 M has been made in the Project financial model as presented in Section 22 to take into account the closure costs.

20.4 Permitting

20.4.1 Exploration

Permitting applicable to the exploration stages is discussed in Section 4.

20.4.2 Development

A comprehensive environmental and social impact assessment will be necessary for the Project in order to obtain necessary permits for construction, operations, and closure. This assessment will be conducted in compliance with Peruvian regulations, including the following key regulations:

- Ley 27446, Ley del Sistema Nacional de Evaluación de Impacto Ambiental
- Reglamento de Proteccion y Gestion Ambiental para las Actividades de Explotacion, Beneficio, Labor General, Transporte y Almacenamiento Minero, D.S. 040-2014, EM
- Reglamento de Consulta y Participacion Ciudadana en el Procedimiento de Aprobacion de los Estudios Ambientales en el Sector Energia y Minas, R.M. 596-2002-EM/DM.

20.4.3 Construction and Operations

Once the environmental and social impact assessment is approved by Peruvian authorities, a variety of permits, licenses, and authorizations will be required to proceed with the construction and operations of the Project. Main permitting requirements include:

- EIA Amendments (if material changes are made to the Project)
- Mine Closure Plan approval, including posting of financial assurance
- Certificate of Non Existence of Archaeological Remains (CIRA)
- Water use authorizations and final licence
- Sanitary authorization, approving wastewater treatment system and discharge
- Sanitary authorization for drinking water treatment system
- Registration as a direct consumer of liquid fuels (fixed or mobile facilities)
- Bi-annual authorization for explosives use (global authorization)
- License to operate explosives magazine

- User's certificate for controlled chemical substances and products
- Operation license for radioactive equipment
- Authorization for the exploitation of construction materials – quarries
- Beneficiation concession
- Authorization to initiate mining exploitation.

20.5 Considerations of Social and Community Impacts

Cotabambas Province is one of seven provinces that make up the Department of Apurimac, under the administration of the Regional Government of Apurimac, Peru.

Cotabambas Province is divided into six political districts; one of these districts is also named Cotabambas. The provincial capital is Tambobamba. The nearest town to the Project is also named Cotabambas.

The District of Cotabambas has a population of approximately 4,250 inhabitants; whereas the District of Coyllurqui, has a population of approximately 7,900 inhabitants.

20.5.1 Potential Social Impacts

The majority of Project components (i.e., open pit, WRF, plant, water dams and camp, and a portion of the TSF) are located in the District of Cotabambas under the conceptual PEA design. The District of Coyllurqui would host the remainder of the TSF.

Agreements with the communities of Ccochapata and Ccalla were obtained to advance the exploration activities around the mineralized zones, and exploration camps were set up in each of these two communities. Exploration activities employed members of Cochapata and Ccalla communities. Section 4 provides details of the agreements obtained with local communities for land use.

A provisional evaluation of the likely land requirements for the Project was completed, and included estimation for facilities including safety buffers around the proposed open pits as per the Peruvian Mining Legislation requirements (MEM, 2010). This indicated that stakeholders from the communities of Ccalla and Ccochapata, the town of Cotabambas and surrounding district are likely to be affected by Project development and some relocation of stakeholders may be required. At this stage of evaluation, the estimations of the area affected and number of stakeholders is preliminary and will need to be refined during future more detailed studies. The proximity of the open pit to the town of Cotabambas in particular will require careful consideration.

Mitigation measures to avoid, reduce, or compensate for potential Project effects will need to be developed and supported by comprehensive environmental and social baseline investigations and engineering studies.

20.5.2 Consultation Considerations

Consultation activities with stakeholders that have conducted to date have supported exploration activities. Successful consultation conducted is evidenced by the minimal interruptions and opposition from local communities during exploration activities that have been conducted by Panoro.

The main stakeholder groups that may potentially be affected by the development of the Project include the following (Figure 20-3):

- Centro Poblado (town of Cotabambas
- Distrito de Cotabambas (District of Cotabambas)
- Comunidad Campesina Ccochapata
- Comunidad Campesina Ccalla;
- Comunidad Campesina de Guaclle
- Comunidad Campesina de Agpitan
- Other communities that may be affected by the western portion of the TSF.

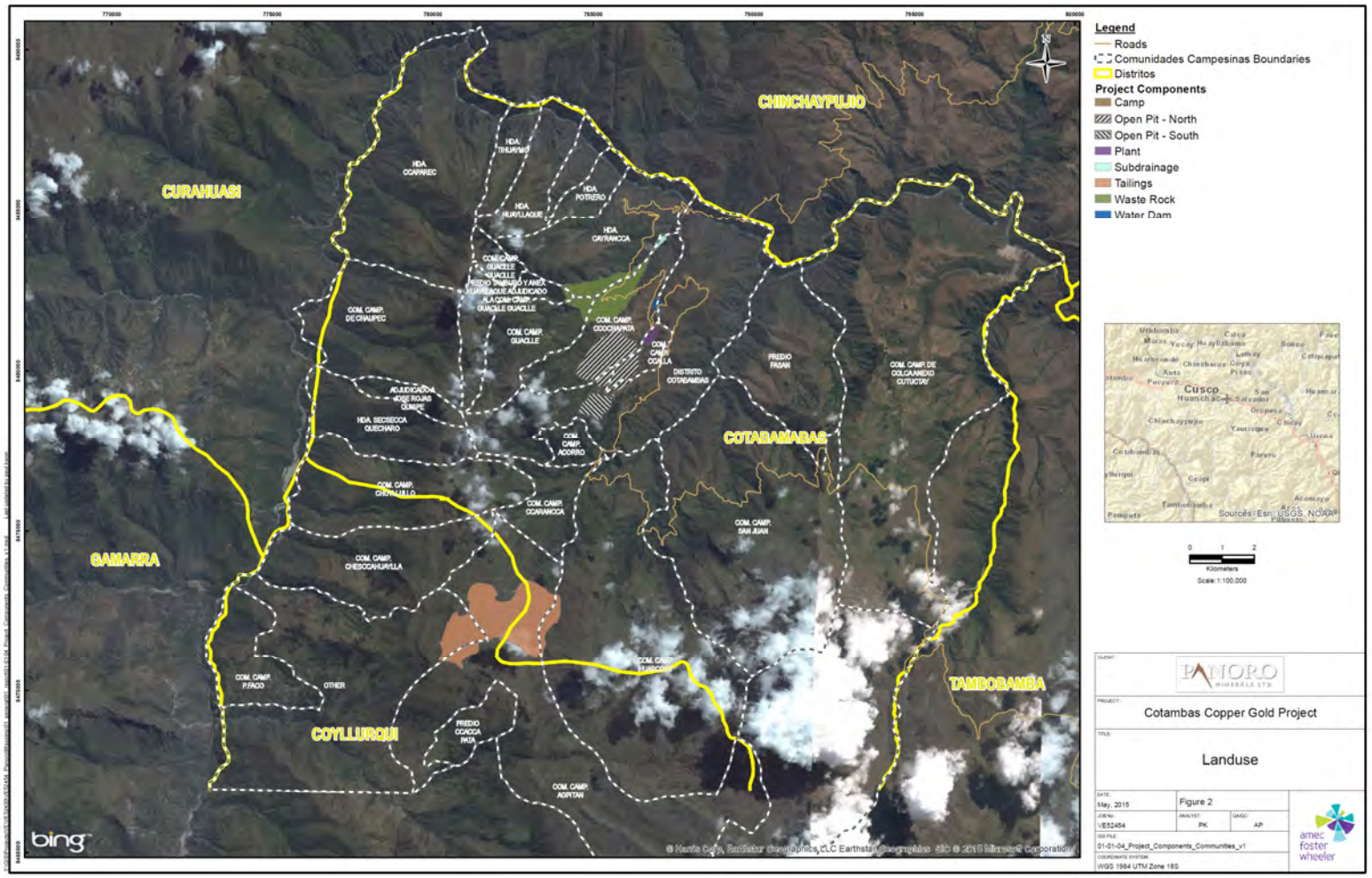
20.6 Discussion on Risks to Mineral Resources

The main risks associated with the development of the Project at a PEA level of understanding include the following:

- A comprehensive resettlement process including extensive consultation with community leadership and local authorities is likely to be required to proceed with Project development. Recent experiences in Perú on resettlement caused by mining development indicate that the costs of resettlement can be very high and delay project execution significantly.
- The mine plan assumes that Panoro will be able to collect surface runoff for mining and processing needs; this assumption is supported by the preliminary water balance. However, any assumptions of impact on stakeholders have not yet been evaluated. Future consideration may require mitigation strategies or possibly compensation payments as required.

Mine waste management currently assumes that mine waste will not be acid generating or incur metal leaching, therefore segregation of waste would not be required. This assumption is based on geochemical testing on a very limited number of waste rock samples. Given the importance of mine waste characterization for mine waste planning and closure planning, more comprehensive geochemical studies are needed.

Figure 20-3: Project Conceptual Major Infrastructure Layout Plan in Relation to Approximate Community Boundaries



Note: Figure prepared by Amec Foster Wheeler, 2015

20.7 Comments on Section 20

A comprehensive environmental and socio-economic impact assessment will be required. Environmental and social baseline information will be required, including consultation with potentially affected stakeholders. Resettlement planning will be an important component of the next stages of Project development. The next stages of Project design should include the application of best design principles that require Project proponents to avoid, minimize, mitigate, or compensate Project effects and to apply design for closure principles to minimize long-term effects to return the land to a productive use.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Basis of Estimate

The Cotabambas PEA capital cost estimate is considered to be a Class 5 estimate in accordance with the Association for the Advancement of Cost Engineers (AACE) capital cost estimate definitions. The accuracy of this estimate is considered to be within -35% to +50%. The level of definition expressed as a percentage of total engineering is in the range of 0 to 2% and engineering completion is within a range of 0 to 1%. A summary of the total initial capital cost for the project considering an own- operate open pit mine approach is presented in Table 21-1. The capital cost estimate is expressed in US Dollars for the third quarter of 2014. Costs associated with escalation beyond the third quarter of 2014, currency fluctuations, interest during construction, and property acquisition or taxes are excluded from this estimate.

The capital cost estimate consists of estimates of the direct and indirect costs for the open pit mine, mineral process plant, on site and off site infrastructure, including auxiliary buildings, TSF, WRF, camp site, electrical power supply, water management, and access road upgrades.

All capital costs were estimated by Amec Foster Wheeler with the exception of the capital cost estimates for the electrical power supply, road and Owner's costs which were estimated by Panoro, and the mining, mine equipment and WRF costs, which were estimated by MMTS. These estimates were integrated to produce a total capital cost estimate for the Project.

The capital cost estimate includes mining equipment costs, construction labor and equipment, permanent materials and equipment, and construction contractor indirect costs for surface facilities, as well as the associated indirect costs, such as management costs and contractor support for commissioning. All costs following the handover of the mineral process plant to the operations group or start of operating revenues are considered as operating expenditures or sustaining capital.

Construction contractors' general and administrative costs such as supervision, general expenses and profit were included as part of the direct cost of the Project.

Engineering, procurement and contract management (EPCM) and supporting facilities costs during construction were included in the capital cost estimate as part of the indirect cost of the Project. EPCM costs were estimated based on benchmarked factors.

Table 21-1: Total Capital Cost Summary

Area	Description	Cost Estimate (US$M)
Mining		
	Mine Equipment	236.3
	Mine Development	127
	Mine Infrastructure	17.9
Tailings Disposal		
	Starter Tailings Dams	4.4
	Tailings Disposal System	73.7
Process Plant		
	Process Plant	505.
Infrastructure		
	On Site Infrastructure	67.2
	Off Site Infrastructure	27.5
Sub-Total Direct Cost		*1059.2*
	Owners Cost	40.0
	Indirect Cost	152
Sub-Total Indirect Cost		*192*
	Contingency	235
Total Capital Cost		***1,486.2***

The capital expenditure of $1.53 B stated in the press release of September 22, 2015 includes both initial capital and estimated closure costs. The initial capital required, without provision for closure, is estimated to be $1.49 B.

Panoro's corporate general and administrative (G&A) costs during construction and commissioning were included as part of the Owner's costs.

The capital cost estimate was based on the following project information:

- Preliminary design criteria
- Preliminary process flow sheet
- Preliminary major mechanical equipment list
- Preliminary general site layout
- Budgetary quotations for major mechanical equipment and benchmark factors to account for non-quoted equipment or minor equipment
- Allowances, factors and prices based on experience and historical database for similar facilities
- Preliminary project schedule.

21.1.2 Labor Assumptions

Labor cost assumptions used by MMTS were provided by Panoro.

All labor costs required for the construction and management during construction of the Project facilities are included in the capital cost estimate.

Construction labor costs are in line with Peruvian regulations and are based on rates established by the Peruvian Construction Chamber (CAPECO) and the civil construction union.

The Owner's team labor costs are considered to be included in the Owner's cost estimate included as a Project indirect cost.

21.1.3 Material Costs

All materials required for the construction of the Project facilities are included in the capital cost estimate. Material costs include freight to the Project site.

21.1.4 Contingency

A contingency of U S $235.0 M has been included in the capital cost estimate, corresponding to 16% of the total initial capital.

