NI 43-101

TECHNICAL REPORT ON THE PEA FOR THE ANTILLA COPPER PROJECT HEAP LEACH AND SX/EW OPERATION

Centred at 18L 719,600 E and 8,413,000 N
(NAD 83)

Submitted to:
Panoro Minerals Ltd.

11 June 2018

Moose Mountain Technical Services
Jesse Aarsen, P.Eng.

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CERTIFICATE & DATE – JESSE J. AARSEN

I, Jesse J. Aarsen, residing at 1122 Duncan Ave E, Penticton BC, V2A 2X1 do hereby certify that:

1) I am a Senior Associate (Mining Engineer) with the firm of Moose Mountain Technical Services (MMTS) with a business address of 1975-1st Avenue South, Cranbrook BC, V1C 6Y3;
2) I am a graduate of the University of Alberta; in 2002, I obtained a Bachelor of Science degree in Mining Engineering Co-op. I have worked as a mining engineer for a total of 14 years since my graduation from university. I have also taken a two-year period for personal travel throughout the world. My relevant experience for the purpose of the technical report includes:
    a) 2002 to 2005 – Employed at a complex coal mine in the Elk Valley (mountainous terrain) working as a short range, long range, dispatch, and pit engineer. Preparations of budget level mine plans and cost inputs, oversaw operation of personal designs and acting in supervisory-role positions as needed.
    b) Since 2007 – Consulting mine engineer specializing in mine planning and project development, who completed mine plans for complex coal operating mines in north-eastern British Columbia and an open-pit copper/molybdenum mine in central British Columbia. Supervisory role in large multi-disciplinary studies for projects in both coal and hard rock settings in Canada and Mongolia. Responsible for building several coal geology and block models and calculation of mineral resources under the supervision of a P.Geo;
3) I am a professional engineer and a member in good standing, registered with the Association of Professional Engineers and Geoscientists of British Columbia (#38709);
4) I visited the project on January 30 to February 01, 2018;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
6) I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the lead author of this report and responsible for the mining aspect of the study. This includes the mining components of the Sections 1, 21, 22, 25, and 26 as well as the entirety of Sections 2, 3, 15, 16, 18 and 24. I accept professional responsibility for those sections of this technical report;
8) I have been involved with the Antilla project during the preparation of the previous Preliminary Economic Assessment and the Technical Report that is based on the Preliminary Economic Assessment;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith;
10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Antilla copper project or securities of Panoro Minerals Ltd., and
11) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 11th day of June 2018

"signed and sealed"

____

Signature of Qualified Person
Jesse J. Aarsen, PEng
Senior Associate (Mining Engineering)

Eur.Ing Andrew J. Carter C.Eng, IOM3, SAIMM, SME

I, Andrew J. Carter, C.Eng., of Salisbury, Wiltshire, UK, do hereby certify:

I am the General Manager of Tetra Tech Mining and Minerals, located at Unit 2, Apple Walk, Kembrey Park, Swindon, SN2 8BL, UK.

This certificate applies to the technical report entitled "Technical Report on the PEA for the Antilla Copper Project Heap Leach and SX/EW Operation" dated June 11, 2018 prepared for Panoro Minerals Limited (the "Technical Report").

I am a graduate of the University of Leeds (B.Sc., Minerals Processing, 1980), a member in good standing of the the Institutute of Materials, Minerals and Mining (#46421) and a Chartered Engineer (#378467, Engineering Council, UK). My relevant experience includes 38 years of experience in Mining, Plant Operations, R&D, Management, Engineering and Consulting. As a Consulting Metallurgist for the Antilla Copper Project in Peru, I am providing technical oversight of the metallurgical test work program currently underway at Aminpro Laboratories, Lima, Peru and have prepared Section 13 of the technical report entitled "Mineral Processing and Metallurgical Testwork." My copper processing experience includes work related to the following projects: La Granja – Rio Tinto, Panagyuishte – Assarel Medet, Ilovitza – Euromax Resources, Silver Bell – Asarco, Canatuan – TVI Pacific, Emba Derho – Sunridge Gold, Mineral Processes – Impala Platinum, Gaisky and Uchalinsky – UMMC, Bor – RTB, Celopech – Dundee Precious Metals, Kupari – Frontera Copper. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").

I am responsible for Sections 1.11 and 13.0 of the Technical Report.

I am independent of Panoro Minerals Limited as defined by Section 1.5 of the Instrument.

I have no prior involvement with the Property.

I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible disclose all material scientific and technical information such that the Technical Report cannot be misleading.

Signed and dated this 11 day of June 2018 at Swindon, Wiltshire, UK.

"Original document signed and sealed by
Eur.Ing Andrew J. Carter, C.Eng."

Eur.Ing Andrew J. Carter, C.Eng.
General Manager
Tetra Tech Mining and Minerals, UK.

CERTIFICATE OF QUALIFIED PERSON

Daniel H. Sepulveda

I, Daniel H. Sepulveda, B.Sc., SME-RM, do hereby certify that:

1. I am Associate Consultant of Moose Mountain Technical Services (MMTS) with a business address of 1975- 1st Avenue South, Cranbrook BC, V1C 6Y3.

2. I graduated with a degree in Extractive Metallurgy from University of Chile in 1992.

3. I am a registered member of the Society of Mining, Metallurgy, and Exploration, Inc. (SME), member No 4206787RM.

4. I have worked as a Metallurgist for a total of 26 years since my graduation from university. My relevant experience includes: employee of several mining companies, engineering & construction companies, and as a consulting engineer.

5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101

6. I am the author for the preparation of the technical report titled “Technical Report on the PEA for the Antilla Copper Project Heap Leach and SX/EW Operation” (the “Technical Report”), dated effective June 11, 2018, prepared for Panoro Minerals Ltd..; and am responsible for sections 1.14, 17, 25.4, and 26.4. I have not visited the project site.

7. I do not have prior involvement with the property that is the subject of the Technical Report.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 11 of June, 2018.

CERTIFICATE & DATE – LUQUMAN A. SHAHEEN

I, Luquman A. Shaheen, residing at 3360 140 St, Surrey, B.C., V4P2A8 do hereby certify that:

1) I am President and CEO of the firm of Panoro Mineral Ltd. (Panoro) with a business address of Suite #1610, 700 West Pender St., Vancouver, B.C., V6C 1G8;
2) I am a graduate of the University of British Columbia; in 1990, I obtained a Bachelor of Applied Science degree in Civil Engineering. I have worked as a civil engineer specializing in the development of mining projects for a total of 28 years since my graduation from university. I also obtained a Master's degree in Business Administration from Simon Fraser University in 2003. My relevant experience for the purpose of the technical report includes:
   a) 1990 to 1993 – Employed by Klohn Leonoff Inc., a consulting company based in the Seattle, Washington, U.S.A. I completed engineering assignment for various copper and gold mining operations and projects throughout the Western United States and in Canada with a focus on infrastructure designs related to waste management, water supply and heap leaching under the supervision of a P.Eng. and/or P.E;
   b) 1993 to 1995 – Employed by Klohn Crippen Ltd, a consulting engineering company based in Vancouver, B.C. I lead the engineering of infrastructure for mining projects and operations located in Northern Ontario including components of mine expansion and reclamation projects including both open pit and underground gold and copper mines. I managed multidisciplinary teams including professionals with P.Eng. designations;
   c) 1995 to 1996 - Employed by Klohn Crippen Ltd and seconded to TVX Hellas, a mining company with a group of mining assets located in Greece. I was part of a multidisciplinary team that carried out the technical design and due diligence for the acquisition of mining complex in Northern Greece. I carried out the assessment of alternatives and designs for various project infrastructure such as tailings storage areas, water supply as well as environmental mitigation and reclamation. The mining complex included three underground mines, two processing facilities and related surface infrastructure.
   d) 1990 to 1998 - Employed by Klohn Crippen Ltd and seconded to Klohn Crippen-SVS S.A, a consulting company based in Lima, Peru. I carried out the engineering studies for a number of proposed and existing mining projects throughout Peru. The scope of the studies included tailings storage, wasterock management, water supply, environmental reclamation and closure and other infrastructure components. I managed teams of engineering specialists with P.Eng. and other professional designations.
   e) 1998 to 2001 – Employed by AMEC Peru S.A., a consulting company based in Lima, Peru and a subsidiary of AMEC Americas Ltd. I completed engineering studies on a variety of existing and proposed mining projects throughout Peru and Bolivia. Studies included conceptual studies, prefeasibility studies, feasibility studies, detailed engineering and construction supervision for both underground and open pit mines. I managed teams of professionals with P.Eng. and P.Geo. designations;
   f) 2001 to 2006 – Employed by AMEC Earth & Environmental Ltd., a consulting company based in Vancouver, Canada. I completed engineering studies of infrastructure for projects in Peru, Bolivia, Canada and the U.S.A. I managed teams

of professionals with P.Eng. and P.Geo. designations;

g) 2006 to 2008 – Panamerican Silver Corp., a mining company based in Vancouver, B.C. I directed the environmental department for the corporate office providing oversight, review and reporting of environmental aspects of existing and proposed mines located in Peru, Mexico, Bolivia and Argentina; and

h) 2008 to 2018 – Panoro Minerals Ltd., a mineral exploration company based in Vancouver, B.C. I have lead the direction of the Company for over ten years. My responsibilities have included the management of three NI 43-101 compliant resource estimates and four NI 43-101 compliant Preliminary Economic Assessments.

3) I am a professional engineer and a member in good standing, registered with the Association of Professional Engineers and Geoscientists of British Columbia (#21675) and am a professional engineer and an inactive member in the States of Washington and Alaska;

4) I visited the project a number of times from 2008 to present;

5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;

6) I, as a qualified person, am not independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

7) I am a co-author of this report and responsible for the Section 19, Market Studies and Contracts of the study. I accept professional responsibility for those sections of this technical report;

8) I have been involved with the Antilla project during the preparation of the previous Preliminary Economic Assessment and the Technical Report that is based on the Preliminary Economic Assessment;

9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith;

10) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Antilla copper-molybdenum project apart from the securities of Panoro Minerals Ltd. which I currently hold, and

11) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 11th day of June 2018

"signed and sealed"

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Signature of Qualified Person
Luquman A. Shaheen, PEng, PE
President & CEO

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled "Technical Report on the PEA for the Antilla Copper Project Heap Leach and SX/EW Operation", dated June 11, 2018.
I, Luis Vela, residing at El Portal #163, La Planicie, La Molina, Lima, Peru, do hereby certify that:

1) I am employed as the Vice President Exploration, with Panoro Minerals Ltd (Panoro) with an office at Alfredo Benavides Avenue #1579, Office 505, Miraflores, Lima, Peru;
2) I am a graduate from the National University of San Agustin in Peru, where I obtained a Bachelor of Geology Sciences degree in 1991 and a Geologic Engineer degree in 1996. I have practiced my profession for 26 years. I have experience in copper porphyries, precious metals, exploration/ mining and pre-feasibility studies in Mexico, Chile, Bolivia and Peru. Prior to Panoro, I worked as the Vice President and Exploration Manager for companies such as Trafigura Beheer Group, Andean Gold Ltd., Minera Peñoles, Andean American Mining, and Minera Aurifera Retamas and have acted in a consulting capacity;
3) I am a professional in geology and a registered Member of the Chilean Mining Commission (CMC #0173);
4) I have regularly visited the Antilla project, with the most recent visit being, January 30 to February 01, 2018;
5) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101 and this technical report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1;
6) I am not independent of Panoro Minerals Ltd. as independence is described by Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 20, 22, 23, 25.1, 26.1, and 26.5 and accept professional responsibility for those sections of this technical report;
8) I have been involved with the Antilla project since August 2011 in my capacity as Vice President Exploration with Panoro;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in accordance therewith; and
10) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Lima, Peru
June 11, 2018

["Signed and Sealed"]
Luis Vela,CMC
VP Exploration (Panoro Minerals Ltd.)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **Technical Report on the PEA for the Antilla Copper Project Heap Leach and SX/EW Operation, June 11th, 2018.**

I, Paul Joseph Daigle, P.Geo., do hereby certify that:

1) I am a Senior Resource Geologist contracted to Coffey Geotechnics Ltd. Trading as Tetra Tech Mining and Minerals (Tetra Tech) with an office at 2 Apple Walk, Kembrey Park, Swindon, Wiltshire, SN2 8BL, United Kingdom.
2) I am a graduate of the Concordia University in 1989, I obtained a Bachelor of Science degree in Geology. I have practiced my profession continuously for over 25 years since graduation. I have been directly involved in exploration and evaluation of porphyry copper deposits which include but are not limited to: the Sheslay copper-gold projects, British Columbia; the Meriguna and Ballyorlo copper molybdenum-gold deposits, Solomon Islands; the Tucumã copper-gold deposit, Pará, Brazil and, most recently, the Cotabambas Project, Apurimac, Perú.
3) I am a member of the Professional Geologist of the Association of Professional Geoscientists of Ontario, Membership Number 1592.
4) I have personally inspected the subject project between the 3rd and 7th of June 2013 for two days.
5) I have read the definition of Qualified Person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
6) I, as a Qualified Person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
7) I am the co-author of this report and responsible for section 12.2 and section 14 and accept professional responsibility for those sections of this technical report.
8) I have had prior involvement with the subject property, completing a Mineral Resource Estimate for the project with an effective date of 19th October 2015;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith.
10) Tetra Tech was retained by Panoro Minerals Limited to report the mineral resources within a constraining shell.
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Antilla Copper-Molybdenum or securities of Panoro Minerals Limited; and
12) That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Original signed and sealed by
'Paul Daigle, P.Geo'

Toronto	Paul Daigle, P.Geo.
June 11th, 2018	Senior Geologist

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1 Summary

1.1 Introduction

This Preliminary Economic Assessment (PEA) Technical Report is written for the Antilla Project (the "Project") and has been prepared by Moose Mountain Technical Services ("MMTS") in conjunction with Panoro Minerals Ltd. ("Panoro"), and Tetra Tech. The Antilla project is 100% owned by Panoro and is located 140 km south west of the city of Cuzco, in the Apurimac region of Southern Peru.

All currency amounts are referred to in U.S. dollars (USD) unless otherwise indicated.

Since the previous PEA, Panoro commissioned MMTS to complete a scoping-level study considering a heap leach process. This PEA presents the results of the scoping study. There has been no new drilling since the previous PEA.

1.2 Property Description and Ownership

The Antilla project is located approximately 140 kilometres southwest of the city of Cusco, approximately 20 kilometres southwest of the District of Sabaino (Antabamba Province) and in proximity to the village of Antilla. The property lies within UTM Zone 18L, South American Datum (SAD) 69 at approximately 719,600 mE and 8,413,000 mN. The Antilla property consists of 12 mining concessions with a total area of 7,500 hectares. The leases, which area owned 100% by Panoro, expire in June 2019 and are renewed yearly.

1.3 Accessibility, Climate, Local Resources, Infrastructure, Physiography

The Antilla project is located in Apurímac Region, Peru and approximately 500 kilometres southeast of Lima, the capital city of Peru. Approximately 150 kilometres southwest of Cusco, approximately 360 kilometres by road, and is most easily accessed from Cusco via Abancay, Antabamba, and the Antabamba valley leading to Panoro's base camp at the village of Antilla.

Abancay, population 51,462 (2007), is the closest major town to the property and can provide most general supplies. Food and fuel can be found in the surrounding villages closer to the project site as well. Mining related equipment and skilled and professional services must be sourced elsewhere. The temperate highland tropical climate zone is characterized by dry winters season lasts from May to October and a rainy summer season occurs from November to April. Precipitation ranges from less than 4 millimetres to 150 millimetres per months in winter and summer seasons, respectively.

The Antilla deposit is situated on the northern slope of the steeply eroded valley (Quebrada Huancaspaco) where elevations vary from 3,100 to 4,200 metres above sea level

1.4 History

In 1999, Southern Peru Copper S.A. (Southern Peru) carried out regional exploration work on the property. In 2002, Cordillera de Las Minas S.A (Cordillera) explored Peru for large copper deposits. Anaconda Peru S.A. (Anaconda), a Peruvian subsidiary of Antofagasta Plc (Antofagasta), transferred ownership of several groups of exploration concessions in southern Peru to Cordillera. Companhia Vale do Rio (Vale), through its subsidiary Compañía Minera Andino-Brasilera (Minera Andino) had the option to acquire a 50% interest in Cordillera by spending US$6.7 million on exploration over three years (Vale 2002). Results of the 2005 campaign were disappointing and led to the dissolution of the joint venture.

1.5 Geological Setting and Mineralization

The Antilla deposit is a copper-molybdenum porphyry deposit, located in the Andahuaylas-Yauri Belt of the high Andes of southern Peru. 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 south-east. The geology of the Andahuaylas-Yauri Belt is dominated by the Andahuaylas-Yauri Batholith and Mesozoic to Early Cenozoic clastic and marine sedimentary rock. The bulk of the property is underlain by quartzite, quartz-arenite and sandstones of the Soraya Formation. Sedimentary rocks are intruded by at least three types of intrusive rock: altered and weakly-mineralized Main Porphyry stocks or aphophyses, narrow Porphyry diorite and narrow, unaltered Late Porphyry dikes. The altered, weakly-mineralized Main Porphyry is exposed as a prominent knob immediately to the west of the mineralized quartz-arenite, and another, smaller diorite intrusive body is exposed to the northwest and southeast of the mineralization.

Four main mineralization types are found at Antilla: primary sulphide, secondary sulphides, a leached cap and an overburden/cover zone overlying the deposit. The secondary sulphide zone forms a relatively continuous, tabular blanket of chalcocite that generally ranges from 60 to 120 metres thick. The secondary sulphide zone is overlain by the leached cap which has an average thickness of 55 metres and generally ranges from 0 to 75 metres thick

1.6 Deposit Types

The mineralization identified to date on the property is consistent with a supergene enrichment blanket underlain by a primary sulphide mineralization, both hosted in the quartzite and sandstones layers, which mineralization is associated with an Andean-type copper-molybdenum porphyry system.

Common features of copper-molybdenum porphyries include stockworks of quartz veinlets, quartz veins, closely spaced fractures and breccias containing pyrite and chalcopyrite with lesser molybdenite, bornite, and magnetite. These features occur in large zones of bulk-mineable mineralization in or adjoining porphyritic intrusions and related breccia bodies. Disseminated sulphide minerals are present, generally in subordinate amounts. The mineralization is typically spatially, temporally, and genetically associated with hydrothermal alteration of the host rock intrusions and extended to the sediments wall rocks.

1.7 Exploration

The property has been explored since its discovery in 2002. Exploration has consisted of geological mapping, geochemical sampling, and geophysical surveys. Exploration work was first conducted by Cordillera de las Minas S.A. (Cordillera) between 2003 and 2005 in the East Block target, where now is located the existing Mineral Resources and the conceptual PEA pit. In the central and eastern portions of the property, a geochemical survey taken 2,850 rock samples has defined a 3 by 6 kilometres area over the known Antilla deposit. This area appears as part of a larger east-west structural trend, where the rock geochemistry sampling has defined five additional exploration targets: Chabuca, The North Block, The Middle Block, and West Block I and II. The outcroppings in the targets expose in surface copper oxides and secondary sulphides mineralization hosted in the sandstones and quartzites layers. There are evidences of the possible connectivity of the mineralization into the same sandstone package occurring in the East Block, Chabuca and the North Block targets. There are a geophysic survey in ground with Mag and IP covering part of the area where the exploration targets occurs.

The exploration model to explore in Antilla is supported by petrographic (Dunn, 2008), lithogeochemistry studies (Mirian Mamani, 2013) and the exploration works done until now. The model consider the quartzmonzonite porphyry as the source of the metals, transported outward by dikes and/or hydrothermal breccias and finally deposited/concentrated in tramps like the arkose sandstones limited by un-permeable quartzites sediments. The lithologic control in specific sediment can be follow by kilometers as show by the extensive distribution of copper anomalies in an area of 3km x 6km. The structural control and folds also represent other group of tramps types. As a late metal concentration process, the vertical movements of the water table developed in Antilla a thick secondary sulphides blanket occurring under the surface and through the leaching of the primary sulphides previously deposited.

To the west extreme of the property, another porphyry-skarn type target named El Piste was discovered, where detailed mapping and sampling is ongoing.

1.8 Drilling

The deposit was drilled by three different companies between 2003 and 2010. Panoro mineras drilled 49 boreholes (9,130.6 metres) between 2008 and 2009. In total, 15,385 metres distributed in 96 core boreholes have been completed on the project. Geotechnical and Geological logging procedures were made and homogenized by Panoro. Tetra Tech is of the opinion that the drilling procedures adopted by Panoro conform to industry standard. Tetra Tech cannot comment on drilling procedures followed by Cordillera and Chancadora. The drilling pattern resulting from the drilling is considered sufficiently dense to interpret the geometry and the boundaries of the copper and molybdenum mineralization with adequate confidence.

1.9 Sample Preparation, Analyses and Security

Panoro used generally recognized industry practices to collect, handle, and assay surface and core samples collected from the Antilla project. Analytical quality control procedures include the use of blank and duplicate samples in all sample batches submitted for preparation and analyses. No certified control samples were used and no check assaying was completed. Sample preparation and analysis were performed by several laboratories including CIMM Peru S.A. in Lima, ALS Chemex in Cusco, Burau Veritas Inspectorate S.A. in Lima, and Certimin S.A. in Lima.

Samples were routinely assayed for copper, molybdenum, silver, lead, zinc, arsenic, and gold. Analyses for copper, molybdenum, silver, lead, zinc, and arsenic were performed by atomic absorption, while gold was analyzed for either by fire assay or atomic absorption.

Tetra Tech reviewed the sample handling and preparation procedures and those used by laboratories contracted by Panoro. In the opinion of Tetra Tech, the sampling preparation, security, and analytical procedures used by Panoro are consistent with generally accepted industry best practices and are, therefore, adequate for an advanced exploration project.

1.10 Data Verification

In accordance with Canadian Securities Administrators' National Instrument 43-101 guidelines, Tetra Tech visited the Antilla project between June 4 and 7, 2013. The purpose of the Tetra Tech's site visit was to review exploration procedures, define geological modelling procedures, examine core, and interview project personnel. During the visit, a particular attention was given to the treatment and validation of historical drilling data.

Tetra Tech aggregated the assay results of the external analytical control samples for further analysis. Blanks data were summarized on time series plots to highlight the performance of the control samples. Paired data were analyzed using bias charts. Overall, Tetra Tech considers that the analytical results delivered by the primary laboratories used by Panoro are reliable and do not present obvious evidence of analytical bias.

1.11 Metallurgy

Several metallurgical testwork programs have been implemented by various organisations on samples of Antilla mineralogical materials since 2006. Preliminary metallurgical characterisation studies were carried out by Laurion and Inspectorate in 2006 and 2011 respectively. In 2013 a more substantive program was undertaken by Certimin which was aimed at recovery of copper and molybdenum from both primary and secondary materials by conventional flotation methods. In 2018 a column leaching program together with supporting testwork, aimed specifically at extraction of copper from secondary sulphides and the supergene zone of the Antilla deposit was implemented by Aminpro, this testwork program is currently ongoing.

Flotation tests conducted by Certimin indicated that the primary mineralization was amenable to a conventional flotation process. The mineralization from the hypogene zone will produce a higher flotation copper recovery compared with the mineralization from the supergene zone. However, copper concentrate from the secondary mineralization will have a higher copper grade on average compared with the mineralization from the primary mineralised zone.

The results for the primary sulphide material produced copper recoveries averaging 85.3%, compared with secondary sulphide material from which only 79.4% of the copper was recovered to a bulk concentrate. However, the concentrate grade produced from secondary sulphides contained 36.3% copper compared with the 20% copper generated from primary material. Molybdenum recoveries from the samples were 77.6% for primary material and 83.3% for secondary material. Molybdenum from both mineralization zones responded well to the test procedures. The impurity levels of the copper concentrates should not attract smelting penalties as defined by most smelters. It was concluded that a conventional mill and concentrator could be implemented for the development of both primary and secondary materials.

As an alternative to a conventional mill and concentrator for the treatment of both primary and secondary sulphides a process based on leaching secondary sulphides alone was considered a viable alternative. This led to a column leach program, together with associated mineralogical and bottle roll leach testwork, being implemented at Aminpro Laboratories, Lima, Peru during March 2018. This testwork program is aimed at assessing the amenability of Antilla to conventional acid heap leaching as well as a simulated bioleach process and is currently ongoing.

Interim results from the Aminpro bulk sample test program indicate copper leach extractions of circa 75 % are achievable in cycle times of between 150 days and 175 days for 3/8" and 1" crush sizes respectively. Future trade-off studies will be required to determine the optimum economic crush size.

The mine plan also indicates the availability of significant quantities of cover and cap materials containing copper values these are inferred as 0.3 Mt and 0.7 Mt grading 0.28 and 0.26 %Cu, respectively. Little work has been undertaken to date regarding recovery of copper from these resources and testing of these materials should also be incorporated in any future development plans.

The testwork conducted to date indicates a phased approach to project development, a first phase based on the implementation of a bioleach process for the recovery of copper from secondary sulphides and a subsequent phase based on a conventional mill and concentrator for the recovery of copper from primary sulphides.

1.12 Resource Estimate

The preliminary economical assessment documented in this technical report is based on mineral resources estimated using generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.

In the opinion of Tetra Tech, the resource evaluations reported herein are a reasonable representation of the global copper and molybdenum mineral resources in the Antilla deposit at the current level of sampling.

The mineral resource model was prepared by Tetra Tech and considers 88 core boreholes (14,293 metres) drilled from 2003 to 2010 by Panoro and the previous operators of the property. The resource estimation work was completed by Paul Daigle, PGeo, a qualified person as the term is defined in National Instrument 43-101. In October 2015, Tetra Tech re-classified and re-defined the mineral resources based on a revised conceptual pit shell. The block model however is unchanged from that defined by Tetra Tech in 2013. The revised Mineral Resource Statement for the Antilla copper deposit is presented in Table 1-1. A cut-off grade of 0.175 percent copper equivalent (CuEq %) is used for mineral resource reporting.

No mineral reserves have been defined for the Antilla deposit.

Table 1-1 Mineral Resources for the Antilla Deposit, October 19, 2015

Domain	Quantity	Grade		
	'000 tonnes	Cu %	Mo %	CuEq%
Indicated*				
Overburden/Cover	5,600	0.25	0.01	0.28
Leach Cap	13,400	0.25	0.01	0.27
Supergene	168,900	0.41	0.01	0.42
Primary Sulphides	103,900	0.24	0.01	0.26
Total Indicated	**291,800**	**0.34**	**0.01**	**0.36**
Inferred*				
Overburden/Cover	500	0.22	0.009	0.24
Leach Cap	13,400	0.21	0.008	0.22
Supergene	25,900	0.34	0.008	0.36
Primary Sulphides	50,700	0.24	0.007	0.25
Total Inferred	**90,500**	**0.26**	**0.007**	**0.28**

**Mineral resources are not mineral reserves and have not demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. Reported at a cut-off grade of 0.175 CuEq%; assuming an open pit extraction scenario, a copper price of US$3.25 per pound and a molybdenum price of US$ 9.00 per pound, and a metallurgical recovery of 90% for copper and 80% for molybdenum.*

1.13 Mining Methods

A scoping level mine plan, pit design, production schedule, and mining costs are developed for the Antilla project based on an open pit mining method. A Lerchs-Grossman (LG) analysis is used to carry out a series of pit optimizations using the resource block model, applying a range of metal prices and estimated costs for mining and processing. An ultimate pit limit is chosen from the optimized LG pit shells and is used to guide the detailed pit designs.

Detailed pit designs include four phases of mining to even out waste stripping and target areas of higher economic return earlier in the project life. The pit designs require cast-blasting and/or dozer push mining of narrow development upper benches to establish operable mining benches for a conventional contractor fleet of haul trucks and excavators/shovels.

The mine plan includes 1 ½ years of pre-production and 16 ½ years of standard open pit mining operations, for a total mine life of 18 years. The production schedule targets 20,000 tonnes/day of leach material, resulting in 7.3 million annual tonnes of crushed leach material placed on the leach pad. Marginally economic leach material is stockpiled to increase leach material grades. The stockpile is reclaimed in later years of the mine schedule when it is economically beneficial to do so. Mine waste rock is used to construct site infrastructure where applicable or is placed in the Rock Storage Facility to the west of the pit area.

The production Schedule is summarized in
Table 1-2, based on an average annual Net Smelter Return cut-off of $8.10/tonne.

Table 1-2 Summary of Mine Production Schedule

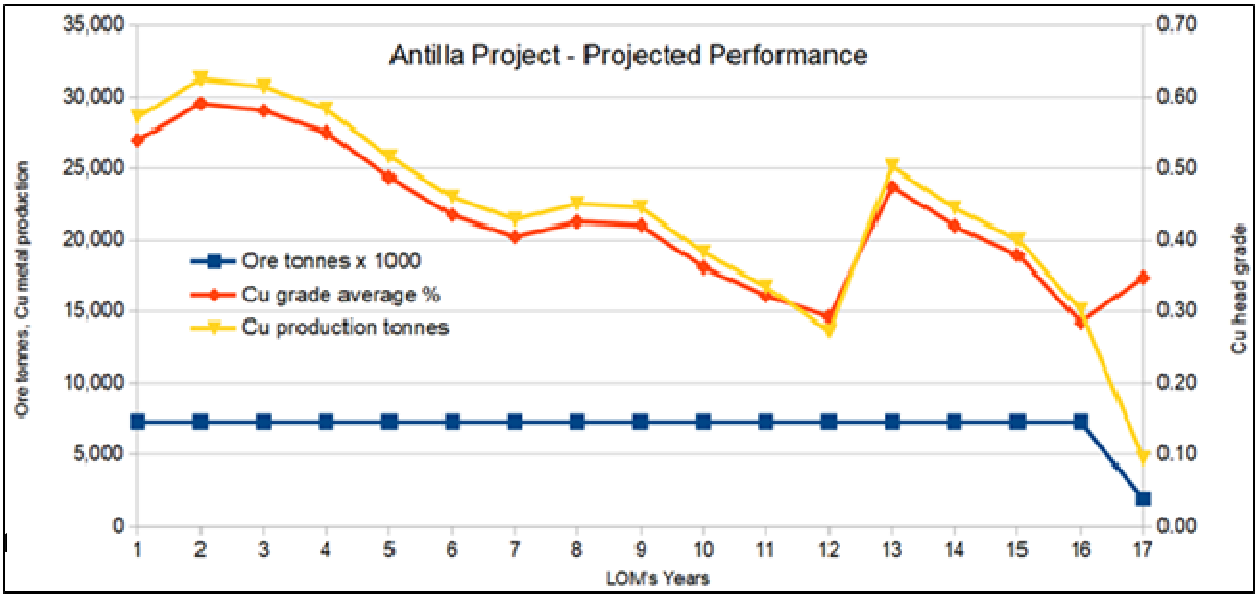
Item	Unit	Typ. Annual Average	LoM Total
Leach Material Total	**kT**	**7,300**	**118,667**
NSR	USD/t	20.90	
Cu Grade	%	0.437	
Pit to Leach Pad	**kT**	**6,953**	**113,110**
NSR	USD/t	21.40	
Cu Grade	%	0.445	
Pit to Stockpile	**kT**	**1,073**	**18,319**
NSR	USD/t	8.75	
Cu Grade	%	0.283	
Stockpile to Leach Pad	**kT**	**327**	**5,557**
NSR	US$/t	10.73	
Cu Grade	%	0.276	
Un-reclaimed Stockpiled Material	kT	-	12,762
Waste Mined	kT	9,335	**159,439**
Total Material Mined	**kT**	**16,952**	**290,868**
Total Material Moved	**kT**	**17,279**	**296,425**

1.14 Production and Processing

The Antilla Project will operate a conventional hydrometallurgical process consisting of a crushing plant, a valley-fill leaching plant, solvent extraction plant, and Electrowinning plant to produce grade-A copper cathodes.

The run-of-mine ore (ROM) is trucked from the open pit to the crushing plant, the crushed ore will be trucked to the heap leaching area to form the ore lifts. Each lift will be irrigated with acid solution. The total capacity of the leaching area is 120 million tonnes over 17 years of life of mine.

The percolating solutions (PLS) at a rate of approximately 650 m3/h is will be collected at the bottom of the valley on the pregnant solution pond. The valley's bottom will be make impermeable with synthetic liner to ensure all percolating solutions will report to the PLS pond. The pregnant leach solution (PLS) will be process in a solvent extraction (SX) circuit followed by an Electrowinning plant to produce grade-A copper cathodes. The raffinate solution generated in the SX plant is recirculated to the leaching circuit after conditioning with acid and make-up water to irrigate the ore. The copper cathodes are trucked off site to an ocean port for sale to markets at an initially rate of 30,000 tonnes per year that will progressively decrease to 5,000 tonnes by year 17 of the life of mine plan. See Figure 1-1.



1.15 Project Infrastructure

The Antilla project will collect runoff water from Huancaspato River passing beside the operational area and the Ticlia lagoons located 7 kms to the northwest, both into the Antilla's property. The nearby electrical power Cotaruse-Las Bambas line is available and connected to the Cotaruse Substation, constructed by Abengoa Peru S.A., a power generator company who operates and maintains the concession on this line. The estimated power requirements for the Antilla project are 13 MW. The project includes cathode transport by road to Marcona Seaport.

1.16 Capital and Operating Costs

The capital cost estimate for the Antilla Project is developed to a level appropriate for a PEA study to estimate overall project viability. As such the level of accuracy is +/-50%, which is suitable for a PEA-level study.

The initial capital costs are summarized in the Table below:

Table 1-3 Capital Cost Estimate Summary

Item	Cost (Million USD)
Mine Equipment	$1.3
Mine Development	$41.1
Crushing, SX, and EW plants	$94.7
Infrastructure	$42.4
Sub-total	***$179.5***
Owner's Cost	$7.8
Indirect Costs	$13.7
Sub-total	***$201.0***
Contingencies	$49.4
Total Initial Capital Cost	**$250.4**

The total life-of-mine (LOM) operating costs are $7.04/tonne. This estimate includes contractor mining, processing, leach material haulage, G&A, re-handle, and closure costs. The LOM average breakdown is summarized in Table 1-4 below.

Table 1-4 Antilla On-site Operating Costs

Item	Cost (USD/tonne)
Mining Costs	$1.63
Processing Costs (including crushing)	$3.85
Average Leach Material Haulage Costs	$0.81
G&A Costs	$0.75
Total On-site Operating Cost	**$7.04**

1.17 Economic Analysis

The PEA project economics are based on a long-term copper price of $3.05/lb. The long-term forecasts are derived from prices periodically published by large banking financial institutions and are applied to Years 4-17 of the mine life. Shorter term copper prices (from the same sources) were used in Years 1-3 of the mine life, reflecting higher price forecasts in the shorter term. Molybdenum is not included in the proposed process recovery and not included in the project economics. A summary of the input parameters for the economic analysis is presented in the table below:

Table 1-5 Input Parameters for the Economic Analysis

Parameter	Value	Unit
Copper Price – long term	3.05	$/lb
Copper Price – short term	3.20/3.15/3.10	$/lb
Copper Payable	96	%
Off-site Costs	0.03	$/lb

A summary of the financial outcomes comparing the base case metal prices to 2 alternative metal prices is presented below. The base case prices are derived from forecasts published by large banking and financial institutions. Alternate scenarios of -$0.30/lb and +$0.20/lb are presented to show the economic sensitivity to varying copper prices.

Table 1-6 Economic Sensitivity to varying Copper Prices

	Pre Tax					After Tax					
	Cu Price (*)										
	From Y4 to LOM										
Scenarios	($/lb)	NPV 5% (million USD)	NPV 7.5% (million USD)	NPV 10% (million USD)	IRR	Payback	NPV 5% (million USD)	NPV 7.5% (million USD)	NPV 10% (million USD)	IRR	Payback
Min	**2.75**	487	383	301	28.8%	2.9	232	169	118	18.7%	3.6
Base	**3.05**	648	**520**	418	34.7%	2.6	393	**305**	236	25.9%	3.0
Max	**3.25**	755	611	497	38.4%	2.5	501	397	314	30.3%	2.7

1.18 Environmental and Social Considerations

Existing environmental liabilities associated with the project are restricted to those associated with an exploration-stage project, and include drilling sites and access roads. Limited environmental baseline studies have been conducted to date, and some social engagement program has been initiated. In order to advance the project, additional environmental baseline studies and a social engagement program would be required. Once a comprehensive project description has advanced to feasibility level, an environmental impact assessment must be conducted. Approval of the environmental impact assessment will initiate the regulatory phase of the project, and contingent to approval of various other permits and authorizations, construction and operations may start after the approval of the environmental impact assessment and receipt of applicable permits.

1.19 Conclusions and Recommendations

Three areas of Geological Potential are recognized at Antilla: 1) next to, and below, to the economic PEA Pit, 2) in an area 6km around the PEA Pit, and 3) in an area 10km to the west in the El Piste zone. All of these represent opportunities to increase the scale of the project.

A PEA open pit mine plan has been developed for the Antilla project incorporating a heap leach recovery method. The average mill feed of the 17 year mine plan is 20ktpd with a LOM average grade of 0.43% Cu. The results of the PEA show improved project economics, a lower capital cost and lower operating costs.

An estimated budget and plan for PFS has been recommended with additional work plans for geotechnical, geochemical, metallurgical testing, mine planning optimization, two drilling programs, environmental characterization, and baseline studies.

2 Introduction

This Preliminary Economic Assessment (PEA) is written for the Antilla project which is 100% owned by Panoro Minerals Ltd. (Panoro) The Antilla project is a copper-molybdenum porphyry deposit, located 140 km south west of the city of Cuzco, in the Apurimac region of Southern Peru. The purpose of this Technical Report is to present the results of the PEA that considers a heap leach and SX/EW operation of the Antilla Project which utilizes a heap leaching process. This Technical Report supersedes the previous Technical Report, dated June 16, 2016.

During 2017, Panoro commissioned Moose Mountain Technical Services to complete a scoping study on the Antilla Project which considered a heap leach process. The results of the scoping study were used to justify the production of a PEA. The lead author of this PEA, Jesse Aarsen, P.Eng. Sr. Associate of MMTS, an independent qualified person as defined by National Instrument (NI) 43-101, conducted a site visit Jan 30 – Feb 01, 2018.

This Technical Report is prepared in accordance with the standards set out in NI 43-101 and is a technical summary of the available geologic, geophysical, geochemical, and diamond drillhole information. The authors, in writing this report, use sources of information as listed in the references section. All currency amounts are in USD unless reported otherwise. All units in this report are metric and Universal Transverse Mercator (UTM).

Several authors contributed to or supervised the completion of this Technical Report and are all Qualified Persons (QP) within the meaning of NI 43-101 standards. Each QP in this report takes responsibility for their work as outlined in their QP certificates included in this report and found in the following chart:

Qualified Person	Firm	PEA Area	Professional Affiliation (and registration number)
Jesse Aarsen, P.Eng.	Moose Mountain Technical Services Ltd.	Mining, Infrastructure	APEGBC (#38709)
Luquman Shaheen, P.Eng.	Panoro Minerals Ltd.	Marketing, Copper Pricing	APEGBC (#21675)
Andrew Carter	Tetra Tech Inc.	Mineral Processing and Metallurgical Testing	EURING (#2920GB) CENG (#378467) MIMMM (#46421) SAIMM (#19580) SME (#4112502)
Daniel Sepulveda	Moose Mountain Technical Services Ltd.	Recovery Methods, Processing Capex and Opex	SME #4206787RM
Luis Vela, CMC	Panoro Minerals Ltd.	Geology, Exploration, Mineral Tenure, Permits	CMC (#0173)
Paul Daigle, P.Geo.	Tetra Tech Inc.	Resources	APGO (#1592)

The responsibilities of each Technical Report section are outlined in the Table below:

Section	Title	Responsible Person(s)
1	Summary	All
2	Introduction	Jesse Aarsen
3	Reliance on Other Experts	Jesse Aarsen
4	Property Description and Location	Luis Vela
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Luis Vela
6	History	Luis Vela
7	Geological Setting and Mineralization	Luis Vela
8	Deposit Types	Luis Vela
9	Exploration	Luis Vela
10	Drilling	Luis Vela
11	Sample Preparation, Analyses and Security	Luis Vela
12	Data Verification	Luis Vela
13	Mineral Processing and Metallurgical Testing	Andy Carter
14	Mineral Resource Estimates	Paul Daigle
15	Mineral Reserve Estimates	Jesse Aarsen
16	Mining Methods	Jesse Aarsen
17	Recovery Methods	Daniel Sepulveda
18	Project Infrastructure	Jesse Aarsen
19	Market Studies and Contracts	Luquman Shaheen
20	Environmental Studies, Permitting and Social or Community Impact	Luis Vela
21	Capital and Operating Costs	All
22	Economic Analyses	Jesse Aarsen (pre-tax) Luis Vela (after-tax)
23	Adjacent Properties	Luis Vela
24	Other Relevant Data and Information	Jesse Aarsen
25	Interpretation and Conclusions	All
26	Recommendations	All
27	References	All

3 Reliance on Other Experts

With respect to legal title and tenure information, the authors have relied on the opinion of Humberto Martinez A., as expressed in a letter referring to Panoro Apurimac S.A. a wholly-owned subsidiary of Panoro, dated 03 April 2018.

The qualified persons have relied on Panoro for taxation estimates applicable to the Antilla project. After-tax estimates are provided by Yves Barsimantov to Jesse Aarsen. After-tax information is used in the economic analysis results presented in Section 1 (Summary), 22 (Economic Analyses), 25 (Interpretation and Conclusions) and 26 (Recommendations).

4 Property Description and Location

The Antilla project is located approximately 150 kilometres southwest of Cusco, in Apurímac Region, Peru and approximately 500 kilometres southeast of Lima, the capital city of Peru (Figure 4-1).

The centroid of the project area is located approximately 14 degrees, 21 minutes southern latitude and 72 degrees, 58 minutes, western longitude. The villages close are Sabaino, approximately 20 kilometres to the northeast, and Antilla, which is in close proximity to the project.