21.1.5 Owner (Corporate) Costs

An estimate of US$40 M for Owner's costs was provided by Panoro and included in the capital cost estimate. This estimate is considered to account for Owner's staff, recruitment, Owner's general and administrative costs for Project support, Owner's management team on site, Owner's team training, and Owner's safety expenses in the areas of site, permitting, social responsibility, and insurance

21.1.6 Sustaining Capital

The sustaining capital cost estimate includes the mine capital and tailings storage facility expansion costs. Table 21-2 presents the summary of the sustaining capital estimate.

Table 21-2: Sustaining Capital Cost Estimate Summary

Description	Cost Estimate (US$M)
Mine Equipment	21.8
Sub-Total Mine Sustaining Capital Cost	*21.8*
Tailings Storage	
Dams Expansion	378.9
EPCM	26.5
Sub-Total Tailing Storage Sustaining Capital Cost	*405.5*
Total Sustaining Caital Cost Estimate	***427.2***

The mine sustaining capital cost includes mine equipment replacement.

The tailings storage facility sustaining capital includes the construction and required EPCM services.

21.1.7 Closure Costs

A provision of US$50.0 M including contingency has been included in the estimate to account for closure costs. This provision was estimated by Panoro and included a 10% contingency. The summary of this estimate is presented in Table 21-3. The closure cost has been equally applied from Year 19 to Year 23.

21.1.8 Capital Cost Summary

Table 21-4 presents the summary of the capital cost estimate that includes initial capital, sustaining capital and closure costs.

21.2 Operating Cost Estimates

21.2.1 Basis of Estimate

The operating cost estimate includes the open pit mining, minerals processing, and general and administrative operating costs of the project. A summary of the operating cost estimate is presented in Table 21-5.

Table 21-3: Closure Cost Estimate Summary

Description	Factor	Cost Estimate (US$M)
Closure Cost	-	45.5
Contingency	10% of direct cost	4.5
Total Closure Cost Estimate		***50.0***

Table 21-4: Capital Cost Estimate Summary

Description	Cost Estimate (US$M)
Initial Capital Cost	
Mining	381.2
Tailings Disposal	78.2
Process Plant	505.2
Site Infrastructure	67.2
Off Site Infrastructure	27.5
Owners Cost	40.0
Indirect Cost	152.0
Contingency	235.0
Total Capital Cost	*1,486.2*
Sustaining Capital Cost	
Mine	21.8
Tailings Storage	405.5
Total Sustaining Capital	*427.2*
Closure Cost	45.5
Contingency	4.5
Total Closure Cost	*50.0*
Total Capital Cost Estimate	***1,963.4***

The capital expenditure of $1.53 B stated in the press release of September 22, 2015 includes both initial capital and estimated closure costs. The initial capital required, without provision for closure, is estimated to be $1.49 B.

Table 21-5: Operating Cost Estimate Summary

Description	LOM Cost Estimate (US$/t)	LOM Cost Estimate (US$M)
Mining	$1.59/t of material mined (excluding pre-stripping) $3.59/t of mill feed processed	1,732
Processing	$4.38/t of mill feed processed	2,116
General and Administration	$0.41/t of mill feed processed	197
Total Operating Cost Estimate		***4,045***

The accuracy of this estimate is considered to be within ±35%. All operating costs estimates are expressed in US Dollars for the third quarter of 2014. Costs associated with escalation beyond the third quarter of 2014, currency fluctuations, marketing and sales costs, royalties, treatment charges/refining charges (TC/RCs), product shipping costs, interest charges, licenses or taxes are excluded from this estimate.

No contingency has been included in this estimate. All process operating costs were estimated by Amec Foster Wheeler.

21.2.2 Mine Operating Costs

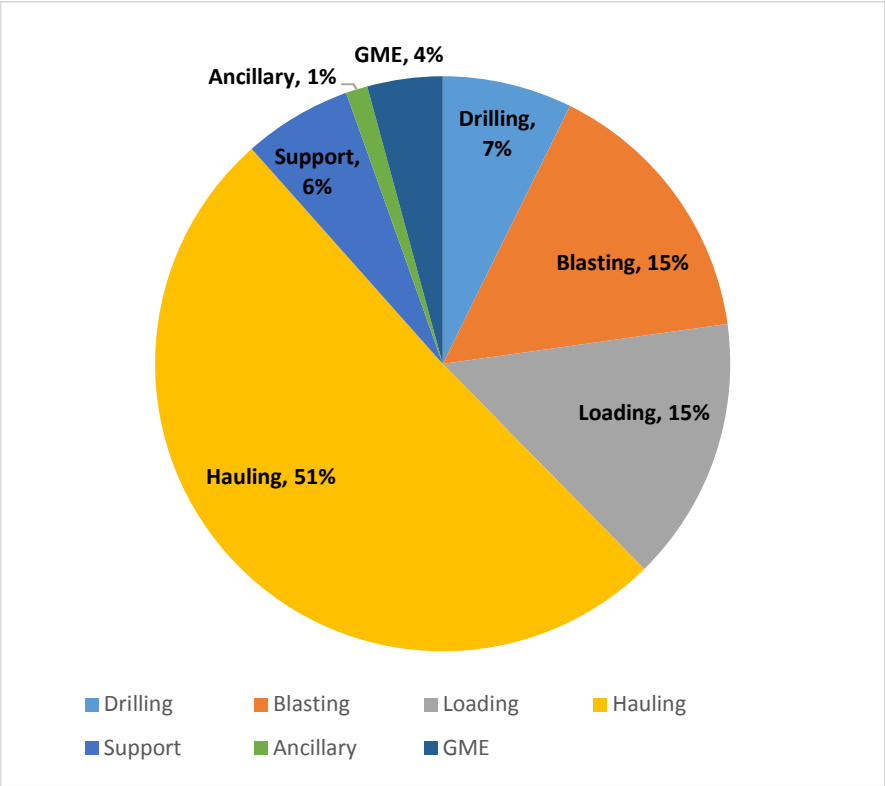
The mine operating costs were estimated by Moose Mountain Technical Services using a cost model built-up from first principles. Mine operating costs are based on a mixture of owner-operated and rental-owner operated haul trucks as well as, production quantities, consumable costs from budgetary quotations obtained by Panoro and benchmarking, and a staffing plan prepared for the Cotabambas Project.

The mine operating cost estimate includes the costs of drilling, blasting, loading, hauling, support and ancillary equipment operations and maintenance, and mine operations supervision required to produce run-of-mine material to feed the process plant.

Figure 21-1 shows the breakdown of the costs in a pie chart. Table 21-6 presents the costs of major consumables utilized in the estimation of the mine operating costs. The costs are based on budgetary quotations obtained by Panoro and benchmarking.

Mine staff wages and salaries are based on benchmarked wages and salaries from similar mine operations in Perú that Panoro and Amec Foster Wheeler are familiar with.

Figure 21-1: Mining Cost Composition



Note: Figure prepared by MMTS, 2015.

Table 21-6: Consumables Unitary Costs

Description	Unit	Cost	Source
Diesel	US$/liter	0.80	Benchmark
Bulk Ammonium Nitrate	US$/t	636	Quoted by Panoro
Bulk Emulsion	US$/t	675	Quoted by Panoro
Blast Stemming / Fuse /Booster	US$/hole	35	Quoted by Panoro
Blast Accessories	US$/hole	2.6	Quoted by Panoro
Tire – 793/830	US$/unit	45,000	Benchmark
Tire – Wheel Loader_844 / WD900-3	US$/unit	15,000	Benchmark
Tire – 793F	US$/unit	45,000	Benchmark
Tire – FEL_994F	US$/unit	52,000	Benchmark
Steel Bar	US$/unit	31,274	Benchmark
Bit	US$/unit	2,000	Benchmark

21.2.3 Process Operating Costs

The process operating cost includes the costs of power, reagents and consumables, labor, maintenance, services and fuel required to process the Cotabambas mineralized material to produce Cu/Au concentrate. Table 21-7 and Table 21-8 present the process operating cost summaries by cost centre and area.

The process operating cost estimate is based on a preliminary mass balance, preliminary mechanical equipment list, preliminary list of reagents and consumables, and a staffing plan prepared for the Cotabambas project.

The estimated LOM process operating cost of the project is US$2,116 M or an equivalent of $4.38/t of mill feed processed.

The cost of power estimated at a total LOM of US$927 M is based on the total estimated power derived from the relevant equipment list and load lists with appropriate factors based on operating time and estimated demand. The cost of power was estimated utilizing the unit cost of US$0.0612/kW-hr. Table 21-9 presents the annual power cost estimate.

The cost of reagents and consumables was estimated at a total LOM of US$746 M. The cost estimate of reagents is based on the process requirements for each area and on the average flow rates to be treated and includes the cost of the flotation and thickening reagents required to produce Cu/Au concentrate. The cost estimate of consumables includes the steel consumption based on the mineral work and abrasion indexes for the crushing and grinding areas. Table 21-10 and Table 21-11 present the annual reagents and consumables cost estimates, respectively.

The cost of labor estimated at a total LOM of US$180 M includes the costs of general, operations, maintenance, metallurgy, and laboratory personnel. The labor cost component is based on a staffing plan and wages and salaries from similar process plant operations in Perú which Amec Foster Wheeler is familiar with. Table 21-12 presents the labor annual cost estimate.

The cost of maintenance was estimated at a total LOM of US$184 M. It includes the maintenance cost for the process plant and thickened tailings pipeline. The cost of spare parts that are considered to be 5% of the mechanical equipment cost included in the capital cost estimate. The cost of maintenance of the tailings disposal pipeline (13.3 km) is considered to be 3% of the pipeline supply and installation costs. Table 21-13 presents the annual maintenance cost estimate.

Table 21-7: Process Operating Cost Summary by Cost Centre

Description	LOM Cost Estimate (US$/t of mill feed processed)	LOM Cost Estimate (US$M)	Cost Distribution
Power	1.92	927	43.8%
Reagents and Consumables	1.54	746	35.3%
Labor	0.37	180	8.5%
Maintenance	0.38	184	8.7%
Services	0.16	76	3.6%
Fuel	0.01	3	0.2%
Total Process Operating Cost Estimate	***4.38***	***2,116***	***100%***

Table 21-8: Process Operating Cost Summary by Area

Description	LOM Cost Estimate (US$/t of mill feed processed)	LOM Cost Estimate (US$M)	Cost Distribution
Crushing	0.15	73	3.5%
Grinding	2.48	1,199	56.7%
Flotation	0.72	348	16.4%
Concentrate Thickening	0.11	53	2.5%
Tailings Thickening	0.70	338	16.0%
Services (Inc. Fuel and Others)	0.20	105	5.0%
Total Process Operating Cost Estimate	***4.38***	***2,116***	***100%***

Table 21-9: Annual Power Cost Estimate

Area	Charge of Operation (kW)	Factor of Electric Charge	Average Demand (kW)	Annual consumption (MW)	kWh/t Milled	Annual Cost Estimate (US$M)
Crushing	1,257	0.95	1,191	9,597	0.33	0.6
Milling	73,500	0.95	69,632	561,175	19.22	34.3
Flotation	9,430	0.95	8,934	72,002	2.47	4.4
Concentrate Thickening and Filtration	791	0.95	749	6,039	0.21	0.4
Tailings Thickening	26,174	0.95	24,796	199,839	6.84	12.2
Services (3%)	3,335	0.95	3,159	25,460	0.87	1.6
Minor Equipment (5%)	5,558	0.95	5,265	42,433	1.45	2.6
Total Annual Power Cost Estimate	***120,045***		***113,727***	***916,545***	***31***	***56.1***

Table 21-10: Annual Reagents Cost Estimate

Reagent	Product Form	Consumption (g/t)	Consumption (t/year)	Unit Cost (US$/t)	Delivery Point	Annual Cost Estimate (US$M)
Aerofloat 208	Liquid	14.00	376.10	$4,475	On Site	1,7
MIBC (Flot Cu Bulk)	Liquid	19.80	531.91	$3,875	On Site	2.1
Sodium Cyanide (Flot Cu Bulk)	Liquid	20.00	537.28	$4,875	On Site	2.6
Lime	Liquid	950.00	25,520.80	$191	On Site	4.9
Z-11 (Flot Cu Bulk)	Solid	6.00	161.18	$2,425	On Site	0.4
Flocculant 1120	Solid	15.00	402.96	$4,375	On Site	1.8
Total Reagents Annual Cost						***13.4***

Table 21-11: Consumables Annual Cost Estimate

Equipment	Consumption	Cost (US$)	Annual Cost (US$M)
Primary Crushing			
Liner	2 set/year	$244,210/set	0.5
Milling			
SAG			
Liners	2.0 set/year	$2,015,200/set	4.03
Balls	320 g/t of mill feed	$1,275/t	11.0
Ball Mill			
Liners	2 set/year	$934,524/set	1.9
Balls	450 g/t of mill feed	$1,234/t	14.9
Screen Panels	2 set/year	$55,050/set	0.1
Hydrocyclones			
Vortex	12 unit/year	$770/unit	0.009
Apex	180 unit/year	$188/unit	0.034
Regrinding			
Ball Mill			
Balls	30 g/t of mill feed	$1,300/t	1.0
Hydrocyclone Regrinding			
Vortex	8.3 unit/year	$650/t	0.005
Apex	50 unit/year	$227/t	0.011
Total Consumables Annual Cost Estimate			***33.5***

Table 21-12: Labor Annual Cost Estimate

Position	Roster	Level	Quantity	Annual Cost Estimate (US$K)
General				
Operative Manager	5x2	9	1	234
Plant Manager	5x2	8	1	207
Maintenance Manager	5x2	8	1	207
Maintenance Planner	14x7	6	2	206
HSSE	14x7	5	8	464
Community Relationships	14x7	5	8	464
Security	14x7	2	25	403
Assistant	14x7	2	6	116
General Sub-Total			*51*	*2,301*
Operations	14x7			
Shift Manager		6	4	412
Crushing				
Supervisors		6	3	335
Technical		4	3	165
Workers		3	4	155
Milling				-
Supervisors		6	3	335
Technical		4	3	165
Workers		3	4	155
Flotation				-
Supervisors		6	3	335
Technical		4	3	165
Workers		3	4	155
Concentrate Thickening and Filtration				-
Supervisors		6	3	275
Technical		4	3	165
Workers		3	4	155
Tailing Thickening				-
Supervisors		6	3	275
Technical		4	3	165
Workers		3	4	155
Warehouse				-
Supervisors		6	3	335
Storekeepers (Flotation Reagents)		3	4	155

Position	Roster	Level	Quantity	Annual Cost Estimate (US$K)
Storekeepers (Sodium Cyanide)		3	4	155
Operations Sub-Total			*66*	*4,208*
Maintenance	14x7			
Plant Maintenance				
Supervisors		6	6	618
Mechanics		5	1	64
Electrics		5	1	64
Instrumentalist		5	12	773
Workers		3	8	309
General Warehouse				-
Supervisors		6	2	206
Workers		3	8	309
Maintenance Sub-Total			*38*	*2,344*
Metallurgy and Laboratory	14x7			
Superintendent		7	1	116
Supervisors		6	2	206
Metallurgists		5	15	966
Preparer of sample		4	12	618
Tester		4	12	618
Workers		3	8	309
Storekeepers (Samples of Process)		3	4	155
Metallurgy and Laboratory Sub-Total			*54*	*2,988*
Total Annual Labor Cost Estimate			***209***	***11,841***

Table 21-13: Annual Maintenance Cost Estimate

Area	Annual Cost Estimate (US$M)
Crushing	0.7
Milling	5.2
Flotation	1.2
Concentrate Thickener	0.2
Tailings (Thickening & Pumping System)	3.8
Total Maintenance Annual Cost Estimate	***11.1***

The cost of services estimated at a total of US$76 M includes the costs of supplies and services associated with the operations of the medical center, dining room, campsite, product transport, tailings transport, and car, cranes and equipment rental. The cost of services is assumed to be 2% of the mechanical equipment cost included in the capital cost estimate. The supplies for chemical laboratory were estimated at US$2.50 per sample for 150 samples per day. Table 21-14 presents the supplies and services annual cost estimate.

The cost of fuel estimated at a total of US$3 M is based on a preliminary fuel consumption estimate per type of vehicle expected to be used and a unit cost of US$3.79/gal. Table 21-15 presents the monthly fuel consumption estimate.

21.2.4 Infrastructure Operating Costs

Infrastructure operating costs are considered to be included in the G&A operating cost.

21.2.5 General and Administrative Operating Costs

The G&A operating cost was estimated at a total LOM of US $197 M based on an average cost of US$0.41/t of mill feed processed applied, which was a benchmarked cost from similar operations in Peru.

21.2.6 Owner (Corporate) Operating Costs

Owner operating costs are considered to be included in the G&A operating cost estimate.

21.2.7 Operating Cost Summary

Table 21-16 presents the operating cost estimate summary.

21.3 Comments on Capital and Operating Costs

No specific costs for land acquisition or provision for relocation of the communities affected by the Project were established. These costs fall within Owners costs. This will need to be further evaluated during the development of the PFS as local communities are engaged.

Owner's costs are developed based on assumed organization and conceptual execution strategies. Changes in execution strategy or organizational structure could impact on the Owner's cost assumptions.

Table 21-14: Supplies and Services Annual Cost Estimate

Description	Quantity	Unit	Unit Cost (US$/unit)	Annual Cost Estimate (US$K)
General				
Medical Center	1	#		247
Dining Room	1	#		247
Campsite	1	#		247
General Sub-Total				*742*
Chemical Laboratory				
Materials Laboratory	150	samples/day	1.25	63
Chemical Tests (Plant Products)	150	samples/day	1.25	68
Chemical Laboratory Sub-Total			*2.5*	*131*
Transporting of Products				
Transporting of Concentrate				2,969
Transporting of Products Sub-Total				*2,969*
Various				
Renting of Vehicles		unit/day		210
Renting of Cranes		unit/day		210
Renting of Equipment		unit/day		210
Others (15%)				111
Various Sub-Total				*742*
Total Supplier and Services Annual Cost Estimate				***4,584***

Table 21-15: Monthly Fuel Consumption Estimate

Equipment	Unit	Km/t	Km/month/v.	Consumption Estimate (L/month)
Trucks	6	2	2,000	6,000
Water Trucks	2	2	2,000	2,000
Buses	4	3	2,000	2,667
Emergency Vehicles	1	7	10,000	1,429
Cranes	2	5	3,000	1,200
Tractors	-	2	400	-
Vans	10	8	2,000	2,500
Cars	-	10	4,000	-
Others (4%)				1,580
Total Fuel Monthly Consumption Estimate				***17,375***

Table 21-16: Operating Cost Estimate Summary

Description	LOM Cost Estimate	LOM Cost Estimate (US$M)
Mining	$1.59/t of material mined (excluding pre-stripping) $3.59/t of mill feed processed	1,732
Processing	$4.38/t of mill feed processed	2,116
General and Administration	$0.41/t of mill feed processed	197
Total Operating Cost Estimate		***4,045***

The assumed contingency amount is at the lower end of the range for a PEA, therefore there is a greater risk that this amount will be insufficient to cover all of the unknown capital costs.

No provision has been made for water treatment capital or operating costs. Based on the information available waste rock is not acid generating. This will need to be further evaluated during more detailed studies.

Assumptions made with respect to geotechnical properties including hazard assessments could require revision during detailed studies once geotechnical information is available, as these data could directly on designs and capital costs of the plant and infrastructure.

Estimates are preliminary and based on limited available information. Changes to the schedule or Project scope could impact the overall cost of the Project.

Final tailings and waste rock design could require additional capital to be expended to achieve production. Current concepts defer tailings raises until after production as part of sustaining capital.

Opportunities to optimize tailings could result in capital reductions. Tailings construction is a significant portion of the overall capital costs.

Factors utilized to establish plant capital costs are within a range of benchmarks of in Perú. The nature of the Project could result in a final cost during detailed design which uses different assumptions and parameters from those used to establish the capital costs at a PEA level.

22.0 ECONOMIC ANALYSIS

The following section is preliminary in nature and partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Dates cited are for illustrative purposes only, as a decision to proceed with mine construction will require formal approvals from appropriate regulatory authorities and from Panoro's Board, and will require support from additional, more detailed, studies, and declaration of Mineral Reserves.

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource estimates
- Assumed commodity prices and exchange rates
- The proposed mine production plan
- Projected recovery rates
- Selection of sites for key infrastructure such as the WRF and TSF
- Infrastructure construction costs and proposed operating costs
- Assumptions as to closure costs and closure requirements
- Project development schedules, including resettlement activities
- Assumptions that Panoro will be able to permit the design and operate to the projected design assumptions
- Assumptions that Panoro will be able to acquire the social licence to construct the envisaged Project and operate it under the design assumptions.

Additional risks to the forward-looking information include:

- The timing and amount of estimated future production
- Costs of production
- Capital expenditures and any requirements for additional capital costs
- Timing of the development of the deposits

- Permitting time lines, government regulation of mining operations
- Environmental risks
- Unanticipated reclamation expenses
- Title disputes or claims and limitations on insurance coverage
- Changes in project parameters as mine and process plans continue to be refined
- Possible variations in quantity of mineralized material, grade or recovery rates
- Geotechnical considerations during mining
- Failure of plant, equipment or processes to operate as anticipated
- Accidents, labour disputes and other risks of the mining industry

22.1 Methodology Used

Financial analysis of the Cotabambas Project was carried out using a discounted cash flow (DCF) approach based on "beginning of period" discounting (the cash flow occurs at the start of each year). This method of valuation requires projecting yearly cash inflows (or revenues) and subtracting yearly cash outflows (such as operating costs, capital costs, royalties and taxes). The resulting net annual cash flows are discounted back to the date of first year of capital expenditure and totaled in order to determine the NPV of the Project at selected discount rates.

Cash flows are assumed to occur at the beginning of each year.

The IRR is expressed as the discount rate that yields an NPV of zero.

The payback period is the time calculated from the start of production until all initial capital expenditures have been recovered. Payback has been calculated on an undiscounted basis.

This economic analysis includes sensitivities to variations in operating costs, capital costs, taxes and metal prices.

All monetary amounts are presented in United States dollars (US$).

22.2 Financial Model Parameters

22.2.1 Resource and Mine Life

Mineralized material will be processed at an average rate of 29.3 Mt/a over a planned production life of approximately 17 years. Prior to mill start-up there is one year of pre-stripping activity.

22.2.2 Smelting and Refining Terms

Smelting and refining terms applied in the financial model are shown in Table 22-1.

22.2.3 Metal Prices

Evaluation of this Project was based on long-term metal prices of US$3.00/lb for copper and US$1,250/oz for gold and $18.50/oz for silver.

22.2.4 Operating Costs

Life of mine total operating costs used for the financial analysis of the Base Case are as shown in Table 22-2.

22.2.5 Capital Costs

The distribution of the estimated project capital costs is shown in Table 22-3.

22.2.6 Royalties

The Peruvian government currently levies a sliding-scale royalty on gross sales from mining operations that ranges between 1% and 12%, which is imposed on operating mining income.

This is not a commercial royalty for the benefit of current or former property owners but is rather a former of taxation.

Panoro considers that a minimum royalty of 1% of mining sales would be applicable to the Cotabambas Project at a PEA stage of evaluation.

The current financial model estimates the total value of royalty payment at US$109.3 M.

22.2.7 Working Capital

A working capital allocation was included in the cash flow model. The allocation varies throughout the Project life, beginning at US$75.2 M and peaks at US$ 92.7 M during the fourth year of production. The assumption is made that all of the working capital can be recovered at Project termination. Thus, the sum of all working capital over life of mine is zero.

Table 22-1: Smelting and Refining Terms

Smelter Terms	Unit	Total
Moisture content	%	8.0%
Concentrate losses	%	0.10%
Land freight	US$/wmt	58.00
Port Charges: Storage & handling	US$/wmt	5.00
Ocean freight	US$/wmt	55.00
Marketing & Other	US$/dmt	0.60
Insurance premium	US$/wmt	0.40
Treatment charge	US$/dmt	85.00
Refining Charges	US$/lb	0.085
Pay factor	%	96.3%

Table 22-2: Operating Cost for Life of Mine (LOM)

Description	LOM Cost Estimate	LOM Cost Estimate (US$M)
Mining	$1.59/t of material mined (excluding pre-stripping) $3.59/t of mill feed processed	1,732
Processing	$4.38/t of mill feed processed	2,116
General and Administration	$0.41/t of mill feed processed	197
Total Operating Cost Estimate		**4,045**

Table 22-3: Project Total Capital Costs

Item	Unit	Amount
Initial Capital	US$M	1,486.2
Mining	US$M	381.2
Tailings	US$M	78.2
Process Plant	US$M	505.2
Site Infrastructure	US$M	67.2
Off Site Infrastructure	US$M	27.5
Indirect Cost + Owner + Contingency	US$M	427.0
Sustaining capital	US$M	427.2
Sustaining Mine	US$M	21.8
Sustaining Tailings	US$M	405.5
Closure	US$M	50
Closure costs	US$M	50
Total Capital Costs	***US$M***	***1,963.4***

The capital expenditure of $1.53 B stated in the press release of September 22, 2015 includes both initial capital and estimated closure costs. The initial capital required, without provision for closure, is estimated to be $1.49 B

22.2.8 Taxes and Depreciation

The corporate income tax rate in Perú is scheduled to be 26% in the year that production is planned to commence. Profit sharing in Perú currently is 8%. Mining companies also pay a new tax in Perú known as the "Special Mining Tax" based on a sliding scale, with progressive marginal rates ranging from 2% to 8.4% of the company´s operating profit.

The depreciation method applied is considered as a function of time based on the progressive obsolescence of the limited assets life, and therefore considers the usefulness of such steadily decreasing over time. The depreciation terms in the Cotabambas financial model for development, equipment and infrastructure are one, five, and 17 years (LOM) respectively.

Based on the above assumptions, the amount of special mining tax, worker tax and income tax for the duration of the Project is US$104 M, US$231 M and US$690 M respectively.

Amec Foster Wheeler does not provide expert advice on taxation matters. Taxes were determined based on information provided by Panoro that was supported by public-domain documentation.

22.2.9 Closure Costs

Provisional closure costs of US$50 M have been included in the model as provided by Panoro Minerals Ltd. A more detailed assessment of closure cost will be developed as the closure plan for the Project is defined during future, more detailed studies.

22.2.10 Financing

The economic analysis is based on 100% equity financing.

22.2.11 Inflation

The economic analysis included no inflation.

22.3 Financial Results

Financial analysis of the Project resulted in a pre-tax NPV at 7.5% discount rate of US$1,052.6 M, an IRR of 20.4% and a payback period of 3.2 years.

On an after tax basis, the financial analysis indicated the Project NPV 7.5% to be US$683.9 M and the IRR to be 16.7%. The project payback period after tax is calculated to be 3.6 years.