Figure 4-1 Location of the Antilla Project in Apurimac Region of Peru

4.1 Mineral Tenure

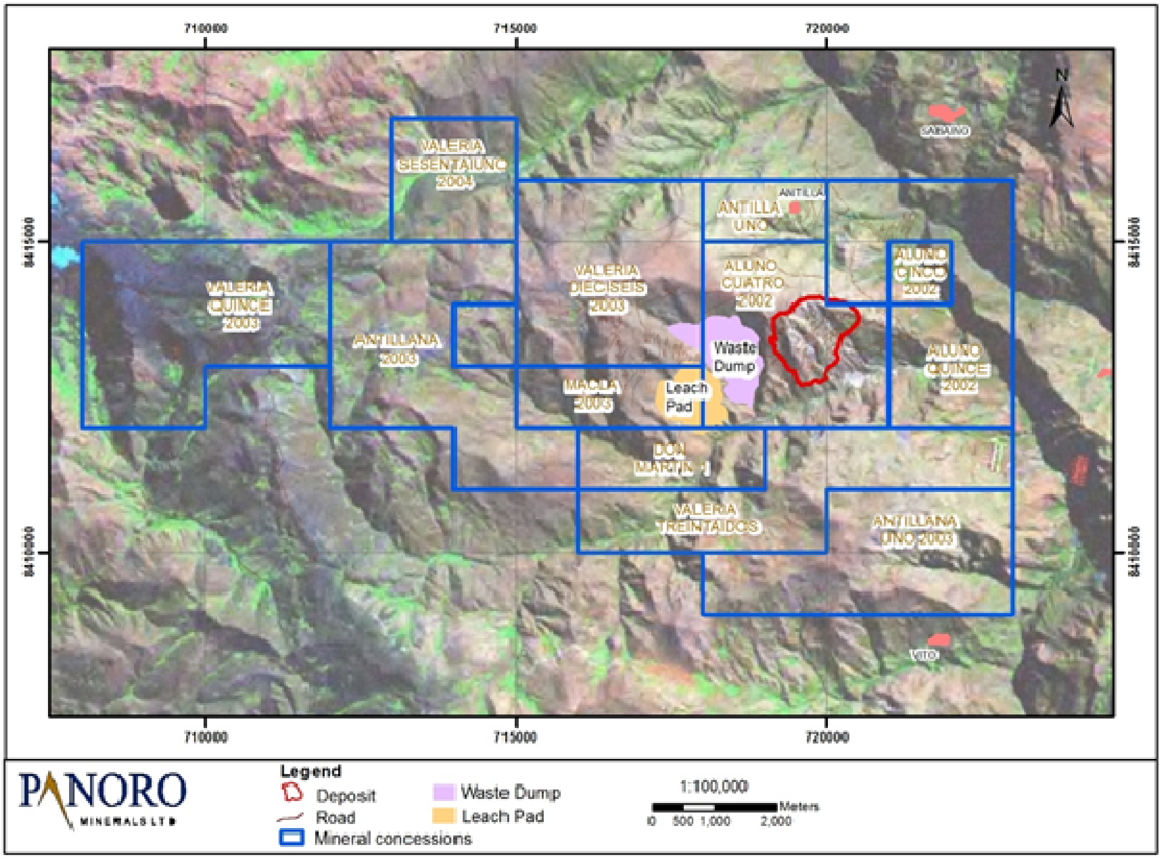
The project area comprises twelve mining concessions with a total area of 7,500 hectares (Table 4-1). The concessions are owned 100% by Panoro Apurimac S.A. a wholly-owned subsidiary of Panoro.

Panoro Operated under a Class B exploration permit. In June 2014, Panoro Apurimac SA informed to The Ministerio de Energia y Minas, the ending of its activities of exploration in the project. Authorities accepted the environmental report with good standing performance.
The property is subject to annual payments to maintain the concessions in good standing and the concessions are renewed on a yearly basis every May. The last annual payment was in April 24, 2018. As the time of writing this report, all concessions were in good standing.

Table 4-1 Mineral Tenure Information

Concession	Concession Name	Has.	Grant Date	Expiry Date
10170402	Aluno Cinco 2002	100	31/01/2003	June 2019
10170302	Aluno Cuatro 2002	800	31/01/2003	June 2019
10200202	Aluno Quince 2002	900	21/03/2003	June 2019
10043803	Valeria Quince 2003	1,000	27/01/2004	June 2019
10043903	Valeria Dieciseis 2003	900	09/09/2003	June 2019
10329903	Valeria Treintaidos	800	04/03/2004	June 2019
10344303	Antillana 2003	1,000	26/02/2004	June 2019
10344203	Antillana Uno 2003	800	16/02/2004	June 2019
10166404	Valeria Sesentaiuno 2004	400	09/08/2004	June 2019
10002003	Macla 2003	300	17/06/2003	June 2019
10313306	Don Martin 1	300	17/11/2006	June 2019
10059709	Antilla Uno	200	16/07/2009	June 2019
Total		**7,500**		

The mineral resources considered for the preliminary economic assessment are all located within the Aluno Quatro 2002 mineral title and the ancillary infrastructure of the conceptual project are located on the Aluno Quince 2002, Aluno Cinco 2002, Valeria Dieciseis 2003, Macla 2003, Don Martin 1, and Valeria Treintaidos mineral titles, as shown in Figure 4-2.



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Panoro entered into an agreement with the local community of Antilla (Comunidad Campesina de Antilla) to allow access to the property for the purposes of carrying out mineral exploration activities. This access agreement did not include construction or pre-production activities on the property.

Panoro knows that there is a dispute related to the Ownership of Superficial Land Rights in an area of 650ha included in the Antilla Community territory. The dispute is within Antilla Community and a Private Family.

MMTS and Panoro are not aware of any significant factors or risks that may affect access, title, or the right or ability to perform work on the property.

4.2 Underlying Agreements

In 2007 Panoro entered into a binding purchase and sale agreement with Cordillera de las Minas S.A. (Cordillera) to acquire all outstanding shares of Cordillera for US$13 million in cash.

Cordillera was the owner of a portfolio of 13 properties, including the Antilla project. The purchase was finalized on June 8, 2007.

In April 2010, Panoro entered into a joint venture agreement with Chancadora Centauro S.A. (Chancadora), whereby Chancadora would make cash payments of US$8 million to Panoro and invest US$17 million into the property to earn a 70% interest.

In September 2010, Chancadora was in breach of the joint venture agreement. As a result the property was under arbitration and an injunction was put in place on the property. No further exploration activities were conducted until January 3, 2013, when the Arbitration Tribunal decided in favour of Panoro and the injunction was officially lifted. On July 3, 2013, the decision was entered into the Register of Mining Rights. There are currently no contractual or judicial limitations on the property.

The project is subject to Peruvian government mining royalties as discussed in Section 3.5. Panoro has advised that the Antilla project is not subject to any private royalties.

4.3 Permits and Authorization

Panoro in the past held the following permits:

- Environmental Assessment Category C
- Permit for surface geophysical and other surface surveys
- Authorization to use water
- Constitutional agreement with the community of Antilla regarding surface and easement rights.

These permits and authorizations expired between 2011 and 2014. MMTS has not been informed on efforts by Panoro to reinstate these permits and others that may be required for exploration and development work on the Antilla property.

4.4 Environmental Considerations

MMTS was informed by Panoro that the property is not subject to environmental liabilities.

4.5 Mining Rights in Peru

The mining industry in Peru is well regulated and governed by straightforward mining laws. The industry is primarily regulated by mining laws enacted by the Peruvian Congress and regulations issued by the executive branch of government. The General Mining Law was approved in 1992, with the aim of attracting foreign investment in the sector. However, since 2000, a number of countervailing laws that focus on sustainable development have been enacted.

The Ministry of Energy and Mines, a principal central government body in Peru, has the authority to regulate mining activities within the Peruvian territory. The ministry also grants mining concessions to local or foreign individuals or legal entities, through a specialized body known as the Institute of Geology, Mining and Metallurgy. Four types of concessions may be granted under Peruvian mining law:

- Mining concession: Grants the right to explore and exploit mineral resources, whether metallic or non-metallic, conferred by the concession.
- Processing concession: Grants the right to process, purify, melt, or refine minerals, whether by means of physical, chemical, or physical-chemical processes.
- General Service concession: Grants the right to render auxiliary services, such as ventilation, sewage, hoisting, or exploitation to one or more mining concessions.
- Mining transport concession: Grants the holder the right to operate a continuous massive transportation system of mineral products between one or more mining unit, and one harbor or processing plant or refinery, using conveyor belts, pipelines, and track cables.

Currently, these concessions are granted on a first-come, first-served basis, without any preference given to the technical and financial qualifications of the applicant. With the exception of mining concessions located in urban expansion areas, the term of a mining concession is indefinite, provided that the title holder fulfills all regulatory obligations including payment of annual license fees of US$3/hectare. Failure to pay the applicable license fees for two consecutive years will result in the termination of the mining concession.

Apart from obtaining a concession from the Ministry of Energy and Mines, a mining company must submit and receive approval for an environmental impact study that includes a social relations plan, certification that there are no archaeological remains in the area, and a draft mine closure plan. In addition, the mining company has to separately obtain water rights from the National Water Authority and surface lands rights from individual landowners.

In April 2012, Peru's government approved a Prior Consultation Law that requires prior consultation with indigenous communities before any infrastructure or projects, especially mining and energy projects are developed in their territories.

On October 31, 2012, Peru's the Commission of Economy of the Congress approved the creation of a new oversight institution, known as the national environmental certification service (SENACE), within the Ministry of Environment. This new institution comprises representatives of six ministries and is chaired by the environment ministry. It has the principal function of reviewing and approving environmental impact assessments for large-scale investment projects. Previously, the Ministry of Energy and Mines was responsible for both awarding mining concession agreements and approving environmental impact assessments. From July 2015 onwards, the national environmental certification service was given the authority to grant environmental certifications for projects that require a Detailed Environmental Impact Assessment.

The corporate income tax rate in Peru for resident companies is 29.5%. In all cases, the fiscal year corresponds to the calendar year.

The tax rate on dividends is 5.0% and on interest from foreign financing is 4.99%, under specific conditions.

Mining companies pay the standard corporate tax on annual income (29.5%). In addition to the corporate income tax, mining companies must pay to the state, on a quarterly basis, a royalty for exploitation of mineral resources based on operational profit.

Under the 2004 Mining Royalty Law, revised in 2011, the royalty is applied to the quarterly operational profit of the mining companies, at an effective rate varying from 1% to 12%, depending on the operational margin identified within 16 separate brackets.

A minimum royalty payment equivalent to 1% of the sales is always required. The law al defines the distribution of royalties among Local Governments, Province Governments, Regional Governments and National Universities.

In addition to the above, there are two industry-specific taxes levied on the mining sector by the central government:

- Special Mining Tax: Mining companies without tax stability agreements with the government (further explained below) are subject to this tax. It applies to the quarterly operational profit at an effective rate varying from 2% to 8.4%, depending on the operational margin identified within 17 separate brackets.
- Special Mining Levy: Mining companies that have taxation stability agreements previously signed with the government are subject to this contribution in the form of a levy. It applies to the quarterly operational profit at an effective rate varying from 4% to 13.12%, depending on the operational margin identified within 17 separate brackets.

The following benefits, among others, are offered to companies undertaking mining activities:

- Foreign currency: The Peruvian constitution establishes equal protection for domestic and foreign investors who may enter into agreements with the government and guarantees free access, possession, and disposal of foreign currency.
- Tax stability: Investors who undertake large mining operations may enter into tax stability agreements with the government, for periods of ten, twelve or fifteen years. Two percentage points are to be added to the corporate income tax rate. Tax stability would include taxes known as *impuestos*. No taxes created subsequently to the date of execution of the stability agreement shall be applicable.

Finally, mining companies must pay taxes to the government for specific services (known as “contributions” – *contribuciones*) related to Environmental and Mining control and audit services. The government institutions in charge of these services are OEFA (*Organismo de Evaluación y Fiscalización Ambiental*) and OSINERGMIN (*Organismo Supervisor de la Inversion en Energía y Minería*) respectively. Each contribution is calculated over the monthly sales of the company, with the following rates:

- Contribution to OEFA: a fixed 0.11% for years 2017, 2018 and 2019, as established by Supreme Decree 097-2016-PCM.
- Contribution to OSINERGMIN: 0.15%, 0.14% and 0.13% for years 2017, 2018 and 2019, respectively, as established by Supreme Decree 099-2016-PCM.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The property is situated approximately 145km by truck to the south of Abancay, capital of Apurimac Region and is most easily accessed from Cusco via Abancay, Antabamba, and the Antabamba valley leading to Panoro's base camp at the village of Antilla (Figure 5-1).

The project site is a further 6.5 kilometres along an access road from the base camp. Access around the project site is limited to drill roads developed in the mid-2000s. These roads are in good condition but have not been maintained.

The main highways in this region of Peru are paved. The secondary highways are generally unpaved but are well maintained. The highways and roads through the mountains are subject to many switchbacks to overcome the high relief, therefore highway distances are longer than they appear. The drive from Cusco to the property is typically eight hours.

There are regular scheduled flights to and from Cusco. Flight time from Lima to Cusco is typically one hour.

Figure 5-1 **Project Access Map**

5.2 Local Resources and Infrastructure

Abancay, population 51,462 (2007), is the closest major town to the property and can provide most general supplies. Food and fuel can be found in the surrounding villages closer to the project site as well. Mining related equipment and skilled and professional services must be sourced elsewhere.

Unskilled labour may be found in the nearby villages.

Panoro has set up a semi-permanent base camp with fixed buildings for a kitchen, offices, and core logging and storage facilities. Weather haven tents are set up for accommodation.

The property is relatively isolated from public infrastructure such as roads, power, and cell phone coverage. A regional road passes the property 17 kilometres to the east; other roads are limited to a small network of access roads. Reliable cellular coverage can be found in the village of Antilla.

Some portions of the property are serviced by less-reliable cellular coverage.

There is no source of electricity on the property except for a low-voltage line that services the village of Antilla. There is a 220 kV substation (Cotaruse S.E.) located approximately 42 kilometres southwest of the property; recently a power line has been constructed from the substation to the Las Bambas copper-molybdenum mine.

The nearest major airport and railhead are in Cusco. Water is sourced from the creeks and rivers in the valleys on and around the property, among them the Antabambas and Mollegamga rivers that cross the property in the east.

5.3 Climate

The project located is a temperate highland tropical climate zone (Cwb; Köppen climate classification) and is characterized by dry winters and rainy summer seasons. Generally, the dry winter season lasts from May to October, while the wet summer season occurs from November to April. Precipitation ranges from less than 4mm to 150mm per months in the winter and summer seasons, respectively.

Daytime temperatures in the dry season range between 18 and 22 degrees Celsius (°C) with highs near 30°C. Night time temperatures tend to be cold. The wet season has moderate variations in temperature with the daytime average ranging between 15 and 18°C and night time lows between 5 and 8°C (Wright 2009).

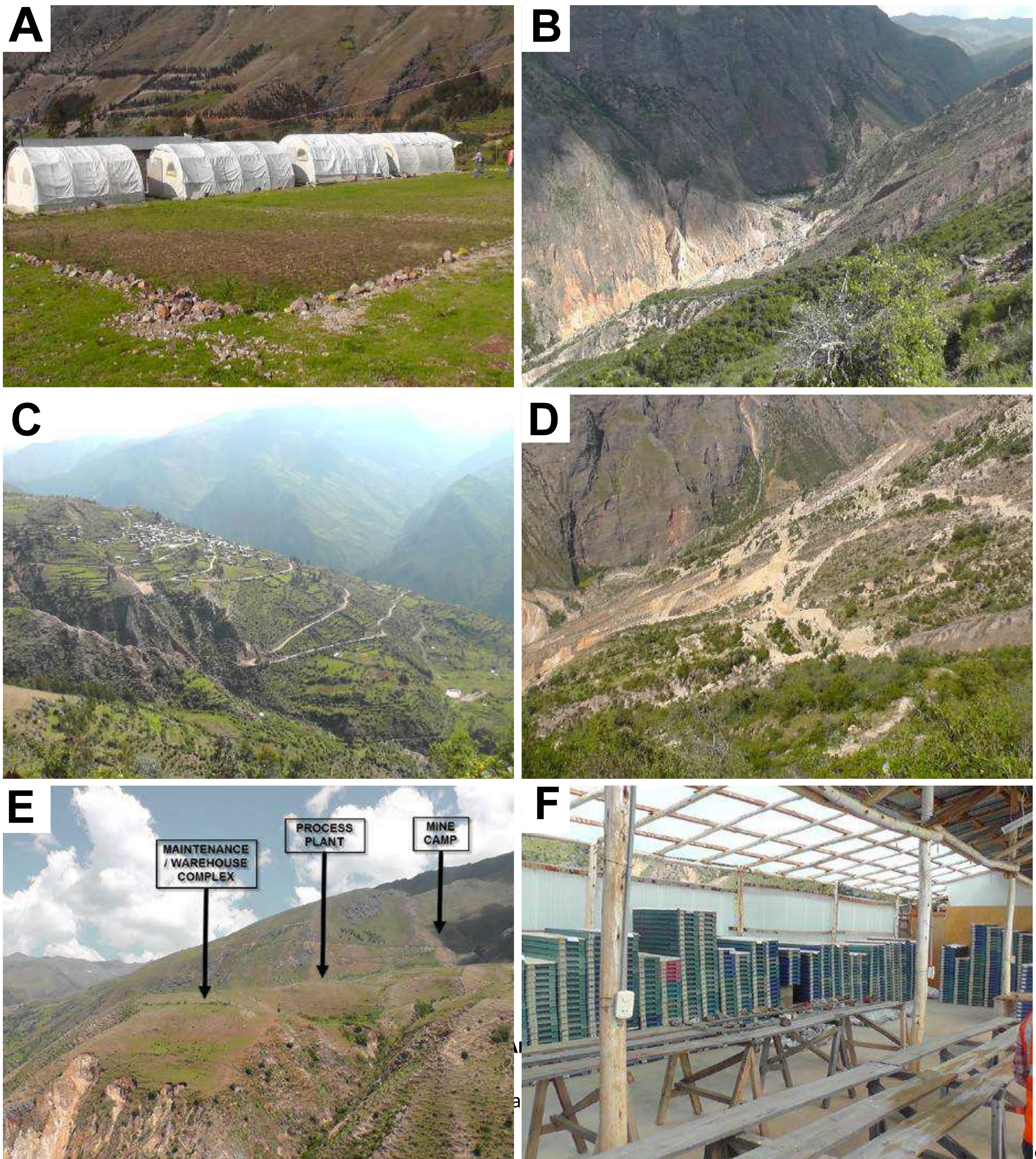
Exploration activities can take place year-round.

5.4 Physiography

The property is located in the high altitudes of the Andean Cordillera where elevations are between 2,500 to 4,500 metres above sea level. Relief on the property is from moderate slopes along ridge tops to very high slopes along the flanks of the ridges. The region is characterized by deeply incised river valleys and canyons such as the Rio Antabamba, which lies 600 metres below the Antilla village (Wright 2009).

The Antilla deposit is situated on the northern slope of the steeply eroded valley (Creek Huancaspaco) where elevations are from 3,100 to 4,200 meters above sea level.

The vegetation on the property is sparse and limited to alpine grass and shrubs in the higher elevations. Eucalyptus trees have been planted along the property access roads to strengthen the road cuts along the steep slopes. For general views of the property refer to Figure 5-2.



C. Village of Antilla
D. Access to drill pads along mountain flank
E. Proposed plant/mine camp location
F. Core logging and storage facility

6 History

The following is modified from Wright (2009).

6.1 Southern Peru Copper S.A., 1999

In 1999, Southern Peru Copper S.A. (Southern Peru) carried out regional exploration work on the property including drilling on an optioned property immediately to the east of what became the Antilla project area. Poor results caused Southern Peru to abandon the project.

6.2 Cordillera de Las Minas S.A., 2002 to 2005

In 2002, Cordillera de Las Minas S.A (Cordillera) explored Peru for large copper deposits. Anaconda Peru S.A. (Anaconda), a Peruvian subsidiary of Antofagasta Plc (Antofagasta), transferred ownership of several groups of exploration concessions in southern Peru to Cordillera. Companhia Vale do Rio (Vale), through its subsidiary Compañía Minera Andino-Brasilera (Minera Andino) had the option to acquire a 50% interest in Cordillera by spending US$6.7 million on exploration over three years (Vale 2002).

In 2002, Cordillera carried out geochemical exploration and followed up anomalous responses to the west of Calvario Hill, where Southern Peru had worked, and staked the first 2,800 hectares of mineral concessions. In 2003, geological mapping and geophysical surveys led to a drilling program in September 2003 that extended into 2004. Ten core boreholes (1,992 metres) were completed, outlining the mineralized zone at Antilla. Three boreholes were abandoned after 20 to 50 metres, recollared, and subsequently drilled to their final depth.

In 2004, Cordillera drilled eight core boreholes testing targets that had been defined during mapping and geophysical surveys in 2003 on the western half of the property. Results were generally disappointing and in 2005 the Cordillera joint venture returned to the eastern part of the property to complete an additional five boreholes (821 metres) in an attempt to extend the known mineralization to the north and southwest. Results of the 2005 campaign were disappointing and led to the dissolution of the joint venture.

Core boreholes from the Cordillera campaigns were logged for descriptive rock type and alteration using graphic logs and geotechnical data such as fracture density, recovery, and rock quality designation (RQD) were recorded. Samples were sent for preparation and analyses to the CIMM Peru SA laboratory in Lima. Samples were assayed for total copper, arsenic, silver, gold, lead, zinc, and sequential soluble copper. No independent analytical quality control 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.

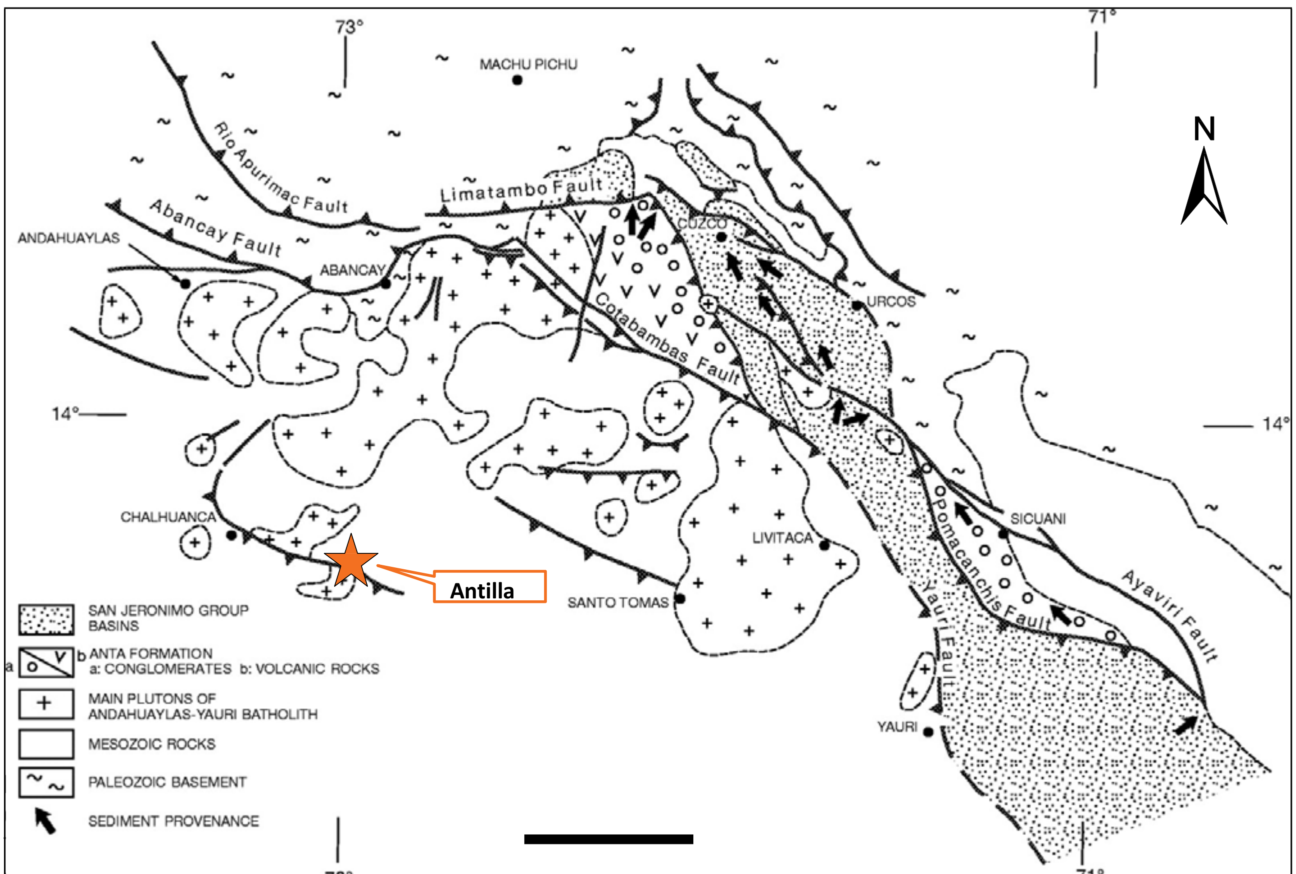
7 Geological Setting and Mineralization

7.1 Regional Geology

The Antilla project is located in the Andahuaylas-Yauri Belt of the high Andes of southern Peru. 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 north-west south-east (Figure 7-1). At the deflection the normal subduction of southern Peru and northern Chile changes to flatter subduction below central and northern Peru.

The geology of the Andahuaylas-Yauri Belt is dominated by the Andahuaylas-Yauri Batholith, which is exposed for approximately 300 kilometres between the towns of Yauri in the southeast and Andahuaylas in the northwest. The batholith width ranges from 25 kilometres at the east end to 130 kilometres near Abancay. It is composed of early mafic to intermediate intrusive rocks with cumulate textures, grading to intermediate intrusive rocks with equigranular to porphyritic textures.

The batholith intrudes Precambrian to Paleozoic basement rocks which are exposed to the northeast. The basement sequence culminates in Permian to Early Triassic age Mitu Group volcaniclastic and clastic rocks.



Western Peruvian basins. The eastern basin is made up of marine clastic and carbonate rock. The western basin, exposed in what is now the Western Cordillera or Cordillera Occidental where the property is located, is a marine transgressional sequence grading from continental deep-water clastic sedimentary rock to limestone. The northeastern edge of the western basin includes the Lagunillas and Yura Groups, made up of middle to late Jurassic quartz-sandstones, quartzite, and shale grading 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 sandstone, quartz sandstone, and quartzite, which hosts the Antilla sulphide deposit.

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

Major mineralization styles in the region include porphyry copper (with subordinate molybdenum and gold), iron-copper skarn, and minor epithermal vein-style mineralization. Since the commissioning of the Tintaya mine by BHP in 1999 at the southeastern end of the belt, major copper deposits have been brought to feasibility at Antapaccay, Las Bambas, and Los Chancas. Fifteen to 20 other copper deposits, including Antilla, are currently being explored by Peruvian and multinational mining and exploration companies (Figure 7-2).

7.2 Property Geology

Quartzite and quartz-arenite of the Soraya Formation outcrop over most of the central and eastern part of the property and host the intrusive rocks and mineralization defined to date. The clastic sedimentary rock are fine- to medium-grained, well laminated on sub-centimetre- to metre-scale and occasionally show other primary depositional features such as cross-bedding. The quartzite and quartz-arenite units can be intercalated with centimetre- to 10-centimetre-scale siltstone or lutite beds.

At the bottom of the canyon in road cuts and behind Calvario Hill, the Chuquibambilla Formation is exposed, comprising outcrops of mudstone, lutite, and arenite.

Sedimentary rocks are intruded by at least two intrusive rock types: altered and weakly-mineralized Main Porphyry stocks or aphophyses and narrow, unaltered Late Porphyry dikes. The altered, weakly-mineralized Main Porphyry is exposed as a prominent knob immediately to the west of the mineralized quartzite, and another, smaller intrusive body is exposed to the southeast of the mineralization (Figure 7-3). The Main Porphyry has medium-grained porphyroblasts of euhedral plagioclase accounting for approximately 25% by volume. Coarse, corroded, or rounded quartz crystals are also common and constitute approximately 5% of the porphyry by volume. Medium- to coarse-grained biotite, hornblende, and orthoclase are also important porphyroblasts and collectively constitute approximately 10% by volume. The remaining 60% of the volume of the Main Porphyry is composed of a groundmass of fine to glassy quartz and feldspar. The composition of the Main Porphyry has a granodiorite to quartz monzodiorite composition.

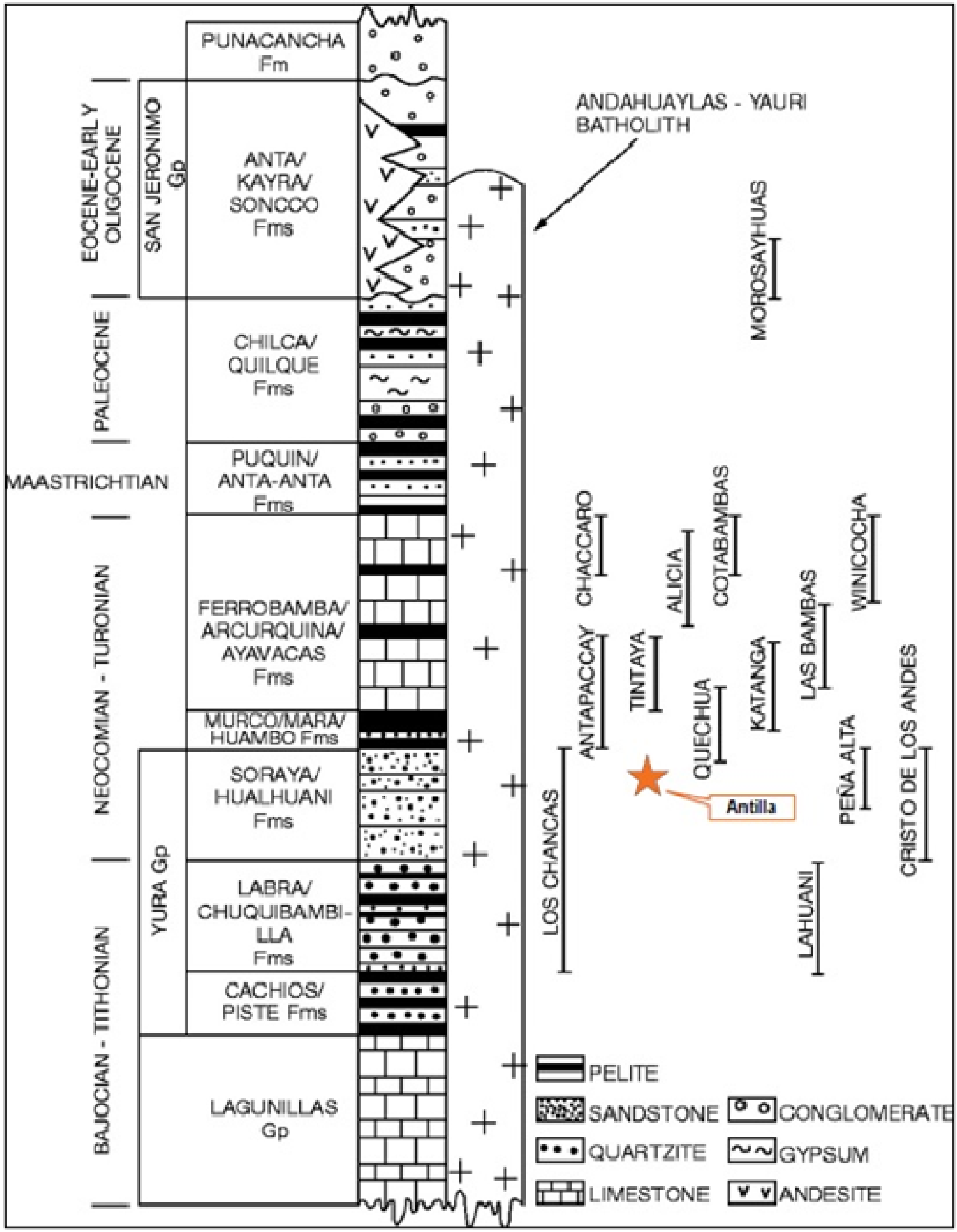
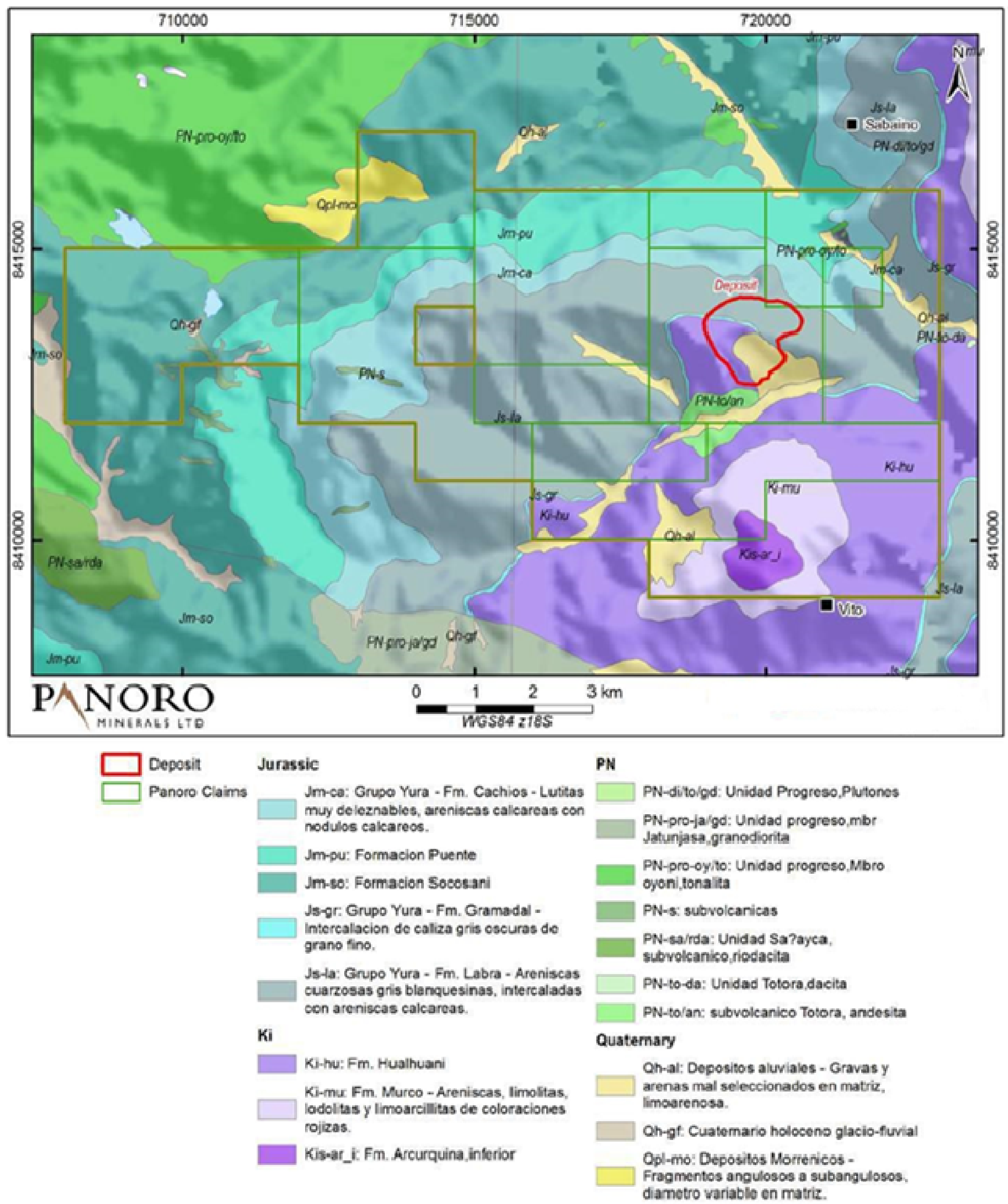


Figure 7-2 Regional Stratigraphic Position of the Antilla Deposit
(modified from Perelló et al., 2003)



...assy ... the rock, and biotite and amphibole porphyroblasts constitute an additional 15%. The Late Porphyry is distinguished from the Main Porphyry by its unaltered, dark-coloured groundmass, relatively low abundance of quartz porphyroblasts, and its tabular dike-like form of emplacement.

Late Porphyry dikes are general north-south-striking and are interpreted to be localized on normal faults that were active during the emplacement of the Andahuaylas-Yauri Batholith. Potassium-argon dating indicates that the bulk of the batholith was emplaced during the middle Eocene to early Oligocene (approximately 40 to 32 Ma, Perelló et al. 2003).

At least two other porphyritic intrusive bodies have been mapped on the property. A diorite porphyry with traces of copper mineralization is exposed on the western block of the property, and unmineralized monzonitic sills are exposed to the north east of the mineralized zone.

7.3 Structural Geology

Regional structural geology is dominated by the Andean Orogeny which, in the project area, is oriented approximately northwest-southeast. Tectonic activity was most active during the Eocene and Oligocene times, referred to as the Incaic pulse, and during an Oligocene to Miocene-age Quechua pulse (Pecho 1981). West-southwest-dipping thrust faults stack repeating packages of Mesozoic to Early Cenozoic sediments on top of each other to form a belt with a width of 300 kilometres. The younger sediments are in turn thrust northeastward on top of Palaeozoic to Precambrian basement (Figure 7-2). Deformation is most intense in the northeastern portion of the Western Cordillera where large north-verging folds are developed in the Ferrobamba formation.

At property scale, a series of steeply-dipping west-northwest-striking faults and conjugate northnortheast-striking normal faults with dextral offsets have been interpreted from outcrop mapping (Figure 7-4 E). The sense and throw of the faults is extremely difficult to determine due to the relatively monotonous sequence of clastic sediments. Reliable indicators of stratigraphic elevation such as marker beds have not been found.

7.4 Mineralization

The most important mineralization encountered to date on the property is a tabular body of fracturecontrolled and disseminated chalcocite and chalcopyrite with minor molybdenite-coated fractures overlain by a barren, leached zone of variable thickness. The tabular zone strikes 50 degrees and dips 20 degrees to the east over an area 1.2 kilometres long and 1.2 kilometres wide. The supergene chalcocite mineralization has a true thickness of 40 to 80 metres. Associated with the chalcocite mineralization is weak sericitization, chloritization, and silicification of arenite and quartzite. The strongest chalcocite mineralization is associated with brittle faults. Below the chalcocite mineralization, low-grade disseminated chalcopyrite, bornite, and molybdenite mineralization occurs. Altered, weakly-mineralized porphyritic felsic intrusives are associated with the hypogene mineralization. Unaltered, unmineralized porphyritic dikes cut the mineralization.

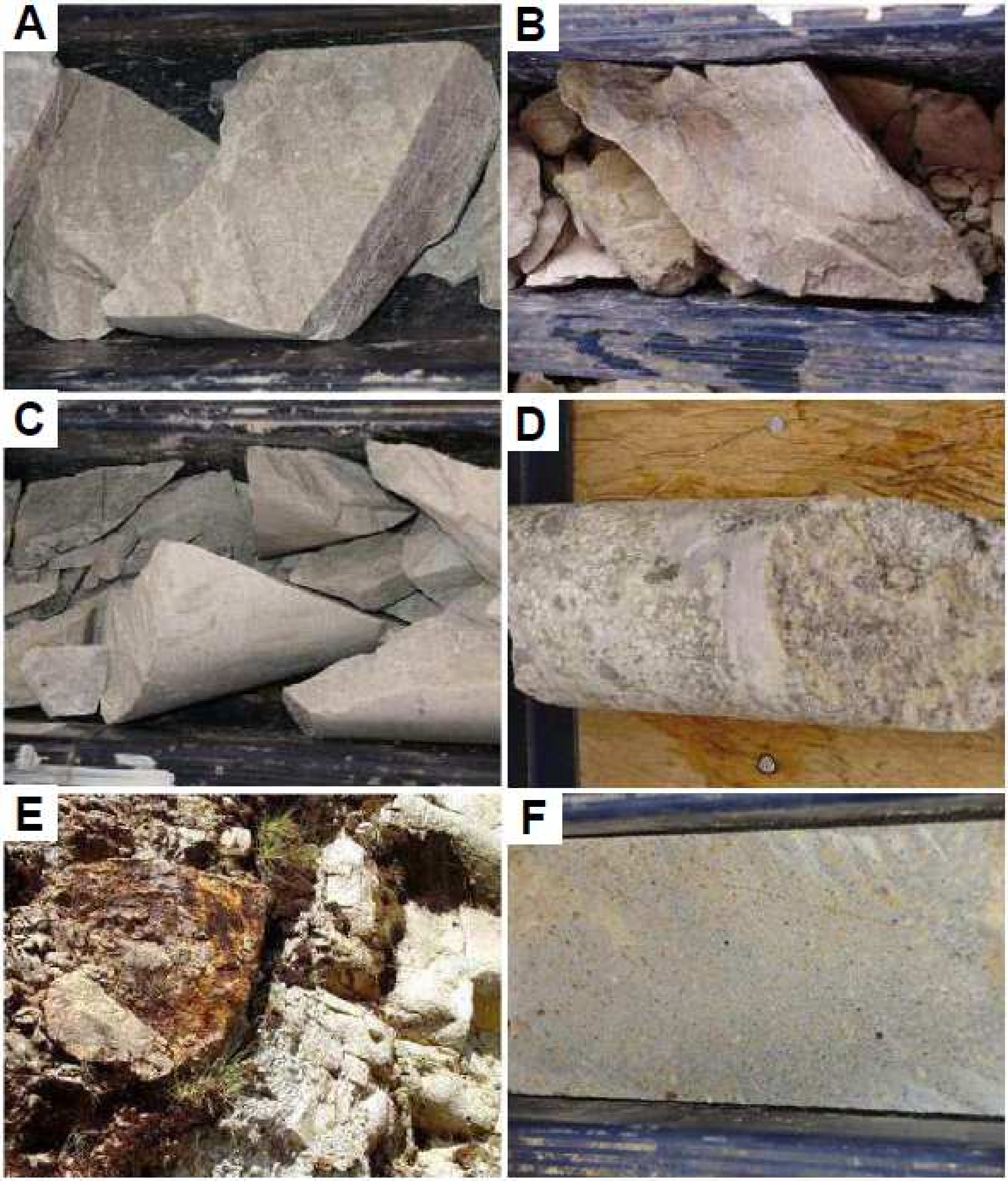


Figure 7-4 Main Rock Types at Antilla

A: Quartzite with bedding laminations at millimetre-scale
B: Delicate bedding features are offset along fractures
C: Fine-grained Lutite
D: Altered Main Porphyry with sulphide mineralization
E: Altered sedimentary rock (left) in contact with Main Porphyry (field of view is 4 metres wide)
F: Quartz aphyric Late Porphyry

7.4.1 Mineralization Style

The most economically significant mineralization encountered to date on the property is fracture-controlled and disseminated chalcocite. The chalcocite occurs as:

- Sooty or scaly coatings 1 millimetre wide, filled to partially open fractures
- Sooty coatings on rock fragments and rock flour encountered in intense fracture or fault zones over widths of 1 to 10 metres
- Selvages on sub-centimeter width quartz veinlets
- Occasionally as disseminated grains or coating disseminated grains of primary chalcopyrite in zones of more intense fracturing and silicification (Figure 7-5)

Chalcocite is enriched in the secondary sulphide enrichment zone.