Table 22-4 provides a summary of the financial analysis for the after-tax scenario.

Figure 22-1 and Figure 22-2 show the annual cash flows and cumulative cash flow on an undiscounted and discounted basis. Figure 22-3 presents the financial analysis sensitivity to the discount rate.

Table 22-5 presents the detailed cashflow on an annualized basis.

22.4 Cash Costs

C1 cash costs are displayed in Table 22-6. The life of mine cash cost per pound of payable copper, after secondary metal credits, is US$1.22/lb.

22.5 Sensitivity Analysis

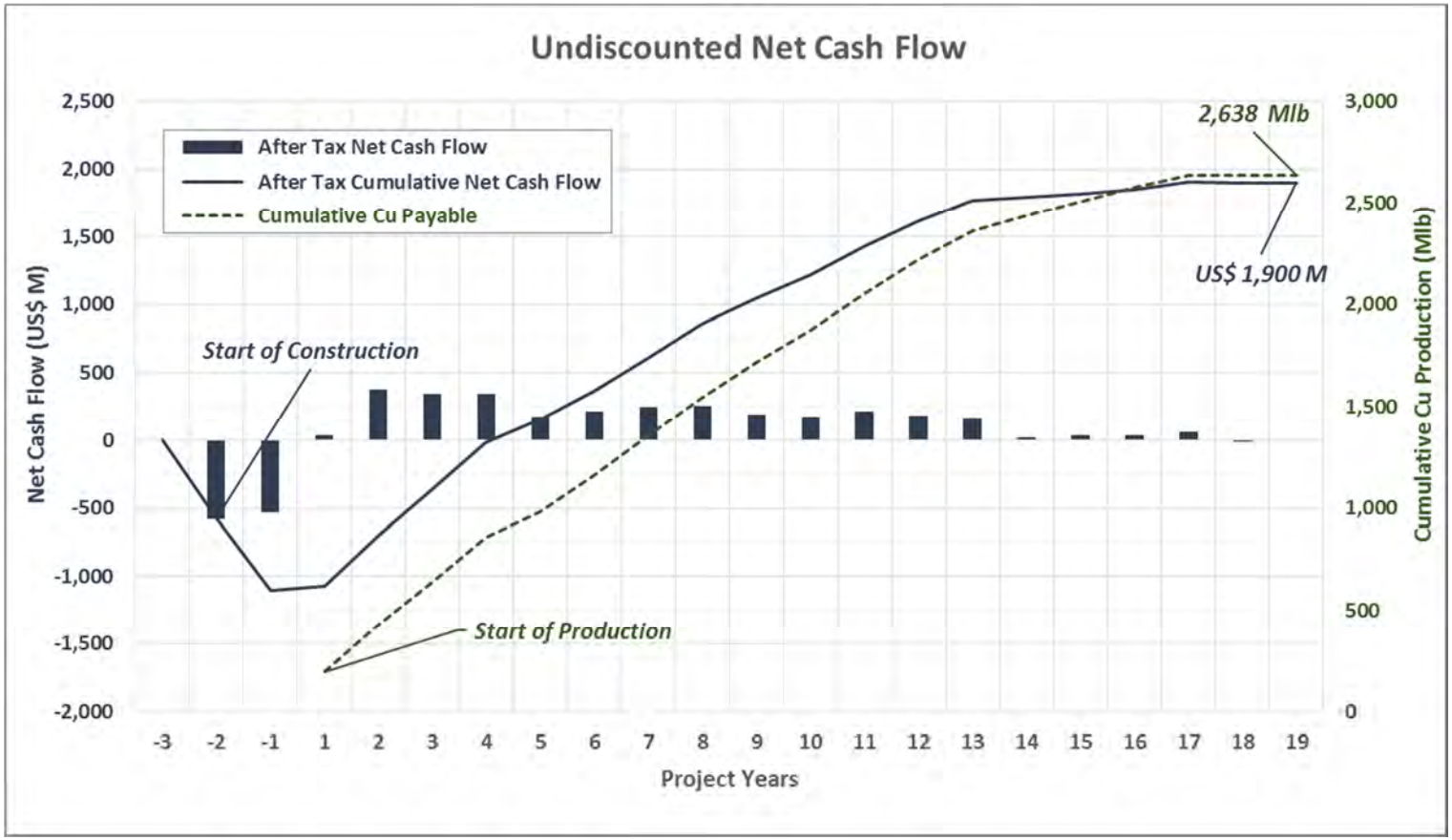
Sensitivity analysis was performed taking into account variations in copper metal prices, copper grade, gold metal prices, gold grade, operating costs and capital costs. The results from the analysis showed that the project NPV sensitivity is (in order from highest to lowest) copper metal price, copper grade, capital costs, operating costs, gold metal price, and gold grade. Gold grade and gold metal price have the same effect on the NPV.

The results are shown graphically for NPV in Figure 22-4.

Table 22-4: Summary of Financial Results (after tax)

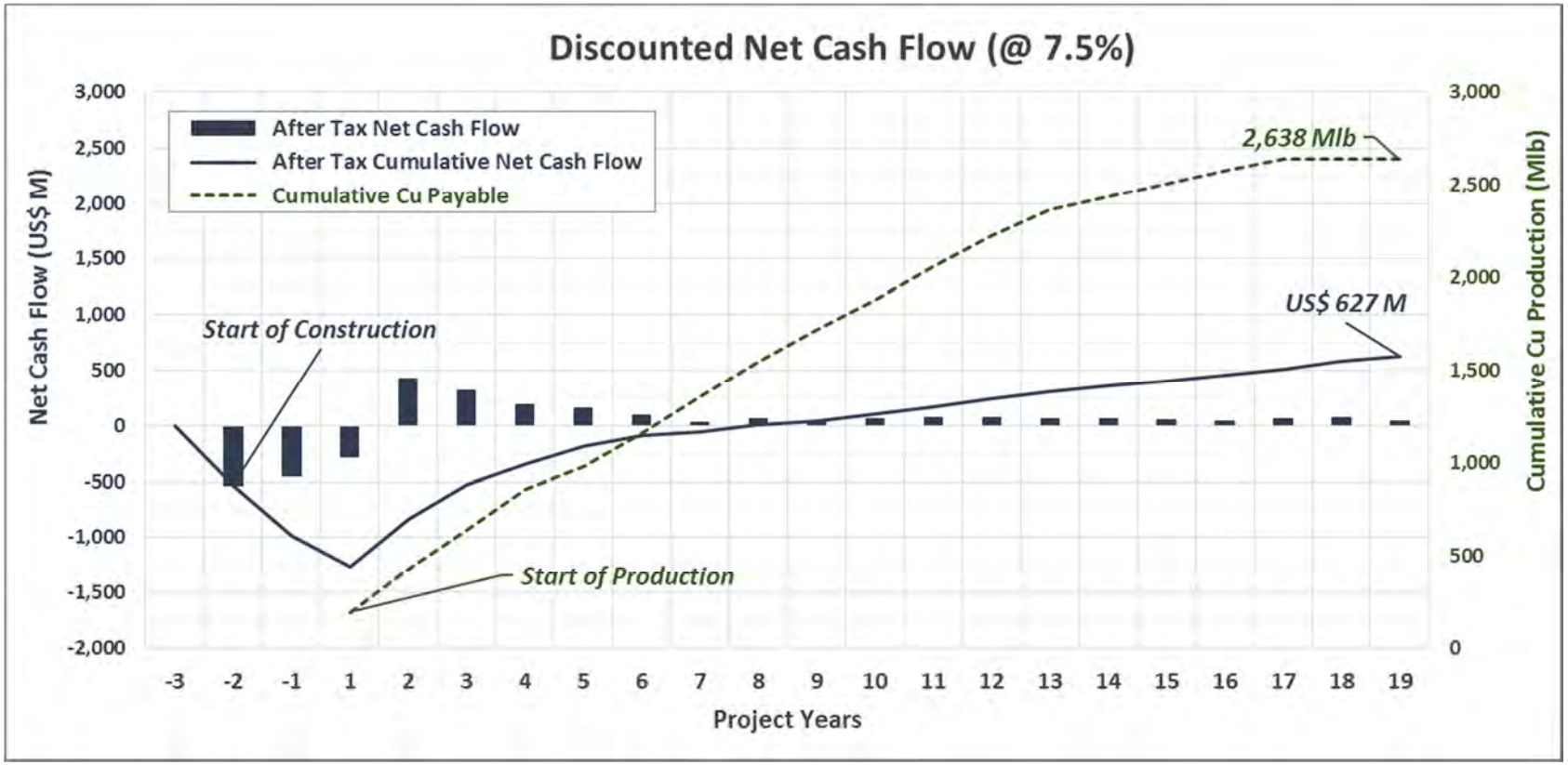
Item	Unit	Total
Payable metal		
Cu	M lb	2,638.3
Au	M oz	1.6
Ag	M oz	17.3
Cash costs (C1)		
Total cash costs	$/lb	2.16
Secondary metal credit	$/lb	(0.94)
Cash costs net of credits	$/lb	1.22
Cash costs (C2		
C1 - Net Direct Cash Cost	$/lb	1.22
Depreciation	$/lb	0.72
C2- Production Cost	$/lb	1.94
Financial		
Cumulative net cash flow	$M	1,900
Internal rate of return	%	16.7%
Net present value @ 7.5%	$M	684
Mine life	Years	19
Payback period	Years	3.6
Total start-up capital	$M	1,486
Total LOM capital	***$M***	***1,963***

Figure 22-1: Undiscounted Net Cash Flow



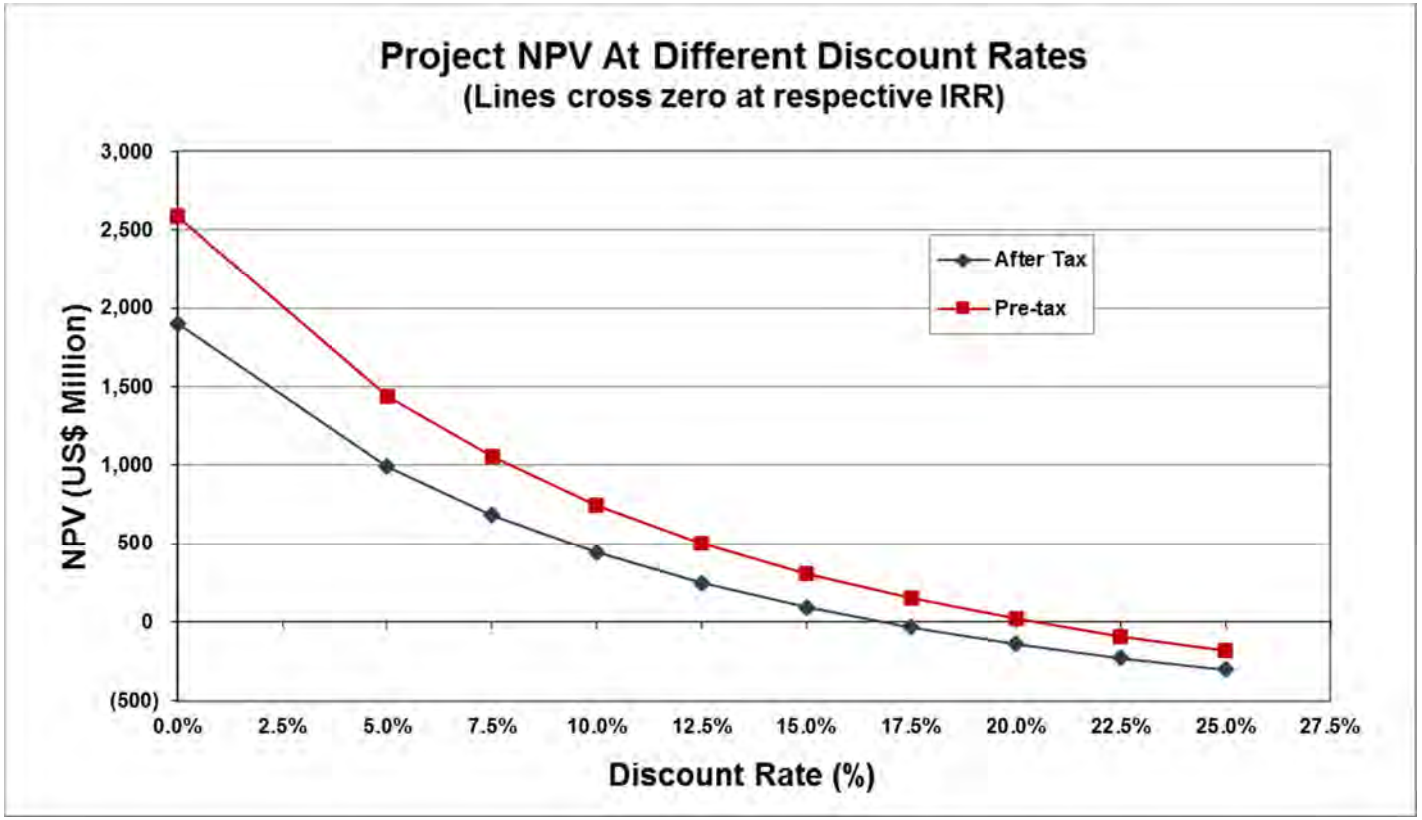
Note: Figure prepared by Amec Foster Wheeler, 2015.