Molybdenite occurs in fine fractures and as grains within sub-centimetre wide quartz veinlets in the primary sulphide, secondary sulphide and Main Porphyry.

Chalcopyrite occurs as disseminated grains and surface coatings along fractures and within quartz veinlets. Disseminated grains were also observed. Chalcopyrite in concentrations of up to 1% occurs in the Main Porphyry and in primary hypogene sulphide zones.

7.4.2 Trace Elements Associated with Mineralization

Copper grade increases three-fold from the primary sulphide zone to the secondary sulphide zone.
The leached zone has copper grades approximately one third of those from the primary sulphide zone and an order of magnitude less than the secondary sulphide zone. The genetic model involving the removal of copper from primary mineralization in what is now the leached zone and re-deposition as chalcocite in the secondary sulphide zone is well supported, given the distribution of copper grades among the mineralization zones. The Main Porphyry is weakly mineralized with copper, and the Late Porphyry contains little or no copper.

Molybdenum grade does not vary significantly between the primary sulphide, secondary sulphide, and leached zones, demonstrating the relative immobility of molybdenum in molybdenite during supergene processes. The highest concentrations of molybdenum occur in the Main Porphyry, a characteristic which is common to other porphyry and skarn deposits in the region.

In general, gold, silver, zinc, and lead concentrations are very low in all mineralization types. These metals do not show significant enrichment or depletion trends between the primary, secondary, and leached zones, and are not especially enriched or depleted in either of the porphyries.

7.4.3 Hydrothermal Alteration Associated with Mineralization

Hydrothermal alteration is restricted to the development of secondary sericite, biotite, and quartz, suggesting that the relatively inert quartzite and low water to rock ration of alteration result in the subtle alteration observed at Antilla. Unaltered quartzite lacks significant quantities of primary aluminosilicates to alter to large quantities of sericite, chlorite, biotite, and clay typical of potassic, phyllic, propylitic, and advanced argillic alteration zones common in other porphyry zones (Figure 7-6).

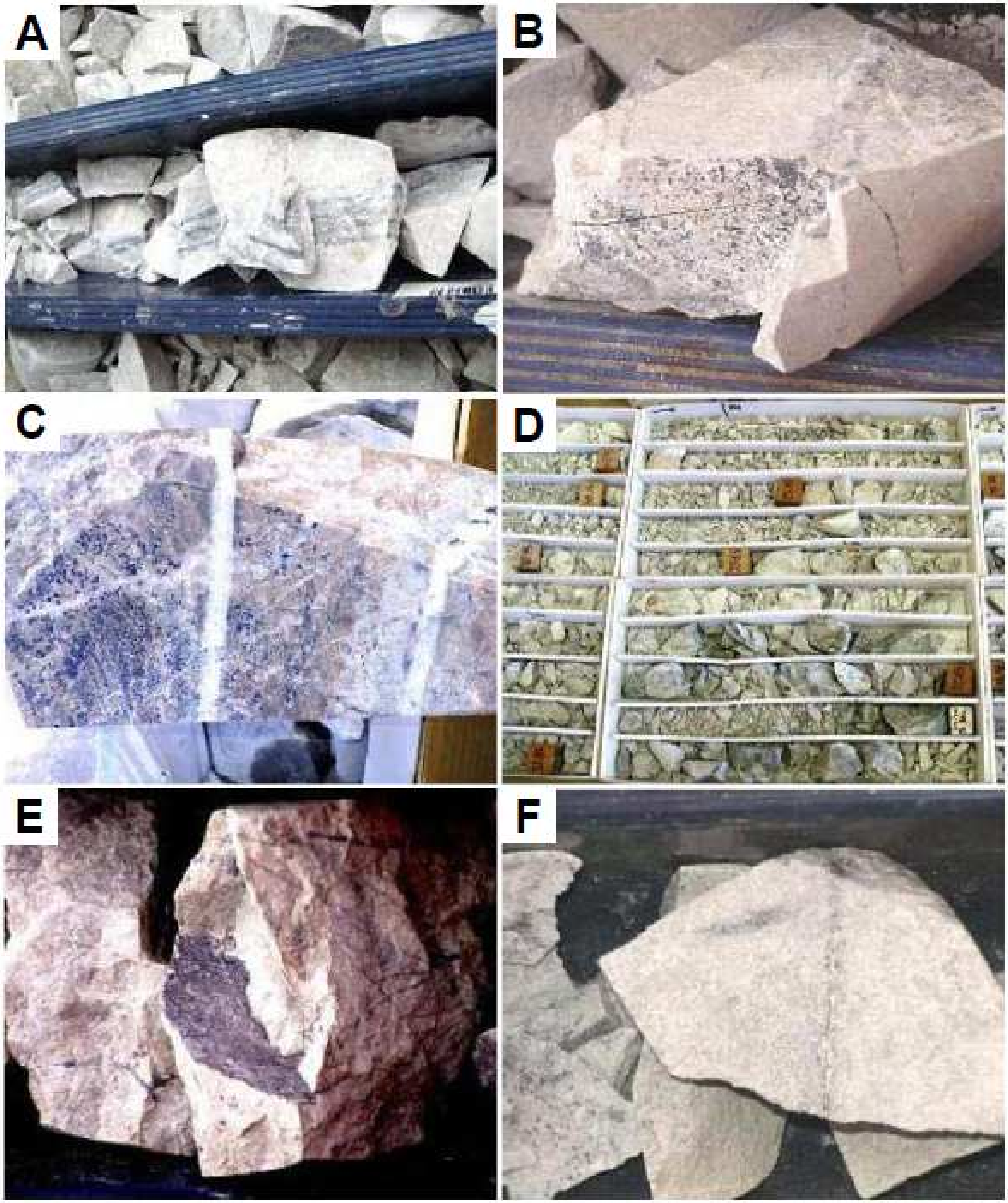


Figure 7-5 Mineralization at Antilla

A: Chalcocite and quartz-filled fractures in quartzite
B: Chalcocite coating a fine late fracture in weakly altered quartzite
C: Chalcocite on fractures surface and in quartz veins
D: Intense fracturing and sooty chalcocite mineralization associated with faulting
E: Molybdenite mineralization on fine fractures
F: Primary chalcopyrite mineralization in fractures below the secondary sulphide zone

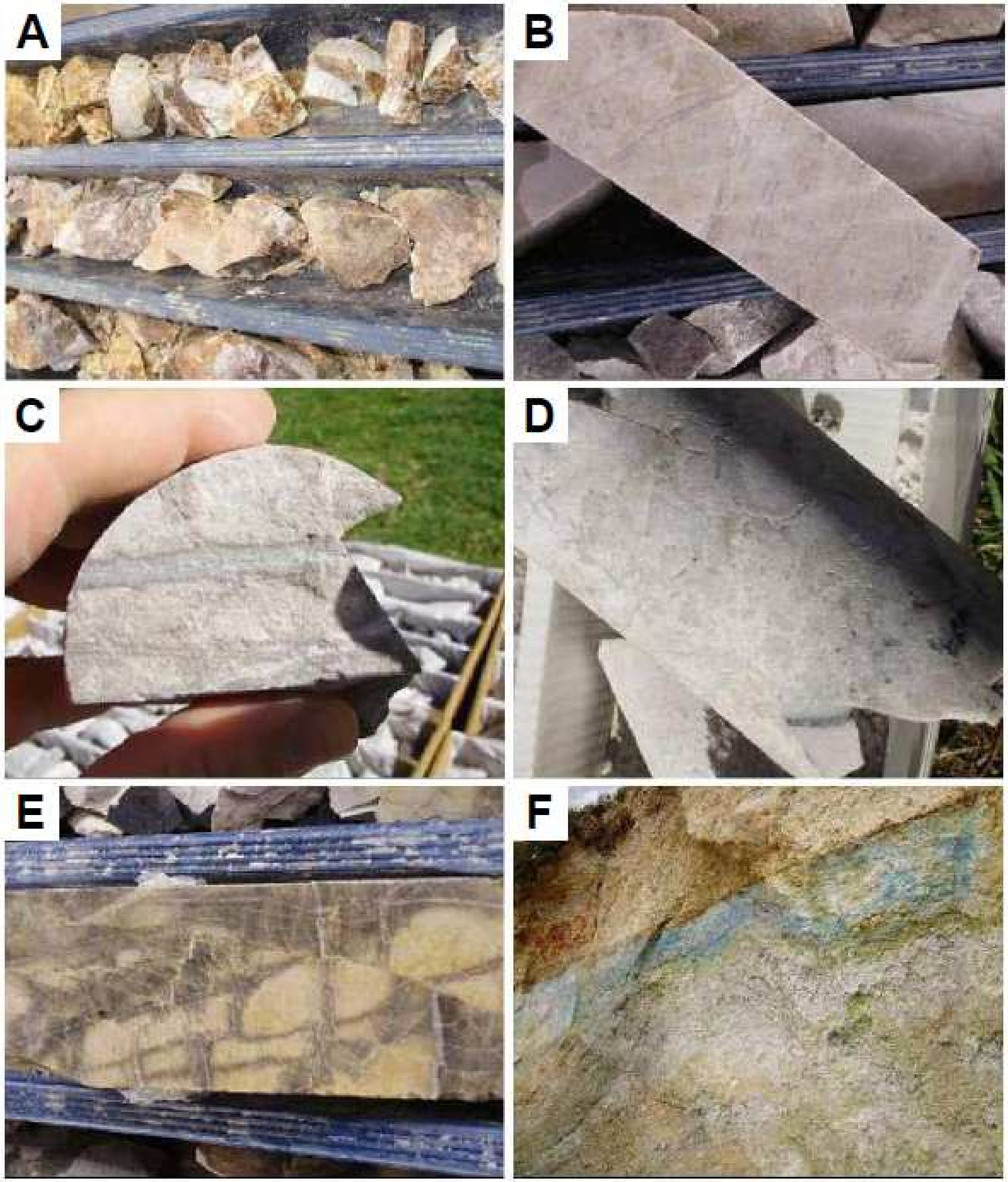


Figure 7-6 Hydrothermal Alteration

A: Blocky, oxidized quartzite from the leached cap
B: Weak biotite, sericite, and silica alteration and quartz veining of quartzite
C: Quartz veins with silicified margins and fine primary sulphides at the centre
D: Quartz breccia with quartz matrix
E: Patchy textured hornfels metamorphism of arenite
F: Remobilized or exotic-style copper oxide mineralization in overburden at the bottom of the slope that hosts the Antilla deposits

7.4.4 Structural Controls on Mineralization

Due to difficulties of stratigraphic correlation within the relatively monotonous quartzite and arenite of the Soraya Formation, a detailed understanding of the structural geology of the Antilla deposit is still under development. However, current genetic interpretations for the Antilla deposit place an emphasis on structural features at regional and local scale as mineralization controls.

The Antilla sulphide deposit occurs along the regional Mollobamba thrust fault in the southwestern part of the Andahuaylas-Yauri Belt (Figure 7-1). Two important regional-scale reverse faults are associated with the Mollobamba fault, the north-east-trending Piste fault, west of the deposit, and the east-trending Matara fault south of the deposit (Lee et al. 2007). These regional scale faults are interpreted to control the emplacement of the Main Porphyry, responsible for the hypogene mineralization, and the Late Porphyry, which cuts the mineralization. Intrusive rocks are interpreted to be located in zones of weakness caused by the intersection of faults in the case of the Main Porphyry, and along extensional or normal faults in the case of the Late Porphyry.

At deposit scale, fault or fracture zones containing relatively high-grade chalcocite mineralization have been intersected in core boreholes. Secondary sulphide mineralization is interpreted to be focused along fault zones that gave access to primary mineralization by meteoric fluids. The fine centimetre- to millimetre-scale fractures that host chalcocite mineralization also tend to increase in frequency near wider property-scale faults.

7.4.5 Zonation of Mineralization

The main mineralization types or zones are similar to many other porphyry deposits. The zones found at Antilla are primary sulphide, secondary sulphide, and oxide in the leached cap overlying the deposit. The secondary sulphide zone forms a relatively continuous, tabular chalcocite-enriched blanket that generally ranges from 60 to 120 metres thick. Borehole ANT-36-08 intersected a secondary sulphide zone 243 metres thick before encountering primary sulphide-style mineralization at 278 metres. The average thickness of the secondary sulphide zone is 92 metres.

The secondary sulphide zone is overlain by a leached cap that has an average thickness of 55 metres and generally ranges from 0 to 75 metres thick. The leach cap appears to thicken to the north and to the west where borehole ANT-64-08 encountered leached cap to a depth of 274 metres. It is interpreted that much of the leached cap overlying the main and southeastern portion of the secondary sulphides has been eroded bringing the secondary sulphide mineralization nearly to surface in some locations.

The tabular secondary sulphide and leached cap zones are underlain by lower-grade primary sulphide mineralization. The depth extent of the primary sulphide mineralization is not known as it has only been tested by five or six boreholes.

Main Porphyry is weakly mineralized and is known to flank the primary and secondary sulphides and oxide zone to the east and west and at the northwest corner. Hornfels alteration, which may indicate proximity to another undiscovered porphyry body, has been encountered in the deepest boreholes from the 2008 drilling program. It is possible that a significant volume of Main Porphyry occurs below the primary and secondary sulphides with the primary and secondary zones occurring in sediments which remain as a roof pendant to a large intrusive body. Conclusive evidence of this interpretation has not been found.

The Late Porphyry occurs as barren dikes cutting mineralization. Mineralization domains have been divided according to the parameters listed in Table 7-1.

A discontinuous veneer of gravel, sand, talus, and colluvium overlies the deposit. Overburden ranges in thickness from 0 to 53 metres, averaging 12 metres. In addition to the mineralization zones, a very small zone of weak exotic-type or remobilized copper oxide mineralization has been found in overburden exposed in a road cut at the bottom of the hill slope, overlying the secondary sulphide blanket.

Table 7-1 Mineralization Domains

Zone	Name	Alteration	Cu Grades	Mo Grades	Characteristics
1	Primary Sulphide	Silicification, biotization, sericitization and hornfels metamorphism	Average = 0.12% Max = 2.00%	Average = 0.01% Max = 0.80%	Absence of chalcocite, minor chalcopyrite, pyrite in veins and fractures; soluble copper <10%
2	Secondary Sulphide	Silicification, sericitization and biotitization	Average = 0.37% Max = 4.42%	Average = 0.01% Max = 0.38%	Presence of chalcocite on fractures, soluble copper is >10% of total copper
3	Oxide/Leached Zone	Limonite staining, bleaching	Average = 0.04% Max = 2.00%	Average = 0.08% Max = 0.26%	Lack of sulphides, limonite on fracture surfaces
4	Main Porphyry	Silicification, sericitization, biotitization	Average = 0.08% Max = 0.59%	Average = 0.01% Max = 0.17%	Quartz porphyroblasts, minor sulphide mineralization
5	Late Porphyry	None	Average = 0.04%	Average = 0.00%	Quartz aphric intrusive, no mineralization

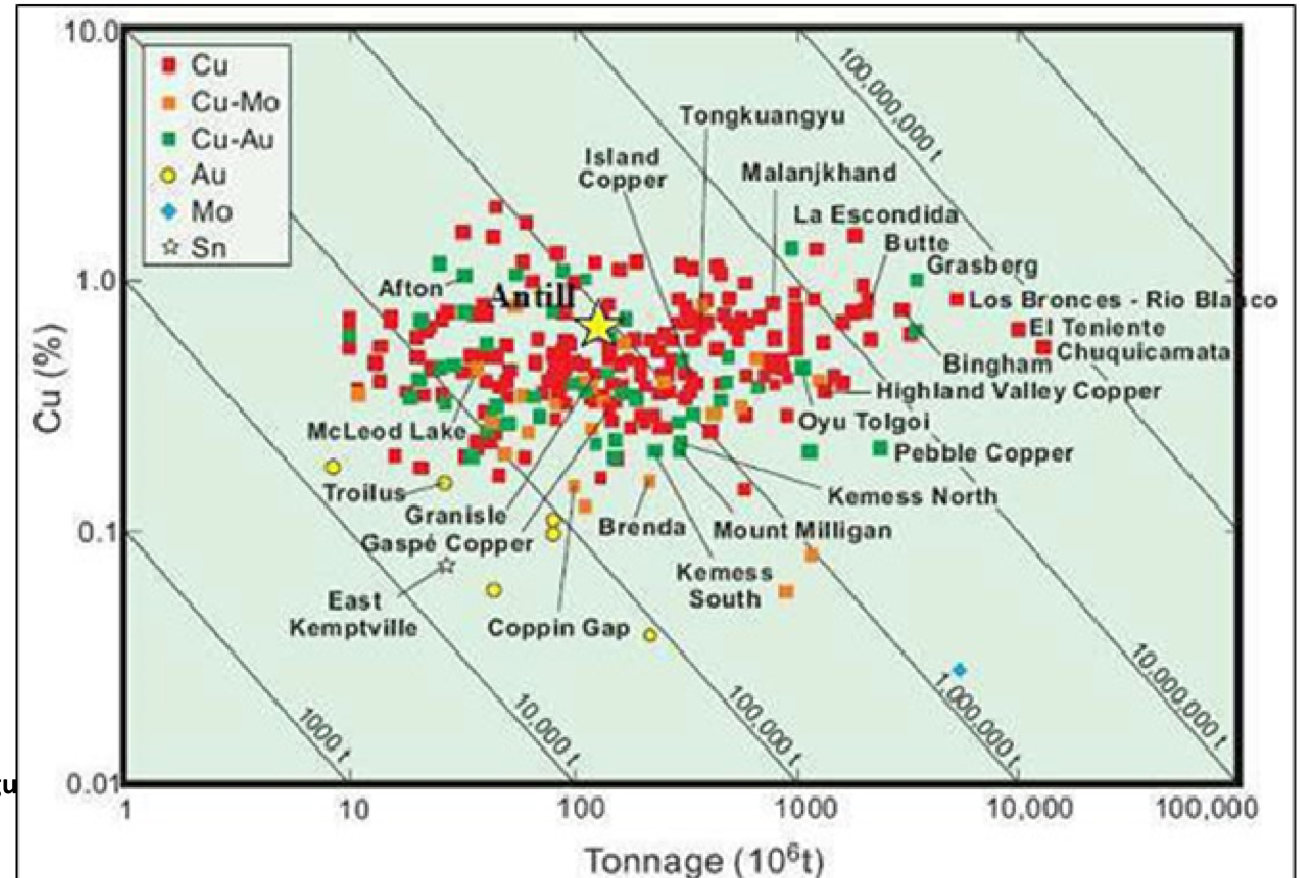
8 Deposit Types

The mineralization identified to date on the property is consistent with a supergene enrichment blanket hosted in a sandstone-quartzite package of Soraya Formation associated with an Andean-type copper-molybdenum porphyry system.

Common features of copper-molybdenum porphyries include stockworks of quartz veinlets, quartz veins, closely spaced fractures and breccias containing pyrite and chalcopyrite with lesser molybdenite, bornite, and magnetite. These features occur in large zones of bulk-mineable mineralization in or adjoining porphyritic intrusions and related breccia bodies. Disseminated sulphide minerals are present, generally in subordinate amounts. The mineralization is typically spatially, temporally, and genetically associated with hydrothermal alteration of the host rock intrusions and wallrocks. Andean-type examples include Antapaccay, Las Bambas, Los Chancas, and Trapiche.

Mineralization at Antilla consists of a tabular body of fracture-controlled and disseminated chalcocite and chalcopyrite with minor molybdenite-coated fractures overlain by a barren, leached zone of variable thickness. Associated with the chalcocite mineralization are silicification, sericitization, biotitization, and chloritization of arenite, quartzite, and sandstone. The strongest chalcocite mineralization is associated with brittle faults. Low-grade disseminated chalcopyrite, bornite, and molybdenite mineralization occurs below the chalcocite mineralization. Altered, weakly-mineralized, porphyritic felsic intrusive rocks are associated with the mineralization. The general geometric and mineralogical characteristics of the deposit are consistent with a supergene enrichment blanket hosted in a Sandstone-Quartzite package associated with an Andean-type copper-molybdenum porphyry system.

Porphyry deposits are defined in general as being large, low- to medium-grade deposits in which primary ore minerals are dominantly structurally controlled and which are spatially and genetically related to felsic to intermediate porphyritic intrusions (Figure 10; Sinclair 2007). Porphyry deposits can contain significant concentrations of one or more of copper, gold, molybdenum, and can also contain silver, tin, tungsten, and rare earth elements. Skarn deposits also occur in the region and are associated with porphyritic intrusions, but the mineralization on Antilla lacks the intense fluid- dominated calc-silicate alteration, intense iron metasomatism, and reactive carbonate host rocks of skarn systems (Figure 8-1).


Cu
Cu-Mo
Cu-Au
Au
Mo
Sn
10.0
1.0
0.1
0.01
Cu (%)
1
10
100
1000
10,000
100,000
Tonnage (10^6t)
1000 t
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100,000 t
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Tongkuangyu
Island Copper
Malanjkhand
La Escondida
Butte
Grasberg
Los Bronces - Rio Bla co
El Teniente
Chuquicamata
Bingham
Highland Valley Copper
Oyu Tolgoi
Pebble Copper
Kemess North
Mount Milligan
Kemess South
Brenda
Coppin Gap
Afton
Antill
McLeod Lake
Troilus
Granisle
Gaspé Copper
East Kemptville

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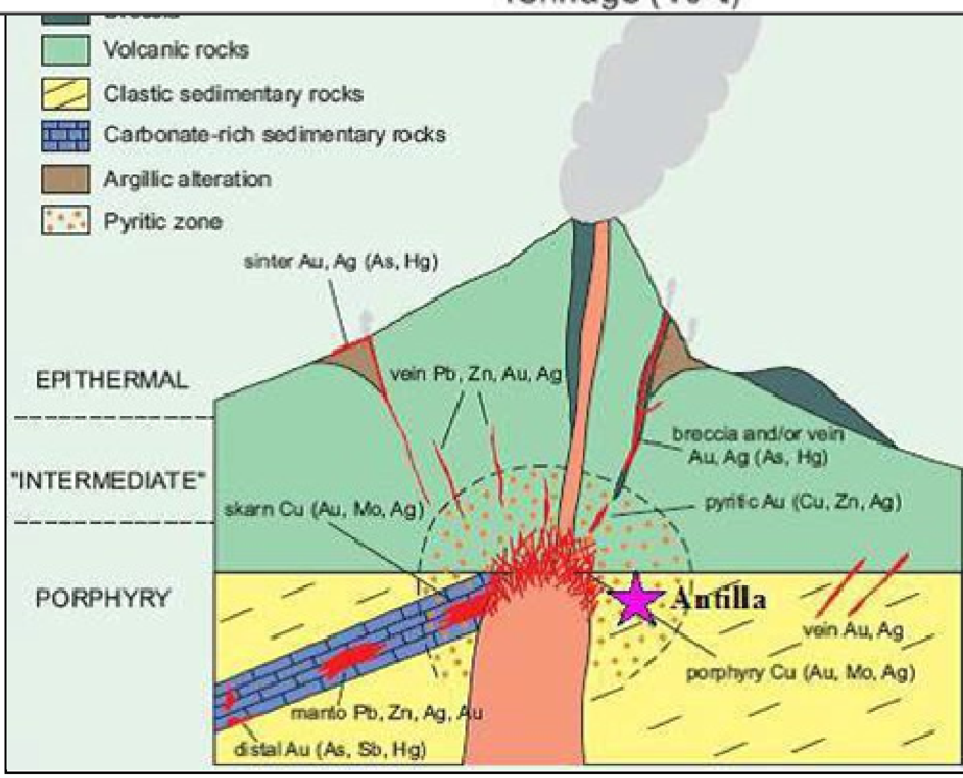

Volcanic rocks
Clastic sedimentary rocks
Carbonate-rich sedimentary rocks
Argillic alteration
Pyritic zone
sinter Au, Ag (As, Hg)
EPITHERMAL
vein Pb, Zn, Au, Ag
breccia and/or vein Au, Ag (As, Hg)
"INTERMEDIATE"
skarn Cu (Au, Mo, Ag)
pyritic Au (Cu, Zn, Ag)
PORPHYRY
Antilla
vein Au, Ag
porphyry Cu (Au, Mo, Ag)
manto Pb, Zn, Ag, Au
distal Au (As, Sb, Hg)

Figure 8-2 Geological Environment of Porphyry Copper Deposits (Wright 2009)

The Antilla deposit has a number of characteristics which are not common in other porphyry systems in the region or in typical porphyry models (such as those discussed by Lowel and Guilbert, 1970; Kirkham and Sinclair, 1975):

- Alteration at Antilla is limited to weak sericitic or phyllic, weak chlorite or propylitic, and weak silicification. The well-defined potassic, phyllic, propylitic, and argillic alteration assemblages typical of porphyry copper deposits are not well developed at Antilla. The relatively weak alteration characteristics are interpreted to be a result of the lack of aluminous mineral phases of the quartzite hosting the mineralization.
- The hypogene or primary sulphide mineralization zone at Antilla contains low amounts of copper minerals. Assays of the primary sulphide zone at Antilla yielded approximately 0.12% copper and 0.009% molybdenum. Based on current drilling information, the Main Porphyry contains approximately 0.08% copper on average. No higher-grade hypogene chalcopyrite mineralization has been encountered. To date, Panoro has not located a higher grade primary porphyry system with which it believes the Antilla mineralization is associated.
- Breccia pipes and hydrothermal breccias occur in the contact zone between the porphyry intrusion and the arenitic host rocks; copper grades between 0.1% and 2.0% have been obtained, especially in the North Block and in the area of the Chabuca target.

A characteristic of the Antilla deposit, and of the other deposits in the Andahuaylas-Yauri Belt, is the lack of typical oxide-style mineralization. Only minor chrysocolla, tenorite and malachite are found. Supergene mineralization consists entirely of secondary sulphides. Two interpretations have been made to explain the lack of oxide mineralization: the lack of pyrite in hypogene mineralization and subsequently the inability to generate sufficient acid to generate oxide mineralization during supergene mineralization, and the relative abundance of carbonate stratigraphy to neutralize acid during supergene enrichment. Carbonates are not locally important at Antilla, but a relatively low quantity of pyrite in hypogene mineralization may support the hypothesis that the low pH required to generate oxide mineralization during supergene enrichment was not attained

9 Exploration

The eastern portion of the property concentrates most of the exploration works done in surface since 2002 by Cordillera to 2013 by Panoro. The works were mapping, geophysics, geochemistry soils and rocks and drilling.

9.1 Geological Mapping

Geological mapping at a 1:5,000 scale has been completed on 4,000 hectares by Cordillera between 2002 and 2004 and was updated in 2008 by Panoro. Then between 2013 and 2014 the mapping was reviewed by Panoro adding more details. The outcroppings are reasonably good and exposures of the multiple Diorite to Monzodiorite porphyries intrusions cutting the Soraya Group sedimentary host rock. Road cuts provide additional exposure in areas that are covered by talus and quaternary gravel, sand, and silt.

Between 2014 and 2015, Panoro completed surface mapping of the western part of the property, where the El Piste target was identified as a result. Reconnaissance-scale mapping has been carried out on the remainder of the property at 1:5,000 scale. Panoro has recently completed more detailed geological mapping both around the resource area and on other exploration targets on the property.

In 2013 Panoro commissioned Seggistem S.R.L. Co. to perform a detailed ground- based topographic survey using differential GPS receivers with a base station for post processing of the data. The resulting topographic surface covers significant area and fit the entire data rise with the geologic mapping, geochemical sampling, drillholes, mineral resource estimation, and infrastructure design for this study.

9.2 Geochemistry

In 2002 and 2003 a systematic rock chips and soils sampling was carried out by Cordillera on a 100 by 50 metres grid. A total of 2,862 samples were taken, including 1,135 rock samples and 1,727 soil samples, including QA/QC samples.

Between 2013 and 2015 Panoro carried out systematic rock chips and litho-geochemical sampling primarily in the eastern and central portions of the property in order to define the geology, alteration, mineralization, and to assess a geologic potential of the project area. A summary of the samples collected by Panoro is presented in Table 9-1. Results of the sampling led to the identification of six anomalies – including Antilla (Figure 9-10).

Table 9-1 Summary of Panoro Sampling, 2008-2015

Year	Rock Chip Sample	Soils Sample	Lithogeoch Sample
2008	103		
2009	6		134
2013	691	11	
2014	858	475	23
2015	99		
Total	**1,757**	**486**	**157**

A total of 4,978 samples were produced in all Campaigns since 2003 to 2015, including 2850 Rock chip samples and 2,128 Soil samples in Table 9-2.

Table 9-2 Geochemistry Sampling by Exploratory Campaign

Company	Total Rock Chip	Total Soil	Total QA/QC	Total Samples
Cordillera	1093	1642	127	
Panoro	1757	486	154	
Total	**2850**	**2128**	**281**	**5259**

In Figure 9-1 note the higher copper grades are located around the pipe and hydrothermal breccia outcropping beside the Chabuca porphyry. This area has been never drilled before and there are surface evidences of the possible connection with the north block zone, where the sandstones dip to the south and toward the mineral resources area.

Also note that the copper anomalies into the PEA pit show not significant size, however the later drilling defined the secondary sulphide blanket continuity over an area of 1.4 x 0.95 km until 90m depth. It means the copper anomalies size in surface have not proportional relation with the real size and extension of the enrichment blanket extended over the sandstone layers. This behavior is observed also in the exploratory drillholes located between the targets of West Block I and II, where drillholes intersected enrichment mineralization in spite of were located outside of the high copper anomalies. See Figure 9-15.

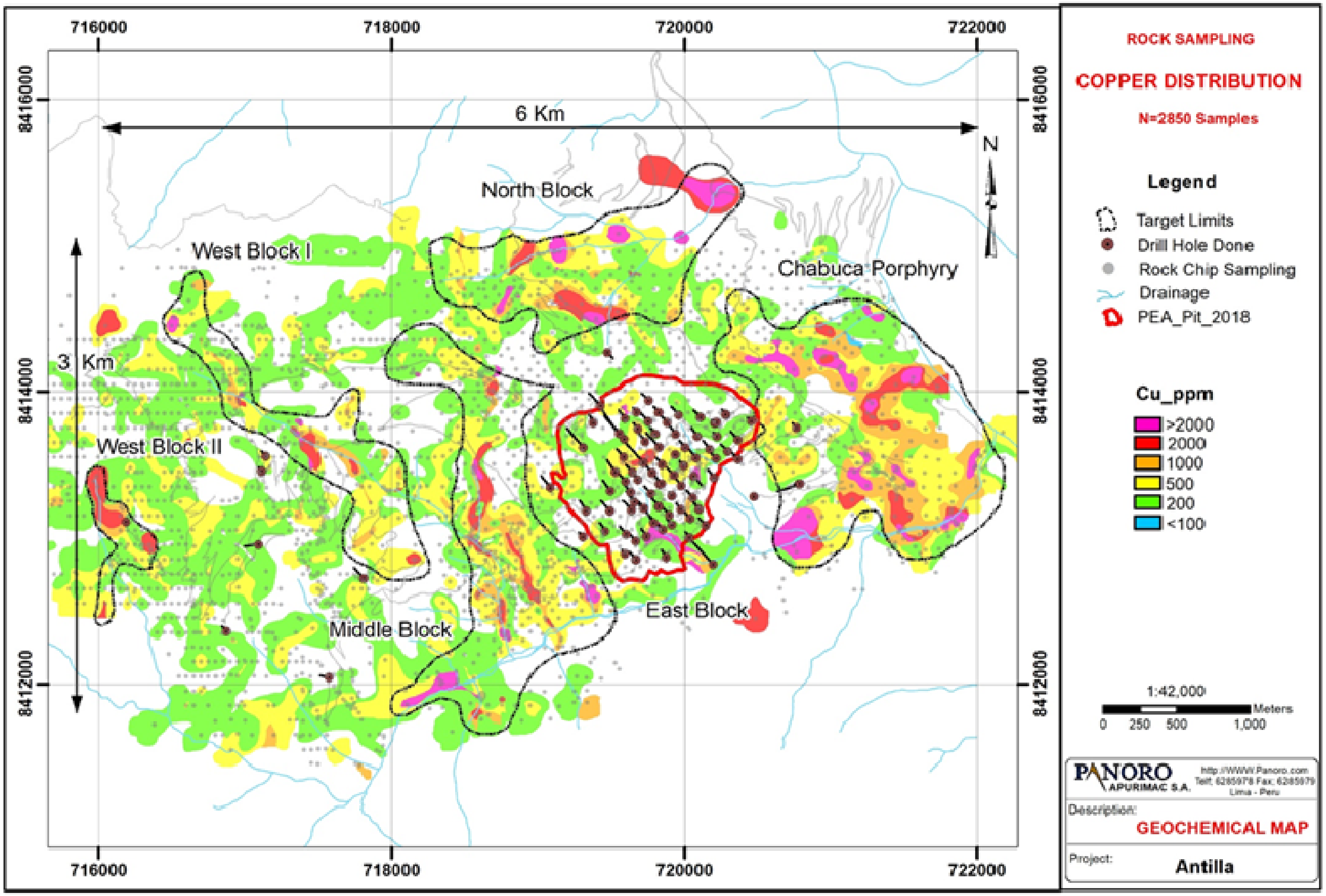


Figure 9-1
the group

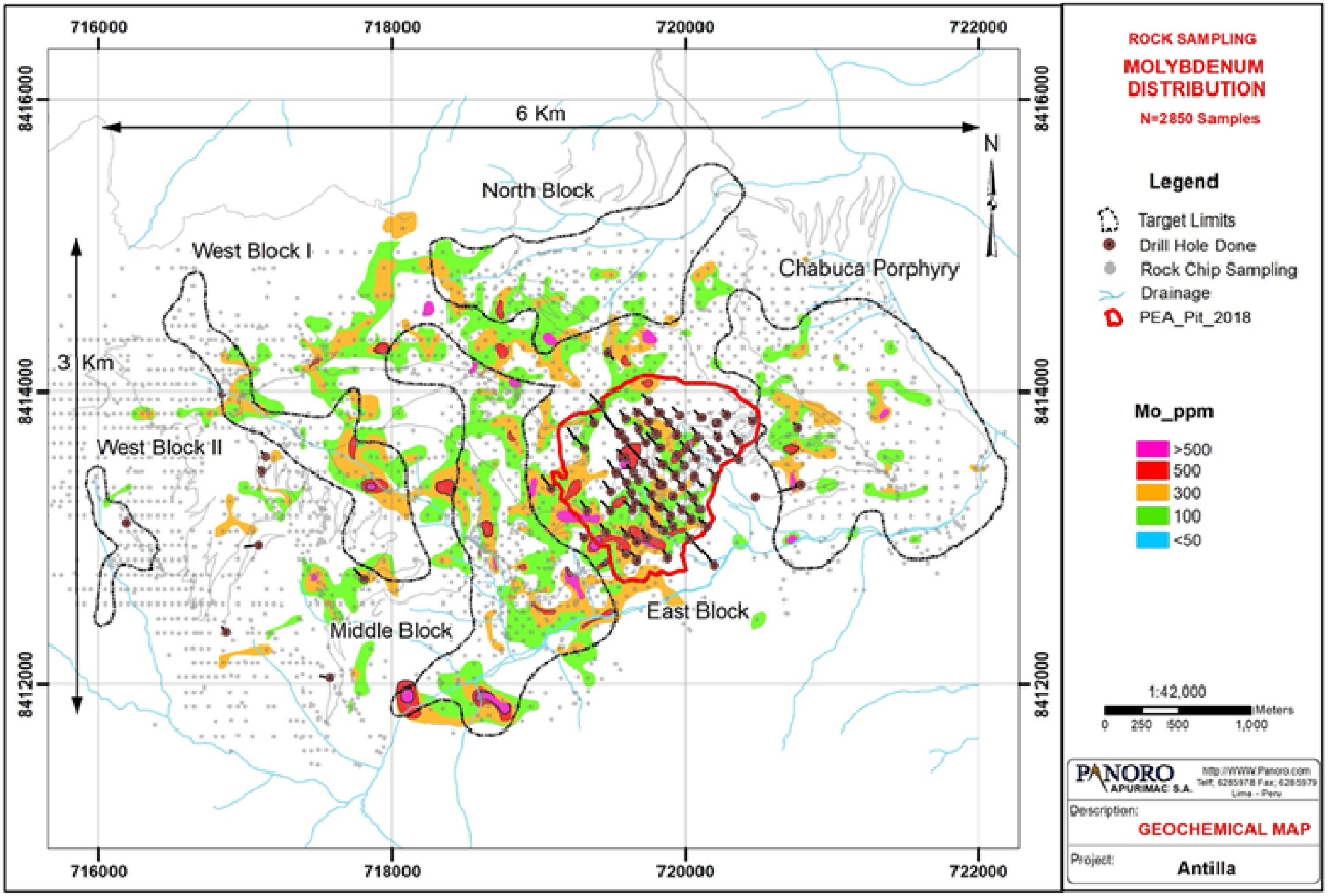


Figure 9-2
configure

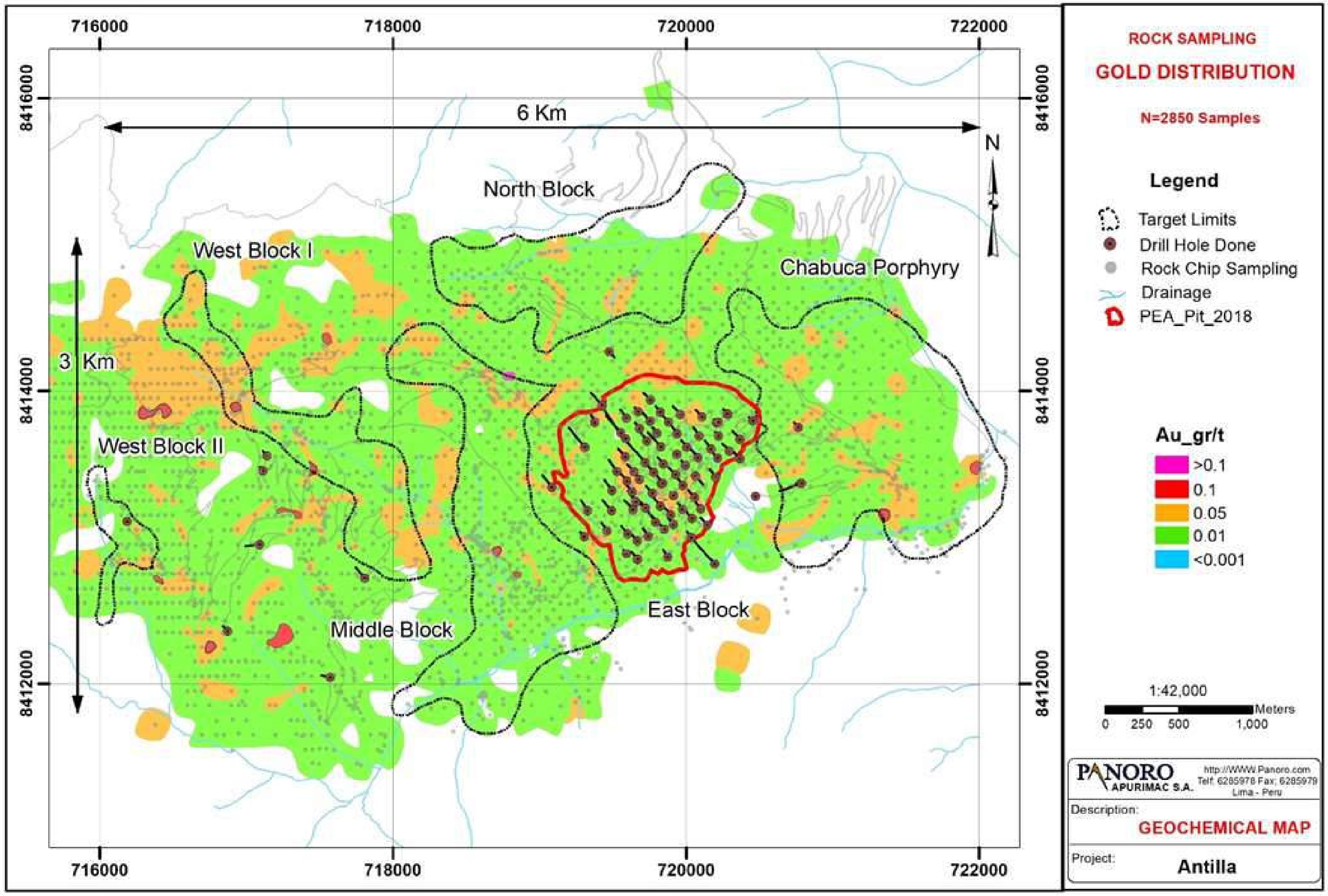


Figure 9-3
sandstone d

9.3 Geophysics

A 214.2-kilometers magnetometer survey and 43.6-kilometers IP (induced polarization and resistivity) survey was carried out by Cordillera in 2003. The survey was executed by Val d'Or Geophysics of Peru.

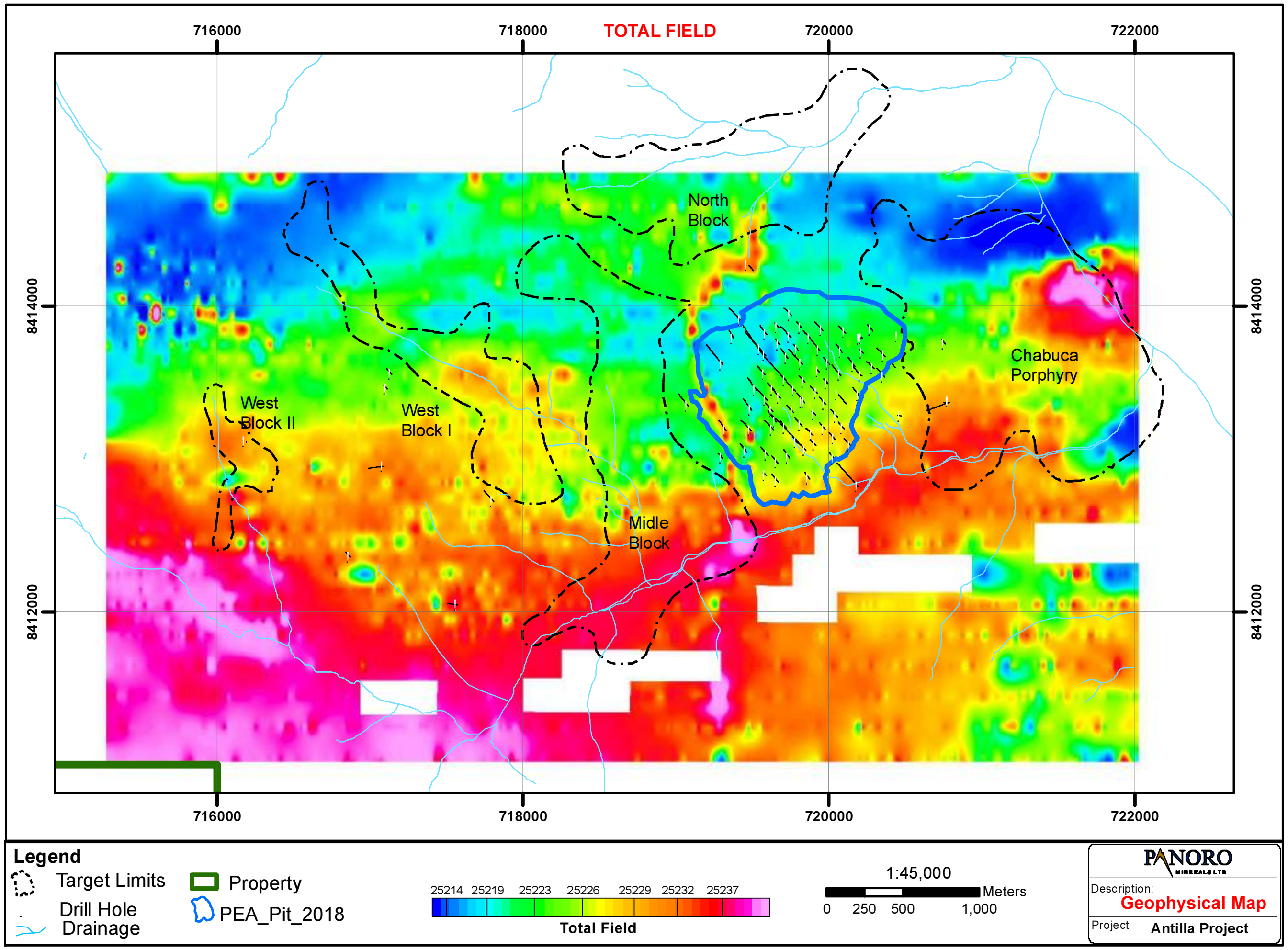
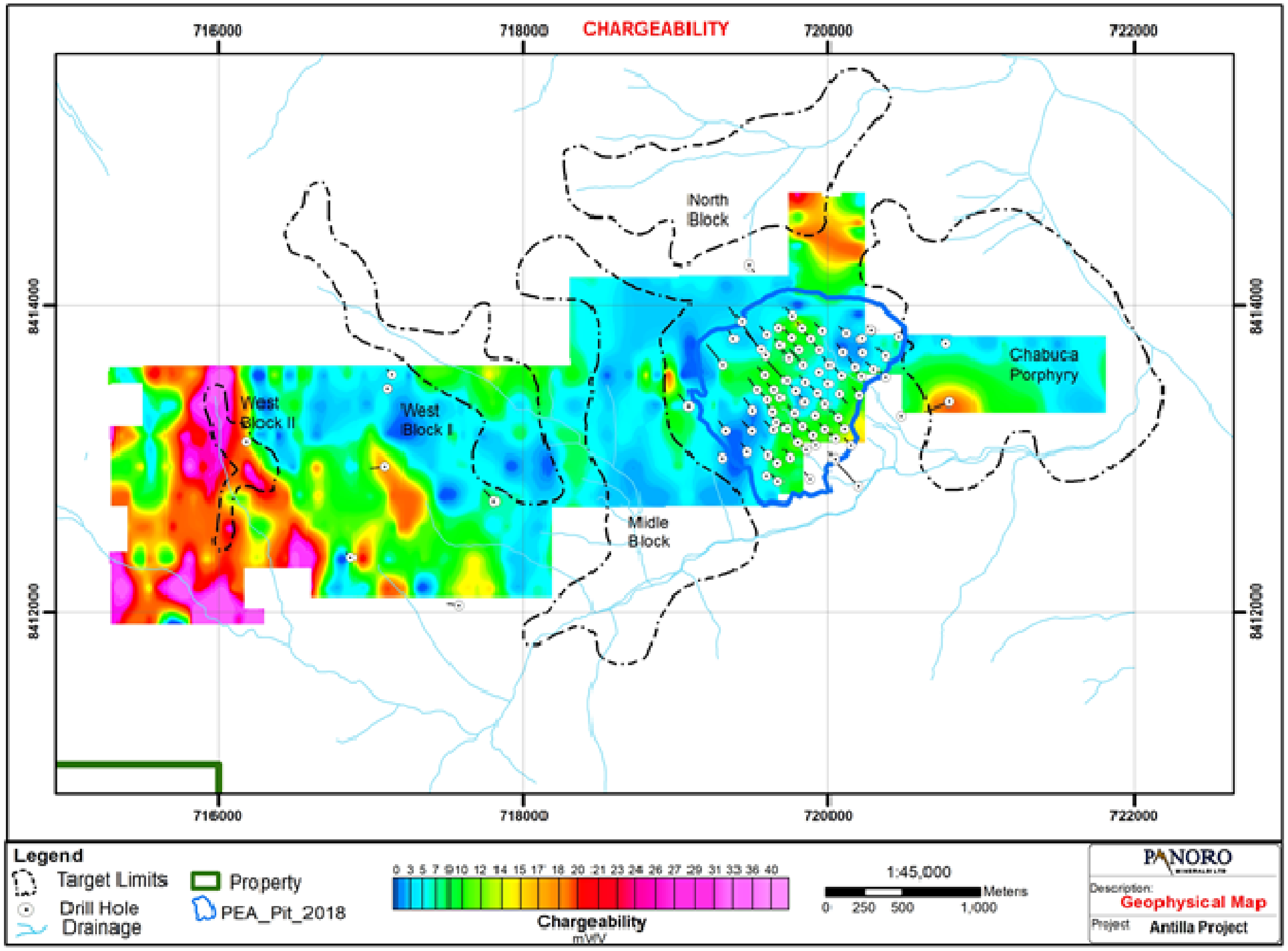


Figure 9-4 Showing Total Field Mag of ground Mag made by Cordillera (2004)


CHARGEABILITY
716000
718000
720000
722000
8414000
8412000
North Block
West Block II
West Block I
Midle Block
Chabuca Porphyry
Legend
Target Limits
Drill Hole
Drainage
Property
PEA_Pit_2018
0 3 5 7 9 10 12 14 15 17 18 20 21 23 24 26 27 29 31 33 36 40
Chargeability
mV/V
1:45,000
Meters
0 250 500 1,000
PANORO
Description:
Geophysical Map
Project
Antilla Project

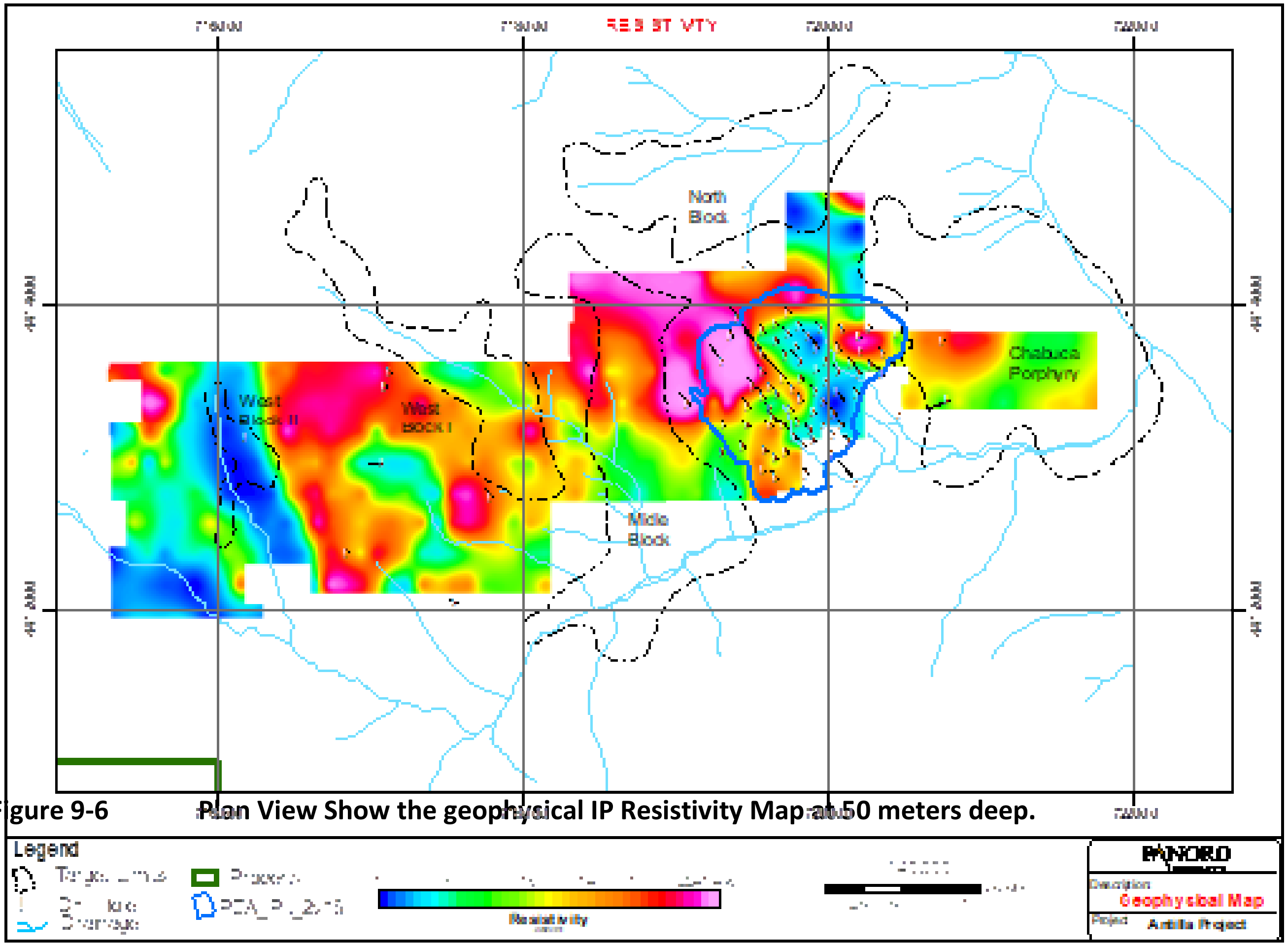


Figure 9-6 Plan View Show the geophysical IP Resistivity Map at 50 meters deep.

9.4 Exploration Model and Geological Potential

In October and November 2008, a petrographic study of 19 core samples and two hand samples was completed by Katherine Dunn of Salmon Arm, British Columbia. The study was of arkose, mudstone, quartzite, hornfels, and intrusion samples. The modal mineralogy and alteration mineralogy were reviewed and documented in a final report (Dunn 2008).

In September 2013, a Lithogeochemistry study was completed by Dr. Mirian Mamani, a consultant PhD. in Geochemistry and Mineral Deposits and Post-doctoral in Sediments Geochemistry in the University of Goettingen, Germany. The preliminary study classifies Antilla as a-magmatic type deposit, where the copper mineralization and mineral resources are hosted into the sedimentary rocks. The composition and grain size in the sandstones layers host the main concentration of Cu-Mo but the ion metals source is represented by the quartz monzonite porphyry. The study also suggests the ion metals were transported to the host rocks by acid magmas as dikes with high concentration of Th-U-K2O.

The sedimentary rocks composition resulted in arkose with high content of CO2 as the best metalogenic agents, with values of Th>4ppm and LOI>1%. The Litogeochemistry of the main & traces elements, and rare earths in the sediments rocks are controlled by the porosity and grain size of the sediments.

In summary, the Exploration Model in Antilla consider the quartzmonzonite porphyry as the source of the metals, transported outward by dikes and/or hydrothermal breccias and finally deposited/concentrated in tramps as arkose sandstones limited by un-permeable quartzites sediments. The lithologic control in specific sediment can be follow by kilometers as show by the extensive distribution of copper anomalies in an area of 3km x 6km. The structural control and folds also may represent other type of tramps. As a late metal concentration process, the vertical movements of the water table developed in Antilla thick secondary sulphides blankets under the surface through the leaching of the primary sulphides previously deposited.

Following this Model, in Antilla has been recognized until now three types of Geological Potential: 1) located next and below to the economic PEA Pit, 2) located 6km around the PEA Pit, and 3) located 10km to the west in El Piste zone. All of them represent high opportunities to grown the operation scale in a near future.

9.4.1 Potential Next to the PEA Pit

For the enrichment secondary sulphides there are 2 types of mineral resources already drilled having potential to be included in a future mine plan. In Figure 9-7, Figure 9-8, and Figure 9-9 the white blue area may need pit design optimization and the orange area may need more drilling definition.

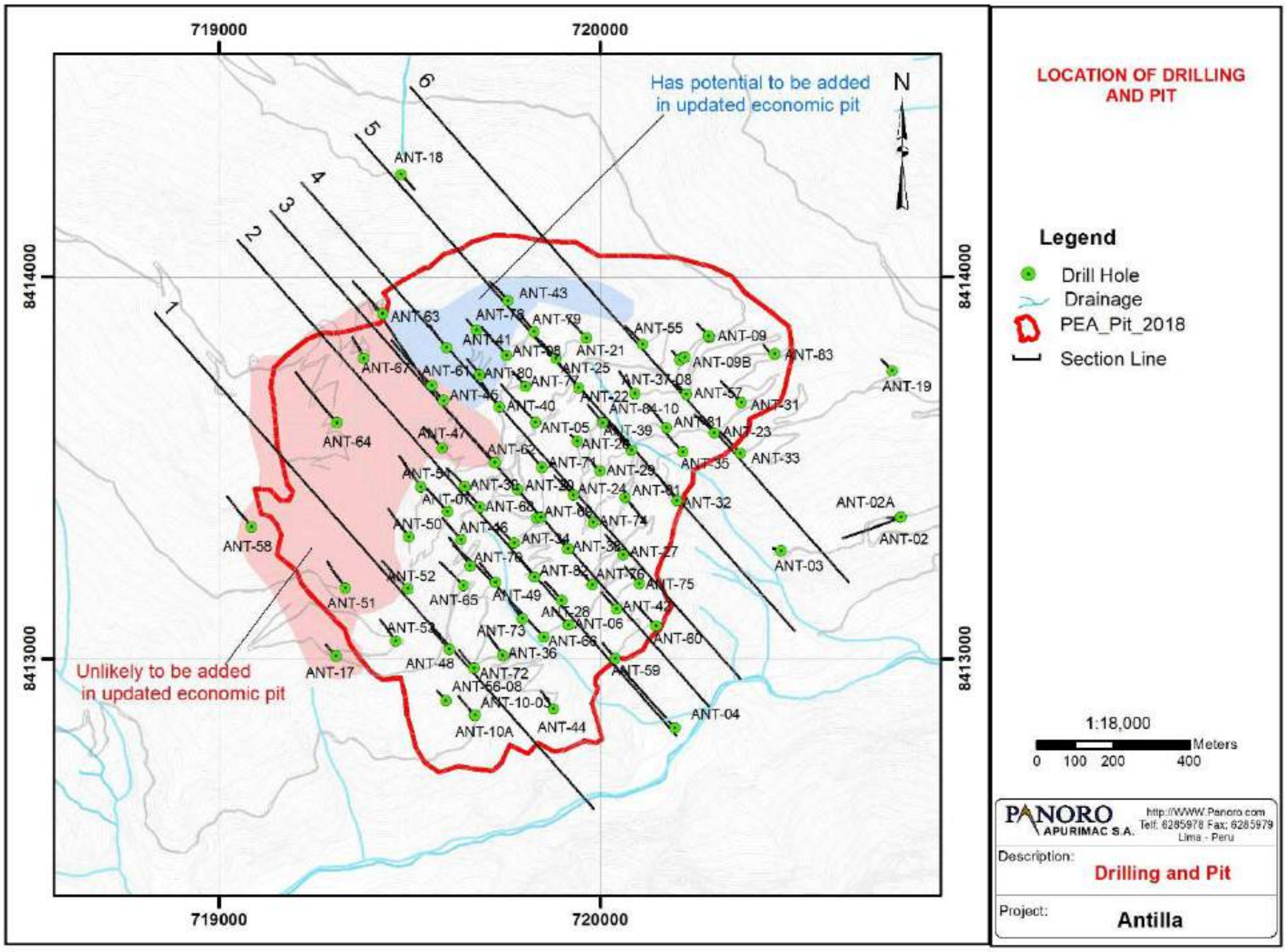
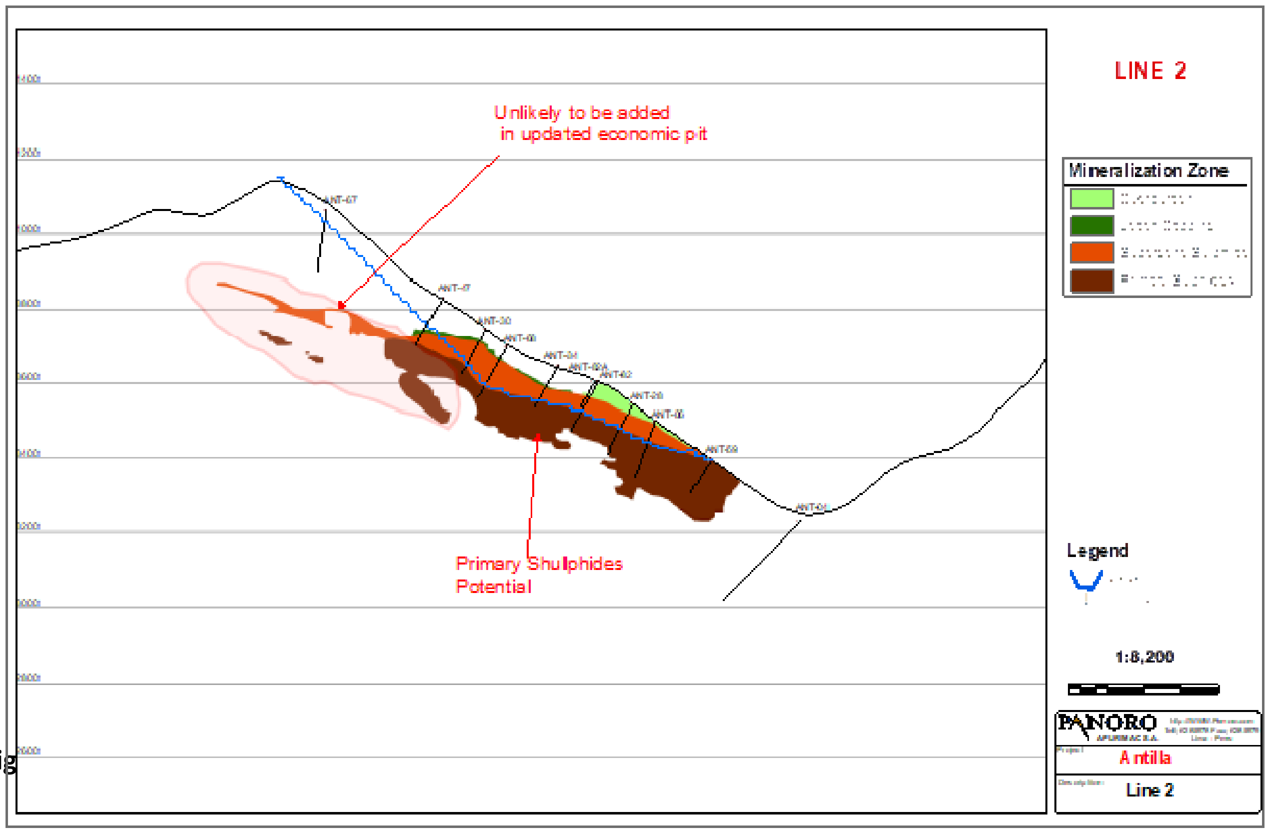


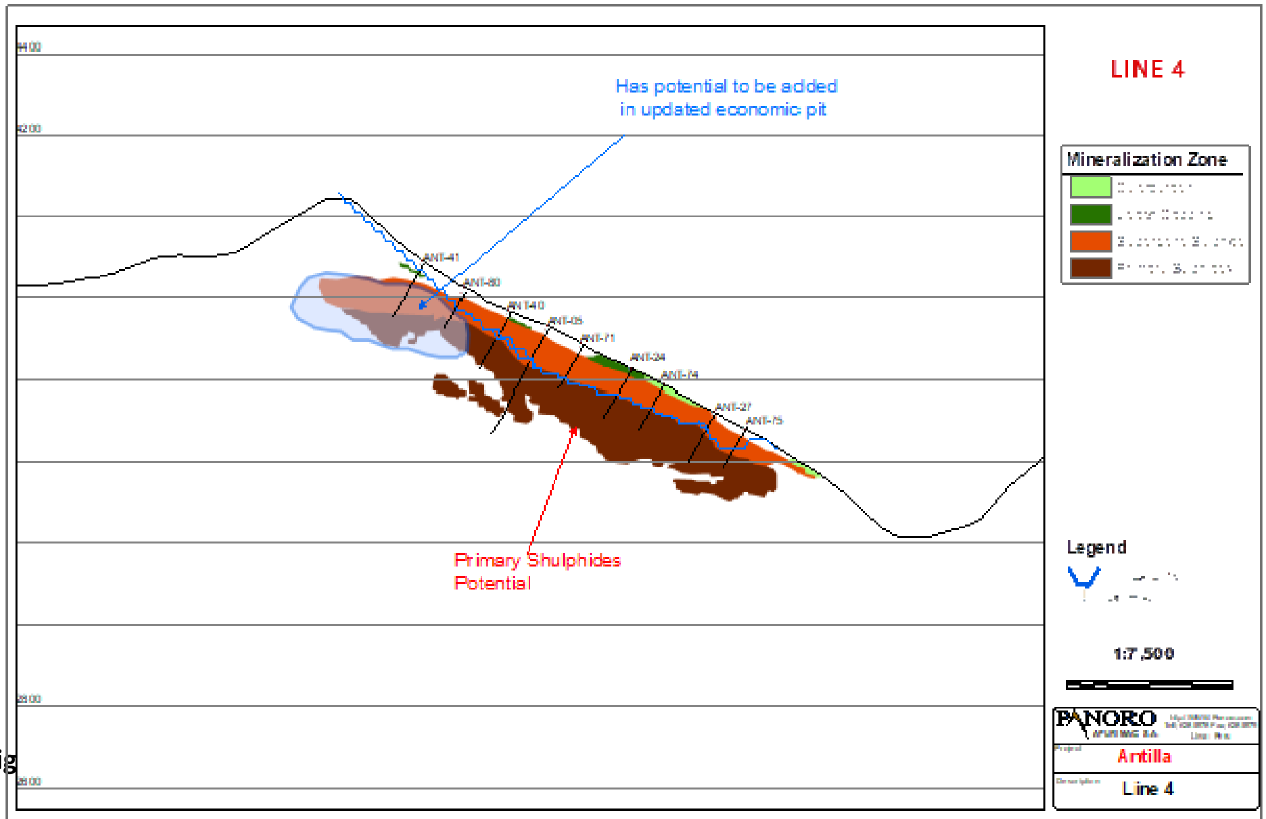
Figure 9-7 **Areas with upside targets next to the PEA Pit**

The PEA mine plan includes 60% of the secondary sulphides contained in the Mineral Resource. There are significant amounts remaining outside the PEA pit that may be utilized with a pit design optimization.

The primary sulphides blanket located immediately below the enrichment (secondary sulphides) blanket represents mineral potential next to the PEA Pit. The mine plan recovers less than 1% of the primary sulphides in the Resource. This potential may represent opportunity for an eventual conventional flotation process. There are some flotation testwork done on this material, but no exist yet leaching testwork. See location on Figure 9-8 and Figure 9-9.



Fig



Fig

The mine plan includes small percentages of the Overburden/Cover material and Leach cap material contained in the Mineral Resource. There is a significant amount outside the pit with mixed mineralogy. No metallurgic leaching testwork has been done on this material.

9.4.2 Potential around the PEA Pit

This potential is related to the five groups of copper anomalies recognized with the rock chip geochemistry survey named: Chabuca, Middle Block, North Block, West Block I and West Block II. The Figure 9-10 and Figure 9-11 shows most of the anomalies areas overlapping the sandstone domain.

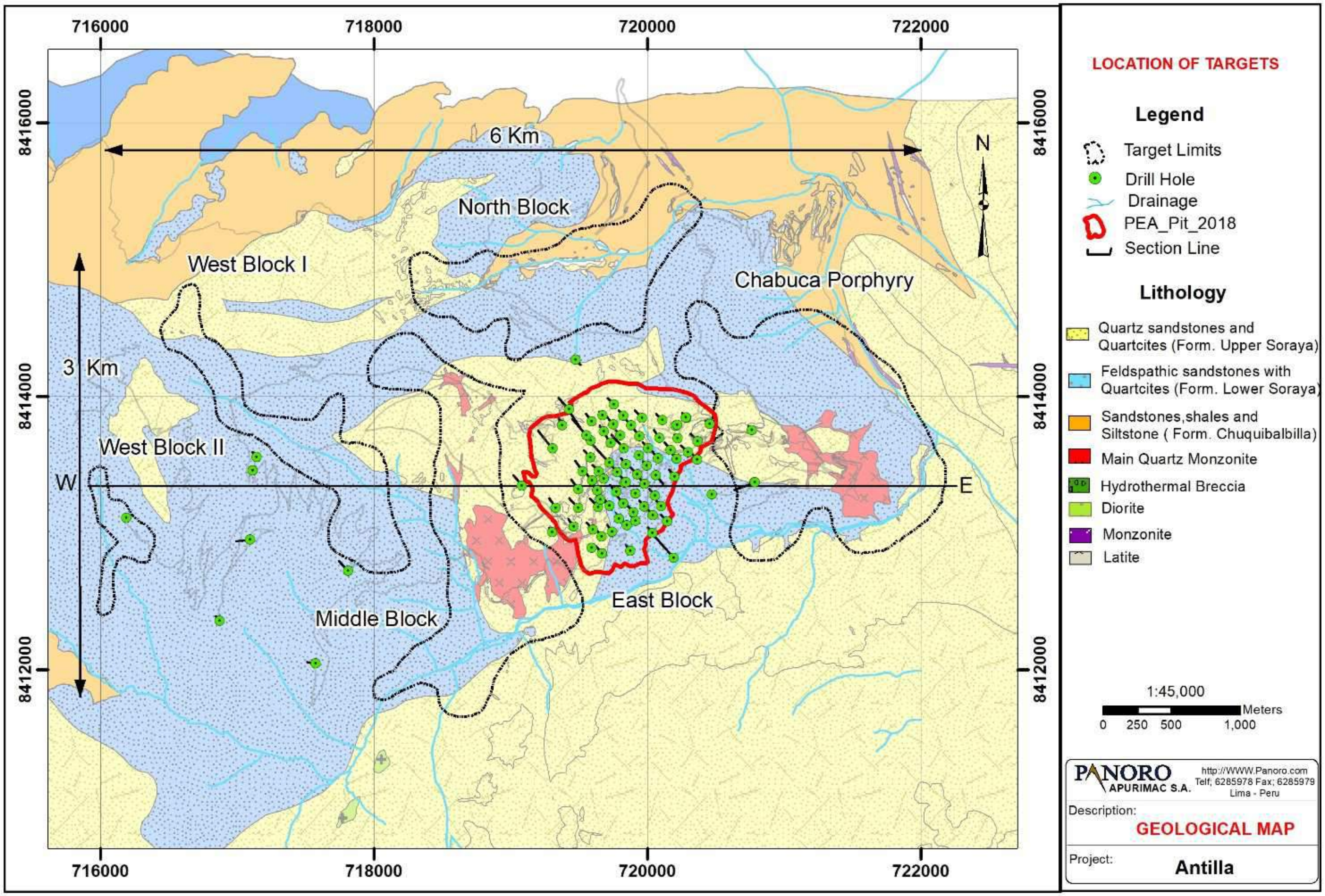


Figure 9-10 **Local geologic map showing the contours of the5 exploration targets around the PEA Pit**

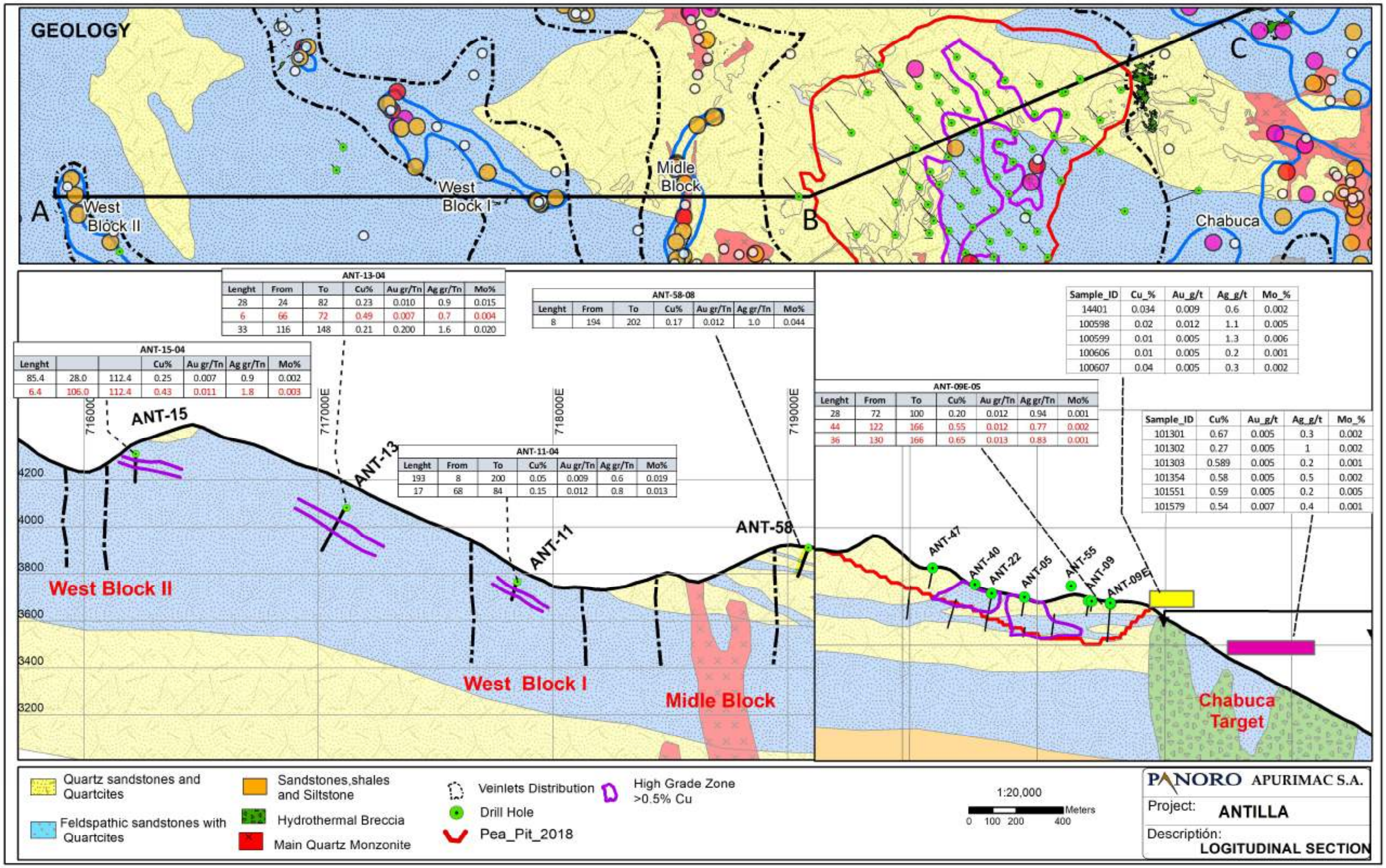


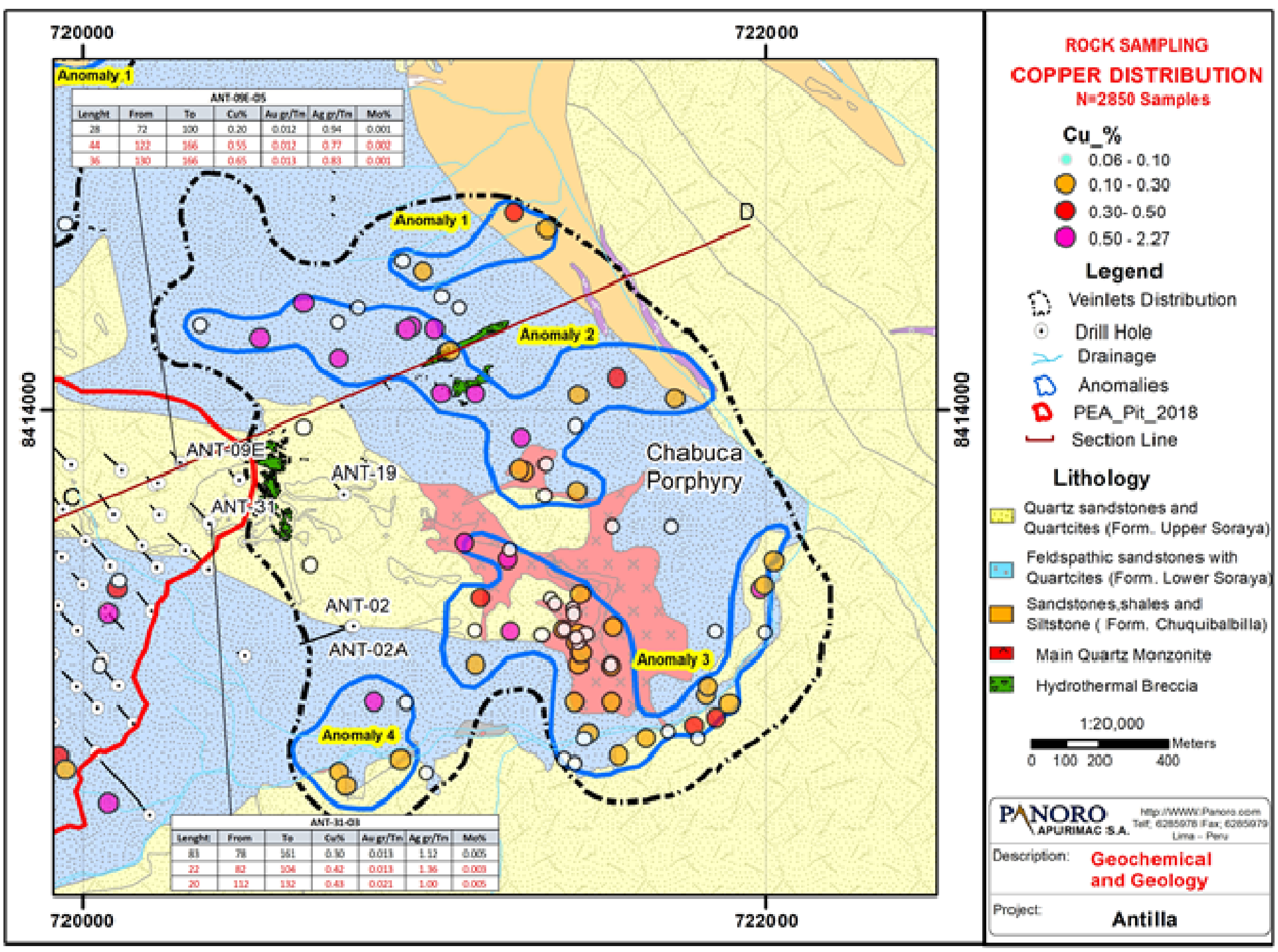
Figure 9-11 Longitudinal Section almost in East-West direction looking to the north, showing explorative drillholes and exploration targets around the PEA Pit

The Chabuca Target

The Chabuca target is located between 200 to 1800m to the East side of the PEA Pit where four copper anomalies were recognized. Anomaly 1 has an Area of 150mx50m with average grade of 0.375% Cu, 0.005 g / t Au and 0.5 g / t Ag. The anomaly 2 has an area of 1000m x100m with average grade of 0.73% Cu, 0.01 g/t Au and 2.74 g/t Ag. The anomaly 3 is split into a 300mx150m with average grade of 1.08% Cu and a second area of 150mx50m with average grade of 2.21% Cu. The Anomaly 4 has an area of 400mx100m with average grade of 0.41% Cu.

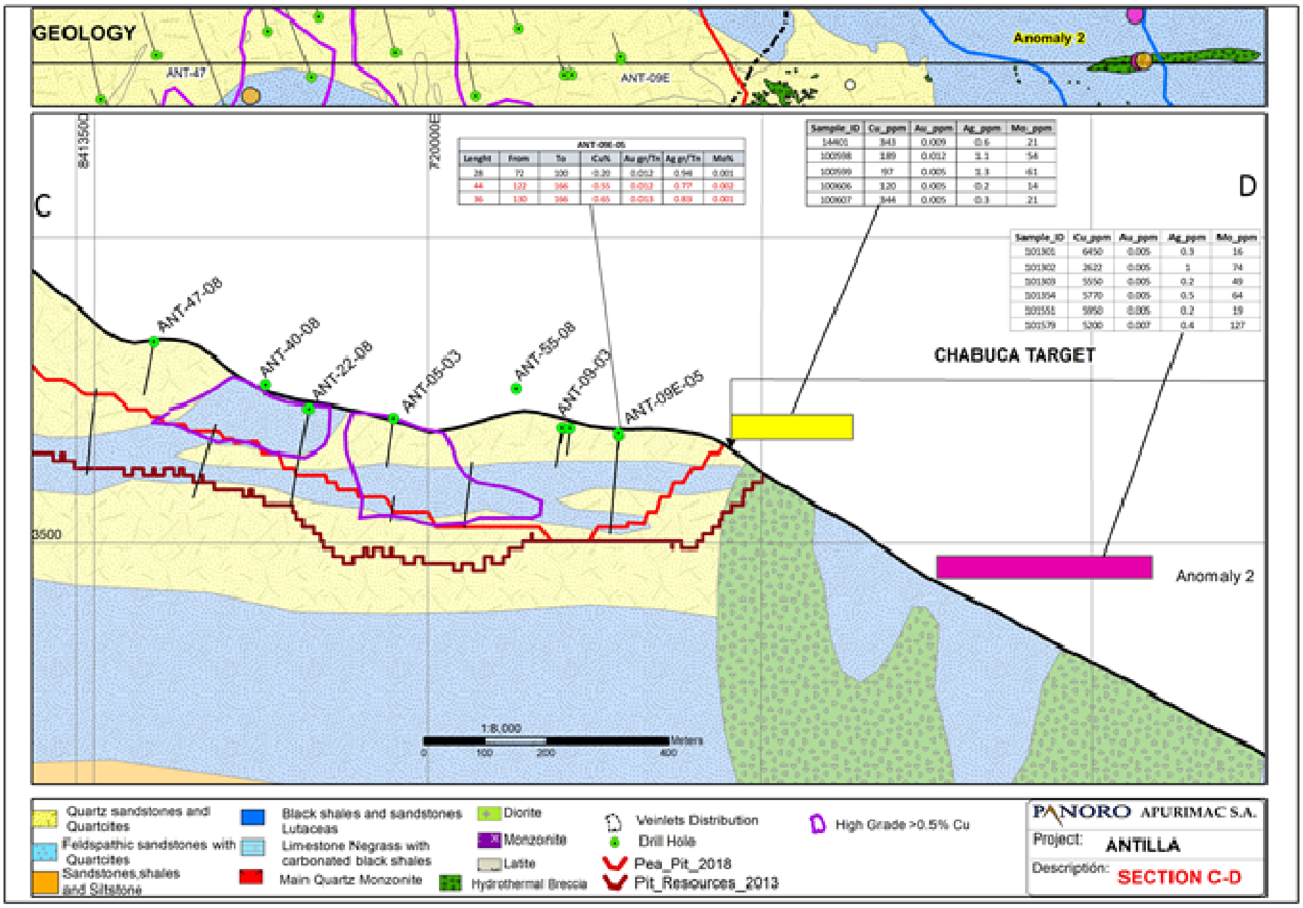
Within the anomalies are observed Breccias pipes with clast and fragments of quartzites, shales, sandstones, and matriz composed by rock dust, malachite, tenorite, chalcantite, iron oxides giving a reddish coloration to the area. The Breccia looks related with the porphyry emplacement and follow an annular shape.

Between the PEA Pit and Chabuca target is done the hole ANT-09E that intercepted until 3 intervals averaging copper grades between 0.20% to 0.65% Cu, and the hole ANT-31 intersected 83 m grading 0.30 %Cu. See Figure 9-12 and Figure 9-13.



ANT-09E-D5						
Lenght	From	To	Cu%	Au gr/Tm	Ag gr/Tm	Mo%
28	72	100	0.20	0.012	0.94	0.001
44	122	166	0.55	0.012	0.77	0.002
36	130	166	0.65	0.013	0.83	0.001

ANT-31-D3						
Lenght	From	To	Cu%	Au gr/Tm	Ag gr/Tm	Mo%
83	78	161	0.30	0.013	1.12	0.005
22	82	104	0.42	0.013	1.36	0.003
20	112	132	0.43	0.021	1.00	0.005



recognized. The sandstone layers are intruded by the quartzmonzonite porphyry in direction North-South. Parallel to the porphyry contact outcrops monomictic breccias up to 70 meters length including copper oxides in fractures and disseminated in the matrix. The primary mineralization is observed in the form of chalcopyrite and molybdenite veins.

Anomaly 1 has an area of 200mx50m with average grade of 0.56% Cu, 0.01 g / t Au, 0.46g / t Ag. The Anomaly 2 has an area of 400mx100m with average grade of 0.32% Cu, 0.01 g / t Au and 0.76 g / t Ag. The anomaly 3 has an area of 300mx50m with average grade of 0.45% Cu 0.01g / t Au and 3.43 g / t Ag. The Anomaly 4 has an area of 500mx100m with average grade of 0.56% Cu, 0.01 g / t Au and 1.03 g / t Ag.

Between the PEA Pit and the Middle Block are located the drillholes ANT-58, ANT-17 and ANT-12 which results are presented in Figure 9-11 and Figure 9-14.