Figure 22-2: Discounted Net Cash Flow



Note: Figure prepared by Amec Foster Wheeler, 2015.

Figure 22-3: Base Case Project Net Present Value at Different Discount Rates



Note: Figure prepared by Amec Foster Wheeler, 2015

Table 22-5: Cash Flow Model Summary for the Cotabambas Project

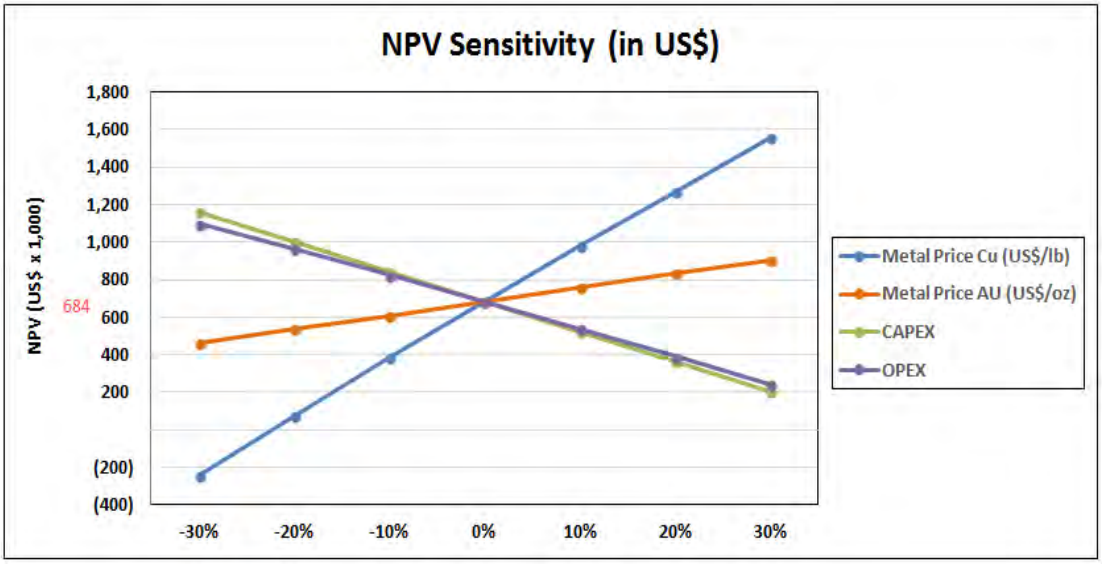
Project year			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Production year			**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**
Metal prices																						
Cu	US$/lb				3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Au	US$/oz				1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00	1,250.00
Ag	US$/oz				18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50
Capital expenditure																						
Construction, Sustaining and Closure	US$000	(1,963,414)	(578,353)	(526,011)	(381,825)	(3,204)	(10,724)	(11,851)	(21,860)	(5,233)	(35,454)	(9,662)	(36,997)	(22,802)	(38,012)	(44,944)	(47,801)	(55,084)	(54,701)	(58,895)	(10,000)	(10,000)
Extracted metal value																						
Concentrate																						
Cu payable	US$000	7,915,044			582,965	698,244	644,997	648,852	368,296	541,439	582,558	567,781	521,880	468,415	541,593	498,624	423,209	216,587	216,587	216,587	176,429	
Au payable	US$000	2,035,545			176,056	164,270	152,987	167,865	156,311	110,631	157,329	152,708	130,806	107,735	121,604	122,018	131,250	48,229	48,229	48,229	39,289	
Ag payable	US$000	320,310			10,780	16,582	19,581	24,132	11,987	21,888	26,579	25,627	21,849	16,409	21,110	23,174	19,668	15,976	15,976	15,976	13,013	
Total	US$000	10,270,898			769,801	879,096	817,565	840,849	536,595	673,959	766,466	746,117	674,534	592,559	684,308	643,815	574,127	280,792	280,792	280,792	228,731	
Smelter deductions																						
Concentrate																						
Cu Refining	US$000	(224,260)			(16,517)	(19,784)	(18,275)	(18,384)	(10,435)	(15,341)	(16,506)	(16,087)	(14,787)	(13,272)	(15,345)	(14,128)	(11,991)	(6,137)	(6,137)	(6,137)	(4,999)	
Au Refining	US$000	(8,142)			(704)	(657)	(612)	(671)	(625)	(443)	(629)	(611)	(523)	(431)	(486)	(488)	(525)	(193)	(193)	(193)	(157)	
Ag Refining	US$000	(8,657)			(291)	(448)	(529)	(652)	(324)	(592)	(718)	(693)	(591)	(443)	(571)	(626)	(532)	(432)	(432)	(432)	(352)	
Total	US$000	(241,059)			(17,513)	(20,889)	(19,416)	(19,708)	(11,384)	(16,375)	(17,853)	(17,391)	(15,900)	(14,146)	(16,402)	(15,242)	(13,047)	(6,761)	(6,761)	(6,761)	(5,508)	
Treatment charge																						
Cu concentrate	US$000	(391,225)			(28,815)	(34,513)	(31,881)	(32,071)	(18,204)	(26,762)	(28,795)	(28,064)	(25,795)	(23,153)	(26,770)	(24,646)	(20,918)	(10,705)	(10,705)	(10,705)	(8,721)	
Product transport																						
Concentrate																						
Land freight	US$000	(290,457)			(21,393)	(25,623)	(23,669)	(23,811)	(13,515)	(19,869)	(21,378)	(20,836)	(19,151)	(17,189)	(19,875)	(18,298)	(15,530)	(7,948)	(7,948)	(7,948)	(6,474)	
Port storage & handling	US$000	(25,039)			(1,844)	(2,209)	(2,040)	(2,053)	(1,165)	(1,713)	(1,843)	(1,796)	(1,651)	(1,482)	(1,713)	(1,577)	(1,339)	(685)	(685)	(685)	(558)	
Ocean freight	US$000	(275,434)			(20,286)	(24,298)	(22,445)	(22,579)	(12,816)	(18,841)	(20,272)	(19,758)	(18,161)	(16,300)	(18,847)	(17,351)	(14,727)	(7,537)	(7,537)	(7,537)	(6,140)	
Marketing & other	US$000	(2,764)			(204)	(244)	(225)	(227)	(129)	(189)	(203)	(198)	(182)	(164)	(189)	(174)	(148)	(76)	(76)	(76)	(62)	
Insurance charges	US$000	(2,003)			(148)	(177)	(163)	(164)	(93)	(137)	(147)	(144)	(132)	(119)	(137)	(126)	(107)	(55)	(55)	(55)	(45)	
Total	US$000	(595,698)			(43,875)	(52,551)	(48,543)	(48,834)	(27,719)	(40,750)	(43,844)	(42,732)	(39,277)	(35,254)	(40,761)	(37,527)	(31,851)	(16,301)	(16,301)	(16,301)	(13,278)	
Net Smelter Return	US$000	9,042,917			679,598	771,144	717,725	740,237	479,288	590,072	675,973	657,930	593,561	520,007	600,375	566,400	508,310	247,025	247,025	247,025	201,224	
Production costs																						
Mining	US$000	(1,732,246)			(155,378)	(171,945)	(166,433)	(182,323)	(150,754)	(151,309)	(106,065)	(112,806)	(111,237)	(91,805)	(80,603)	(93,444)	(74,249)	(28,425)	(19,254)	(18,793)	(17,424)	
Process	US$000	(2,115,834)			(89,746)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(128,115)	(104,363)	
G&A	US$000	(196,731)			(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(11,700)	(9,531)	
Total	US$000	(4,044,811)			(256,825)	(311,760)	(306,248)	(322,138)	(290,569)	(291,124)	(245,880)	(252,621)	(251,052)	(231,620)	(220,418)	(233,259)	(214,064)	(168,240)	(159,069)	(158,608)	(131,318)	
Closure & salvage																						
Closure costs	US$000	50,000																10,000	10,000	10,000	10,000	10,000
Taxation																						
EBITDA	US$000	4,998,105	0	0	422,774	459,384	411,477	418,098	188,719	298,949	430,094	405,309	342,509	288,387	379,957	333,141	294,246	78,785	87,956	88,417	69,906	0
Depreciation 1 - Detailled	US$000	(1,898,879)	0	0	(399,777)	(232,918)	(233,705)	(234,630)	(188,719)	(67,614)	(23,272)	(24,178)	(28,336)	(31,051)	(36,538)	(43,744)	(53,206)	(64,100)	(78,768)	(88,417)	(69,906)	0
EBIT	US$000	3,099,226	0	0	22,996	226,466	177,772	183,469	0	231,334	406,822	381,130	314,173	257,336	343,419	289,397	241,040	14,685	9,188	0	0	0
Royalty	US$000	(109,282)	0	0	(230)	(5,010)	(3,355)	(3,464)	0	(6,749)	(18,112)	(16,359)	(12,329)	(9,433)	(14,556)	(10,959)	(8,486)	(147)	(92)	0	0	0
Special Mining Tax	US$000	(103,739)	0	0	(460)	(5,994)	(4,397)	(4,539)	0	(6,992)	(15,627)	(14,315)	(11,183)	(8,805)	(12,800)	(10,089)	(8,061)	(294)	(184)	0	0	0
Income before Profit Sharing and Income Tax	US$000	2,886,205	0	0	22,306	215,463	170,019	175,466	0	217,593	373,083	350,457	290,661	239,097	316,062	268,349	224,493	14,244	8,912	0	0	0
Worker Tax	US$000	(230,896)	0	0	(1,785)	(17,237)	(13,602)	(14,037)	0	(17,407)	(29,847)	(28,037)	(23,253)	(19,128)	(25,285)	(21,468)	(17,959)	(1,140)	(713)	0	0	0
Carryover NOL available	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

Project year			**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**
Production year			**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**
Carryover NOL used	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Carry forward NOL	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Income before Income Tax	US$000	2,655,309	0	0	20,522	198,226	156,418	161,429	0	200,186	343,236	322,420	267,408	219,969	290,777	246,881	206,534	13,105	8,199	0	0	0
Income tax payable	US$000	(690,380)	0	0	(5,336)	(51,539)	(40,669)	(41,971)	0	(52,048)	(89,241)	(83,829)	(69,526)	(57,192)	(75,602)	(64,189)	(53,699)	(3,407)	(2,132)	0	0	0
Net Income	US$000	1,964,929			15,186	146,687	115,749	119,457		148,137	253,995	238,591	197,882	162,777	215,175	182,692	152,835	9,697	6,067			
EBIT		3,099,226			22,996	226,466	177,772	183,469		231,334	406,822	381,130	314,173	257,336	343,419	289,397	241,040	14,685	9,188			
Total Taxes (Royalty+SMT+Income tax)		(1,134,297)			(7,810)	(79,779)	(62,022)	(64,012)		(83,197)	(152,827)	(142,540)	(116,291)	(94,559)	(128,244)	(106,705)	(88,205)	(4,987)	(3,120)			
Capital expenditure																						
Construction & sustaining	US$000	(1,963,414)	(578,353)	(526,011)	(381,825)	(3,204)	(10,724)	(11,851)	(21,860)	(5,233)	(35,454)	(9,662)	(36,997)	(22,802)	(38,012)	(44,944)	(47,801)	(55,084)	(54,701)	(58,895)	(10,000)	(10,000)
Working capital	US$000				(75,175)	(15,903)	2,380	(4,045)	13,171	(3,396)	10,537	(1,407)	1,256	5,864	1,424	(2,402)	6,218	15,344	2,293	115	43,727	
Construction, sustaining & working capital	US$000	(1,963,414)	(578,353)	(526,011)	(457,000)	(19,107)	(8,344)	(15,896)	(8,689)	(8,630)	(24,917)	(11,069)	(35,742)	(16,938)	(36,588)	(47,346)	(41,584)	(39,740)	(52,409)	(58,779)	33,727	(10,000)
Net project cash flow																						
Pre-tax	US$000	2,590,774	(578,353)	(526,011)	38,474	427,939	379,400	384,208	166,858	262,567	331,054	336,936	258,747	228,218	289,303	245,681	211,938	22,121	32,266	29,522	59,906	(10,000)
After tax	US$000	1,900,394	(578,353)	(526,011)	33,138	376,400	338,731	342,236	166,858	210,518	241,812	253,107	189,220	171,026	213,701	181,492	158,240	18,714	30,134	29,522	59,906	(10,000)
Summary of Cash Flow		**After Tax**	**Pre-tax**																			
Cumulative net cash flow																						
Undiscounted	US$000	1,900,394	2,590,774																			
Net present value																						
Discounted at 5.0%	US$000	991,613	1,440,744																			
Discounted at 7.5%	**US$000**	**683,934**	**1,052,550**																			
Discounted at 10.0%	US$000	441,110	746,926																			
Discounted at 12.5%	US$000	247,190	503,501																			
Internal rate of return	**US$000**	**16.7%**	**20.4%**																			
Payback period	**Years**	**3.6**	**3.2**																			

Table 22-6: Summary C1 Cash Costs

Summary of Cash Costs	Unit	LOM total	Cost per tonne milled (US$/t)	Cost per pound Cu payable (US$/lb)
Cash Costs				
Mining	US$000	1,732,246	3,59	0.66
Process	US$000	2,115,834	4.38	0.80
G&A	US$000	196,731	0.41	0.07
Smelter deductions	US$000	424,513	0.88	0.16
Treatment charges	US$000	391,225	0.81	0.15
Refining charges	US$000	241,059	0.50	0.09
Transport & Marketing	US$000	595,698	1.23	0.23
Sub-total	US$000	5,697,306	11.79	2.16
Credits				
Au	US$000	(2,120,359)	(4.39)	(0.80)
Ag	US$000	(355,900)	(0.74)	(0.13)
Sub-total	US$000	(2,476,259)	(5.13)	(0.94)
Adjusted cash costs				
Total	***US$000***	***3,221,047***	***6.67***	***1.22***

Figure 22-4: Sensitivity of After-Tax NPV Discounted at 7.5% (in US$)



Note: Figure prepared by Amec Foster Wheeler, 2015

23.0 ADJACENT PROPERTIES

This section is not relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Risks and Opportunities

Although no a formal risk and opportunity analysis has been performed, the following major risks and opportunities have been identified for the Project from internal discussions and reviews and are applicable to the current level of Project knowledge and the PEA-level of evaluation.