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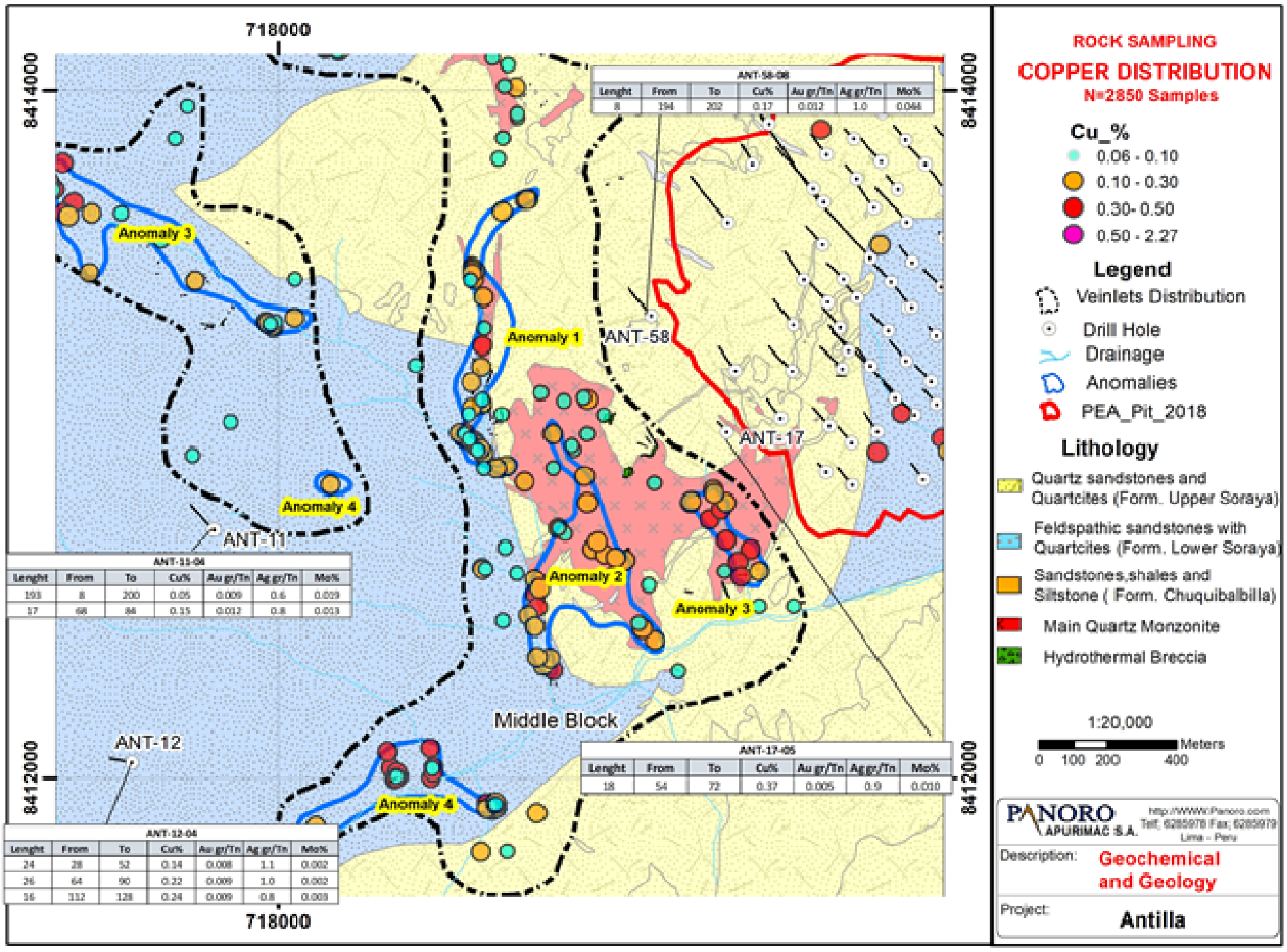
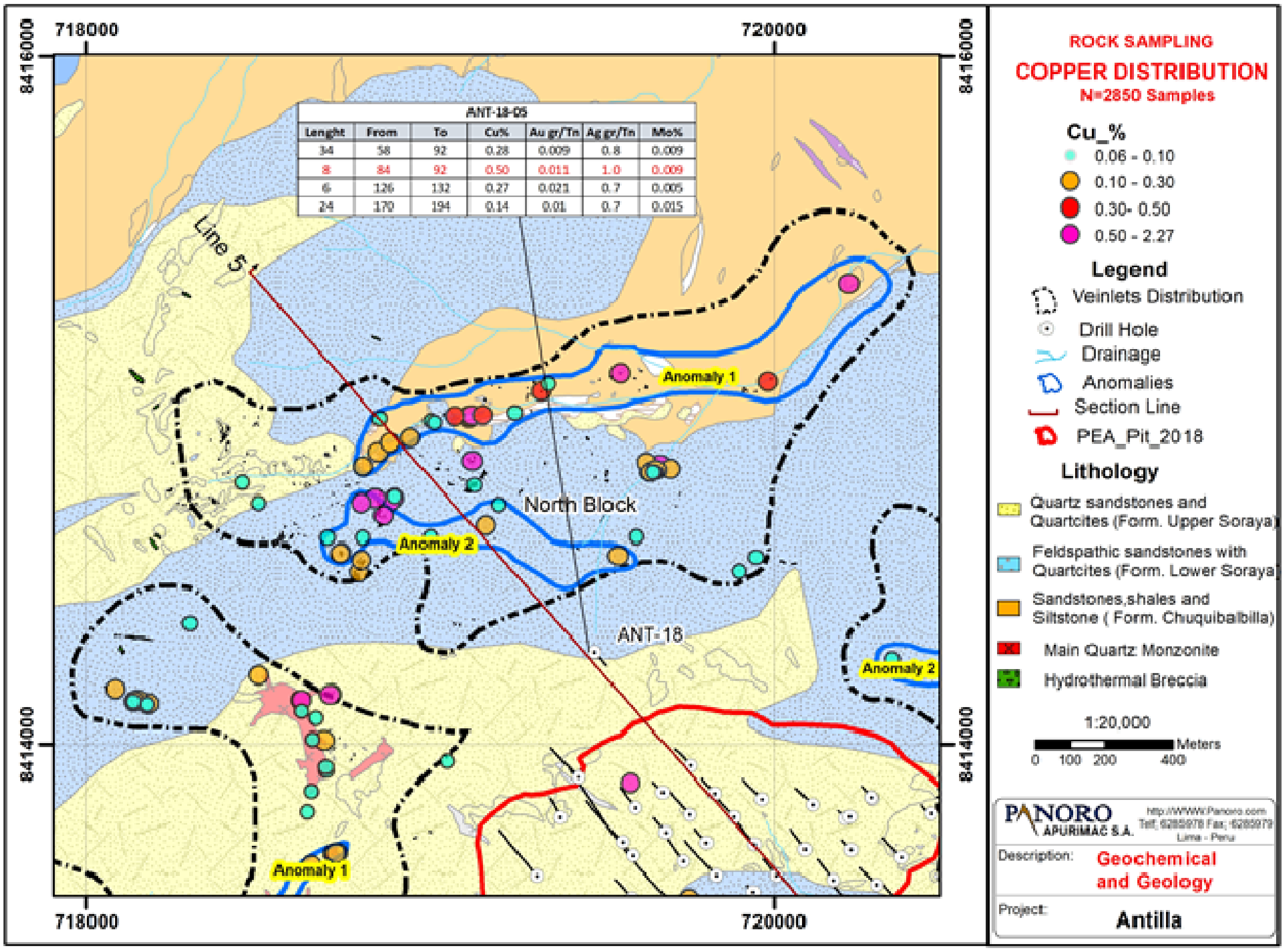


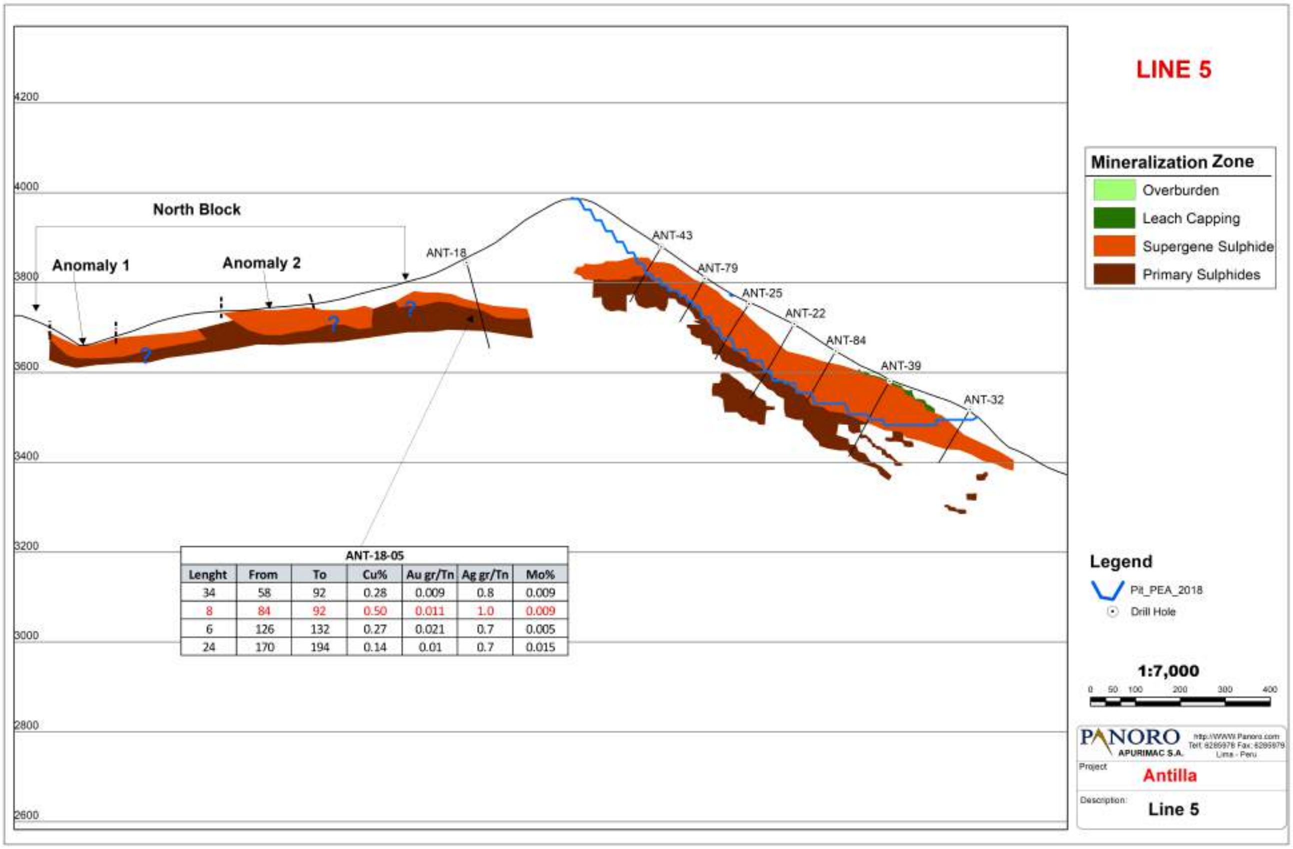
anomalies almost in East-West direction following the contact between the sandstones of Soraya formation with the shales and siltstones of the Chuquibambilla formation representing an important stratigraphic control to explore. Breccia pipes and tectonics are recognized, containing rock dust and iron oxides in the matrix. The mineralization associated with the breccias includes malachite, chalcantite, goethite and jarosite.

The mineralization associated with the breccias includes malachite, chalcantite, goethite and jarosite. The Anomaly 1 has an area of 1300mx100 with average grade of 1.35% Cu, 0.012 g / t Au, 0.36g / t Ag. The anomaly 2 has an area of 200mx50m with average grade of 0.83% Cu, 0.01 g / t Au and 0.3 g / t Ag.

To the south of the North Block target is done the drillhole ANT-18 intersecting the same secondary sulphides blanket of the PEA Pit. The Figure 9-15 and Figure 9-16 suggest the North block as the north extension of the PEA Pit mineral.


718000
720000
8416000
8414000
ANT-18-05
Lenght | From | To | Cu% | Au gr/Tn | Ag gr/Tn | Mo%
34 | 58 | 92 | 0.28 | 0.009 | 0.8 | 0.009
8 | 84 | 92 | 0.50 | 0.011 | 1.0 | 0.009
6 | 126 | 132 | 0.27 | 0.021 | 0.7 | 0.005
24 | 170 | 194 | 0.14 | 0.01 | 0.7 | 0.015
Line 5
Anomaly 1
Anomaly 2
North Block
ANT-18
ROCK SAMPLING
COPPER DISTRIBUTION
N=2850 Samples
Cu_%
0.06 - 0.10
0.10 - 0.30
0.30- 0.50
0.50 - 2.27
Legend
Veinlets Distribution
Drill Hole
Drainage
Anomalies
Section Line
PEA_Pit_2018
Lithology
Quartz sandstones and Quartcites (Form. Upper Soraya)
Feldspathic sandstones with Quartcites (Form. Lower Soraya)
Sandstones,shales and Siltstone (Form. Chuquibambilla)
Main Quartz Monzonite
Hydrothermal Breccia
1:20,000
Meters
0 100 200 400
PANORO APURIMAC S.A.
Description:
Geochemical and Geology
Project:
Antilla

Fi



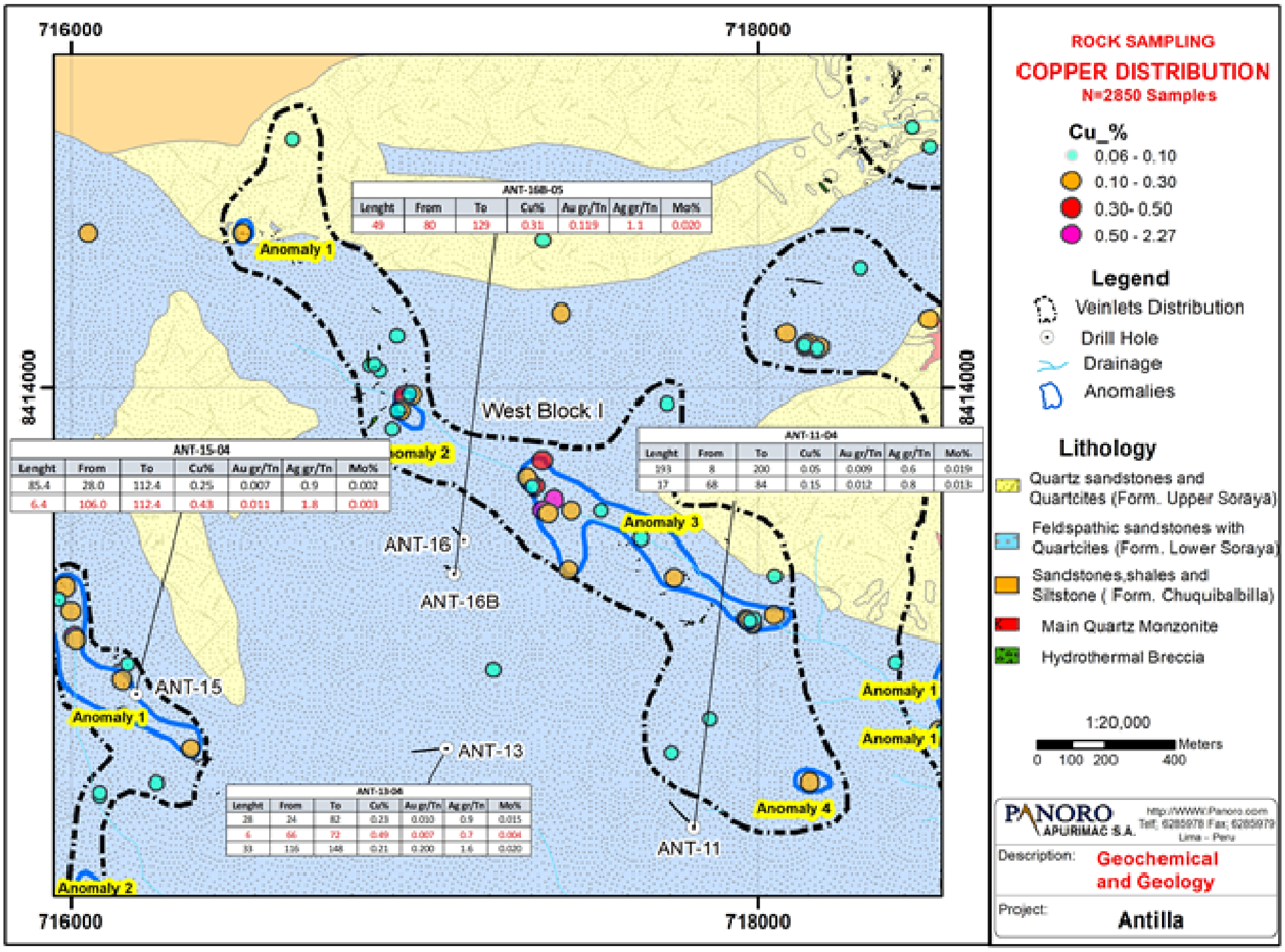
ANT-18-05						
Lenght	From	To	Cu%	Au gr/Tn	Ag gr/Tn	Mo%
34	58	92	0.28	0.009	0.8	0.009
8	84	92	0.50	0.011	1.0	0.009
6	126	132	0.27	0.021	0.7	0.005
24	170	194	0.14	0.01	0.7	0.015

Figure 9-16 Cross section linen 5 traced in Figure 9-14, describe the possible extension of the PEA Pit mineralization toward the North Block target.

The West Block I Target

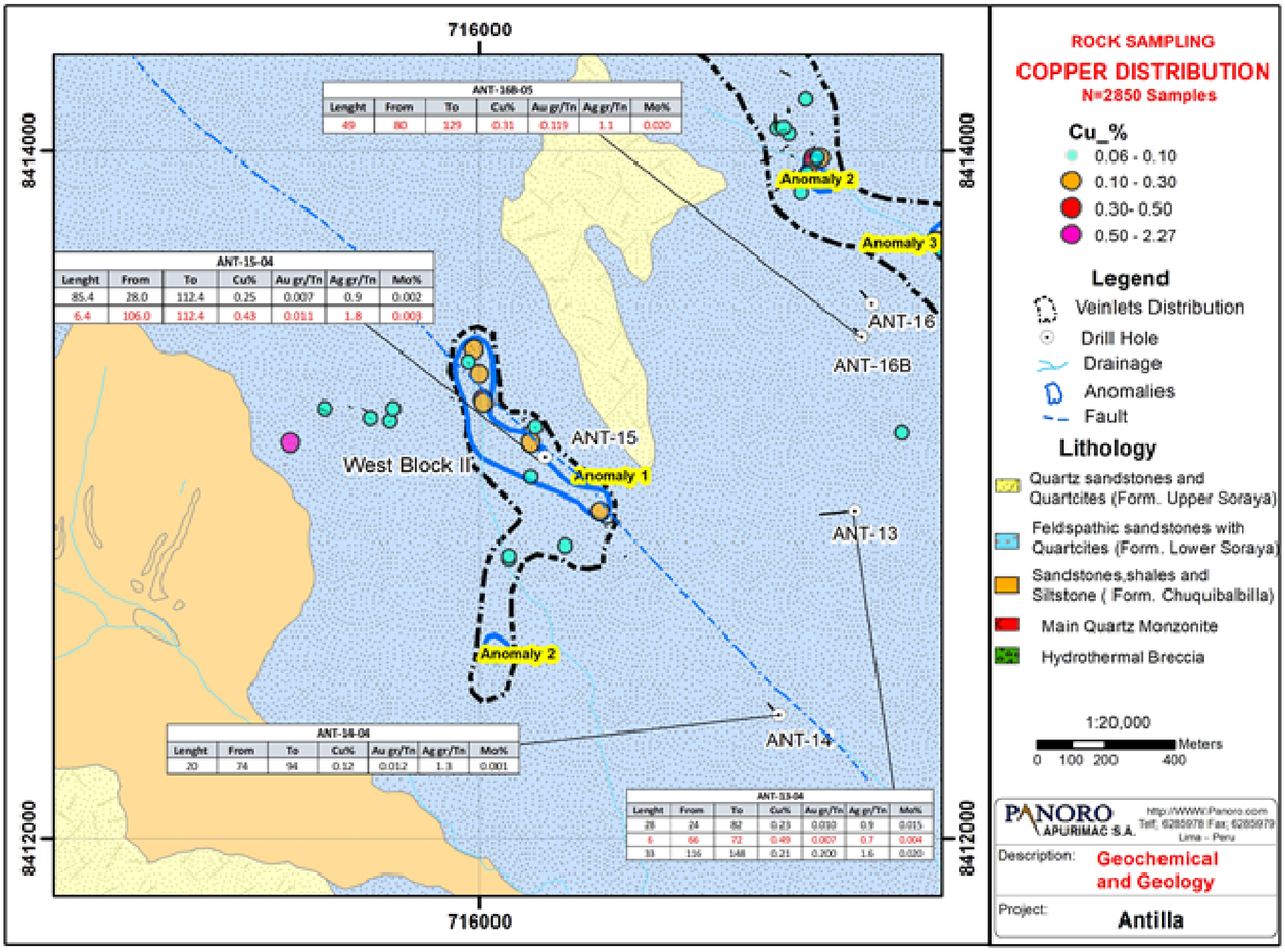
The West Block I target is located 1km to the west of the PEA Pit, with the Middle Block target located in the area between. Two high copper anomalies were recognized where the mineralization is composed by quartz molybdenum veinlets, chalcopyrite-pyrite-molybdenum veins, with predominant NS directions, and quartz veins with goethite boxwork. The Anomaly 1 has an area of 350mx100 with average grade of 0.51% Cu, 0.036g / t Au, 2.1g / t Ag. The second anomaly has average grades of 0.51% Cu, 0.036 g / t Au and 2.1 g / t Ag.

The drillholes ANT-11, ANT-13 and ANT-16B are done outside of the copper anomalies; however the mineralization with attractive copper grades were intersected into the sandstones layers. See results in Figure 9-11 and Figure 9-17.

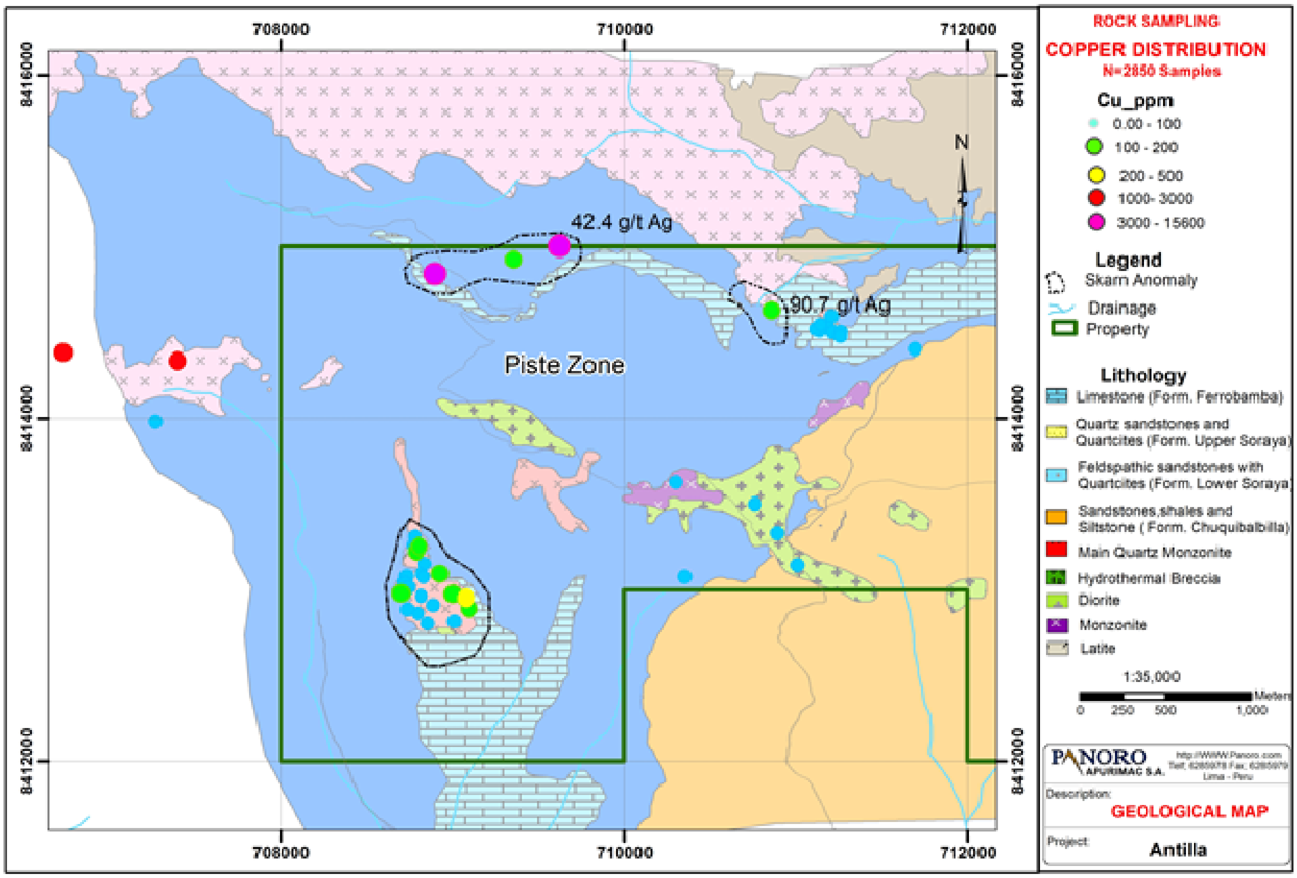


mineralization occurs disseminated in the sandstones, including chalcopyrite-pyrite-covelite-calcosine and copper oxides associated with faulting in 145 ° direction. The Anomaly 1 has an area of 500mx100m with average grade of 5.93% Cu, 0.03 g / t Au, 3.30g / t Ag.

The nearest drillholes done are the ANT-15 and ANT-14 which result are show in the Figure 9-11 and Figure 9-18.



the contact between granodioritic intrusives with limestones of the Ferrobamba formation and sandstones of the Chuquibambilla formation generating potential areas for additional exploration. The referential rock sampling shows Cu values between 200 ppm to 15000 ppm and two samples reporting 42.4 and 90.7 Ag ppm.


708000
710000
712000
8416000
8414000
8412000
N
42.4 g/t Ag
90.7 g/t Ag
Piste Zone
ROCK SAMPLING
COPPER DISTRIBUTION
N=2850 Samples
Cu_ppm
0.00 - 100
100 - 200
200 - 500
1000- 3000
3000 - 15600
Legend
Skarn Anomaly
Drainage
Property
Lithology
Limestone (Form. Ferrobamba)
Quartz sandstones and Quartcites (Form. Upper Soraya)
Feldspathic sandstones with Quartcites (Form. Lower Soraya)
Sandstones,shales and Siltstone (Form. Chuquibalbilla)
Main Quartz Monzonite
Hydrothermal Breccia
Diorite
Monzonite
Latite
1:35,000
Meters
0 250 500 1,000
PANORO APURIMAC S.A.
Description:
GEOLOGICAL MAP
Project:
Antilla

10 Drilling

The 2008 drilling program focused on defining the mineralization on the eastern part of the property. During this program, Panoro re-logged core from the boreholes completed by Chancadora Centauro S.A. (Chancadora) and carried out surface mapping of outcrops and road cuts on the property. Since 2008, Panoro has not undertaken any drill programs. The latest drilling program on the property was carried out in 2010 by Chancadora under the Antilla joint venture agreement. As of August 2010, 96 core boreholes (15,386 meters) had been drilled on the property (Table 10-1). Between 2003 and 2010 a total of five drilling programs were completed on the property. Borehole locations are shown in Figure 10-1.

Table 10-1 Summary of Drilling Programs on the Property

Year	N°. Holes	Metres	Targets
2003	12	1,983.10	Reconnaissance of main mineralized zone, holes collared 500 m apart
2004	12	1,378.90	Reconnaissance of Punkuccasa, Carachara, Cayarani, and Hualhuani areas, 3 km west of the main mineralized zone
2005	4	650.10	Reconnaissance holes at the edge of the zone defined in 2003
2008	49	9,130.60	Drilling on 100 m centres to define mineralization in the main zone
2010	19	2,242.80	Infill drilling on the Antilla deposit
Total	**96**	**15,385.50**	

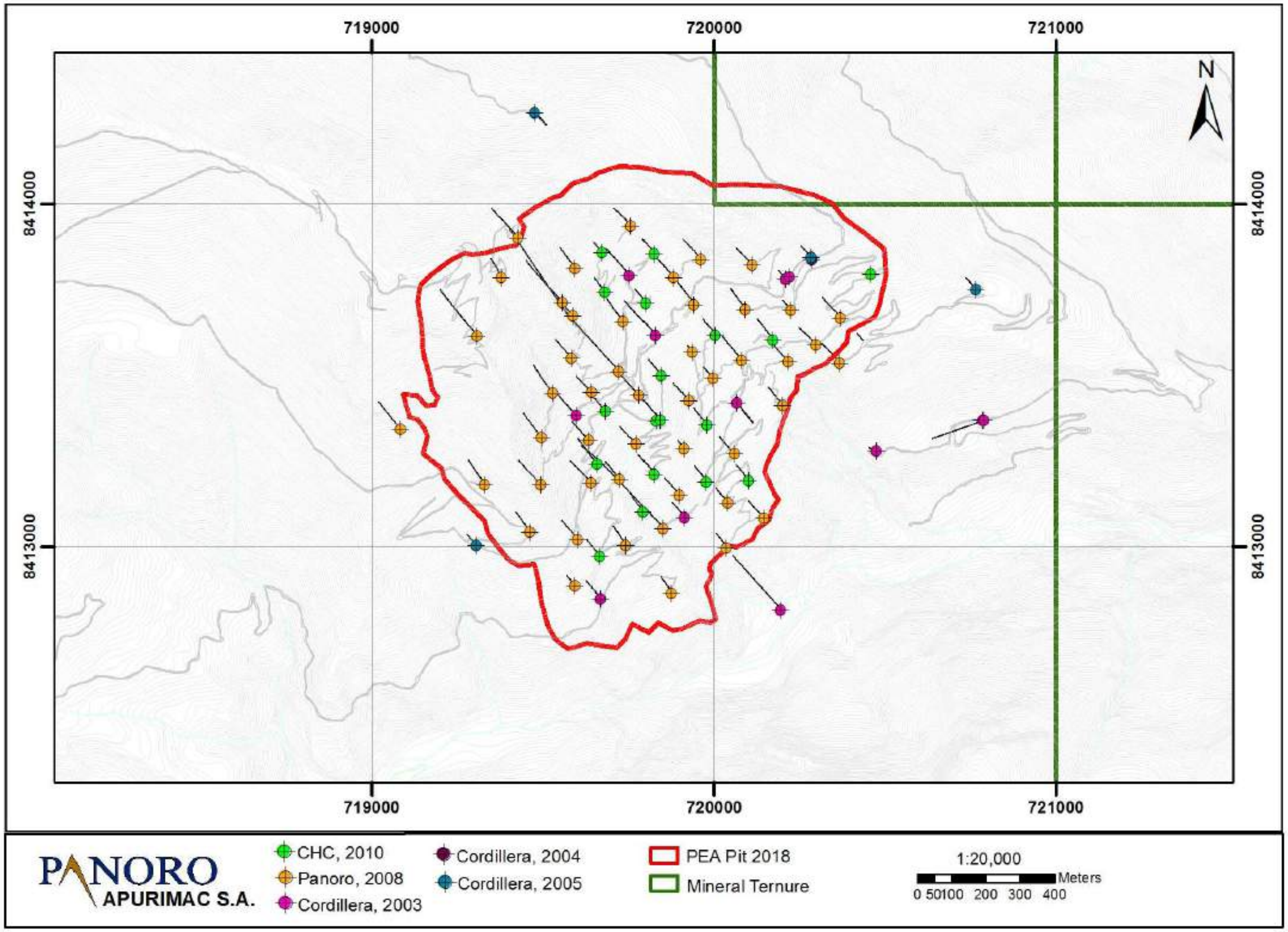


Figure 10-1 Map Showing the Distribution of Drilling in Relation to Conceptual Pit Outline

Only the drill program by Cordillera de las Minas S.A. (Cordillera) in 2003, Panoro in 2008, and Chancadora in 2010 intersected copper mineralization. Selected intersects from these boreholes are shown inTable 10-2.

Table 10-2 Selected Borehole Sulphide Mineralization Intersections

Borehole	From (m)	From (m)	Lengh (m)	Cu (%)	Mo (%)	Au (ppm)	Ag (ppm)	Mineralization Type
Cordillera Campaigns								
ANT-01-03	2	54	52	0.79	0.003	0.01	1	Secondary Sulphide
ANT-05-03	18	84	66	0.67	0.008	0.01	0.9	Secondary Sulphide
ANT-06-03	10	76	66	0.89	0.014	0.01	0.7	Secondary Sulphide
ANT-06-03	150	165	15	0.26	0.027	0.01	0.4	Primary Sulphide
ANT-07-03	18	98	80	0.68	0.008	0.01	0.9	Secondary Sulphide
ANT-09E-05	139	166	30	0.72	0.001	0.01	0.8	Secondary Sulphide
ANT-10-03	20	48	28	0.39	0.023	0	0.9	Primary Sulphide
Panoro 2008 Campaign								
ANT-20-08	60	94	34	0.75	0.007	0.01	0.6	Secondary Sulphide
ANT-22-08	56	98	42	0.8	0.005	0.01	1.2	Secondary Sulphide
ANT-24-08	18	74	56	0.7	0.017	0.01	0.6	Secondary Sulphide
ANT-25-08	26	64	38	0.56	0.004	0.01	0.9	Secondary Sulphide
ANT-26-08	6	68	62	0.63	0.012	0.01	0.7	Secondary Sulphide
ANT-28-08	26	86	60	0.85	0.023	0.01	3.2	Secondary Sulphide
ANT-30-08	32	96	64	0.75	0.002	0.01	0.5	Secondary Sulphide
ANT-34-08	60	82	22	0.51	0.039	0.01	0.5	Secondary Sulphide
ANT-37-08	62	136	74	0.54	0.001	0.01	0.5	Secondary Sulphide
ANT-38A-08	37	67	30	0.75	0.029	0.01	1.1	Primary Sulphides
ANT-38C-08	34	130	96	0.72	0.029	0.01	0.8	Secondary Sulphide
ANT-39-08	6	82	76	0.57	0.002	0.01	0.9	Secondary Sulphide
ANT-41-08	58	149	91	0.56	0.013	0.01	0.5	Secondary Sulphide
ANT-43-08	44	62	18	0.67	0.011	0.01	1.1	Secondary Sulphide
ANT-46-08	76	102	26	0.63	0.006	0.01	0.5	Secondary Sulphide
ANT-49-08	28	70	42	0.93	0.017	0.01	0.8	Secondary Sulphide
ANT-51-80	66	99	33	0.49	0.016	0.01	0.6	Secondary Sulphide
ANT-61-08	196	226	30	0.65	0.009	0.01	1	Secondary Sulphide
ANT-62-08	60	102	42	0.52	0.012	0.01	1	Secondary Sulphide
ANT-65-08	46	90	44	0.51	0.006	0.01	1.1	Secondary Sulphide
ANT-66-08	9	64	55	0.42	0.033	0.01	0.5	Secondary Sulphide

**Reported core length intervals*

10.1 Drilling by Cordillera de las Minas (2003 to 2005)

The 2003, 2004, and 2005 drilling campaigns by Cordillera were reconnaissance exploration programs intended to test for large porphyry-type targets carried out by contract drilling companies and supervised by Cordillera staff geologists. Borehole spacing was wide and collar surveying was limited to the use of handheld global positioning system (GPS) receivers. Logging was largely descriptive, featuring graphic logs for rock type, texture, structure, alteration, and mineralization, and focused on regional stratigraphic context. Analytical quality control practices relied on internal laboratory duplicates and did not meet industry best practices. Boreholes were surveyed with a Sperry Sun or Flexit down-hole directional survey instrument.

Core sampling for the Cordillera campaigns is described in detail in Lee et al. (2007). The authors report that core was sampled at continuous 2-metre down-hole intervals, independent of logging for mineralization intensity or rock type. Sample intervals were marked by the logging geologist and core was split with a rotary diamond-carbide saw. Half of the core was placed into pre-numbered sample bags; the other half was transferred into corrugated plastic boxes for storage.

The corrugated plastic boxes used to archive the core were not ideal for long-term storage or transport. Shifting and disruption of the core and sample tags and blocks made validation of the sampling intervals difficult.

10.2 Drilling by Panoro (2008)

Panoro contracted Bradley MDH from Lima, Peru, a subsidiary of Bradley Group Ltd., to perform the drilling for the 2008 drill campaign, which was supervised by Panoro personnel. Two rigs, a Bradley MDH LF-70 and LD-250, were used to drill conventional NQ-sized boreholes. Boreholes were drilled from surface platforms that were located with a handheld GPS receiver on 100-metre spaced grid lines with an azimuth of 150 degrees. The boreholes have a dip of 45 to 75 degrees to the northwest to provide high-angle intersections with the secondary sulphide zone. Drills were aligned with a compass. Boreholes generally range from 95 to 200 meters in length. However, boreholes ANT-62-08 and ANT-66-08 were drilled to just over 750 meters depth to test primary mineralization and hornfels alteration at depth.

The contractor performed down-hole surveys using a Sperry Sun instrument at 30-metre intervals. Following drilling, casings were pulled and a cast concrete monument was set on the borehole collar. Panoro contracted Global Mapping Peru (Global Mapping) based in Lima, Peru to survey borehole collars using a total station GPS. Global Mapping visited the property twice to survey completed boreholes during the 2008 campaign.

10.3 Drilling by Chancadora Centauro (2010)

In 2010 Chancadora in a joint venture with Panoro completed an infill drilling program comprising 19 core boreholes (2,243 meters). Borehole lengths varied from 25 to 169 meters with an average length of 116 meters. The aim of the program was to confirm high grade intersections encountered by previous drilling and to reduce the borehole spacing to 100 meters. The decreased borehole spacing was aimed at upgrading mineral resource classification in the in-pit area.

The drilling was conducted by a subsidiary of Chancadora and, according to Panoro; drilling procedures matched those employed by Panoro in 2008.

10.4 Geotechnical Logging

During the Cordillera drill programs in 2003, 2004, and 2005, geotechnical logging was restricted to the collection of rock quality determinations (RQD) and core recovery data. Recovery averaged 88% and RQD averaged 18%. The criteria and methodology for the collection of these data are not known.
At the beginning of the 2008 drilling campaign, Panoro contracted Knight Piésold of Lima to develop and train Panoro staff in geotechnical logging procedures. During the program, Knight Piésold staff visited the property to review data and logging, and maintain logging standards for the program. Geotechnical logging was carried out prior to geological logging and sampling for all boreholes drilled during the 2008 campaign. Logging information was recorded on standardized paper log sheets. The geotechnical database comprises 22 logged parameters consisting of measurements or scores and calculated values for RQD, recovery, and rock mass rating (RMR).

Core recovery in all campaigns is good for all zones, averaging greater than 87% for each of the Primary Sulphide, Secondary Sulphide, and Leached/Oxidized zones, and averaging 93% for all zones (Table 10-3. RQD is relatively low. The Primary Sulphide and Secondary Sulphide zones have RQDs of approximately 20, the Leached/Oxidized zone of 10. RQD in the intrusive domains is higher, ranging from 29 to 63. Uniaxial compressive strength (UCS) is reasonably consistent ranging from 32.4 MPa in the Leached/Oxide Zone to 42.8 MPa in the Main Porphyry.

Table 10-3 Geotechnical Summary for the 2008 Panoro Drill Campaign

Domain	Core Recovery (%)	UCS (Mpa)
Primary Sulphide	95.4	37.8
Secondary Sulphide	98.3	38.2
Leached/Oxidized	87.3	32.4
Main Porphyry	100	42.8
Late Porphyry	95.3	42.1
Total	**93.2**	**36.0**

10.5 Geological Logging

Geological logging during the 2003, 2004, and 2005 drilling campaigns was recorded on graphic log sheets. Intensities of structural features, mineralization, and alteration were marked by coloured pencil lines in columns down the borehole. Rock types were marked with graphic figures for clay, sand, bedding, or intrusive symbols. It is difficult to translate graphic geological logs to database records to plot sections for geological modelling and resource estimation. As a result, at the end of 2008, Panoro re-logged boreholes completed by Cordillera between 2003 and 2005.

Geological logging for the 2008 Panoro program was recorded on standardized log sheets with fields for interval depths, mineralization zone type (primary, secondary-sulphide, leached, and oxidized), texture, breccia bodies and veining, structure filling, alteration intensity by mineral (sericite, silica, clay, biotite, potassic feldspar, albite, calcite, magnetite, chlorite, epidote), iron and copper oxide and sulphide mineral intensity, and mineralization style. Fields for observations, graphic strip log, and rock code were also recorded.

10.6 Comments

Tetra Tech is of the opinion that the drilling procedures adopted by Panoro conform to industry standard. Tetra Tech cannot comment on drilling procedures followed by Cordillera and Chancadora. The drilling pattern resulting from the drilling is considered sufficiently dense to interpret the geometry and the boundaries of the copper and molybdenum mineralization with adequate confidence.

11 Sample Preparation, Analyses and Security

Cordillera de las Minas S.A. (Cordillera) used CIMM Peru S.A. in Lima, Peru for all analyses of core. MMTS was unable to determine whether the laboratory was accredited to any quality standards at the time.

Panoro used ALS Chemex's preparation laboratory in Cusco for the preparation of core samples. No information on the accreditation of ALS Cusco was available; however, ALS Chemex laboratories typically operate under a global management system that is accredited ISO 9001:2008.

In 2008 Panoro used Bureau Veritas Inspectorate S.A (Inspectorate) in Lima, Peru to assay composite core samples for total copper. MMTS was not able to determine whether the Inspectorate laboratories are accredited to ISO standards. Furthermore, Panoro submitted one sample to Inspectorate in Vancouver for metallurgical testing.

Panoro used ALS Chemex in Lima, Peru for geochemical analysis of core samples as well as for specific gravity determinations. The laboratory is accredited to ISO 9001:2008 by IQNet and ICONTEC international (Registration number CO-SC-5462-1) and to ISO 17025 by the Standards Council of Canada (Accredited Laboratory Number 670) for a host of geochemical analyses, but not for the determination of specific gravity.

Panoro used Certimin S.A. (Certimin) of Lima, Peru for geochemical analysis of rock chip samples taken between 2013 and 2015. Certimin is accredited to ISO 9001:2008 and ISO 17025 accredited by the Instituto Nacional de Calidad (INACAL), Peru (Registration Number LE-022). MMTS was unable to determine whether Certimin's ISO 17025 certification covers those methods used to analyze Panoro's samples.

Panoro submitted samples to Certimin of Lima, Peru for metallurgical testwork.

11.1 Sample Preparation and Analyses

11.1.1 Sampling by Cordillera de las Minas (2003 to 2005)

For the Cordillera drilling programs in 2003, 2004, and 2005, samples were prepared and analyzed at the CIMM laboratory in Lima. Results for total copper, cyanide soluble copper, sulphuric acid soluble copper, residual copper, molybdenum, silver, lead, zinc, and arsenic by atomic absorption (AA), and gold by fire assay were reported.

All core, pulps and coarse crushed rejects from the Cordillera drilling programs were transported to what is now the Panoro core logging and storage facility at Cotabambas where they are stored in a secure building.

11.1.2 Sampling by Panoro (2008)

For the 2008 drilling program, Panoro maintained a chain-of-custody of core from the core tube at the drill site to the ALS Chemex's sample preparation facility in Cusco. Panoro staff supervised drilling, 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 Lima.

Samples were prepared by the ALS Chemex sample preparation facility in Lima. 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 millimeters. A 250-gram sub-sample of the crushed material was pulverized to 85% passing 75 micrometres (preparation code PREP-31). The pulps were sent to the ALS Chemex chemical laboratory for analyses.

Samples were analyzed at the ALS Chemex chemical laboratory in Lima by atomic absorption spectroscopy for total copper, molybdenum, lead, zinc, arsenic, and silver (Code AA62). Gold was assayed by atomic absorption using a 30-gram aliquot (Code Au-AA23).

At the conclusion of the 2008 Panoro drilling program, 2,715 reject samples from mineralized intersections were combined into 140 composite samples, which were analyzed for sequential copper and total copper at Inspectorate using a four acid digestion and atomic absorption spectroscopy (analytical code Sp-135).

11.1.3 Sampling by Chancadora Centauro (2010)

No information is available regarding sample preparation and analytical procedures used by Chancadora Centauro S.A. (Chancadora). However, according to Panoro, Chancadora used the same procedures employed by Panoro in 2008.

11.1.4 Sampling by Panoro (2013 to 2015)

For the 2013 to 2015 soil and rock chip sampling, Panoro maintained a chain-of-custody of all samples; 1648 rock chip and 486 soil sample in total; from the field to Certimin's sample preparation and analytical facility in Lima. Panoro staff supervised all sampling, transported the samples to Panoro's exploration facility, and processed all samples. Bagged samples were stored in a locked contained beside the core shed along with core samples until a batch could be dispatched by pickup-truck to Lima

Samples were prepared by Certimin's sample preparation facility in Lima. Samples were registered and assigned a LIMS code upon reception. Sample preparation (code IC-PMM-01) consisted of drying the samples at 60 to 100°C, followed by crushing to 90% passing ¼ inch (6 millimetres) and further crushing to 90% passing 10 mesh. At this stage, a sub-sample of 200 to 300 grams was split off and pulverized to 85% passing 200 mesh; the resulting pulp was bagged and passed on for analysis.

Samples were analyzed by inductively coupled plasma optical emission spectrometry (ICP-OES) (code IC-VH-17) for total copper, molybdenum, lead, zinc, arsenic, and silver; gold was assayed by atomic absorption using a 30- or 50-gram aliquot (code IC-EF-01).

11.2 Specific Gravity Data

Between 2003 and 2005, Cordillera measured specific gravity on core. Two consecutive measurements were taken from each 2-metre core interval. MMTS was unable to determine the methodology or technique used for specific gravity determinations. In total, more than 3,600 specific gravity measurements were completed by Cordillera. However, Panoro did not consider these data for mineral resource modelling.

During the 2008 Panoro, specific gravity was measured on 283 core samples using a water displacement method. Determinations were carried out on 10- to 15-centimetre long pieces of core taken at 20-metre intervals down each borehole.

The procedure involved the determination of the sample's weight after being saturated in water. The sample was then suspended from a wire hanger from the bottom of the balance and the weight of the submersed, water-saturated sample was measured.

Specific gravity was also measured on a suite of 22 samples by ALS Chemex in Lima for validation. Specific gravity was measured using a water displacement methodology with paraffin coating. The ALS Chemex determinations compare well with the Panoro measurements (Table 11-1).

Table 11-1 Summary of Specific Gravity Determinations Zone

Zone	No of measurements	Average	Maximum	Minimum
Primary	41	2.46	2.83	2.15
Secondary	132	2.43	2.79	2.13
Oxide	78	2.42	2.73	2.07
Dike	3	2.4	2.44	2.38
Late Dike	8	2.41	2.53	2.28
Total	**262**	**2.43**	**2.83**	**2.07**

11.3 Quality Assurance and Quality Control Programs

11.3.1 Cordillera de las Minas (2003 to 2005)

No information was available regarding an analytical quality assurance or a quality control program implemented by Cordillera during 2003 to 2005.

11.3.2 Panoro (2008)

Panoro implemented an analytical quality control program consisting of blank and duplicate samples. No standard reference material was used as control sample. Panoro did not submit samples for umpire check assaying. Only limited information was available to Tetra Tech regarding the frequency of blank and duplicate sample insertion into the general samples stream, and the total number of quality control samples submitted during the 2008 drilling program. Crushed quartz samples (external source) were used by Panoro as blank reference material.

11.3.3 Chancadora Centauro (2010)

Chancadora implemented an analytical quality control program consisting of using control samples (blanks, certified reference material, field duplicates, pulp duplicates and preparation duplicate samples) inserted in each batch of samples submitted for preparation and assaying. The insertion rates for the quality control samples was 1.1% for blanks, 1.6% for certified standards, 1.1% for field duplicates, 0.8% pulp duplicates, and 0.8% for preparation duplicate samples.

No further information is available regarding the analytical quality assurance or a quality control program implemented by Chancadora in 2010.

11.4 Tetra Tech Comments

In the opinion of Tetra Tech, the sampling preparation, security and analytical procedures used by Panoro are generally consistent with widely accepted industry best practices and are, therefore, adequate.

12 Data Verification

12.1 Verifications by Panoro

In 2008 Panoro submitted an unknown number of pulps and coarse rejects from the Cordillera de las Minas S.A. (Cordillera) drilling program for re-assay. Panoro inserted standard reference material samples with this sample stream. According to Panoro, results compared well with the original assay results.

MMTS is unaware of any other data verification procedures employed by Panoro during exploration.

12.2 Verifications by Tetra Tech

Upon receipt of Panoro database for the project, all relevant data underwent extensive data verification. From a total of 96 boreholes, Tetra Tech chose 23 for validation, representing approximately 24% of the boreholes in the database. Seventeen of these boreholes represent the most recent drilling on the property (2010 drilling program), which also coincides with all the available density data; the boreholes were chosen for this reason. The remaining six boreholes were chosen at random from the earlier drill campaigns.

12.2.1 Collar Data

Collar data were provided in comma separated values (CSV) file format. No original survey information for the collar locations was provided, and therefore no verification was possible.

12.2.2 Lithology Data

Lithological data were provided in CSV file format. The lithology of the 23 boreholes chosen for review represents 14% of the lithology database. Where possible, these data were compared to the scanned paper logs. For more recent data, the original Excel spreadsheet logs were used. There were discrepancies between the original logs and the database. However, all of these differences are attributable to recent updating, re-logging, and consolidation of lithological data. Once these changes were considered, 100% of the data verified matched the originals.

12.2.3 Assays Data

Assay data were provided in CSV file format. The assay results for the 23 boreholes chosen for review represent 20% of the assay database. Assay results for these boreholes were verified against the original laboratory certificates and Excel spreadsheets. One hundred percent of the verified samples matched the original data. Discrepancies were noted in the handling of samples that yielded results below the detection limit. In order to obtain a numerical value, some of these assay results were set to a value equal to the lower detection limit, while in other cases the results were set at half the lower detection limit. Tetra Tech recommends adopting a consistent way to handle samples yielding results below the detection limit throughout the database.

12.2.4 Down-hole Survey Data

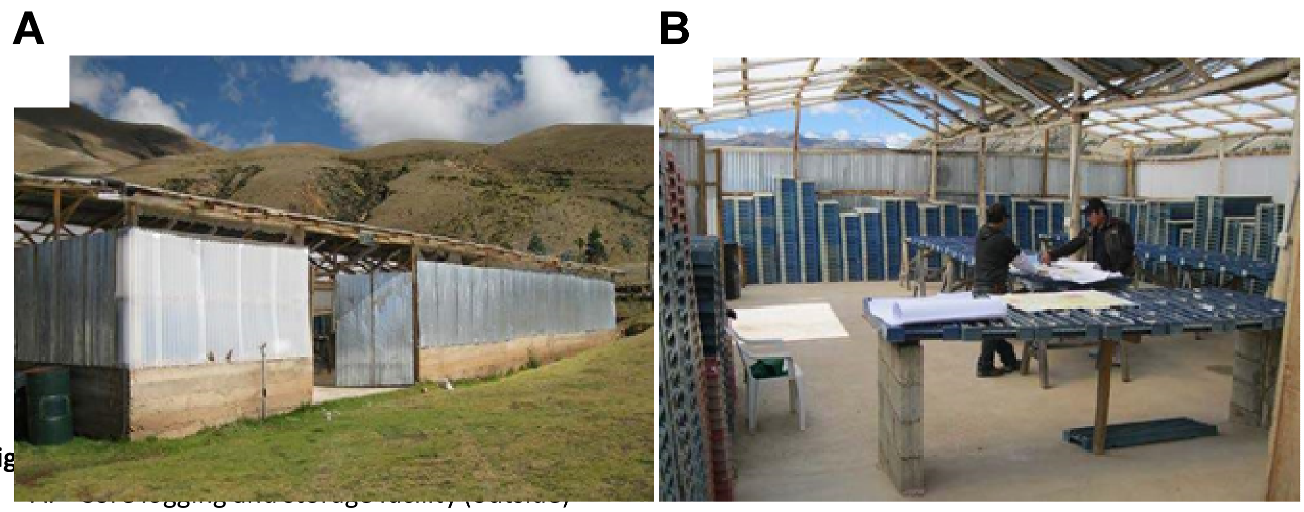
Down-hole survey data were provided in CSV file format. No original down-hole survey information was provided and therefore no verification was possible.

12.2.5 Site Visit

The qualified person responsible for the mineral resource statement reported herein is Paul Daigle, PGeo, Senior Geologist with Tetra Tech. Mr. Daigle conducted the site visit to the property between June 4 and 7, 2013. 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 Luis Vela Arellano, Vice President Exploration for Panoro; John Romero Villanueva, Chief Project Geologist for Panoro; and Edwin Mayta, Manager Technical Services, for Panoro.

Project Site and Borehole Locations

The Antilla base camp and project site were visited on June 5, 2013. The base camp is located adjacent to the village of Antilla and is made up of several permanent cinder block buildings (kitchen and office), semi-permanent wood and corrugated tin structure (logging, sampling, and storage facility) and Weather haven tents (accommodation). The base camp is clean and well-maintained. The core logging and sampling facility is clean and well-maintained. Core boxes are stacked by borehole. The plastic core boxes are sturdy and made to be stackable. The core boxes are marked in black marker showing borehole number, box number, and sample interval. Sawhorses and beams are set up for core logging and review of core. The author was able to review boreholes ANT-48-08 and ANT-69-08 (Panoro drill program) that are still stored on site. Figure 12-1 illustrates the core logging and storage facility.



Fig

A. Core logging and storage facility (outside)
B. Core logging and storage facility (inside)

Eleven borehole collars were sited using a handheld GPS receiver. The position of all checked borehole collars is consistent with the borehole coordinates in the logs and in the database. The project site was clean of drilling debris.

Collars are clearly marked on the ground. The collar is fitted with PVC pipe and cemented into place. The borehole number is engraved in the cement and, at some borehole locations, marked on a nearby boulder or outcrop.

Core Storage Warehouse, Cusco

The Antilla core is stored either at the Antilla base camp or in one of three warehouses in Cusco. The author 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 some of the Antilla core and most of the core from Panoro's Cotabambas project.

The warehouse also serves as a storage depot for exploration, field, and camp supplies as well as for equipment for the various projects. The warehouse is kept clean and has a wooden core tables along its length for viewing core.

12.2.6 Verifications of Analytical Quality Control Data

Panoro made available to Tetra Tech external analytical quality control data collected by Panoro.

Tetra Tech aggregated the assay results for the external quality control samples for further analysis. Blanks data were summarized on time series plots to highlight the performance of the control samples. Paired data were analyzed using bias charts. No other analytical quality control data were available; Panoro did not use certified reference material as a control sample.

Performance of the blank samples was adequate, with a couple of 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 lab. Figure 12-2 displays the blank performance.

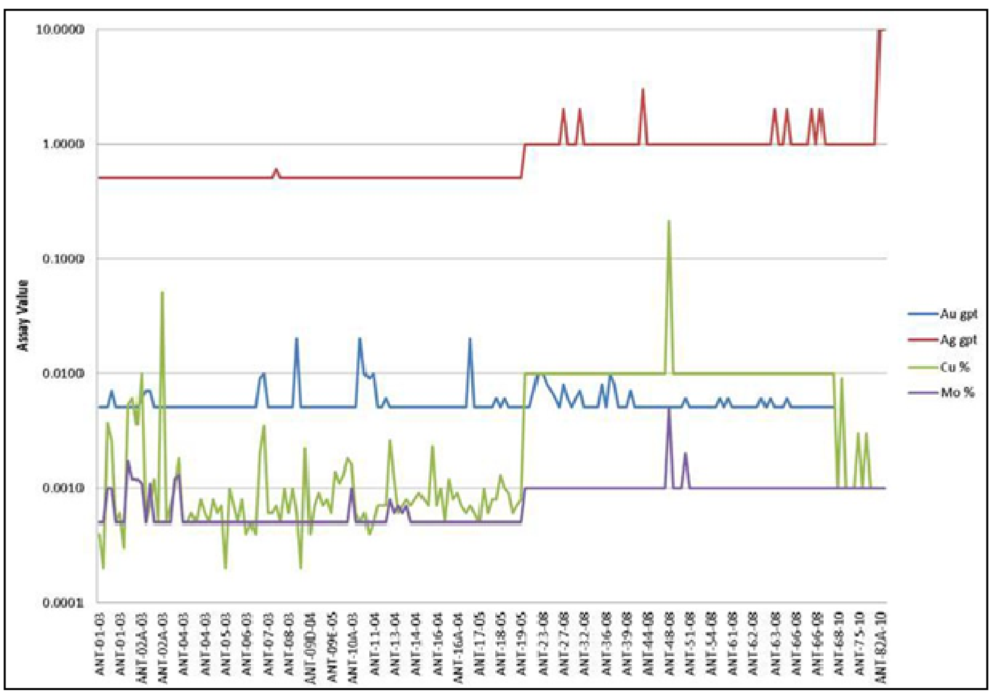


Figure 12-2 Blank Performance for Various Metals

Three duplicate samples were used by Panoro: split core, reject, and pulp. All three performed well, with less than 6% of the samples falling outside of a two the standard deviation error envelope for each type. Figure 12-3 displays the duplicate control graphs.

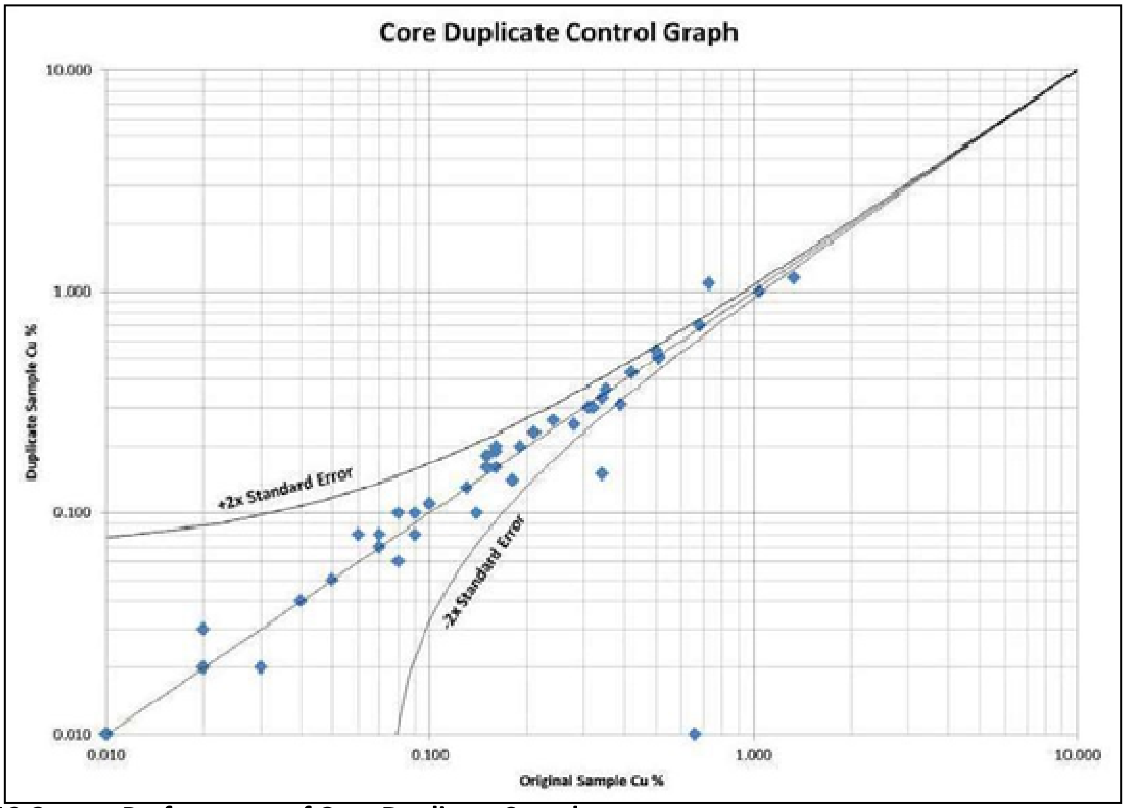


Figure 12-3 Performance of Core Duplicate Samples

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Several metallurgical testwork programs have been implemented by various organisations on samples of Antilla mineralogical materials since 2006. Preliminary metallurgical characterisation studies were carried out by Laurion and Inspectorate in 2006 and 2011 respectively. In 2013 a more substantive program was undertaken by Certimin which was aimed at recovery of copper and molybdenum from both primary and secondary materials by conventional flotation methods. In 2018 a column leaching program together with supporting testwork, aimed specifically at extraction of copper from secondary sulphides and the supergene zone of the Antilla deposit was implemented by Aminpro, this testwork program is currently ongoing. The results to date from the various testwork programs are summarised in the sections below.

13.2 Laurion, 2006

The following was taken from (Wright, 2009).

A preliminary assessment of flotation and two acid heap leach process options for the Property were carried out by Laurion Consulting Ltd. (Laurion) in 2005/2006 (Fox 2006). The assessment was based on drill core logging and total and soluble copper assays from the 2003, 2004, and 2005 CDLM drill campaigns.

Froth flotation of sulphide minerals in ground ore was considered as the most favourable process option for the Property (Fox 2006). Laurion identified several advantages of flotation including the recovery of molybdenum and silver, and the suitability of the method for recovering the transitional sulphide-secondary sulphide mineralization that is observed at the Property. Laurion pointed out that typical porphyry copper mineralization can be treated with a coarse primary grind, roughers, and scavengers to achieve a final recovery of 90% for copper. A 30,000 t/d concentrator could be configured with a primary gyratory crusher, a semi-autonomous grinding (SAG) or ball mill circuit and hydro-cyclones. Large tank cells can be used in rougher and scavenger flotation and a ball mill Vertimill™ used to regrind concentrate. Tank cells and column cells can be used for cleaner flotation. Thickening and pressure filtration of the concentrates could also be used to increase copper grade and decrease impurities in the final concentrate. In the preliminary assessment recovery of 90% of copper and 40% of molybdenum by flotation were given as reasonable targets for metallurgical recovery.

Conventional, or Cuprochlo heap leach with solvent extraction and Electrowinning (SX-EW) are often used on lower-grade oxide and chalcocite ores. These processes have the advantage of lower capital costs and often have lower operating costs than flotation (Fox 2006). The major down-side of the leach methods is the low recoveries anticipated for chalcocite and chalcocite-chalcopyrite mineralization and the inability to recover molybdenum and other by-product credits.

13.3 Inspectorate, 2011

In 2011, Panoro retained Inspectorate, a Bureau Veritas Company (Inspectorate), based in Vancouver, Canada, to conduct several preliminary bench scale flotation tests on samples from the Antilla deposit. A series of six flotation tests were carried out on samples of roughly 2 kg. There are limited descriptions of these tests; however, summary results show an approximate 90% recovery for copper.

13.4 Certimin, 2013

In June 2013, Panoro retained CERTIMIN S.A. (Certimin), a laboratory in Lima, Peru to undertake metallurgical test work on the main mineralization of the Antilla deposit. In June 2013, Certimin received two composite samples from the Antilla deposit. Sample A was taken from the primary sulphide zone and Sample B was taken from the supergene enrichment zone. The composite samples were mostly made up of sample rejects from the drill core sample analyses.

The metallurgical test work included comminution (grindability) tests and flotation tests. The flotation test work consisted of copper-molybdenum bulk flotation conditioning optimization tests, copper-molybdenum separation tests and bulk flotation locked cycle tests.

Characterization of the head samples, concentrates, and tailings was also undertaken. The results discussed in this section are summarized from the report prepared by Certimin (Certimin 2013).

13.4.1 Head Assays

Table 13-1 summarizes the assay results of the two samples. Sample A from the primary sulphide zone contained 0.29 % copper and 115 ppm molybdenum. Sample B from the supergene zone had a higher copper content, assaying at 0.56 % copper and 97 ppm molybdenum. The copper chemical analysis results showed that Sample B contained a significant amount of secondary copper minerals. The acid soluble copper (copper in oxide forms) represents approximately 10 % and 25 % of the total copper in Samples A and B, respectively. Gold and silver contents are low.

Table 13-1 Summary of Head Assay of Metallurgical Samples

Sample	Zone	Weight (kg)	Cu (%)	Cu_{CN-} (%)	Cu_{Res} (%)	Cu_{SolH+} (%)	Mo (ppm)	Ag (g/t)	Au (g/t)	Fe (%)	S_{total} (%)
Sample A	Primary Sulphides	1,245	0.29	0.05	0.21	0.03	115.4	0.60	0.02	1.08	0.73
Sample B	Supergene	1,145	0.56	0.32	0.10	0.14	97.4	0.97	<0.01	0.98	0.73

Note: Cu_{CN-} cyanide leachable copper; Cu_{Res}: residual copper; Cu_{SolH+}: acid soluble copper

13.4.2 Preliminary Comminution Test Work

The results of the grindability tests show that both the samples were low in grinding energy requirements. The Bond ball mill work index was 10.4 kWh/t for Sample A and 8.9 kWh/t for Sample B. The hardness must be confirmed with further testing on fresh samples.

13.4.3 Flotation Test Work

13.4.3.1 Open Circuit Bulk Flotation

Certimin conducted preliminary flotation test work to investigate the effects of various process conditions on copper and molybdenum recoveries. The process conditions tested for rougher and cleaner flotation included primary grind size, pulp pH, reagent suite, and regrind size. The tested primary grind size varied from 80% passing 94 µm to 121 µm for Sample A and from 80% passing 76 µm to 115 µm for Sample B.

A total of ten rougher flotation tests were run on each sample. For Sample A, the rougher flotation recoveries varied between 86.5% and 91.0% for copper and between 77.6% and 89.4% for molybdenum.

For Sample B, the rougher flotation recoveries varied between 83.3 % and 86.9 % for copper and 83.7% to 87.8% for molybdenum.

The regrinding tests showed that the concentrate grades of the cleaner flotation improved after the bulk rougher concentrates were reground. Further tests are required to determine the optimum regrind size for the mineralization.

The test results from the open circuit flotation tests conducted at a primary grind size of 80% passing approximately 100 µm are summarized in Table 13-2. The flowsheet used for the tests is plotted in Figure 13-1.

Table 13-2 Open Circuit Flotation Test Results

Sample	Test ID	Product	Grade				Recovery				
			Cu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	Mass (%)	Cu (%)	Mo (%)	Au (%)	Ag (%)
Sample A	M.A.-P17	Cleaner Concentrate	25.30	7,968	0.4	37.6	0.96	81.1	71.2	37.3	37.2
		Rougher Concentrate	3.94	1,275	0.1	7.3	6.90	91.0	82.2	58.1	51.8
Sample A	M.A.-P18	Cleaner Concentrate	25.80	8,739	0.4	37.9	0.96	81.1	74.4	37.5	36.7
		Rougher Concentrate	4.36	1,484	0.1	8.3	6.29	90.1	83.1	53.1	52.7
Sample B	M.B.-P20	Cleaner Concentrate	40.30	5,936	0.3	38.7	1.06	74.5	73.5	36.1	41.6
		Rougher Concentrate	8.86	1,343	0.1	9.4	5.49	84.7	86.1	48.7	52.1
Sample B	M.B.-P21	Cleaner Concentrate	45.90	3,899	0.4	39.2	0.87	69.5	41.1	30.9	36.0
		Rougher Concentrate	8.04	1,040	0.1	7.8	6.08	85.5	77.0	60.2	50.1

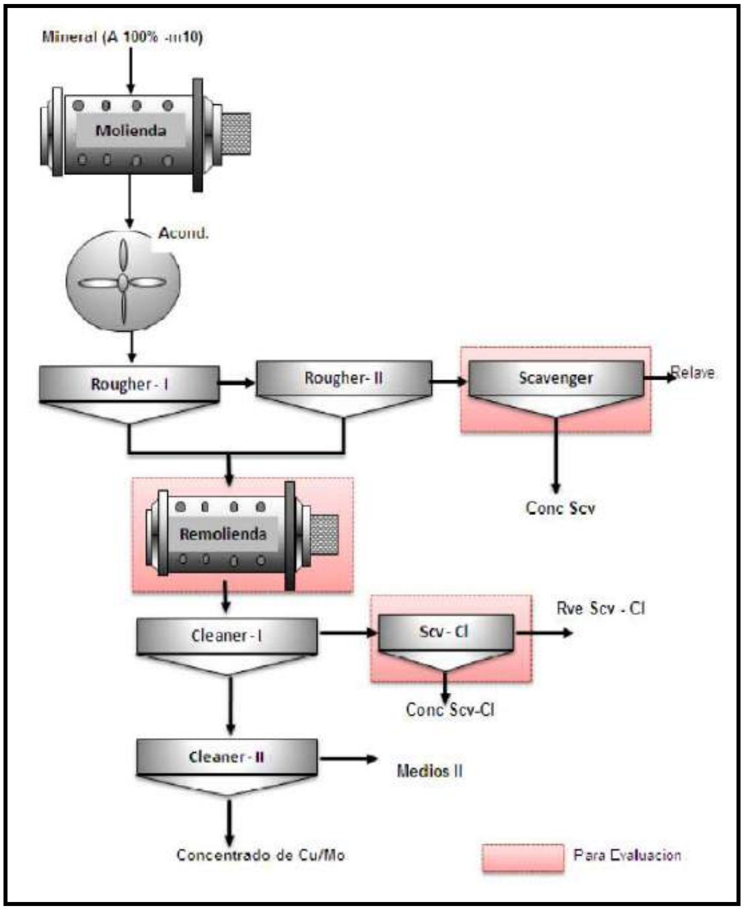


Figure 13-1 Batch Open Circuit Flotation Flowsheet

The results indicate that the copper and molybdenum sulphides are amenable to flotation processing. The test results are summarized below:

- For Sample A, approximately 90.5% of the copper and 82.6% of the molybdenum were recovered to the bulk rougher flotation concentrate. On average the cleaner concentrate contained 25.6% copper and 8,354 ppm molybdenum. The copper and molybdenum recoveries to the cleaner concentrate were 81.1% and 72.8%, respectively.
- For Sample B, the copper and molybdenum reporting to the bulk rougher flotation concentrate were 85.1% and 81.6%, respectively. On average the bulk cleaner concentrate contained 43.1% copper and 4,918 ppm molybdenum. The copper and molybdenum recoveies to the cleaner concentrate were reduced to 72.0% and 57.3%, respectively.

13.5 Bulk Flotation Locked Cycle Tests

Two different flowsheets were tested using locked cycle test procedures to investigate metallurgical responses of the two samples:

- A One Tailings Flowsheet (Flowsheet One): recirculating the first cleaner tailings to the rougher flotation feed (Figure 13-2)

- A Two Tailings Flowsheet (Flowsheet Two): adding cleaner scavenger flotation; the cleaner scavenger flotation tailings were directly discharged as the final tailings while the cleaner scavenger flotation concentrate was returned to the first cleaner flotation feed (Figure 13-3).

The locked cycle test results produced by Flowsheet One are shown in Table 13-3 and Table 13-4.

Table 13-3 One Tailings Flowsheet (Sample A; Locked Cycle Test 1)

Products	Grade				Recovery				
	Cu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	Mass (%)	Cu (%)	Mo (%)	Au (%)	Ag (%)
Cu-Mo Bulk Concentrate	20.00	6,032	1.12	34.3	1.24	85.3	77.6	73.1	46.3
Rougher Tailings	0.04	22	0.01	0.5	98.76	14.7	22.4	26.9	53.7
Calculated Head	0.29	97	0.02	0.9	100.00	100.0	100.0	100.0	100.0

Table 13-4 One Tailings Flowsheet (Sample B; Locked Cycle Test 1)

Products	Grade				Recovery				
	Cu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	Mass (%)	Cu (%)	Mo (%)	Au (%)	Ag (%)
Cu-Mo Bulk Concentrate	36.3	6,527	0.51	37.9	1.25	79.4	83.3	54.2	49.0
Tailings	0.12	17	0.01	0.5	98.75	20.6	16.7	45.8	51.0
Calculated Head	0.57	98	0.01	1.0	100.00	100.0	100.0	100.0	100.0

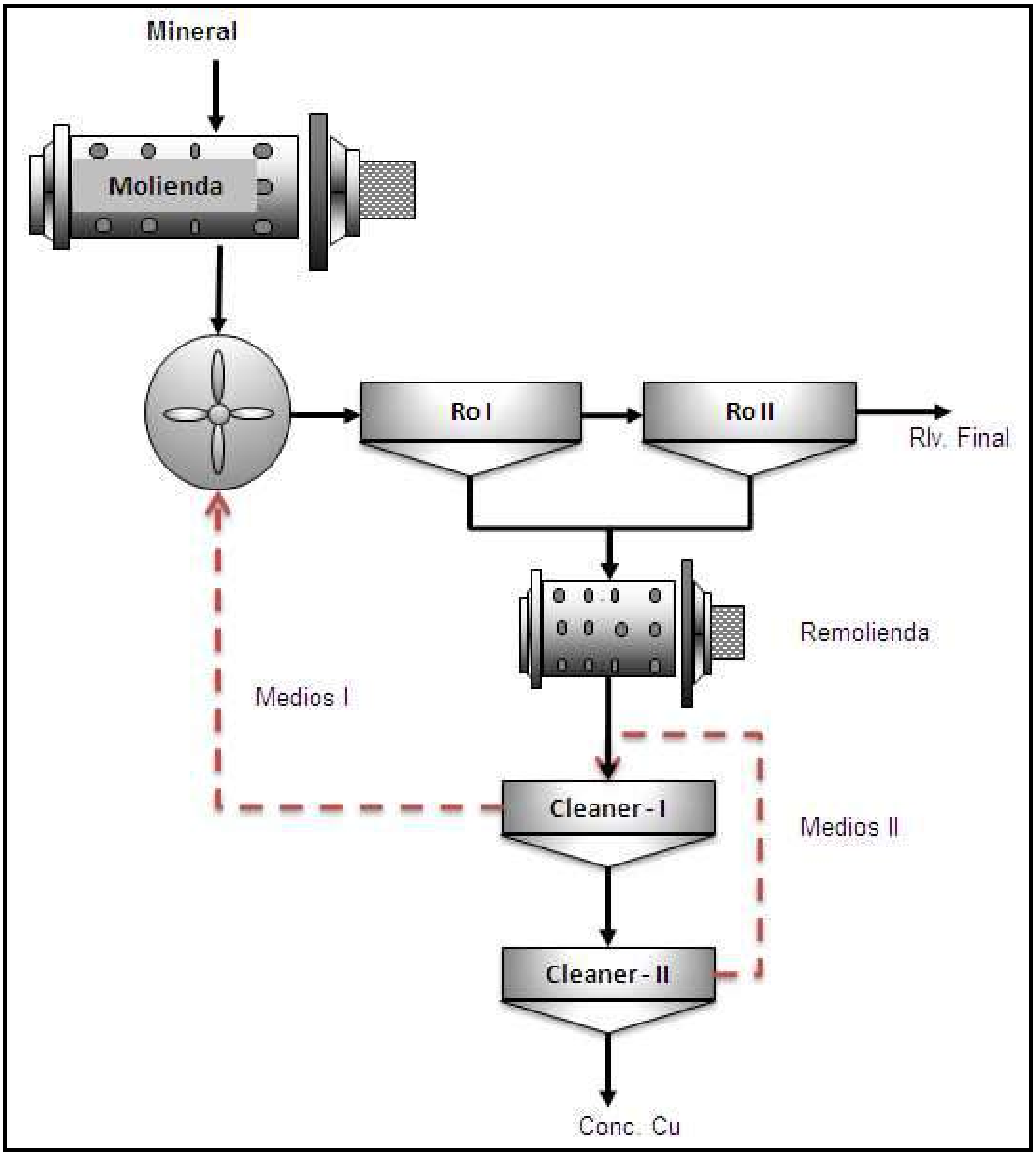


Figure 13-2 Locked Cycle Flotation - One Tailings Flowsheet (Flowsheet One)

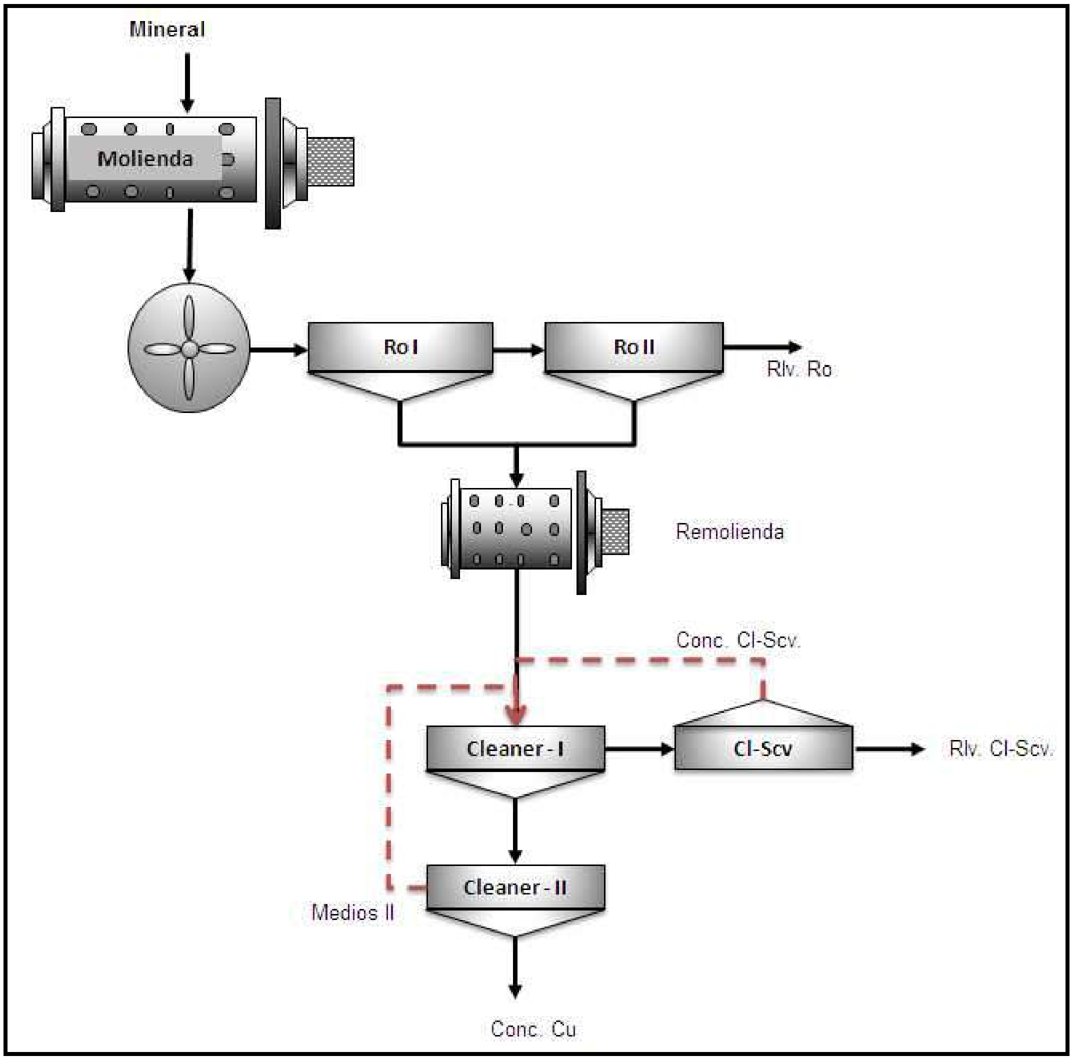


Figure 13-3 Locked Cycle Flotation - Two Tailings Flowsheet (Flowsheet Two)

The results indicate that Sample A produced a higher copper recovery, averaging 85.3%, compared to Sample B from which only 79.4% of the copper was recovered to the bulk concentrate. However, the concentrate grade produced from Sample B contained 36.3% copper, much higher than the 20% copper generated from Sample A. Molybdenum recoveries from the samples were 77.6% for Sample A and 83.3% for Sample B.

Compared to Flowsheet One, copper grades of the bulk concentrates produced from the alternative flowsheet (Flowsheet Two) were higher, especially for Sample A. However, the copper and molybdenum recoveries for the alternative flowsheet were lower. The test results are shown in Table 13-5 and Table 13-6.

Table 13-5 Two Tailings Flowsheet (Sample A; Locked Cycle Test 2)

Products	Concentrate Grade				Recovery				
	Cu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	Mass (%)	Cu (%)	Mo (%)	Au (%)	Ag (%)
Cu Concentrate	25.20	8,044	0.55	42.1	0.96	81.5	75.8	35.7	41.4
Cleaner Scavenger Tailings	0.42	87	0.10	1.8	5.78	8.3	4.9	32.6	10.8
Rougher Tailings	0.03	21	0.01	0.5	93.26	10.2	19.3	31.7	47.8
Calculated Head	0.30	102	0.02	1.0	100.00	100.0	100.0	100.0	100.0

Table 13-6 Two Tailings Flowsheet (Sample B; Locked Cycle Test 2)

Products	Concentrate Grade				Recovery				
	Cu (%)	Mo (ppm)	Au (ppm)	Ag (ppm)	Mass (%)	Cu (%)	Mo (%)	Au (%)	Ag (%)
Cu Concentrate	37.50	6,058	0.39	38.2	1.20	78.3	82.1	44.4	47.1
Cleaner Scavenger Tailings	0.83	112	0.03	1.0	4.47	6.4	5.7	10.6	4.5
Tailings	0.09	11	0.01	0.5	94.33	15.3	12.2	45.0	48.4
Calculated Head	0.58	89	0.01	1.0	100.00	100.0	100.0	100.0	100.0

These locked cycle test results indicated that gold and silver grades of the bulk concentrates were low.

The bulk concentrates produced from the locked cycle tests were subjected to multi-element analysis. The assay results, as shown in Table 13-7, indicate that the impurity levels in the copper concentrates produced from the mineralization should not attract smelting penalties as defined by most smelters.

Table 13-7 Bulk Concentrate Multi-Element Analysis Results

Element	Unit	Sample A Flowsheet One	Sample A Flowsheet Two	Sample B Flowsheet One	Sample B Flowsheet Two
Ag	ppm	33.4	40.3	36.8	35.5
Al	%	2.74	2.4	1.64	1.51
As	ppm	366	421	465	473
Ba	ppm	103	100	100	96
Be	ppm	0.6	0.6	<0.5	<0.5
Bi	ppm	<5	<5	<5	<5
Ca	%	0.7	0.67	0.57	0.5
Cd	ppm	8	10	10	10
Co	ppm	85	38	72	75
Cr	ppm	496	335	1,070	827
Cu	ppm	>10,000	>10,000	>10,000	>10,000
Fe	%	>15.0	>15.0	>15.0	>15.0
Ga	ppm	<10	<10	<10	<10
K	%	1.39	1.25	0.85	0.78
La	ppm	12.1	8.5	13	11.8
Mg	ppm	0.17	0.17	0.09	0.08
Mn	ppm	418	467	173	138
Mo	ppm	6,337	9,081	6,828	7,041
Na	%	0.18	0.19	0.07	0.07
Nb	ppm	<1	<1	<1	<1
Ni	ppm	250	148	410	439
P	%	<0.01	<0.01	<0.01	<0.01
Pb	ppm	177	219	125	115
S	%	>10.0	>10.0	>10.0	>10.0
Sb	ppm	65	82	36	34
Sc	ppm	7.1	7.4	8.3	8.1
Sn	ppm	22	28	19	21
Sr	ppm	48	46.9	41.9	39.5
Ti	%	0.1	0.09	0.08	0.08
Tl	ppm	<2	<2	<2	<2
V	ppm	49	46	34	30
W	ppm	<10	<10	<10	<10
Y	ppm	8.1	6.6	6.5	6
Zn	ppm	1,200	1,380	1,880	1,840
Zr	ppm	24.3	18.5	12.2	8.7

13.6 Copper-Molybdenum Separation Flotation

Preliminary test work was conducted to investigate molybdenum separation from the copper-molybdenum bulk concentrates that were produced from batch open circuit tests. The flotation separation procedure comprised floating molybdenum and suppressing copper minerals using sodium hydrosulphide (NaSH). The molybdenum rougher flotation concentrates were upgraded using three stages of cleaner flotation; excluding Test Sep No.01 in which the molybdenum flotation concentrates was upgraded by four stages of cleaner flotation. The test results are summarized in Table 13-8. The test results showed that the molybdenum concentrates contained approximately 32% molybdenum for Sample A and 40% molybdenum for Sample B. The results indicated that the molybdenite could be separated from the copper-molybdenum bulk concentrate by a conventional flotation process. However, further test work is required to improve molybdenum concentrate grade including optimizing reagent regime and regrinding arrangement for intermediate molybdenum concentrate(s).