24.1.1 Risks

A list of project risks and sensitivities is provided in this sub-section.

Delays in obtaining and preparing baseline information and in approvals process for the EIA may result in changes to projected timelines and nominated construction dates.

Water for the Project is considered to be obtained directly from creeks and rivers on and around the Project area. If needed, water could also be pumped from the Apurimac River. Although water storage facilities are planned, long unexpected dry seasons could make water unavailable for the Project at times during construction or operations.

No provision has been included for treatment of the contact water coming from the process plant, TSF and WRF prior to discharge to the surroundings. Future studies or permitting requirements may indicate the need for water treatment, thus increasing the cost and potentially impacting the Project schedule.

The Project is likely to require the relocation of some stakeholders. Delays and costs associated with obtaining stakeholder consents and the relocation process could impact the project cost and schedule. Small settlements that are potentially still unidentified in the Project site surroundings due to difficult geography could also potentially have an impact on the Project implementation. The unknown nature and the significant variability between agreements that have been concluded with resettled parties for other comparable projects in Perú makes it impractical to evaluate with certainty within the financials for this aspect until further engagement with the potentially affected stakeholders has occurred.

High rainfall during the Andean winter from November to April can make access road conditions more difficult and periodic delays due to road closures can be expected.

No evaluation of the impact of road improvements on the stakeholders or environment has yet been undertaken and as such could provide impediments to execution. Road upgrades must be completed prior to construction and as such require an alternative permitting strategy to be established.

Current construction costs are based on assumed costs from similar projects for earthworks and civil works, as geotechnical information is not available at this time.

Different soil conditions than assumed could have either a negative or positive impact on the project cost and schedule. Minimal contingencies have been applied at this time due to the unknown nature of the underlying soils.

The increase of copper price and utilization of constant recovery relative to grade has produced a decrease in the marginal benefit from the blocks, increasing the amount of low-grade material (<0.2% total copper). Testwork to data on sulphide composites with grades between 0.54% Cu to 0.22% Cu indicated copper recovery is relatively insensitive to grade in this range and this generally corresponds approximately to the average annual grades of considered in this PEAs mine plan. It is possible however that the expected recovery values are not reached, and this low-grade material, particularly material stockpiled and reclaimed at the end of the mine life, may not provide some or all of the projected economic values assumed in the PEA. Variability testing is planned in the next phase of work to establish or confirm the constant grade-recovery relationship assumed in this study.

The copper recovery of mixed oxide is estimated from the amount of recoverable copper sulphide contained in this type of mineralization inferred by copper sequential analysis classification in the resource model. No testwork has been conducted yet directly on samples of this material or blends with sulphide to confirm those assumed in this PEA and this is planned in the next phase.

The locations and designs for the WRF and TSF are based on assumed geotechnical and hydrogeological parameters, as such studies or investigations are not available at this time. Any changes would affect the cost estimates presented in this study.

Project execution is based on the assumption that Project financing and approvals will be available immediately, such that the development of baseline and feasibility study level work can proceed. Any delay in initiating these activities would impact the assumed production start date.

Cost estimates are provisional in nature and in some cases have limited contingencies applied. Changes in development and design could result in re-estimates of the infrastructure components associated with tailings, waste rock, water management, power line and road upgrade. Changes in design assumptions, and further information that will become available from baseline data, in particular in relation to Panoro's social and environmental responsibilities, may result in changes to the estimated Owner's costs.

Project economics were estimated on the basis of Amec Foster Wheeler long-term guidelines rates for copper at $3.00/lb, gold at $1,250/oz and silver at $18.50/oz, as of January 2015, derived from prices periodically published by a number of large banking and financial institutions. There is a risk in the future that the metal prices may be lower than assumed for this Project.

The smelter terms, concentrate transportation, and insurance were based on estimates. During more detailed studies, it is recommended to contact potential shipping companies and smelters to obtain firm estimates for land freight, storage and handling, ocean freight, transport insurance, smelting and refining charges, and a pay factor deduction.

24.1.2 Opportunities

The final pit has two components, the North pit and the South pit. There is potential to use the South pit as a rock storage site, reducing the costs associated with transportation of waste and reducing the environmental impact.

Currently, material that is stockpiled is placed in the WRF area. Placing stockpile material closer to the mill will greatly reduce the haul distances for placement of material into the stockpile and reclamation of material from the stockpile to the mill. This will reduce mining costs for stockpiled material.

Haul distances to the WRF could be reduced by mining out a slot through the ridge between the North pit and the WRF reducing haulage costs for waste material,

Phase 3N is low grade (30% lower Cu grade than Phase 4N) and very high strip ratio. Changing the phase limits in the North pit will allow targeting of higher grade material sooner.

Implementing a variable cut-off grade strategy has increased annual material movement. There is an opportunity to change the mine equipment fleet to larger, more efficient sizes and thereby reduce the unit mining costs.

No underground mining option has been analyzed for this Project. This could be explored during future detailed studies.

During more detailed studies, the use of hypogene and supergene mineralization types should be reinstated when performing metallurgical sample selection. The two mineralization types should be scheduled separately in the mine plan as there was some indication in historical batch testing that the recovery and particularly product grade of supergene mineralization was slightly higher than considered in the PEA and represents minor Project upside potential in metallurgical recoveries and final concentrate grades.

The Project retains exploration potential, and should additional mineralization be found that can support Mineral Resource estimation, this represents upside potential for future studies.

The TSF currently represents the largest capital expenditure item for the Project and as such there are significant opportunities if associated costs can be reduced.

24.2 Project Implementation

It has been assumed for the development of the study that an EPCM contractor will be responsible for delivering the Project facilities and will provide the overall management of contractors and suppliers, as well as the major guidelines for projects systems and procedures to be implemented for project delivery. After commissioning the Project facilities will be handed over to the Owner's operations team.

The Owner's operations team will develop and run the mine during the overall life of the Project.

All stakeholder relocations should be completed before construction earthworks are started.

24.2.1 Project Personnel

The Owner objective is to source personnel and suppliers from Perú. If insufficient skilled and experienced personnel are available locally, then personnel will need to be sourced elsewhere outside Perú.

24.3 Drill Spacing Study

A drill spacing study was completed by Amec Foster Wheeler in 2014 to assess what likely spacings should be considered when planning infill drill programs to support potential upgrade of mineralization to higher confidence Mineral Resource categories.

Amec Foster Wheeler guidance is that for an Indicated Mineral Resource, the drill-hole spacing should be sufficient to predict tonnage, grade, and metal on annual production basis, with ±15% relative precision at the 90% confidence level. In the case of a Measured Mineral Resource the ±15% relative precision must be achieved on a quarterly production volume, which, assuming independence of errors between quarterly production increments, is equivalent to a ±7.5% relative precision at the 90% confidence level on annual production volume.

This study indicated that the classification in Table 24-1 can be derived.

The current drill spacing suggests that there are a number of blocks, particularly in the Primary (Hypogene) Zone, that represent upside for potential upgrade of mineralization to higher confidence Mineral Resource categories. The design of future drilling programs should place a high emphasis on increasing the information density available in these block areas.

Table 24-1: Maximum Drill-hole Spacing Criteria for Resource Classification

Domain	Measured	Indicated
Oxide/Mixed	60 x 60 m	100 x 100 m
Primary	60 x 60 m	100 x 100 m

25.0 INTERPRETATION AND CONCLUSIONS

The QPs have reached the following conclusions and made the following interpretations as a result of the completion of the PEA study and the underlying Mineral Resource estimate.

25.1 Project Setting

- Mining activities should be capable of being conducted year-round
- There is sufficient suitable land area and surface rights available within the mineral claims for any future tailings disposal, mine waste disposal, and installations such as a processing plant, and related mine infrastructure
- The infrastructure assumptions in the PEA are within the claim boundaries.

25.2 Mineral Tenure, Surface Rights and Royalties

- Information from legal experts and Panoro experts provided to Amec Foster Wheeler supports that Panoro holds 100% of the Project
- Information from legal experts supports that the mining tenure comprising the Property is valid and is sufficient to support declaration of Mineral Resources
- Panoro holds no surface rights in the area, and any future mining activities will require agreements to be negotiated with both local communities and individual surface rights holders
- Exploration activities to date have been completed under the appropriate Peruvian permits
- Panoro will need to apply for additional permits as appropriate under applicable Peruvian laws to allow future mining operations
- Notwithstanding the information contained in Section 4, there is no guarantee that title to any of the Project will not be challenged or impaired, and third parties may have valid claims affecting the Project, including prior unregistered liens, agreements, transfers or claims, including aboriginal land claims, and title may be affected by, among other things, undetected defects. As a result, there remains a risk that there may be future constraints on Panoro's ability to operate the Project, or Panoro may be unable to enforce rights with respect to the Project.

25.3 Geology and Mineralization

- Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization, and the mineralization style and setting is sufficient to support Mineral Resource estimation
- The deposit is considered to be an example of a porphyry copper system
- The deposit type used for exploration targeting is appropriate to the mineralization identified and the regional setting.

25.4 Exploration and Drilling

- The exploration concepts being applied to the exploration programs at Cotabambas are consistent with the Project setting and mineralization identified to date
- The Project retains exploration potential, and Panoro has a number of geophysical, geochemical, structural and principal component analysis targets that could support further work
- A total of 155 drill holes (74,988 m) have been completed. Of this total, 24 holes were drilled by Antofagasta, 10 holes by CDLM and the remainder by Panoro
- The quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected in the 1996–2014 exploration and infill drill programs are sufficient to support Mineral Resource estimation. There are no known sampling or recovery factors that could materially impact the accuracy and reliability of the results.

25.5 Sample Preparation and Analysis

- Sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource and mine planning purposes at the PEA level
- Bulk density determination procedures are consistent with industry-standard procedures, and there are sufficient bulk density determinations to support tonnage estimates
- Analysis is performed by accredited third-party laboratories.

25.6 Data Verification

- Data verification indicates that the data to support Mineral Resource estimates are acceptable, conform with industry-standard practices, and are acceptable for Mineral Resource and mine planning purposes at the PEA level

- The process of data verification performed by the relevant QPs indicates that the data collected from the Project during the 2006 to 2014 work programs adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposit, and adequately support the geological interpretations, and the analytical and database quality
- QA/QC with respect to the results received to date for the 2006–2014 exploration programs is acceptable, and protocols are acceptable.

25.7 Metallurgical Testwork

- Since 2012, preliminary comminution, hydrometallurgical and flotation test work has been completed. The metallurgical samples tested were based on drill hole core composites generated from different mineralization zones
- Metallurgical recoveries are projected to be, by mineralization type (zone):

Mineralization Type	Recovery			Concentrate Grade		
	Cu %	Au %	Ag %	Cu %	Au g/t	Ag g/t
Hypogene Sulphide	87.5	62.0	60.4			
Supergene Sulphide	87.5	62.0	60.4			
Mixed Oxide Cu-Au	60.0	55.0	—			
Oxide High Au	—	65.0	—			
*Overall Life-of-Mine Average **	*80.4*	*61.3*	*52.2*	*27.0*	*11.5*	*130*

Note: Average life-of-mine figures from PEA overall annual mine plan schedule

- Test work to date has focused on the global metallurgical characterization of composites of the main mineralization types, and no variability testing has been conducted at this stage
- No deleterious elements that could have a significant effect on potential economic extraction were noted
- Molybdenum is indicated in the concentrate but the head grade is too low globally to result in the production of a saleable by-product or credit
- Sulphur levels of annual oxide/sulphide concentrate blends should be assessed in future work to confirm they meet minimum sulphur contract specifications.

25.8 Mineral Resource Estimate

- Estimations of Mineral Resources for the Project conform to industry standard practices, and meet the requirements of CIM (2014)

- At a 0.2% CuEq cut-off, Tetra Tech estimated a total Indicated Mineral Resource of 117 Mt at 0.42% copper, 0.23 g/t gold, 2.74 g/t silver and 0.0013% molybdenum, and a total Inferred Mineral Resource of 605 Mt at 0.31% copper, 0.17 g/t gold, 2.33 g/t silver and 0.0019% molybdenum
- The following factors could affect the Mineral Resources: commodity price and exchange rate assumptions; pit slope angles and other geotechnical factors; assumptions used in generating the conceptual pit shell, including metal recoveries, and mining and process cost assumptions, and the ability to obtain relevant permits and social licence to operate.

25.9 Mine Plan

- A PEA open pit mine plan is developed for the Cotabambas project.
- The PEA mine plan is based on a subset of the Mineral Resources. No mining dilution was incorporated in the plan. The assumed mill feed throughput is 80,000 t/d (29.3 Mt/a)
- Two open pits will be mined, in seven pit phases, utilizing 12 m high benches. Cotabambas has a projected life of mine of 18 years, considering one year of pre-stripping and 17 years of feed to the plant
- A WRF with a maximum capacity of 369 Mm3 will be located in Guaclle Creek. Waste will be transported by truck
- The operating fleet will consist of a conventional diesel fleet. Equipment requirements were estimated using first principle calculations
- Haulage requirements are based on measured annual haulage profiles from the two pits. The average haulage distance for mineralized material to the primary crusher is 3.3 km. The average haulage distance for waste by trucks is between 3 and 4 km
- The mine plan assume that Cotabambas will operate seven days a week, twenty-four hours per day with four crews rotating to fill the mine roster of 12 hours per shift
- Mine infrastructure includes shops, warehouse, washing and welding bays, fuel stations, explosives magazine and administrative offices
- Main consumables for mine operations include diesel fuel, ANFO, emulsion and tires.

25.10 Recovery Plan

- The plant design consists of a plant with a nominal processing capacity of 80,000 t/d and it includes crushing, grinding, flotation, concentrate dewatering and tailings disposal

- Mineralized material will be processed in a conventional copper porphyry flotation concentrator plant to produce a copper, gold and silver concentrate, with copper as the main payable metal in the concentrate and gold and silver providing important economic by-product credits
- The process considers a primary crusher, a SAG mill and a ball mill prior to the flotation plant consisting of rougher flotation, regrinding and three cleaner stages. Flotation tailings will be thickened and pumped to a tailings storage facility for disposal and final product concentrate is dewatered for storage and transport to port by trucks
- It is planned that 80 kt/d of mill feed will be crushed in a primary crushing circuit, and milled through a primary grind SAG mill and secondary grind ball mills to produce fine material at P80 106 µm. Only four BWI tests have been done to date on composites of the main hypogene material, and this indicates some variability in hardness within the deposit
- The selection of the conventional SAG/ball grinding circuit and equipment is based on a typical process flowsheet for this type of mineralization and scale of the Project. However a comminution trade-off study and further testing needs to be done
- Milled feed will be processed in a conventional rougher, regrind and cleaner copper flotation plant to produce about 270,000 t/a of copper concentrate with a grade of 27.0% Cu and 11.5 g/t Au. Concentrate will be dewatered to about 8% moisture, and tailings will be pumped to a TSF at 62 wt% solids for disposal
- No trade-off studies or geotechnical investigations have been completed to support the plant site selection and these should be completed in the next study stage
- The concentrate road transport option selected for the basis of this PEA considers a Cotabambas–Challhuahuacho–Espinar–Imata–Arequipa–port of Matarani route, with a total distance of 598 km
- About 80% of process water in the plant flotation tailings stream will be recovered and recycled to the process from tailings thickeners (75%) located at the plant site and as reclaim from the thickened tailings storage facility (5%). Water make-up requirements for the plant are estimated to be about 364 L/s
- Plant power and consumables were estimated. The power requirements have been factored from the installed power indicated in the major equipment list, and the consumables have been estimated from a combination of mass balances and metallurgical testwork, as well as comparable industry benchmarks.

25.11 Infrastructure

- Two routes are proposed to be used to transport goods to and from the Project. The route from the port of Callao will be utilized to transport personnel and minor equipment during construction and through-out operations. The Matarani port route will be utilized to transport major equipment during construction. Copper concentrate will also be transported by truck utilizing this route to Matarani port where it will be offloaded and shipped to the end user
- The 66 km stretch of road from Pamputa to Cotabambas (using the Matarani port route) will require upgrading for Project use
- Thickened high density tailings will be pumped to a facility with a capacity of 483 Mt, located 6.2 km and about 1 km higher elevation than the plant. This configuration of the TSF has three retention dams with a dam volume of 38.23 Mm3
- The conceptual water management was developed based on water balance use of the non-contact water pond (fresh water pond). The pond capacity will be approximately 3Mm^3 and the minimum volume will be 0.1 Mm^3. Contact water will be collected below the waste rock facility or within an event pond within the plant footprint. No allowance for a water treatment plant has been considered at this time.
- On-site infrastructure will include a security office, training building for 15 persons, medical centre, 400-person accommodations camp, diner, fuel station, administrative plant offices, plant warehouse, maintenance workshop, process plant sample warehouse, laboratories, mine administrative offices, mine operation offices, mine control booth, explosive warehouse, six-bay truck shop (including spares warehouse), mine and geology sample warehouse, and a provisional laydown area
- An annual power consumption of 120 MW has been estimated. The Project will be connected to the national electrical network through the extension of an existing substation located in Abancay. The Project considers this extension and a 61 km 220 kV transmission line from Abancay to Cotabambas. The electrical design assumes that space will be available for expansion of the existing Abancay substation.

25.12 Environmental, Social and Permitting Considerations

- The baseline environmental information available was collected by Schlumberger Water Services in 2012
- Preliminary geochemical testing has been conducted for waste rock, indicating that the rock does not have the potential to generate acid. Leaching tests show some potential for the release of manganese

- Site run-off will constitute the main source of water for the Project. Mine water will be recycled as much as possible and evaporation and seepage losses will be minimized in order to reduce fresh water requirements for the Project and avoid potential effects on other surface and groundwater users in the vicinity of the Project
- The mine plan assumes that Panoro will be able to collect surface runoff for mining and processing needs; this assumption is supported by the preliminary water balance. However, any impacts of the assumptions on stakeholders have not been evaluated, and future consideration may require mitigation strategies or possibly compensation payments as required
- The EIA for the Project will include a conceptual closure plan to obtain the environmental approval. A provision of $50 M has been made in the Project financial model to take into account the closure costs
- A comprehensive environmental and social impact assessment will be necessary for the Project in order to obtain necessary permits for construction, operations, and closure
- Once the environmental and social impact assessment is approved by Peruvian authorities, a variety of permits, licenses, and authorizations will be required to proceed with the construction and operations of the Project
- There are likely to be some stakeholder relocations required. A comprehensive resettlement process including extensive consultation with community leadership and local authorities will be required to proceed with the development of the Project. Recent experiences in Perú on resettlement caused by mining development indicate that the costs of resettlement can be very high and delay project execution significantly
- Mitigation measures to avoid, reduce, or compensate for potential Project effects will need to be developed and supported by comprehensive environmental and social baseline investigations and engineering studies

25.13 Market Studies

- No market studies have specifically been conducted on the Cotabambas Project. The final sulfide copper concentrate produced is expected to be shipped to smelters located in China, Japan or India. The copper concentrate is also expected to realize important payable silver and gold credits at various times in the mine life. No credit for Mo has been considered.
- Testwork results indicate the copper concentrate will be relatively clean and can reasonably be expected to be marketable with a Cu grade in the range of 25 to 28%. No deleterious elements that could have a significant effect on potential economic

extraction have been detected in the concentrates produced to date by the various test programs.

- No contracts are in place for the Project.

25.14 Capital Costs

- The accuracy of this estimate is considered to be within -35% to +50%. The level of definition expressed as a percentage of total engineering is in the range of 0 to 2% and engineering completion is within a range of 0 to 1%.
- The capital cost estimate is:

Description	Cost Estimate (US$M)
Initial Capital Cost	
Mining	381.2
Tailings Disposal	78.2
Process Plant	505.2
Site Infrastructure	67.2
Off Site Infrastructure	27.5
Owners Cost	40.0
Indirect Cost	152
Contingency	235
Total Capital Cost	*1,486.2*
Sustaining Capital Cost	
Mine	21.8
Tailings Storage	405.5
Total Sustaining Capital	*427.2*
Closure Cost	
Closure Cost	45.5
Contingency	4.5
Total Closure Cost	*50.0*
Total Capital Cost Estimate	***1,963.4***

The capital expenditure of $1.53 B stated in the press release of September 22, 2015 includes both initial capital and estimated closure costs. The initial capital required, without provision for closure, is estimated to be $1.49 B.

25.15 Operating Cost Estimate

- The accuracy of this estimate is considered to be within ±35%.
- The operating cost estimate is:

Description	LOM Cost Estimate	LOM Cost Estimate (US$M)
Mining	$1.59/t of material mined (excluding pre-stripping) $3.59/t of mill feed processed	1,732
Processing	$4.38/t of mill feed processed	2,116
General and Administration	$0.41/t of mill feed processed	197
Total Operating Cost Estimate		**4,045**

25.16 Economic Analysis

- The Project is forecasted to provide a pre-tax a NPV at 7.5% discount rate of US$ 1,052.5 M, an IRR of 20.4% and a payback period of 3.2 years
- Financial analysis showed the after tax Project NPV and 7.5% to be US$684 M and the IRR to be 16.7%. The project payback period after tax is 3.6 years
- The life of mine cash cost per pound of payable copper, after secondary metal credits, is US$1.22/lb
- Sensitivity analysis was performed taking into account variations in copper metal prices, copper grade, gold metal prices, gold grade, operating costs and capital costs. The results from the analysis showed that the project NPV sensitivity is (in order from highest to lowest) copper metal price, copper grade, capital costs, operating costs, gold metal price, and gold grade. Gold grade and gold metal price have the same effect on the NPV.

25.17 Risks and Opportunities

- No formal risk and opportunity analysis has been performed; however a list of indicative major risks and opportunities were identified for the Project from internal discussions and reviews, based on the PEA-level evaluation and the current status of Project knowledge
- The key risks are:
  - The Project is likely to require the relocation of some stakeholders.
  - The unknown nature and the significant variability between agreements that have been concluded with resettled parties for other comparable projects in Perú makes it impractical to evaluate with certainty within the financials for this aspect until further engagement with the affected stakeholders and communities has occurred

  - Availability of water for Project purposes
  - Additional requirement for a water treatment plant and water treatment costs
  - Lack of Project and site-specific geotechnical and hydrogeological data; changes to assumptions in the PEA as a result of site-specific data could result in changes to the estimated costs, and potentially to the site locations selected for infrastructure such as the plant, TSF and WRF
  - Impact of proposed road upgrades on stakeholders
  - Permitting timeline assumptions.

- Key opportunities could include:
  - Exploration upside potential
  - There is potential to use the South Pit as a rock storage site, reducing the costs associated with transportation of waste and reducing the environmental impact
  - There is potential to reduce the waste and stockpile material haulage costs.

26.0 RECOMMENDATIONS

A two-phase work program is recommended. The first phase primarily consists of the required data collection activities to support a pre-feasibility study and environmental permitting requirements, but also incorporates exploration activities that may not necessarily be required for PFS completion. The second phase consists of a PFS. Phase 1 is estimated at about $26 M, and Phase 2 at US$2–3 M.

26.1 Phase 1

26.1.1 Exploration

Exploration-stage reconnaissance sampling in conjunction with geological mapping is suggested to be conducted over the Chaupec, Jean Louis, Añarqui and Chuyllullo conceptual porphyry and skarn targets to determine whether additional work, such as drill testing, is warranted.

An initial-stage exploration drilling program is recommended for the Guaclle, Cayrayoc, Ccochapata, Maria Jose and Buena Vista areas.

Infill and step-out drilling is proposed for the Ccalla and Azulccacca zones. The infill drilling will be completed to support potential confidence classification upgrades. Step-out drilling will focus on the areas to the northeast and southwest of the zones, and at depth in the area southeast of the Ccalla zone.

Table 26-1 summarizes the exploration program proposed. The planned exploration program has been revised from that envisaged in the 9 April 2015 technical report. There are fewer drill holes planned, and costs have been updated to reflect current market conditions.

Amec Foster Wheeler notes that continuation of exploration drilling in any one target area would be predicated on the results of each drill hole, with programs in the target area being discontinued if negative or discouraging results were returned.

Table 26-1: Exploration Program

		Estimated Quantity (m)	Unit Price (US$/m)	Total Estimated Cost (US$M)
Exploration	Guaclle, Cayrayoc, Ccochapata, Maria Jose, and Buena Vista, including logging, assaying, QA/QC, density determination, survey	30,000; assumes 5,000 per target area	300	9.0
	Geological and reconnaissance mapping; geochemical sampling			0.5
Infill Drilling	Azulccacca and Ccalla deposits, including logging, assaying, QA/QC, density determination, survey	15,000	300	4.5
Step-Out Drilling	Northeast and southwest of the zones, and at depth in the area southeast of the Ccalla zone; including logging, assaying, QA/QC, density determination, survey	5,000	300	1.5
	Total Estimated Cost			*15.5*

26.1.2 Geology and Mineral Resource Estimates

Amec Foster Wheeler recommends the following work program in support of the Phase 2 completion of a pre-feasibility study:

- Panoro should consider development of a core recovery model, based on existing information, which can be used to provide indirect geotechnical information and also highlight zones where sampling may be difficult and logging interpretation may be uncertain
- The existing comprehensive veinlet density and vein type log information should be reviewed to refine estimation domains.
- The existing lithological model should be reviewed using information obtained from the lithogeochemical and structural studies completed on the project, in order to refine the definition and interpretation of the porphyry units for the purposes of more tightly defining estimation domains
- Structural and alteration models based on the results of the lithogeochemical and structural studies should be constructed as part of the next geological model update

- The mineralization zones (primary, oxide, mixed, supergene and leached cap) models should be reviewed by applying sequential copper domain criteria to support future mineral resource estimates and geometallurgical domaining.
- The program is estimated at US$100,000 to $250,000 to complete, depending on whether Panoro completes the work in house, or contracts third-party consultants.