Table 13-8 Copper and Molybdenum Separation Test Results

Sample	Test ID	Product	Grade (%)		Recovery (%)	
			Cu	Mo	Cu	Mo
Sample A	Sep No 01	Mo Cleaner Concentrate	3.57	32.2	0.3	74.0
		Mo Rougher Concentrate	22.9	2.96	24.3	95.9
		Mo Rougher Tailings	25.4	0.05	75.7	4.1
	Sep No 02	Mo Cleaner Concentrate	4.2	32.5	0.4	93.8
		Mo Rougher Concentrate	21.0	6.28	11.4	98.8
		Mo Rougher Tailings	24.9	0.01	88.6	1.2
Sample B	Sep No 03	Mo Cleaner Concentrate	3.7	38.6	0.1	85.4
		Mo Rougher Concentrate	30.4	4.06	11.9	98.5
		Mo Rougher Tailings	42.3	0.01	88.1	1.5
	Sep No 04	Mo Cleaner Concentrate	3.3	41.5	0.1	84.4
		Mo Rougher Concentrate	31.1	3.47	16.6	98.9
		Mo Rougher Tailings	42.8	0.01	83.4	1.1

13.6.1 Related Test Work

13.6.1.1 Acid-Base Accounting Tests

Certimin conducted acid-base accounting (ABA) tests on samples of flotation tailings generated from the locked cycle tests. As shown in Table 13-9, the test results indicated that the tailings from the two mineralization samples may present a high-acid generation potential.

Table 13-9 ABA Test Results (Flotation Tailings)

Sample – Flotation Tailings		AP (kg $CaCO_3$/t)	Pulp (pH)	NP (kg $CaCO_3$/t)	NP/AP	NNP (kg $CaCO_3$/t)	S(t)%	$S(SO_4^{2-})$(%)	S^{2-}(%)
Sample A	Locked Cycle Test 1	4.48	8.46	2.03	0.45	-2.45	0.20	0.05	0.14
	Locked Cycle Test 2	2.33	8.58	2.13	0.92	-0.19	0.11	0.03	0.07
Sample B	Locked Cycle Test 1	10.8	8.63	1.92	0.18	-8.88	0.40	0.05	0.35
	Locked Cycle Test 2	2.73	8.43	1.57	0.58	-1.16	0.12	0.03	0.09

Note: AP – potential acid generation; NP – neutralization potential; NNP – net neutralization potential; S(t) – total sulphur; S (SO_4^{2-}) – sulphate sulphur; S (S^{2-}) – sulphide sulphur

13.7 Solubility Tests

The soluble content in the head samples was determined to be 0.839 kg/t for Sample A and 0.755 kg/t for Sample B. It appeared that the solubilization did not have a negative effect on the process of flotation.

13.8 Certimin Summary

The preliminary flotation tests indicated that the mineralization was amenable to conventional flotation process. The mineralization from the primary sulphide zone will produce a higher flotation copper recovery, compared with the mineralization from the supergene zone. However, on average copper concentrate from the supergene zone mineralization will have a much higher copper grade compared with the mineralization from the primary sulphide zone. Molybdenum from both mineralization zones responded well to the test procedures. The impurity levels of the copper concentrates should not attract smelting penalties as defined by most smelters.

The acid generating potential of Sample B is particularly advantageous for a proposed bio-leach process for the treatment of supergene material.

If a mill and concentrator were to be considered as part of a later phase development or part of an integrated treatment facility a comprehensive testwork and development program on primary zone material will be required. This will likely include,

- Definition of more detailed lithological and mineralogical domains
- Identification and selection of metallurgical variability samples
- Advanced grinding tests, Bond, JK, SPI, SMC, MacPherson
- Grinding variability testwork
- Flotation optimisation testwork
- Flotation variability testwork
- Locked cycle flotation testwork
- Concentrate and tailings filtration tests
- Concentrate and tailings thickening tests
- Flotation pilot testwork
- Development of geometallurgical mine development plan

Market studies should also be conducted to investigate smelting terms, especially the terms for gold and silver because on average low gold and silver contents in copper concentrates are anticipated.

13.9 Aminpro, 2018

As an alternative to a conventional mill and concentrator for the treatment of both primary and secondary materials a process based on leaching secondary sulphides alone was considered a viable alternative. This led to a column leach program, together with associated mineralogical and bottle roll leach testwork, being implemented at Aminpro Laboratories, Lima, Peru during March 2018. This testwork program was aimed at assessing the amenability of Antilla to conventional acid heap leaching as well as a simulated bioleach process. It is generally accepted that in leaching of secondary copper sulphide minerals the primary oxidative pathway is via a ferric iron leaching route, the role of bacteria being to re-oxidize the ferrous iron generated back to ferric iron oxidant. Consequently, it is possible to simulate the process by instituting a ferric leaching cycle using chemical oxidants as opposed to bacteria. This saves on the time required to identify, isolate, adapt and generate a suitable bacterial consortium which may be deferred to later more substantive studies. The following sections summarize the results achieved to date; however, the column leach program is still ongoing and will only be finalized by the end of September 2018.

13.9.1 Sample Identification and Selection

The bulk composite sample selected for the AMINPRO Laboratory testwork program was based on quarter core samples selected by Panoro from drillhole intervals representative of the Antilla supergene mineralized zone. The total mass of combined core was 1,000 kg and was generally representative of the first 5 years of mine production based on the mine plan outlined in an earlier MMTS scoping study. Figure 13-4 shows the location and spatial distribution of the selected drillhole intervals within the conceptual pit shell. Table 13-10 shows the selected drillhole intervals grouped by benches of 12 m height.

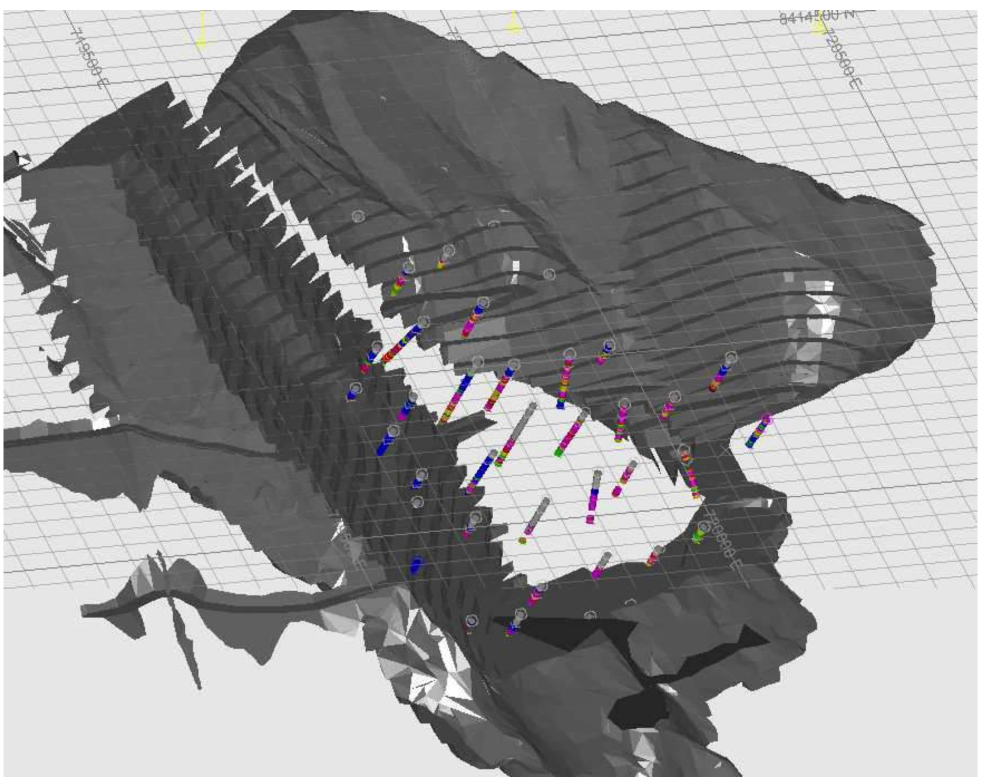


Figure 13-4 Drillhole interval location and spatial distribution

Table 13-10 Drillhole Intervals by Benches Indicative of the First 5 Years of Mine Production

Bench	Hole	From	To	Length	Average Cu%	Bench	Hole	From	To	Length	Average Cu%
3472	ANT-36-08	58	66	8	0.30	3592	ANT-20-08	122	135.2	13.2	0.41
	ANT-66-08	14.3	28	13.7	0.81		ANT-24-08	25.55	39	13.45	0.92
	ANT-73-10	50.3	56	5.7	0.10		ANT-26-08	46	57.6	11.6	0.26
3484	ANT-01-03	66	68	2	0.37		ANT-29-08	2.1	10.15	8.05	0.97
	ANT-36-08	44	58	14	0.60		ANT-46-08	92	105	13	0.47
	ANT-66-08	13	14.3	1.3	0.41		ANT-48-08	30.8	32	1.2	0.25
	ANT-72-10	60	65	5	0.38		ANT-68-10	102	105.65	3.65	0.77
	ANT-73-10	36	50.3	14.3	0.99		ANT-69A-10	79.8	94.4	14.6	0.91
	ANT-76-10	34	36	2	0.39		ANT-70-10	48	62	14	0.41
3496	ANT-01-03	54	66	12	0.57		ANT-71-10	80.7	87.4	6.7	1.82
	ANT-27-08	4	18	14	0.21		ANT-84-10	34	46.4	12.4	0.72
	ANT-28-08	40	46	6	0.70	3604	ANT-20-08	108	122	14	0.38
	ANT-36-08	30	44	14	0.56		ANT-22-08	102	106.6	4.6	0.20
	ANT-72-10	45.1	60	14.9	0.08		ANT-24-08	17.3	25.55	8.25	0.96
	ANT-73-10	22	36	14	0.56		ANT-26-08	33.1	46	12.9	0.70
	ANT-76-10	20	34	14	0.53		ANT-46-08	78	92	14	0.56
3508	ANT-01-03	40	54	14	1.24		ANT-68-10	88	102	14	0.20
	ANT-27-08	2.55	4	1.45	0.09		ANT-69A-10	68.5	79.8	11.3	0.55
	ANT-28-08	26	40	14	1.66		ANT-70-10	37.6	48	10.4	0.28
	ANT-35-08	32	46.3	14.3	0.42		ANT-71-10	66	80.7	14.7	0.54
	ANT-39-08	69.9	83	13.1	0.50		ANT-84-10	20	34	14	0.82
	ANT-72-10	31.5	45.1	13.6	0.33	3616	ANT-20-08	94	108	14	0.28
	ANT-73-10	8	22	14	0.76		ANT-22-08	88	102	14	1.26
	ANT-76-10	15.6	20	4.4	0.74		ANT-26-08	20	33.1	13.1	1.59
3520	ANT-01-03	28	40	12	0.70		ANT-46-08	64	78	14	0.20
	ANT-35-08	18	32	14	0.19		ANT-68-10	73.85	88	14.15	0.28
	ANT-38C-08	72	81.2	9.2	0.76		ANT-71-10	52.6	66	13.4	0.26
	ANT-39-08	56	69.9	13.9	0.48		ANT-84-10	12	20	8	0.30
	ANT-48-08	101.4	104.2	2.8	0.95	3628	ANT-20-08	80.5	94	13.5	0.51
	ANT-72-10	18	31.5	13.5	0.72		ANT-22-08	74	88	14	0.66
	ANT-73-10	7.4	8	0.6	0.01		ANT-26-08	14.6	20	5.4	0.37
	ANT-74-10	52.9	66	13.1	1.28		ANT-46-08	49.8	64	14.2	0.26
	ANT-81-10	82	88	6	0.12		ANT-68-10	60	73.85	13.85	0.86
	ANT-82A-10	82	86	4	0.24		ANT-71-10	39.5	52.6	13.1	0.91
3544	ANT-01-03	3.45	14	10.55	0.64	3640	ANT-05-03	72	82	10	0.65
	ANT-29-08	46	59	13	1.00		ANT-07-03	76	80	4	0.92
	ANT-38C-08	48	60	12	0.89		ANT-20-08	68	80.5	12.5	0.75
	ANT-39-08	28	42	14	0.51		ANT-22-08	60	74	14	0.61
	ANT-48-08	74	88	14	0.42		ANT-68-10	46	60	14	1.36
	ANT-49-08	44	58	14	1.05		ANT-71-10	24	39.5	15.5	0.47
	ANT-74-10	24	38.9	14.9	0.50	3652	ANT-05-03	58.4	72	13.6	0.62
	ANT-81-10	54.2	68	13.8	0.31		ANT-07-03	62	76	14	0.59
	ANT-82A-10	54	68	14	0.61		ANT-20-08	58	68	10	0.83
	ANT-84-10	88	90.1	2.1	0.51		ANT-22-08	54	60	6	0.35
3556	ANT-24-08	67.5	70.3	2.8	0.33		ANT-68-10	35	46	11	0.83
	ANT-29-08	35	46	11	0.40		ANT-71-10	19.9	24	4.1	0.50
	ANT-34-08	86	94	8	0.23	3664	ANT-05-03	44	58.4	14.4	0.99
	ANT-38C-08	36	48	12	1.42		ANT-07-03	48	62	14	1.29
	ANT-39-08	14.3	28	13.7	0.56		ANT-30-08	77.6	92	14.4	0.71
	ANT-48-08	60	74	14	0.35		ANT-62-08	92	100	8	0.42
	ANT-49-08	30	44	14	1.28	3676	ANT-05-03	32	44	12	0.52
	ANT-65-08	72	76	4	0.37		ANT-07-03	33.8	48	14.2	0.23
	ANT-70-10	90	104	14	0.47		ANT-30-08	64	77.6	13.6	1.43
	ANT-74-10	11.6	24	12.4	1.31		ANT-62-08	76	92	16	0.41
	ANT-81-10	41.2	54.2	13	0.83	3688	ANT-05-03	17.7	32	14.3	0.59
	ANT-82A-10	50.15	54	3.85	0.19		ANT-07-03	20	33.8	13.8	0.68
	ANT-84-10	74	88	14	0.35		ANT-25-08	56	66	10	0.41
3568	ANT-24-08	53.1	67.5	14.4	0.68		ANT-30-08	50	64	14	0.44
	ANT-29-08	22	35	13	0.71		ANT-62-08	59.8	76	16.2	0.56
	ANT-34-08	72	86	14	0.50	3700	ANT-07-03	17.2	20	2.8	0.23
	ANT-38C-08	34.9	36	1.1	0.63		ANT-25-08	42	56	14	0.62
	ANT-39-08	2.1	14.3	12.2	0.49		ANT-30-08	36	50	14	0.64
	ANT-48-08	46	60	14	0.10		ANT-40-08	54	62.8	8.8	0.95
	ANT-49-08	26	30	4	0.32		ANT-77-10	48	60	12	0.23
	ANT-65-08	57.9	72	14.1	0.73	3712	ANT-25-08	28	42	14	0.49
	ANT-69A-10	108	117.9	9.9	0.56		ANT-30-08	32.1	36	3.9	0.39
	ANT-70-10	76	90	14	0.48		ANT-40-08	40	54	14	0.25
	ANT-81-10	33.6	41.2	7.6	0.70		ANT-62-08	36.6	44	7.4	0.27
	ANT-84-10	60	74	14	0.39		ANT-77-10	34	48	14	0.37
3580	ANT-24-08	39	53.1	14.1	0.54	3724	ANT-25-08	25.25	28	2.75	0.32
	ANT-26-08	57.6	62	4.4	0.72		ANT-40-08	26	40	14	0.46
	ANT-29-08	10.15	22	11.85	1.32		ANT-77-10	22	34	12	0.23
	ANT-34-08	59.25	72	12.75	0.42	3736	ANT-40-08	12	26	14	0.54
	ANT-65-08	46.6	57.9	11.3	0.42		ANT-79-10	46.5	52	5.5	0.52
	ANT-69A-10	94.4	108	13.6	0.28						
	ANT-70-10	62	76	14	0.59						
	ANT-84-10	46.4	60	13.6	0.66						

Quarter core intervals representing each bench were shipped to Aminpro Laboratories where each bench sample was homogenized, prepared and split according to the requirements of the subsequent laboratory metallurgical testwork program. A reject sample of 750 kg is saved in Aminpro Laboratories in the event any future testwork or repeat tests are required. Results from the testwork program are summarized in the following sections.

13.10 Quantitative Mineralogy

A quantitative mineralogical investigation program was conducted using QEMSCAN on head samples split from the Antilla bulk sample. This program included Modal Mineralogy, Grain Size Distribution Data, Mineralogical Associations and Locking, Liberation, Classified and BSE images. The essential mineralogical characteristics of the ore is shown in Table 13-11 and Table 13-12 and Figure 13-5 to Figure 13-7 inclusive..

Table 13-11 Modal Mineralogical Analysis

		Mineral Wt%	
		Client ID	Head - Antilla
		ActLabs ID	1
Sulphides	Cu Carriers	Chalcocite	0.92
		Chalcopyrite	0.41
		Enargite	0.05
		Sphalerite	<0.01
		Molybdenite	0.02
		Galena	<0.01
		Arsenopyrite	<0.01
		Pyrite	1.98
Silicates		Zircon	0.07
		Quartz	77.57
		K-Feldspar	9.56
		Plagioclase	0.26
		Muscovite	4.60
		Biotite	0.39
		Kaolinite	1.75
		Silicate low Al	1.37
		Smectite	0.27
		Serpentine	<0.01
Oxides & Hydroxides		Fe Oxy/Hydroxide	0.07
		Rutile	0.23
Phosphate		Monazite	0.01
		Xenotime	<0.01
		Al Phosphate	0.02
Others		Others	0.42
		Total	100

The Modal Analysis shows the sulphides chalcocite, chalcopyrite and enargite to be the copper carriers. Of note is that 81.6 % of the copper is associated with chalcocite which is readily leachable, 2.7 % with enargite which is substantially leachable and 15.8 % with chalcopyrite, which is only partially leachable. Pyrite is a significant source of both ferric iron and acid, essential to any potential bioleach process. Of note are soluble iron oxides, a potential additional source of iron. Feldspars and phosphates are sources of essential bacterial nutrients such as potassium and phosphorous. Nitrogen requirements will be largely met by blasting residues, and CO_2 from the air. Arsenic associated with enargite and arsenopyrite would normally be fixed in a bioleach process as scorodite. The ultimate destiny of arsenic in the process will be addressed in subsequent studies.

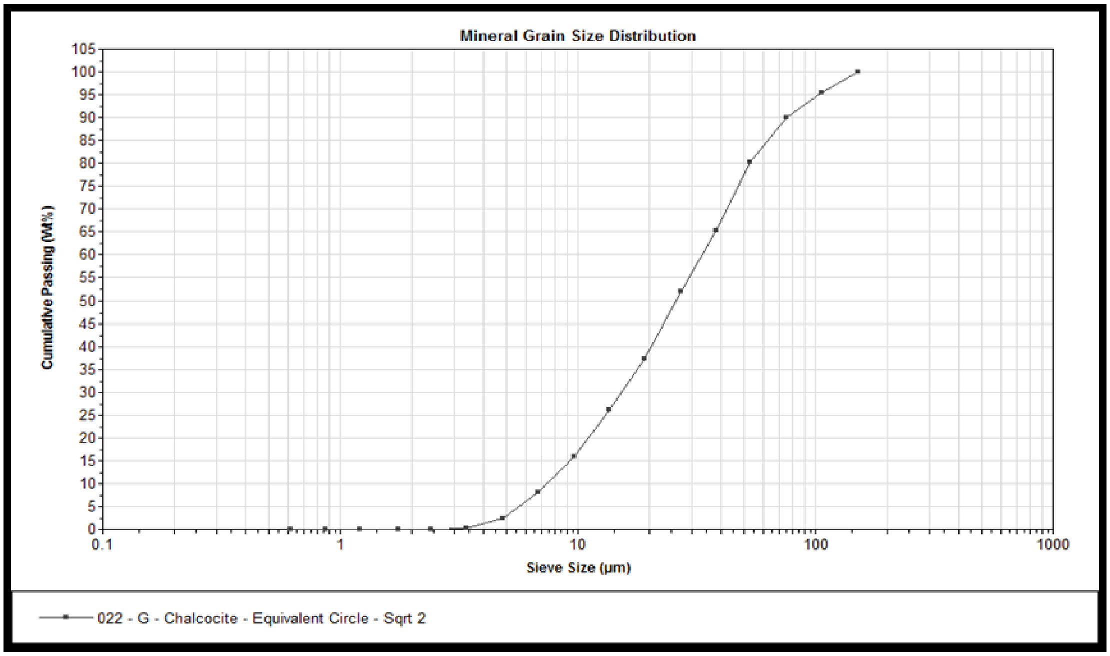


Figure 13-5 Chalcocite grain size distribution

The mineral grain size distribution data shows the chalcopyrite to be characteristically finer than both the chalcocite and pyrite with a characteristic size (D_{50}) of approximately 15 µm. This is potentially of more significance in processing of primary hypogene material rather than supergene. Chalcocite and pyrite particles are much coarser with characteristic sizes of 25 µm and 35 µm respectively.

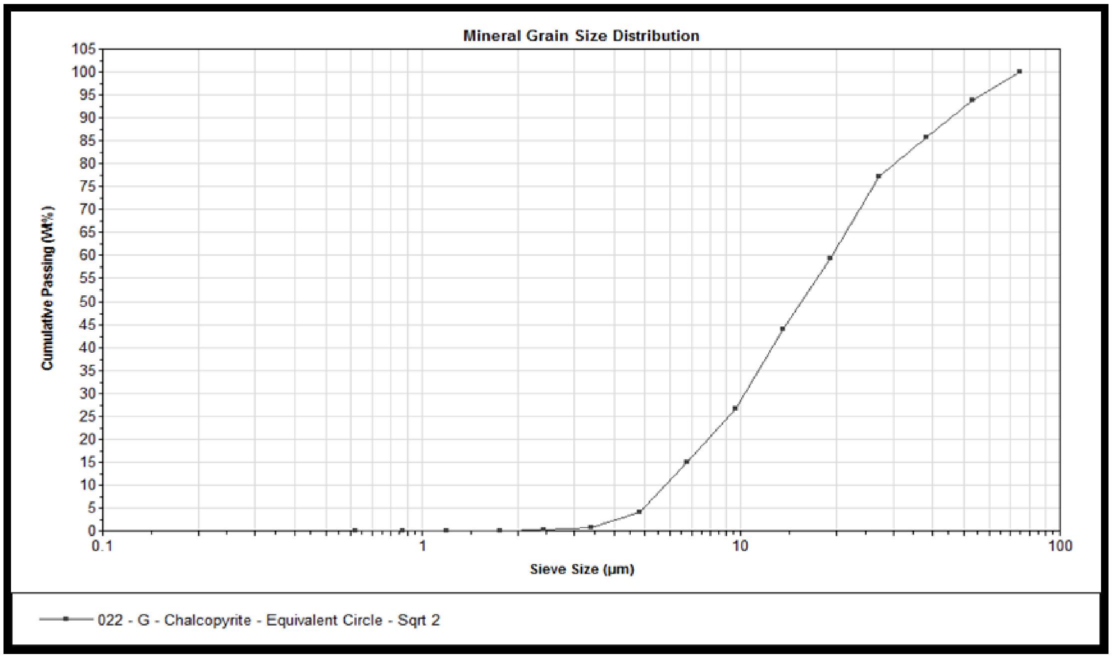


Figure 13-6 Chalcopyrite grain size distribution

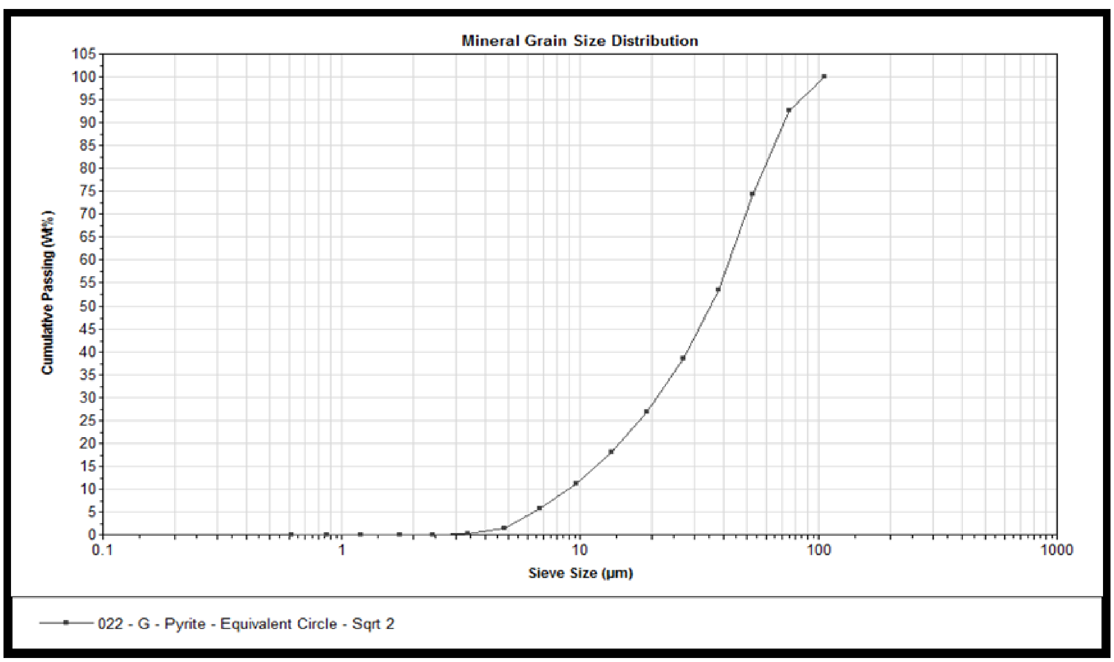


Figure 13-7 Chalcocite grain size distribution

Table 13-12 Mineral Summary Liberation Data

Chalcocite Liberation by Composition (Wt%)						
Sample		0% < x <= 20%	20% < x <= 50%	50% < x <= 80%	80% < x <= 95%	95% < x <= 100%
1	Head – Antilla	1.93	4.61	6.78	19.23	**67.44**

Chalcopyrite Liberation by Composition (Wt%)						
Sample		0% < x <= 20%	20% < x <= 50%	50% < x <= 80%	80% < x <= 95%	95% < x <= 100%
1	Head – Antilla	9.59	6.21	15.33	11.61	**57.25**

Pyrite Liberation by Composition (Wt%)						
Sample		0% < x <= 20%	20% < x <= 50%	50% < x <= 80%	80% < x <= 95%	95% < x <= 100%
1	Head – Antilla	0.41	1.3	1.46	5.99	**90.84**

Liberation classes are defined as,

- Free: A mineral with > 95 % surface exposure
- Liberated: A mineral with ≥80 % but < 95 % surface exposure
- Middlings: A mineral with ≥ 50 % but < 80 % surface exposure
- Sub-middlings: A mineral with ≥ 20 % but < 50 % surface exposure
- Locked: A mineral with < 20 % surface exposure

The liberation data indicates that both pyrite and chalcocite will be substantially available for both oxidation and extraction. Although chalcopyrite is more locked the main limitations in bioleach extraction of copper from chalcopyrite are chemical, kinetic and thermodynamic.

13.11 Head Assays

The current testwork program is focussed copper extraction and so chemical analysis has been limited to copper, iron and sulphur head assays. The assays for the bulk sample of the current testwork program are shown in Table 13-13 below.

Table 13-13 Chemical Analysis and Mineral Head Assays (Assay by Sequential Copper Analysis)

CODIGO AMINPRO	CODIGOAHK	Cu ASS (%)	Cu residual (%)	Cu Sol Aci (%)	Cu Sol CN (%)	Fe (%)	S (%)	RATION CuSA+CuSCN/CuT
MR17004PAMC2-1-1-025	158902/1	0.65	0.06	0.07	0.50	1.21	1.33	76.99

**Note according to the chemical assays, the mineral of Antilla can be typified as a mineral clearly Secondary Sulphide*

The assay data indicates that copper extractions of up to 77 % are potentially achievable. Of note is that 99% of the mineral for leaching in the mine plan is composed of secondary sulphide.

13.12 Bottle Roll Tests

13.12.1 Sulphuric Acid

An initial sulphuric acid extraction test was carried out on a sample of head ore ground to 10# at 40 % solids, 50 g/l H_2SO_4 and for an initial period of 72 hours. The test was subsequently extended to 160 hours. Cumulative copper extraction after 72 h and 160 h based on solids assays was 30.93 % and 46.26 % respectively. Similarly, iron extractions were 4.12 % and 5.05 % for the same periods. Sulphuric acid consumption was 16.54 kg/t. Even after 160 hours the sample continued to leach but kinetics were very slow, as acid oxidation of chalcocite is limited by oxygen mass transfer, which is restricted in bottle roll tests. The metal extraction curves are shown in Figure 13-8 below.

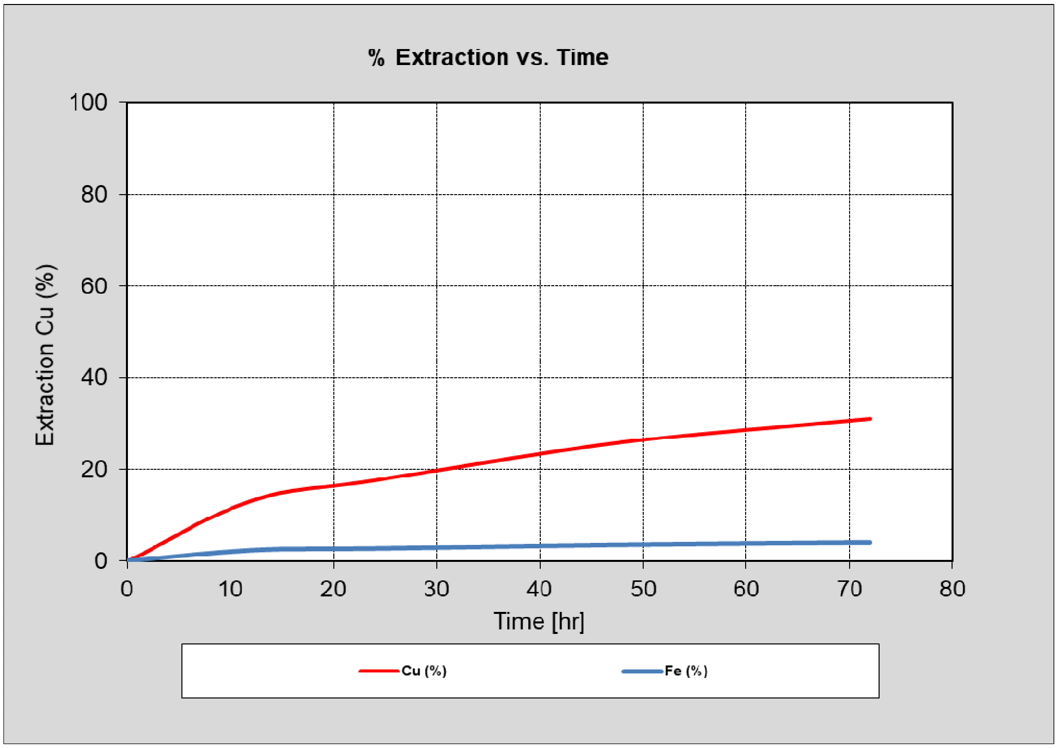


Figure 13-8 Sulphuric Acid Bottle Roll Extraction Curves

Subsequently an additional series of test were carried out at a range of acid concentrations from 5 g/l to 50 g/l H_2SO_4. Increasing the acid concentration above 20 g/l had only a marginal effect on metals extraction, Table 13-14 and Figure 13-9 below. Acid consumptions during these tests ranged from 6.39 kg/t and 9.14 kg/t.

Table 13-14 Summary of Supplementary Acid Bottle Roll Tests

Sample	H2SO4	pH	mV	Time (hr)	Cu ex	Fe ex	Acid Consumption (Kg/t)
1	5.0	1.3	387.0	0	0.00	0.00	0.00
	5.0	1.4	315.7	12	11.18	1.82	3.65
	5.0	1.2	357.7	24	14.96	2.04	1.37
	5.0	1.1	336.4	48	22.45	2.36	0.76
	5.0	1.0	363.2	72	26.51	2.65	0.61
2	10.0	1.0	355.9	0	0.00	0.00	0.00
	10.0	1.0	337.4	12	11.80	2.00	4.11
	10.0	1.0	352.4	24	16.08	2.23	1.52
	10.0	0.7	337.3	48	24.01	2.79	0.61
	10.0	0.8	364.1	72	28.35	3.11	0.30
3	20.0	0.8	344.3	0	0.00	0.00	0.00
	20.0	0.7	354.8	12	13.13	2.35	5.18
	20.0	0.7	351.3	24	15.86	2.64	1.52
	20.0	0.4	326.7	48	23.14	3.28	1.37
	20.0	0.6	375.4	72	30.13	4.16	0.91
4	50.0	0.5	333.3	0	0.00	0.00	0.00
	50.0	0.4	373.3	12	13.15	2.39	6.39
	50.0	0.5	347.2	24	17.60	2.78	0.61
	50.0	0.1	322.2	48	25.86	3.60	1.22
	50.0	0.3	367.7	72	30.93	4.12	0.91

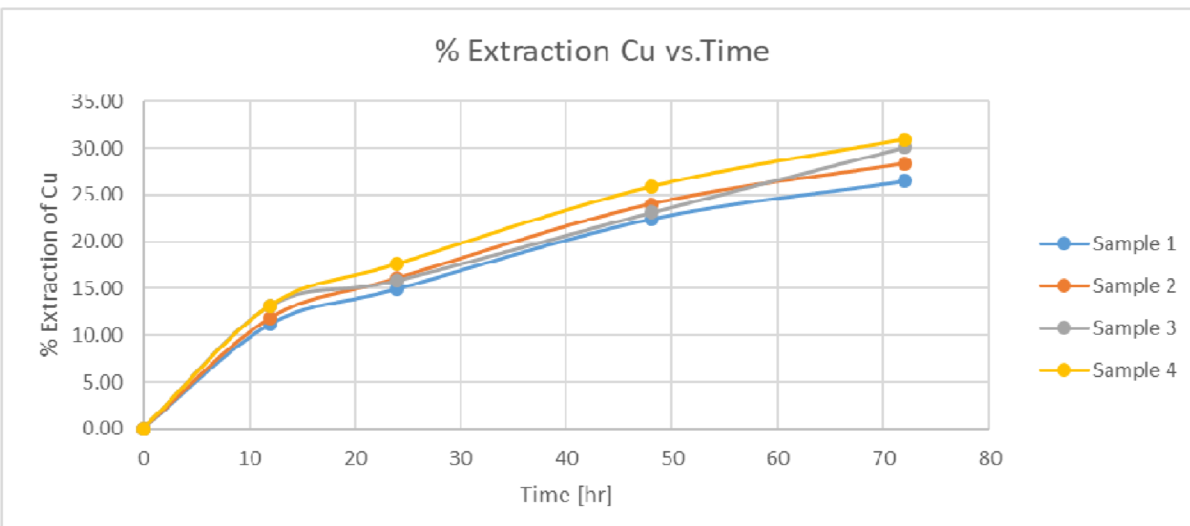


Figure 13-9 Supplementary Cu Acid Extraction Tests

13.13 Ferric Leach Tests

The objective of this series of tests was to evaluate the effect of ferric iron on leaching kinetics and metals extraction compared with acid leaching. A series of ferric copper extraction tests was conducted on samples of ore. The samples had a head grade of 0.7 % of Cu_T and 1.42 % Fe_T. The tests were conducted at a grind of 100% -10#, 1.5 L/S and 10g/l of H_2SO_4. The effect of ferric concentration was evaluated at concentrations of 1, 2, 5 and 10g/l. The leach extraction tests were conducted for 144 hours with periodic monitoring of acid consumption, pH, ORP and sampling of solutions at 12, 24, 48, 96, 120 hours as shown in Table 13-15 and Table 13-16.

Table 13-15 Ferric Leach Extraction Tests Conditions

Test	Sample	Grind [100% minus]	[%]	Solution	[H_2SO_4]	[Fe^{+3}]	Time
	weight [g]	[microns]	Solids	[ml]	[g/l]	[g/l]	[hr]
B-1	1000	1700	40	1500	10	1	144
B-2	1000	1700	40	1500	10	2	144
B-3	1000	1700	40	1500	10	5	144
B-4	1000	1700	40	1500	10	10	144

Table 13-16 below shows a summary of the bottle roll tests. Table 13-17 shows the extraction results and Table 13-18 shows extraction based on residues analyses. It can be observed that the highest extractions of Cu were obtained at concentrations of 5 g/l and 10 g/l. It is also apparent that the extraction of copper is a function of ferric concentration below 5 g/l of Fe^{3+}.

Table 13-16 Ferric Leach Test Results

Test	Time	pH	mV	Cu	Fe^{+3}	Rec. Cu	Acid consumption
	[hr]			[ppm]	[ppm]	[%]	[kg/t]
B-1	0	0.78	382	0	1000	0	0.0
	12	0.94	379	1095	1133	24	2.7
	24	0.86	374	1274	1091	29	2.4
	48	0.85	372	1480	1107	35	1.6
	72	0.79	370	1559	1078	38	1.5
	96	0.75	368	1716	1303	43	2.0
	120	0.74	367	1765	1018	45	1.8
	144	0.70	366	1869	1005	49	1.3
B-2	0	0.76	398	0	2000	0	0.0
	12	0.95	389	1456	1995	32	1.8
	24	0.86	380	1508	1948	35	1.6
	48	0.85	378	1629	1958	39	1.0
	72	0.76	378	1729	1871	42	1.0
	96	0.76	375	1871	1820	47	1.3
	120	0.73	371	2028	1766	52	1.2
	144	0.70	372	2159	1794	57	0.9
B-3	0	0.89	429	0	5000	0	0.0
	12	0.89	425	1822	4959	41	1.4
	24	0.79	422	1850	4852	43	1.3
	48	0.80	420	2070	4596	49	1.3
	72	0.75	416	2287	4380	56	0.2
	96	0.77	406	2525	4183	63	0.4
	120	0.76	399	2546	3836	66	0.8
	144	0.72	395	2658	3714	70	1.0
B-4	0	0.80	478	0	10000	0	0.0

Test	Time	pH	mV	Cu	Fe^{+3}	Rec. Cu	Acid consumption
	[hr]			[ppm]	[ppm]	[%]	[kg/t]
	12	0.85	474	1794	9921	40	1.3
	24	0.75	468	1902	9708	44	1.3
	48	0.76	460	2080	9417	50	1.3
	72	0.75	455	2283	8662	56	0.2
	96	0.75	446	2424	8268	61	0.5
	120	0.69	443	2473	7986	64	0.8
	144	0.68	436	2504	7702	67	0.8

Table 13-17 Ferric Leach Tests Copper Extractions Based on Solution Analysis

Test	Total Time	Acid consumption	Solution Rec. Cu
	[hr]	[kg/t]	[%]
B-1	144	13.3	49.3
B-2	144	8.8	56.9
B-3	144	6.3	70.5
B-4	144	6.2	66.9

Table 13-18 Ferric Leach Tests Copper Extractions Based on Residue Analysis

Test	Head	Residue	Rec. Cu in solids
	Cu [%]	Cu [%]	[%]
B-1	0.70	0.36	48.57
B-2	0.70	0.30	57.14
B-3	0.70	0.19	72.86
B-4	0.70	0.19	72.86

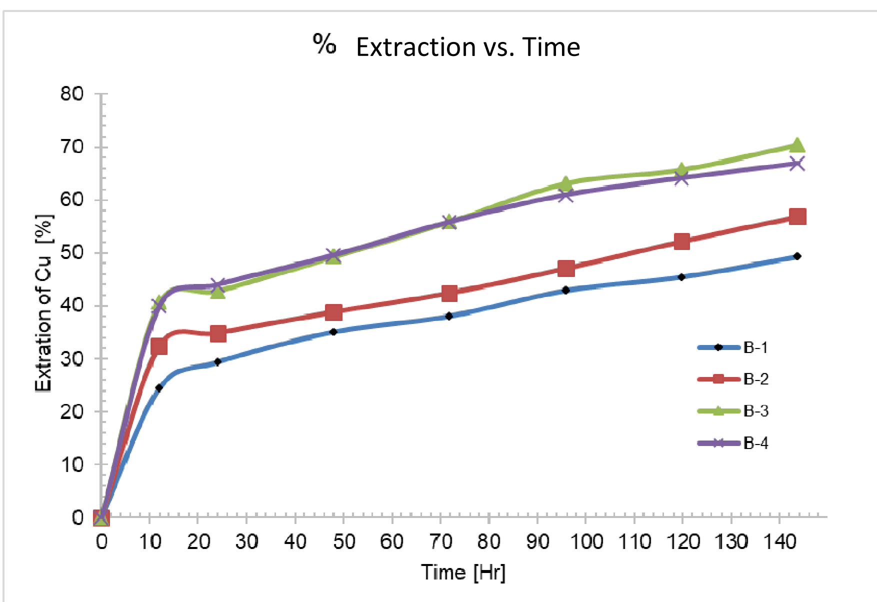


Figure 13-10 Ferric Leach Extraction Curves

The ferric leach tests indicated sulphuric acid consumption was higher at lower ferric concentrations and ranged between 13.3 kg/t and 6.2 kg/t. The tests also showed that kinetics was relatively slow but that copper continued to be extracted even after 144 hours under leach. Further, that Cu extractions and rate are maximised at ORP's above 420 mV and ferric concentrations of 5 g/l.

13.14 Column Leach Tests

A series of locked cycle column leach tests have been implemented to simulate a bioleach moderated ferric leach cycle. Four crush sizes are being evaluated namely 100 % minus 1", ¾", ½" and 3/8". The columns were initially acid cured and pH stabilised to a pH of 1.2 over a period of 5 days. The columns were then placed under a conventional sulphuric acid leach for a period of 18 days. Thereafter the columns were irrigated with a synthetic raffinate solution at a ferric concentration of 0.8 g/l. Leach kinetics were unacceptably slow at the low ferric concentration and resultant low ORP, as supported by the ferric leach bottle roll tests. The ferric recycle concentration in the synthetic raffinate was then increased to 5 g/l and the ORP in pregnant solution rose to +430 mV, after which there was an immediate and rapid response in leach kinetics, Figure 13-11.