26.1.3 Metallurgical Testwork and Process Design

Amec Foster Wheeler recommends the following work program in support of the Phase 2 completion of a pre-feasibility study. The planned program remains the same as that envisaged in the 9 April 2015 technical report; however, costs have been updated to reflect current market conditions.

The relatively fine sulphide primary grind of P80 106 µm selected should be reviewed as test work indicates that recovery is relatively insensitive to primary grind size in the range up to 130 to 150 µm. Additional baseline/variability work and economic grind recovery trade-off is recommended in future work to confirm an opportunity to specify a coarser primary grind than that considered in this study thereby reducing grinding related costs.

Gravity-flotation did not appear to offer a material advantage over flotation only in any testing phase on the composites tested and has not been included in this studies flowsheet. The inclusion of gravity should however be re-evaluated in future variability testing and associated with oxide blend mill feed high in gold produced early in the mine plan as an economic trade-off for producing doré on site versus smelter concentrate credit.

Flotation testing is required in future test work on representative samples of the Mixed zone and Oxide Au and as blends with sulphide to confirm the recoveries estimated by Amec Foster Wheeler for these zones in this study. The use of alternative hydroxamate collectors to enhance copper oxide recovery in blends should be investigated in future programs.

Future prefeasibility testing should include some batch flotation and comminution variability testing as only zone composites have been tested to date.

Additional leach amenability bottle roll and column testing is recommended on representative samples of oxide (excluding Mixed oxide) to support an updated economic assessment trade-off study of Cu oxide leaching.

The PFS metallurgical test program should consider the following testing on representative hypogene sulphide, supergene, mixed copper oxide and oxide Au zone composites and blends relevant to the mine plan and individual variability samples:

- Sample chemical, copper speciation and ICP analysis

- GRG gravity characterization
- Batch rougher and cleaner flotation testing (optimize process flowsheet parameters)
- LC flotation testing and concentrate product characterization
- Variability sample batch rougher and cleaner flotation testing (about 30 samples)
- Comminution testing (JK, HPGR, CWI, BWI, Abrasion Index)
- Variability sample comminution testing (about 30 samples)
- Tailings/concentrate product dewatering and rheology characterization
- Oxide bottle roll amenability and column testing.

The PFS engineering should include preliminary process and infrastructure design. Trade-off studies should be completed during the PFS to finalize the process flowsheet and allow preliminary plant facility location selections including:

- Plant siting options analysis and preliminary geotechnical investigations
- Comminution configuration options including HPGR versus SAG
- Gravity gold concentrate production
- Oxide leaching preliminary economic assessment.

The metallurgical testwork program is estimated at US$600,000 to complete.

26.1.4 Mining

Moose Mountain Technical Services recommends the following optimization studies prior to a pre-feasibility study:

- High level scoping analysis to lock down the general arrangement footprint and choose the base case for a pre-feasibility level mine plan
- Optimization of the mining fleet utilized
- Investigate potential for underground mining.

The program is estimated at approximately US$25,000 to complete.

26.1.5 Infrastructure

Amec Foster Wheeler recommends the following work program in support of the Phase 2 completion of a pre-feasibility study:

- Hydrogeological studies and seismic hazard studies
- Development of a site water balance based on hydrological studies

- Development of contact water and fresh water management strategy and associated infrastructure to a level that will support PFS designs
- Confirmation of water treatment requirements based on further testing and hydrological data.
- Optimization of the tailings facility location and TSF design to a level that will support PFS designs
- Evaluation and development of the tailings pipeline routing and requirements
- PFS level design of WRF including all associated water management
- An estimate was previously developed for the supply of energy to the Project. This should be updated to reflect any changes to the grid and ensure accurate pricing.
- A comprehensive route study that further defines the upgrades required to the access road is required
- Confirmation of available port facilities and costs associated with concentrate handling at port will need to be addressed.

The program is estimated at US$550,000 to US$675,000 to complete, depending on whether Panoro completes the work in-house, or contracts third-party consultants. The planned program remains the same as that envisaged in the 9 April 2015 technical report; however, costs have been updated to reflect current market conditions.

26.1.6 Baseline Studies

A comprehensive set of baseline studies to encompass environmental and social baseline data that can be used in the PFS, and to provide the basis for an EIA is estimated at US$2 M.

The planned program remains the same as that envisaged in the 9 April 2015 technical report; however, costs have been updated to reflect current market conditions.

26.2 Phase 2

Once the elements of the Phase 1 work are to hand, Amec Foster Wheeler recommends Panoro consider commencement of a pre-feasibility study. Information obtained during the infill and exploration drilling should be incorporated into an updated resource model and estimate that would support the PFS analysis.

In Amec Foster Wheeler's experience, a comprehensive pre-feasibility study, excluding environmental, social and permitting components, can range between US$2 M and US$3 M.

27.0 REFERENCES

AACE, 1997: Recommended Practice N° 17R-97: Cost Estimate Classification System, August, 1997, 6 p.

Amec Foster Wheeler, 2015: Cotabambas Copper-Gold Project, Preliminary Economic Assessment, Apurímac, Perú: report prepared by Amec Foster Wheeler for Panoro, 2015.

Aminpro 2014: Project 2013-021, Oxide Mineral Leach Tests: report dated July 2014.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2003: Estimation of Mineral Resources and Mineral Reserves, Best Practice Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 23, 2003, http://www.cim.org/committees/estimation2003.pdf.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2010: CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 2010, http://www.cim.org/UserFiles/File/CIM_DEFINITON_STANDARDS_Nov_2010.pdf.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2014: CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, May, 2014

Canadian Securities Administrators (CSA), 2011: National Instrument 43-101, Standards of Disclosure for Mineral Projects, Canadian Securities Administrators, http://www.osc.gov.on.ca/en/SecuritiesLaw_ni_20110624_43-101_mineral-projects.htm.

Certimin, 2012: Informe Metalúrgico Final – Pruebas Metalúrgicas Mineral De Cobre Proyecto Cotabambas de Panoro Apurímac S.A.: internal metallurgical report dated July 2012. 109 p.

Certimin 2014a: FEB4007 R14, Hypogene Cu-Au and Cu-Mo Sample Sulphide Batch Flotation: report dated February 2014.

Certimin 2014b: ABR4003 R14, Cu–Ag and Cu–Au Oxide Sample Batch Flotation: report dated April 2014.

Certimin 2014c: MAY4000 R14, Cu–Au and Cu–Ag Oxide and Hypogene Sulphide Blend Batch Flotation: report dated May 2014.

Certimin 2014d: MAY4015 R14, Cu-Au Hypogene Sulphide and Oxide Blend LC Confirmation Flotation: report dated May 2014

CIA, 1991: Base Map of Perú: Downloaded from: https://www.cia.gov/library/publications/the-world.../geos/pe.html. 1 p.

CMJ, 2012: The 101 on Peruvian mining: Article published in the Canadian Mining Journal, February, 2012. 1 p.

Congreso de la República del Perú, 2014: Ley N° 30296 Capítulo I: Modificación de las Tasas del Impuesto a las Rentas del Trabajo y de Fuente Extranjera: December, 2014.

DEPMEM, 2004: Plan Nacional de Electrificacion Rural (PNER) Periodo 2004-2013: Ministry of Energy and Mines document discussing the National Rural Electrical Distribution Plan for the Period 2004-2013. 130 p.

Ernst and Young, 2015: Peru Mining and Metals Investment Guide 2015–2016: report posted to http://www.ey.com/Publication/vwLUAssets/Peru-Business-and-Investment-guide-2014-15/$FILE/Peru%C2%B4s%20Business%20and%20investment%20guide%202014-2015-2.pdf.

Krige, D.G. 1996: A Practical Analysis Of The Effects Of Spatial Structure And Data Available And Used, On Conditional Biases In Ordinary Kriging: 5th Int. Geostatistics Congress, Wollongong, Australia, 1996.

Lee, C. Nowak, M. and Wober, H. H., 2007: Independent Technical Report on the Mineral Exploration Properties of the Cordillera de las Minas S.A. Andahuaylas-Yauri Belt, Cusco Region, Perú: report prepared by SRK Consulting (Canada) Inc. for Panoro, effective date 9 March 2007, 125 p.

Long, S., 2003: Assay Quality Assurance-Quality Control Program for Drilling Projects at the Pre-Feasibility to Feasibility Report Level (4th edition): internal Amec Foster Wheeler document, 82 p.

Mamami, 2014: Estudio Litogeoquímico del Depósito Mineral de Cotabambas, Perú report prepared for Panoro, 2014.

Mayta, E., Vela, L., 2015: Acceso Externo: Proyecto Cotabambas: internal report prepared for Panoro, 23 April, 2015, 5 p

Ministerio de Energia y Minas, 2012: Resolucion Directoral N° 194-2012-MEM-AAM: Ministry of Energy and Mines approval of Panoro's modification to semi-detailed

EIA for a drill program on the Property, Signed and sealed by Dr. Manuel Castro Baca, Director General Asuntos Ambientales Mineros of the MEM. 10 p.

Ministerio de Energia y Minas 2001: Reglamento de Seguridad e Higiene Minera D.S. 046-2001-EM.

Morrison, R.S., Huang, J., and Daigle, P., 2014: Technical Report and Resource Estimate of the Cotabambas Copper-Gold Project, Perú: report prepared by Tetra Tech WEI Inc for Panoro, effective date 29 October 2013, amended 7 July 2014, 130 p.

NCL Ingenieria Y Construccion, 2001: Proyecto Cotabambas, Estimacion de Recursos Geologicos: internal report prepared for Minera Anaconda, Perú SA.

O'Brien, M., 2014: Cotabambas Drillhole Spacing Study: internal report prepared by Amec Foster Wheeler for Panoro, 11 September 2014, 21 p.

Outotec, 2014 Test 111586T1, Sulphide and Blend Flotation Dewatering, Report, July 2014.

PANAPEX S.A., 2014: Estudio de Pre Factibilidad para el Suministro de Energía Eléctrica, Mediante la Implementación de Líneas de Transmisión y Sub Estaciones Necesarias para los Proyectos de Exploración Minera Cotabambas y Antillas, July, 2014.

Panoro Minerals Ltd. 2015: Cotabambas Project: Preliminary Economic Assessment NI 43-101 Technical Report – Taxation Notes: Notes and reports collected by Panoro Minerals Ltd for Amec Foster Wheeler, 11 May 2015, 45 p.

Paredes, A., 2014: Environmental Section, Cotabambas Copper Project Preliminary Economic Assessment: draft memorandum prepared by Amec Foster Wheeler, 22 July 2014, 8 p.

Perelló J., Carlotto V. Zarate A.,Ramos P., Posso H., Neyra C., Caballero A., Fuster N., and Muhr R., 2003: Porphyry Style Alteration and Mineralization of the Middle Eocene to Early Oligocene Andahuaylas – Yauri Belt, Cuzco region, Perú; Economic Geology, vol.98, 2003, pp 1575–1605.

Perelló J., Neyra,C., Posso H., Zarate A., Ramos P., Caballero A., Martini R., Fuster N., and Muhr R., 2004: Cotabambas; Late Eocene Copper – Gold Mineralization Southwest of Cuzco, Perú: Economic Geology Special Publication 11, pp 213–230.

Perello, J., Posso, H., Neyra, C., 2001: Geologia y Recursos del Proyecto Cotabambas, Departmento de Apuromac, Perú.

Rodriguez, 2014: Estudio Estructural del Pórfido Cotabambas, Perú: report prepared for Panoro, 2014.

Sillitoe, R., Porphyry copper systems: Economic Geology, v. 105, pp. 3–41.

Sinclair, W.D., 2006. Consolidation and Synthesis of Mineral Deposits Knowledge - Porphyry Deposits: report posted to Natural Resources Canada website 30 January 2006, 14 p.

Singer, D.A., Berger, V.I., and Moring B.C: Porphyry Copper Deposits of the World: Database and Grade and Tonnage Models: U.S. Geological Survey Open-File Report 2008-1155, version 1.0, 45 p.

SWS, 2012: EIA Semi Detallado de la Modificación del Proyecto Cotabambas-Ccalla, Schlumberger Water Services, July 2012: Semi-detailed environmental impact assessment study completed by Schlumberger Water Services for Panoro dated July 2012.

Vale, 2002: CVRD and Antofagasta Announce Joint Venture: Press release dated July 19, 2002. 1 p.

VDG del Perú S.A.C., Val d'Or Geofisica, 1996–2004: Reports on all Geophysical Surveys carried out on the properties of CDLM: unpublished reports for Cordillera de las Minas.

Wakefield, T., and Torres, A., 2014: Cotabambas Principal Component Analysis; report prepared by Amec Foster Wheeler for Panoro, 15 April, 2015, 52 p.

Wright, C., Arrias, W., and Vilela, M., 2009: Restated, Amended Technical Report for the Antilla Property Apurimac, Perú: report prepared by AMEC (Perú) S.A. for Panoro, effective date 1 June, 2009, amended 23 August, 2009, 100 p.

Wright, C., and Colquhoun, W., 2012: Cotabambas Copper-Gold Project, National Instrument 43-101 Technical Report on a Mineral Resource Estimate, Apurímac, Perú: report prepared by AMEC (Perú) S.A. for Panoro, effective date 24 July, 2012, 120 p.