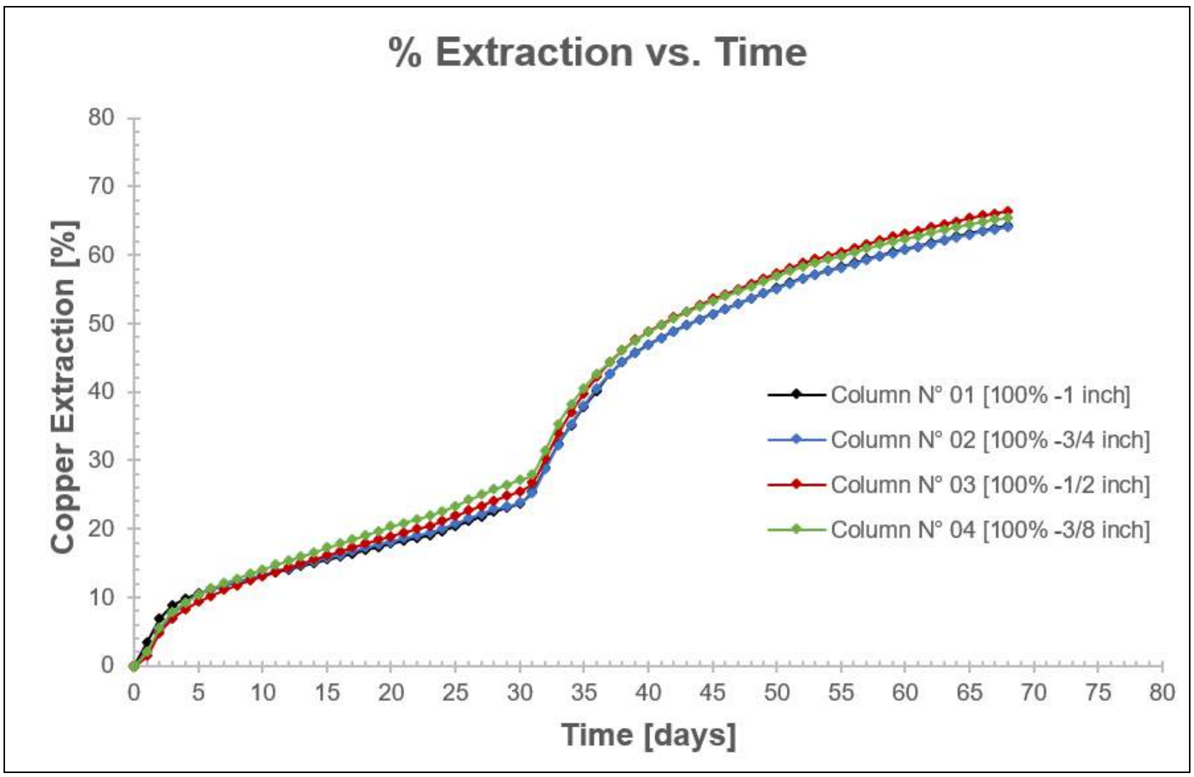


Figure 13-11 Column Leach Extraction Curves

After a total of 68 days under leach the following interim results have been obtained,

- Column # 1 (100% - 1") Cu_T recovery 64.3 % acid consumption 5.9 kg/t
- Column # 2 (100% - ¾") Cu_T recovery 64.1 % acid consumption 6.2 kg/t
- Column # 3 (100% - ½") Cu_T recovery 66.5 % acid consumption 6.3 kg/t
- Column # 4 (100% - 3/8") Cu_T recovery 65.5 % acid consumption 6.7 kg/t

The columns will remain under leach for up to 200 days, which is typical of industrial scale copper bioleach operations. It is anticipated that ultimate column leach extractions will meet or exceed those of the bottle roll tests owing to the extended leach cycle.

13.15 Metallurgical Extraction and Recovery Data

The current test work program is aimed at verifying metallurgical extraction data for the supergene enrichment zone based on a simulated bioleach treatment scenario. Theoretical estimates of copper extraction are provided in Table 13-19 below. Historical estimates for the Cover, Leach Cap and Primary Sulphide domains remain the same.

Table 13-19 Theoretical Bioleach Copper Extractions

Mineral	Formula	% Cu	%Fe	%As	%S	Wt% in Mineral	Wt% Cu	Cu dist'n	% Oxidation	%Cu Extraction
Chalcocite	Cu2S	79.85			20.15	0.92	0.73	81.6%	95	77.47
Digenite	Cu9S5	78.10			21.90					
Covellite	CuS	66.46			33.54					
Chalcopyrite	CuFeS2	34.63	30.43		34.94	0.41	0.14	15.8%	65	10.25
Enargite	Cu3AsS4	48.41		19.02	32.57	0.05	0.02	2.7%	80	2.15
					Subtotal	**1.38**	**0.90**	**100.0%**		**89.87**

**Heap bio-leach scenario*

It should be noted that the copper extractions for chalcocite and enargite will be the same regardless of whether a mesophyllic or thermophyllic bacterial consortia is employed in the process that is implemented. However, significant contribution from chalcopyrite would only be expected if a thermophile based process were implemented.

In the current estimate any contribution from either enargite or chalcopyrite has been discounted and only the contribution from chalcocite has been considered.

A distinction should be drawn between extraction and recovery. The extraction is the percentage of metal lixiviated during the leaching process. The recovery is the amount of metal produced relative to that contained in process feed material after application of downstream efficiency factors. Typically, a factor of 94% would be allowed for downstream SX/EW efficiencies to arrive at an overall recovery number. On this basis an overall copper recovery of 72.5% for the supergene material is indicated. This figure is in line with results from the bottle roll ferric leach tests. There is also a realistic expectation that this figure will be exceeded with further process development once the contribution from all copper minerals has been fully optimized and assessed. Table 13-20 shows preliminary estimates of overall copper recoveries by mineral domain.

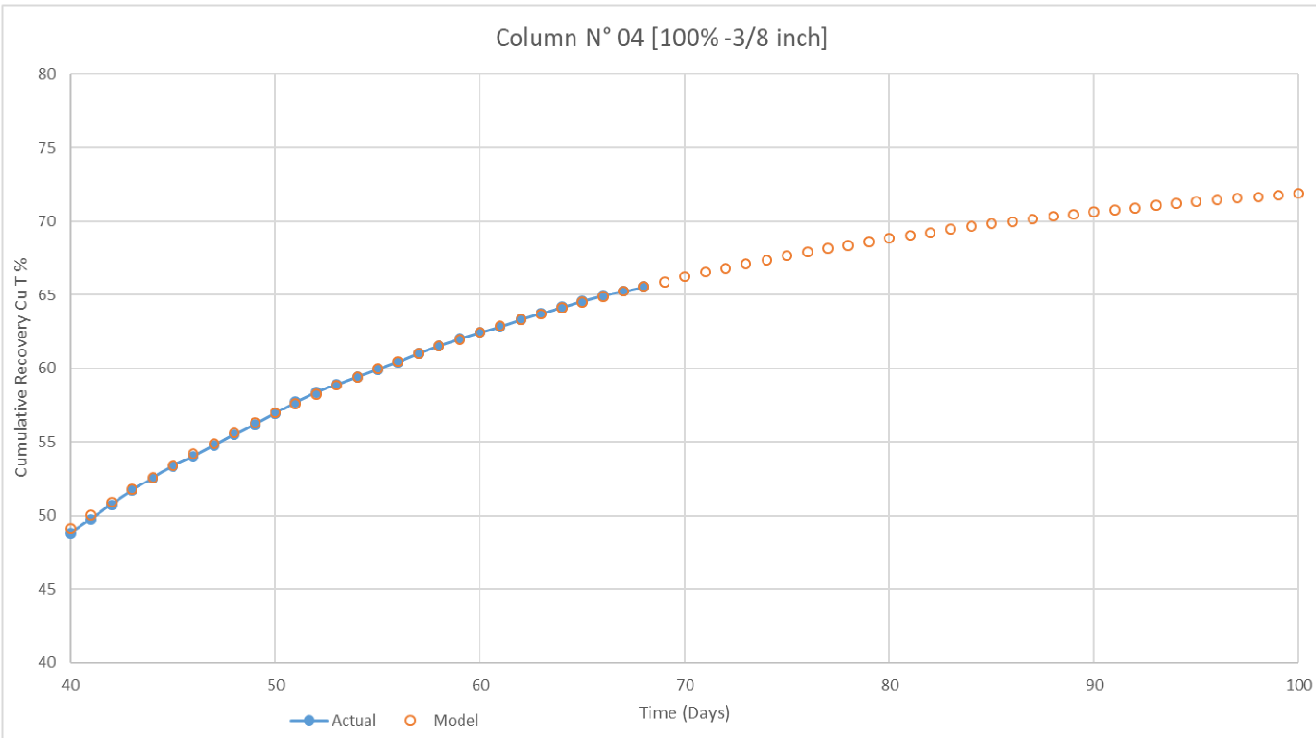
Table 13-20 Estimated Copper Recoveries

Mineralized Domain	Overall Cu Recovery (%)
Cover*	31.1
Leach Cap*	38.0
Supergene	72.5
Primary Sulphide*	21.2

**Estimates taken verbatim from earlier internal technical communications*

13.16 Copper Leach Extraction Modelling

Figure 13-12 below shows the predictive model developed for the Aminpro test columns. As can be seen the model is able to predict leach extractions with a high degree of accuracy. For Column #4 the model currently indicates that for 3/8" material a leach extraction of 74.5 % Cu will be achieved after 175 days under leach. This includes a period of 30 days during which ferric solution was not applied. The results for Column #3, ½" material, are very similar. Columns #1, and #2, the 1" and ¾" materials, lag by about 2 % over 68 days compared with columns #3 and #4. These predictions are supported by the earlier bottle roll tests.



for cycle times of between 150 days and 175 days for 3/8" and 1" crush sizes respectively. Future trade-off studies will determine the economic crush size. These predictions remain to be confirmed in the laboratory.

For comparison, column leach curves for different mineralogical domains of a typical copper deposit are shown in Figure 13-13 below (after E. Lasillo and W. J. Schlitt, Practical Aspects Associated with Evaluation of a Copper Heap Leach Project, 1999).

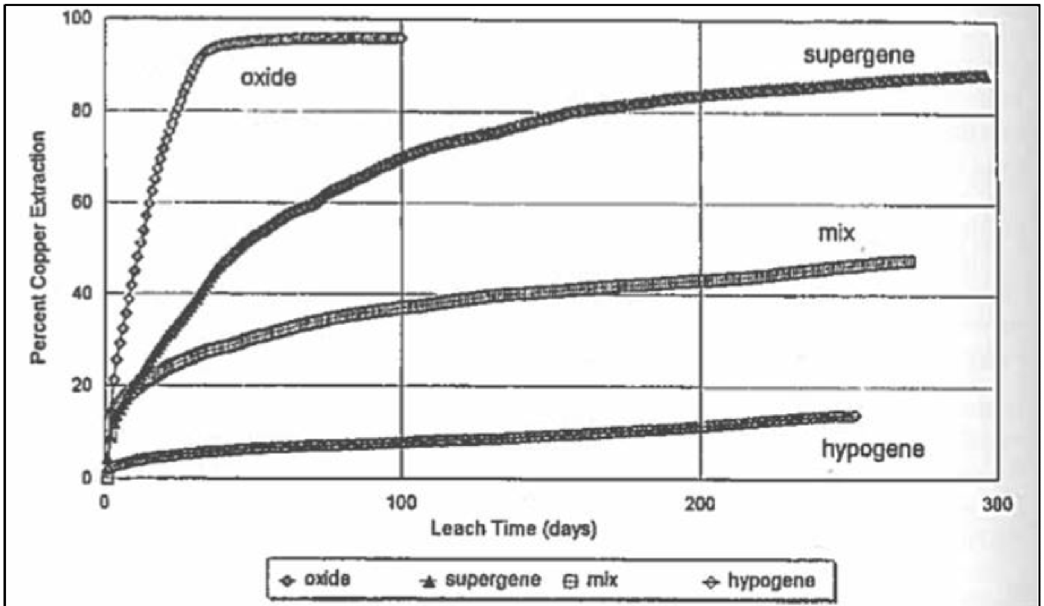


FIGURE 1 Column leach tests conducted in open cycle on material crushed to −⅜ inch. Lixiviant used consisted of 10 gpl sulfuric acid and 3 gpl ferric.

Figure 13-13 Typical Column Leach Curves for Different Mineralogical Domains of a Copper Deposit

13.17 Summary of Aminpro Testwork

The amenability of Antilla supergene material to conventional acid heap leaching is poor, as the leach kinetics is extremely slow. It is unlikely that a project can be successfully implemented based on an acid leach process as the cycle time will be prohibitive.

However, the supergene material is amenable to a ferric leach cycle and acceptable leach extractions have been demonstrated in the laboratory. Although column tests are still underway cycle times comparable with existing commercial operations are anticipated. A process based on a conventional copper bioheap leach is indicated.

Future process development work should focus on,

- Identification, isolation and adaptation of suitable bacterial consortia
- Bioleach amenability tests using, mesophile, moderate thermophile and thermophilic bacteria
- Bioleach column optimisation tests using the selected bacteria
- Potential of using thermophiles for leaching primary ore
- Locked cycle column tests
- Iron and acid balances and internal process requirements
- Understanding arsenic mineralogy, arsenic generation, stabilisation and control
- Neutralisation and residue stabilisation tests
- Supporting environmental testwork
- Ore variability testing
- Process modelling
- Development of geometallurgical based mine model, acid/base accounting,
- Crushing and agglomeration and "engineered" feed material

- Potential for an integrated process facility comprising bioheap and conventional mill and concentrator technologies for both secondary and primary ores
- Potential large-scale column or site trial heap testwork

The mine plan also indicates the availability of significant quantities of cover and cap materials containing copper values these are inferred as 0.3 Mt and 0.7 Mt grading 0.28 and 0.26%Cu, respectively. Little work has been undertaken to date regarding recovery of copper from these resources and testing of these materials should also be incorporated in any future development plans.

14 Mineral Resource Estimates

14.1 Introduction

This section discloses the mineral resource statement for the Antilla copper-molybdenum deposit, prepared Tetra Tech. The effective date of the mineral resource statement is October 19, 2015. The Mineral Resources Statement is reported in accordance with Canadian Securities Administrators' National Instrument 43-101 and has been estimated in conformity with the generally accepted CIM *Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines*. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

A reporting cut-off grade of 0.175% copper equivalent (CuEq) was selected for the Antilla deposit. This cut-off grade reflects current cut-off grades for similar deposits in the region. Tetra Tech considers this CuEq% cut-off to be reasonable for this deposit.

14.2 Resource Database

Panoro supplied all of the digital data. These data were compiled from core drilling programs and analytical results conducted on the property since 2003. The data were verified and imported into Gemcom GEMS version 6.5 Resource Evaluation Edition.

The entire borehole data set included the header, survey, assay, and lithology files for 96 core boreholes (15,385.0 metres). Table 14-1 summarizes the number of boreholes and their lengths on the property. Out of all drilling completed on the property, 88 boreholes (14,293 metres) occur within the deposit area and were considered for geology and mineral resource modelling.

Table 14-1 Summary of Boreholes

Company	Year	No. of Boreholes	Total Length (m)
Cordillera	2003-2005	20	**2,919.**
Panoro	2008	49	**9,130.**
Chancadora	2010	19	**2,242.**
Total		88	14,292.

14.2.1 Specific Gravity

There are 47 density measurements available for the three mineralization domains modelled. Average densities were assigned each domain. The Overburden/Cover domain was assigned a value of 2.00.

Table 14-2 summarizes the specific gravities used for the various lithologies and rock type domains.

Table 14-2 Summary of Specific Gravity by Lithological Domain

Lithology Domain	Rock Code	Specific Gravity	Count	Mean	Minimum	Maximum	Standard Deviation
Cover/Overburden [COV]	100	2.00	-	-	-	-	**-**
Leach Cap [LC]	200	2.51	2	2.51	2.46	2.55	**0.07**
Supergene [SE]	300	2.69	3	2.69	2.36	2.96	**0.13**
Primary Sulphides [PS]	400	2.70	1	2.70	2.42	2.80	**0.11**
Country Rock [CR]	**99**	**2.68**	**4**	**2.68**	**2.36**	**2.96**	**0.13**

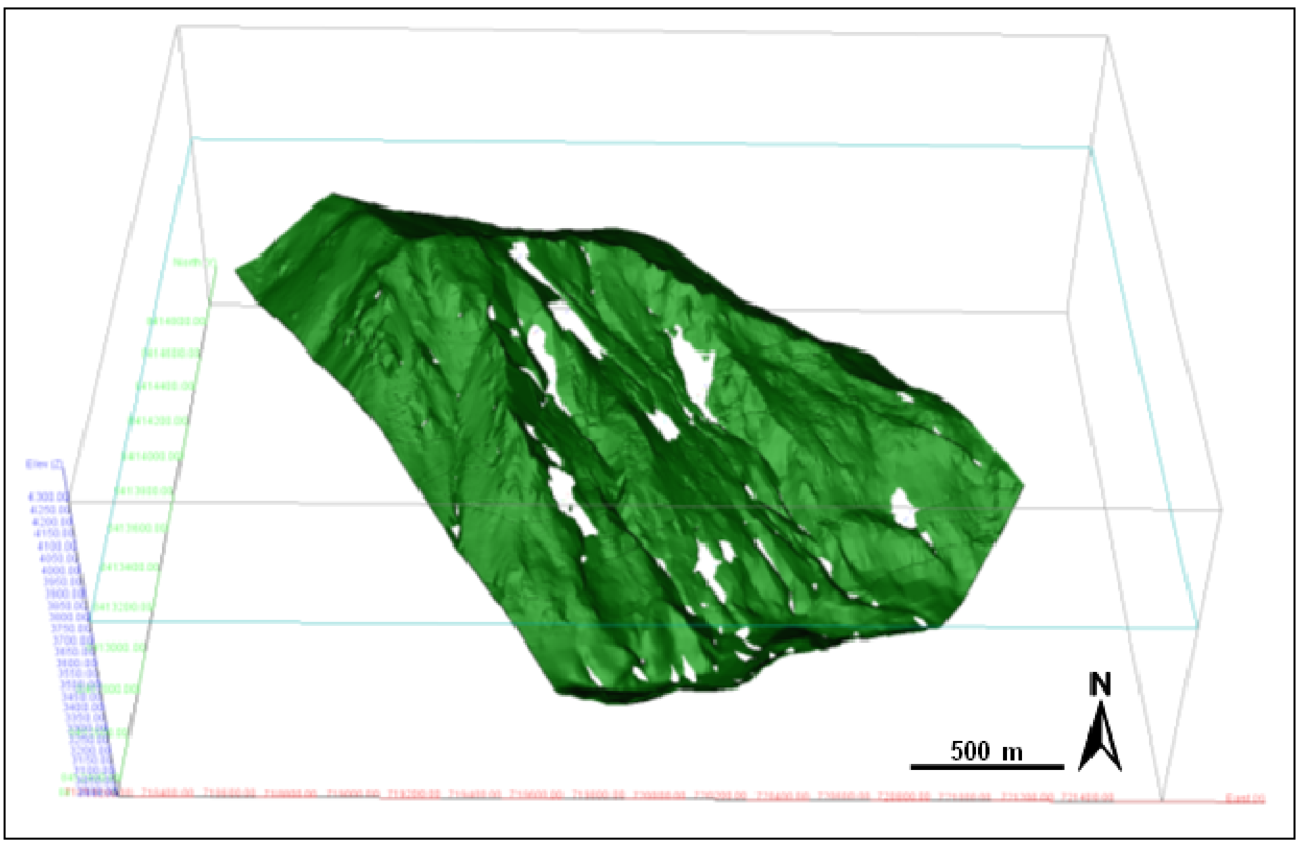
14.3 Solid Body Modelling

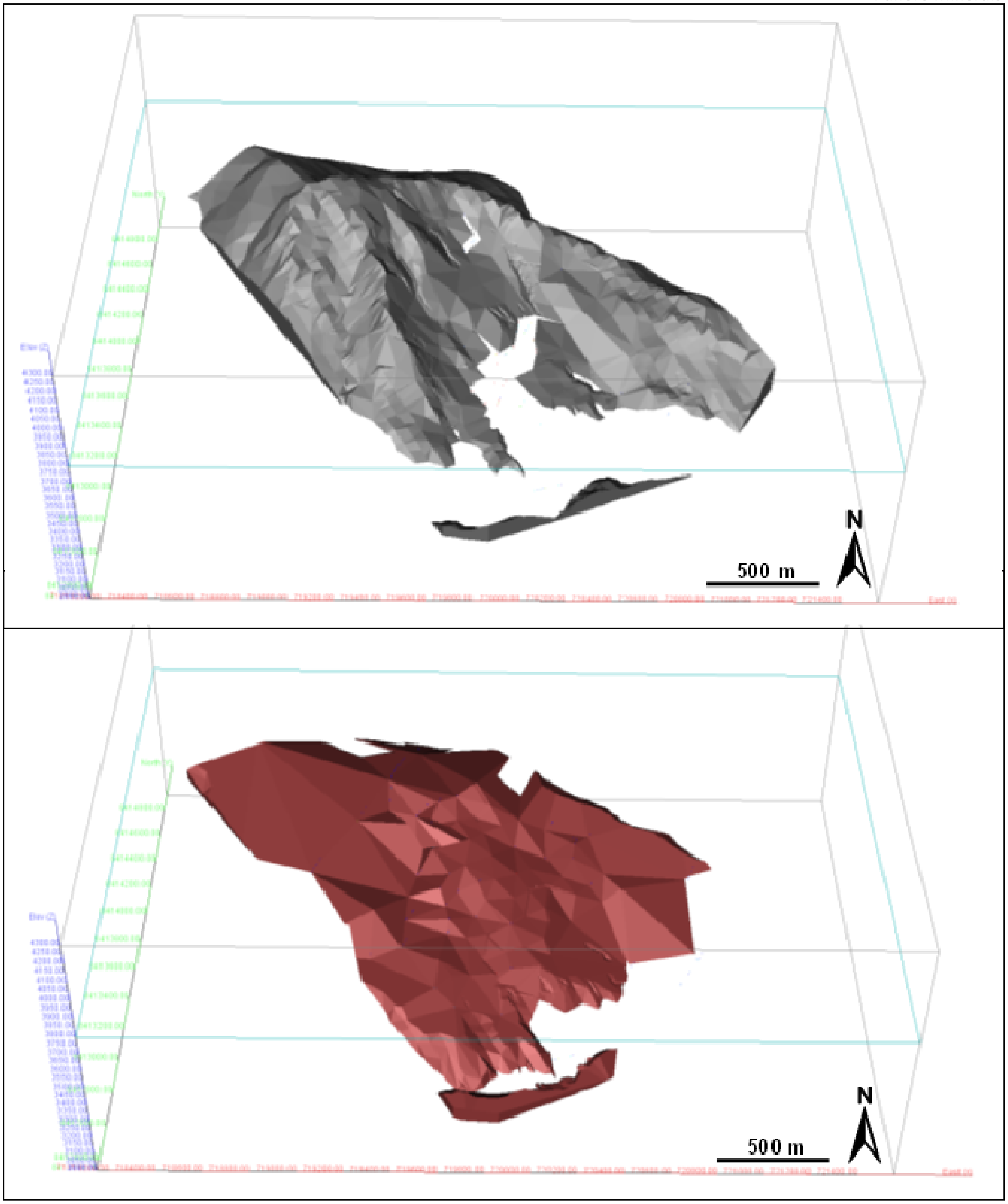
Tetra Tech used Panoro's initial wireframes as a guide to build the final mineralized wireframes. The wireframes were built by creating surfaces between lithological contacts from borehole intersections. In addition, a contact between the Supergene and Primary Sulphide domain was generated. The 3D wireframes were clipped to topography. The topographic surface was supplied by Panoro. The lateral extent of the wireframes was to a nominal 200 metres beyond the limit for the boreholes.

A country rock wireframe was built surrounding the mineralized wireframes and below the topographic surface.

Figure 14-1 to Figure 14-6 illustrate the various 3D wireframes for the mineralized and country rock domains in perspective view. The following domains were modelled:

- Cover/Overburden (100)
- Leach Cap (200)
- Supergene (300)
- Primary Sulphide (400)
- Country Rock (99)

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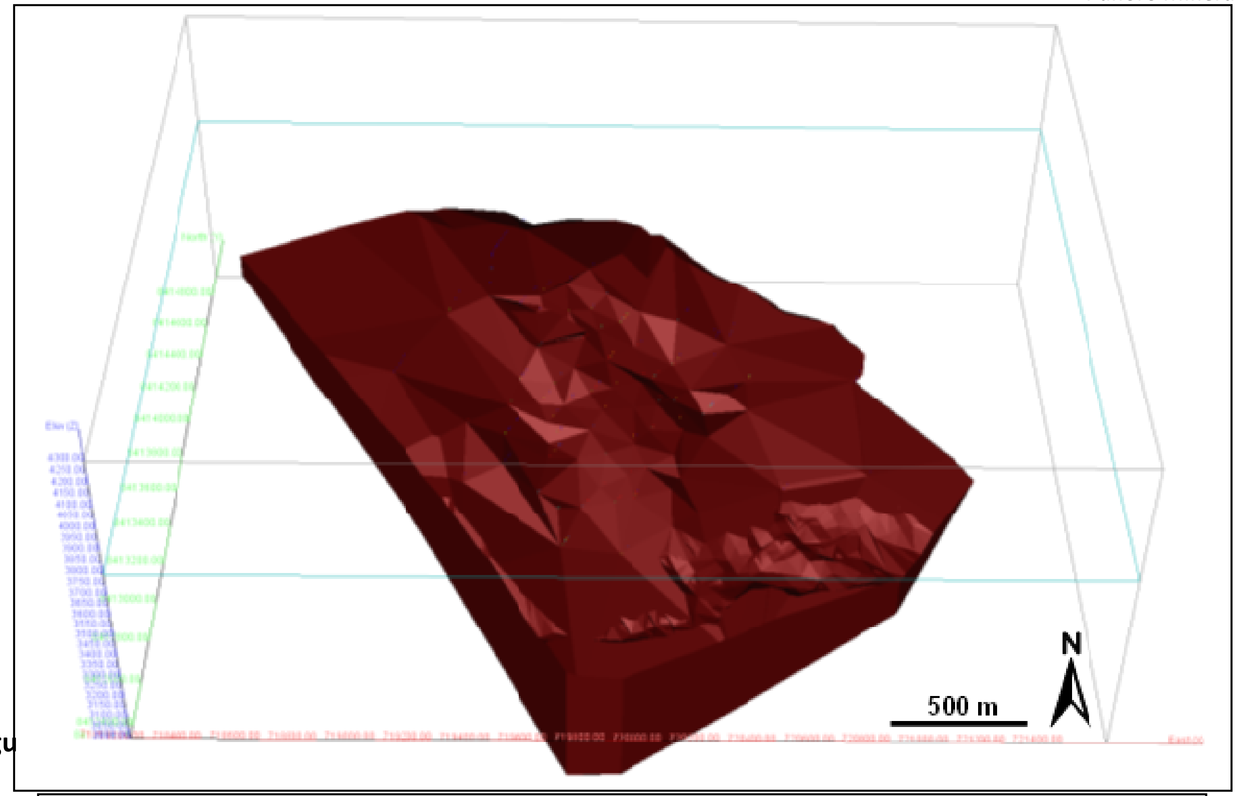

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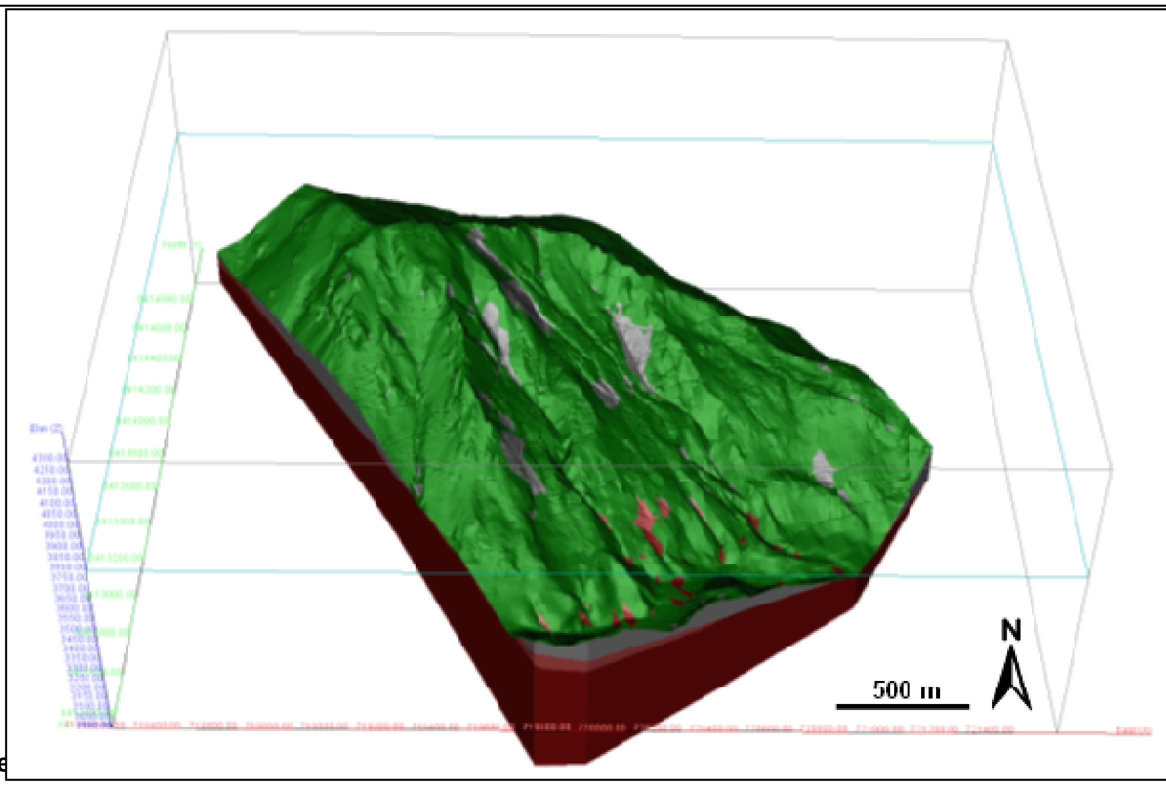

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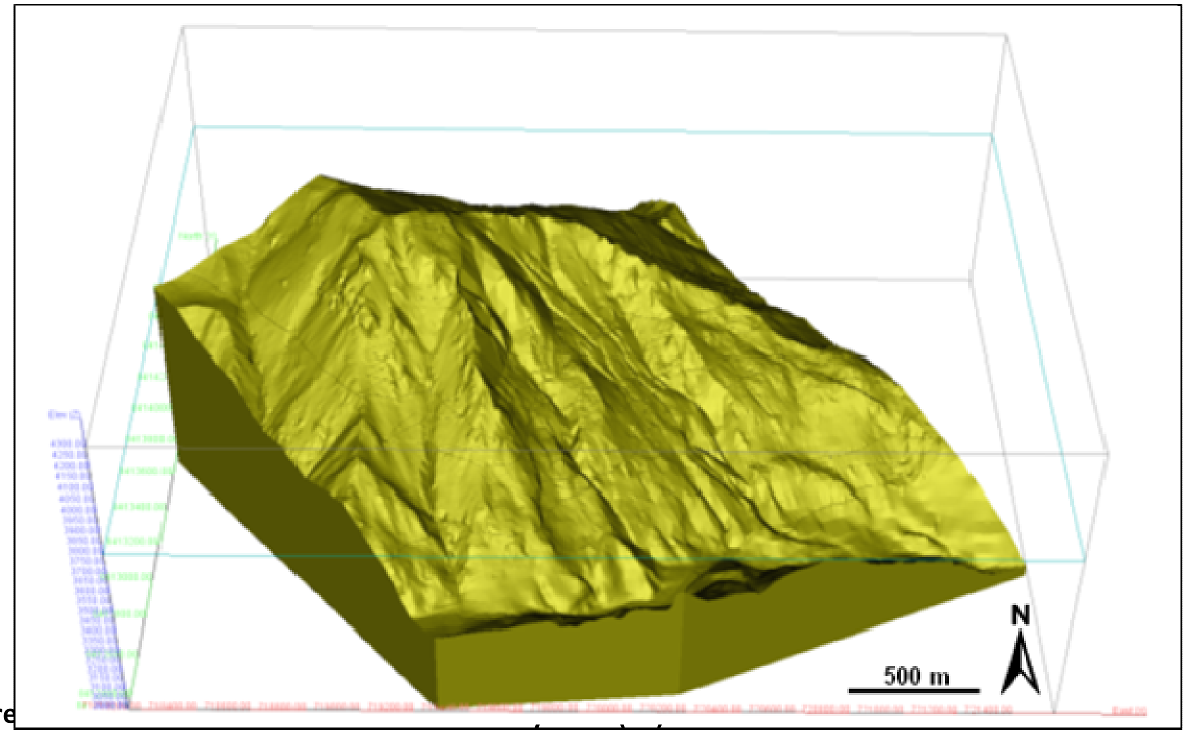

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14.4 Exploratory Data Analysis

Exploratory data analysis is the application of various statistical tools to characterize the analytical data. In this case, the objective is to understand the population distribution of the grade elements through the use of such tools as histograms, descriptive statistics, and probability plots.

14.4.1 Raw Assays

Raw assay statistics for copper, molybdenum, gold and silver in the modelled domains are shown in Table 14-3. Only those values greater than zero were used in the statistical analysis. Gold and silver are relatively low and near the detection limits of the analytical method used. Since these metals do not contribute significantly to the value of the sulphide mineralization, they have been omitted from the mineral resource estimation.

Table 14-3 Raw Assay Statistics by Domain

	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)		Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)
Covertura [100]					**Supergene [300]**				
Count	69	69	69	69	Count	2,153	2,153	1,568	2,153
Minimum	0.00	0.00	0.01	0.50	Minimum	0.00	0.00	0.01	0.50
Maximum	0.86	0.05	0.03	2.00	Maximum	5.09	0.26	0.08	95.00
Mean	0.05	0.00	0.01	0.93	Mean	0.45	0.01	0.01	1.18
Std. Dev.	0.12	0.01	0.00	0.31	Std. Dev.	0.47	0.02	0.01	2.61
Variance	0.02	0.00	0.00	0.10	Variance	0.22	0.00	0.00	6.83
COV	2.61	1.44	0.57	0.34	COV	1.04	1.74	0.72	2.21
Leach Cap [200]					**Primary Sulphides [400]**				
Count	1,674	1,674	1,588	1,674	Count	2,706	2,706	2,417	2,706
Minimum	0.00	0.00	0.01	0.50	Minimum	0.00	0.00	0.01	0.08
Maximum	2.03	0.26	0.17	74.00	Maximum	2.27	0.79	1.21	55.00
Mean	0.04	0.01	0.01	1.49	Mean	0.16	0.01	0.01	1.12

Std. Dev.	0.12	0.01	0.01	2.91		Std. Dev.	0.18	0.03	0.04	1.42
Variance	0.02	0.00	0.00	8.47		Variance	0.03	0.00	0.00	2.00
COV	3.43	1.62	0.98	1.96		COV	1.18	2.87	3.73	1.27

14.4.2 Capping

Cumulative probability plots, Parrish decile analysis and descriptive statistics were used to assess the need for capping of the assay grades for the Antilla deposit. Typically, a step-change in the profile or a separation of the data points is present if there are different populations in the data set (Figure 14-7). High value outliers will show up in the last few percent of a cumulative probability plot (typically in the 97 to 100% range) and the break in the probability distribution may be selected to set a capping level.

Tetra Tech found that capping of raw data was deemed necessary. Capping values selected are shown in Table 14-4.

Table 14-4 Summary of Capping Grades

Metal	Capping Value	Samples Capped
Cu (%)	1.9	**40**
Mo (%)	**0.1**	**23**

14.4.3 Composites

The raw uncapped data within the Antilla deposit were composited on 4-, 6-, and 8-metre composites. Statistics show little change in the mean but a lowering of the coefficient of variation. The 6-metre composites were selected as the most reasonable for the interpolation of the Antilla deposit.

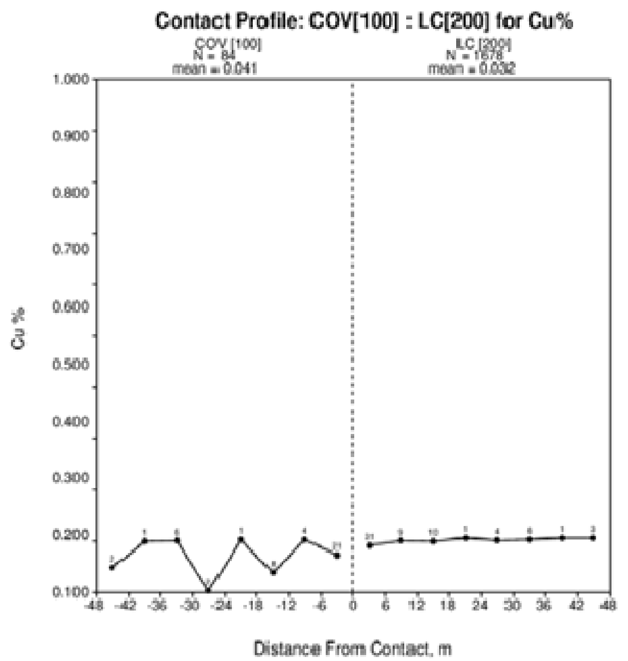
In the Geovia GEMS project, the table COMP_6M was created for composite data. Composite data, once calculated, was tagged with their associated rock code and rock type. The composites were then extracted into a point area named 6m_Comp. A total of 2,124 composite data points were extracted from the borehole data. All composite data were used in the interpolation of the Antilla deposit. Table 14-5 presents the comparison between the capped data 6-metre composite data (no zeroes) for the mineralized domains.

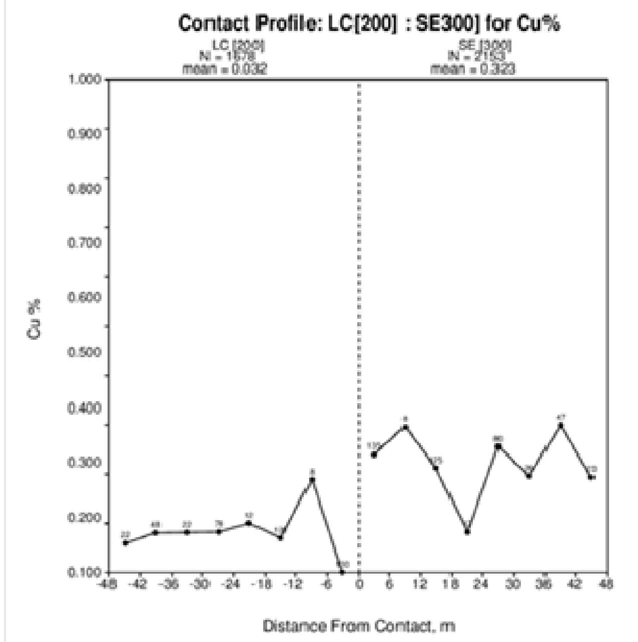
Table 14-5 Statistics for Capped 6-metre Composites Data

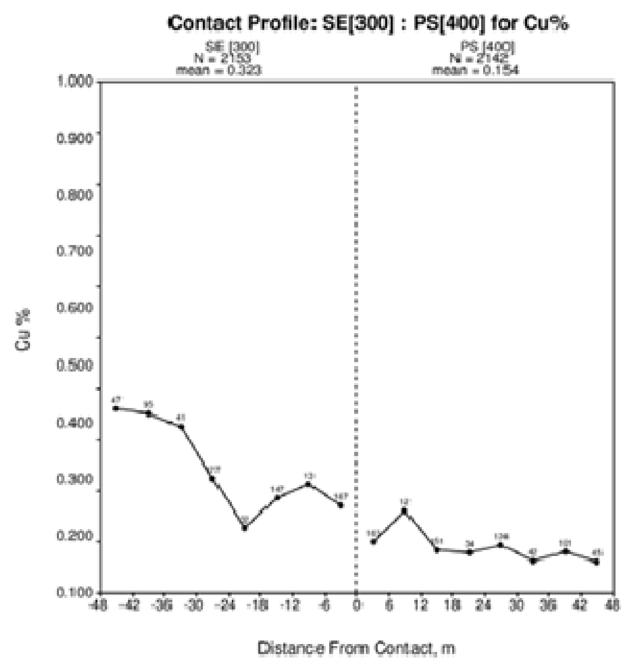
	Cu (%)	Mo (%)			Cu (%)	Mo (%)
Covertura [100]				**Supergene [300]**		
Count	38	38		Count	747	747
Minimum	0.01	0.00		Minimum	0.01	0.00
Maximum	0.80	0.05		Maximum	1.63	0.09
Mean	0.09	0.01		Mean	0.43	0.01
Std. Dev.	0.19	0.01		Std. Dev.	0.30	0.01
Variance	0.04	0.00		Variance	0.09	0.00
COV	2.04	1.47		COV	0.70	1.11
Leach Cap [200]				**Primary Sulphides [400]**		
Count	600	600		Count	739	739
Minimum	0.00	0.00		Minimum	0.00	0.00
Maximum	0.80	0.04		Maximum	1.07	0.06
Mean	0.04	0.01		Mean	0.18	0.01
Std. Dev.	0.09	0.01		Std. Dev.	0.14	0.01
Variance	0.01	0.00		Variance	0.02	0.00
COV	2.29	0.97		COV	0.79	1.03

14.4.4 Contact Plots

Contact plots were created over each domain boundary with the mineralized domains. The contact plots illustrate a soft boundary exists between the coverture and leach cap domains. The supergene and primary sulphide domains show a gradual contact with regards to copper, where higher copper in the supergene diminish into the primary sulphide domain. Figure 14-7 presents the contact plots for each of these domains using copper percent grades.







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ulphide

14.5 Statistical Analysis and Variography

Samples used for variography are a function of geological interpretation and sample populations. For the Antilla deposit, all composite data within the mineralized domains were used in determining variograms. Variograms were established using the 6-metre composite samples within the combined Covertura and Leach Cap domains; and combined Supergene and Primary Sulphide domains.
Variography was completed in variogram analysis in GEMS.

Experimental variograms were developed on 50- to 100-metre lag distances for copper, molybdenum, and gold. The ranges of the experimental variograms appear to reach the sill at approximately 100 to 250 metres. Up to two spherical structures were used for spatial modelling.

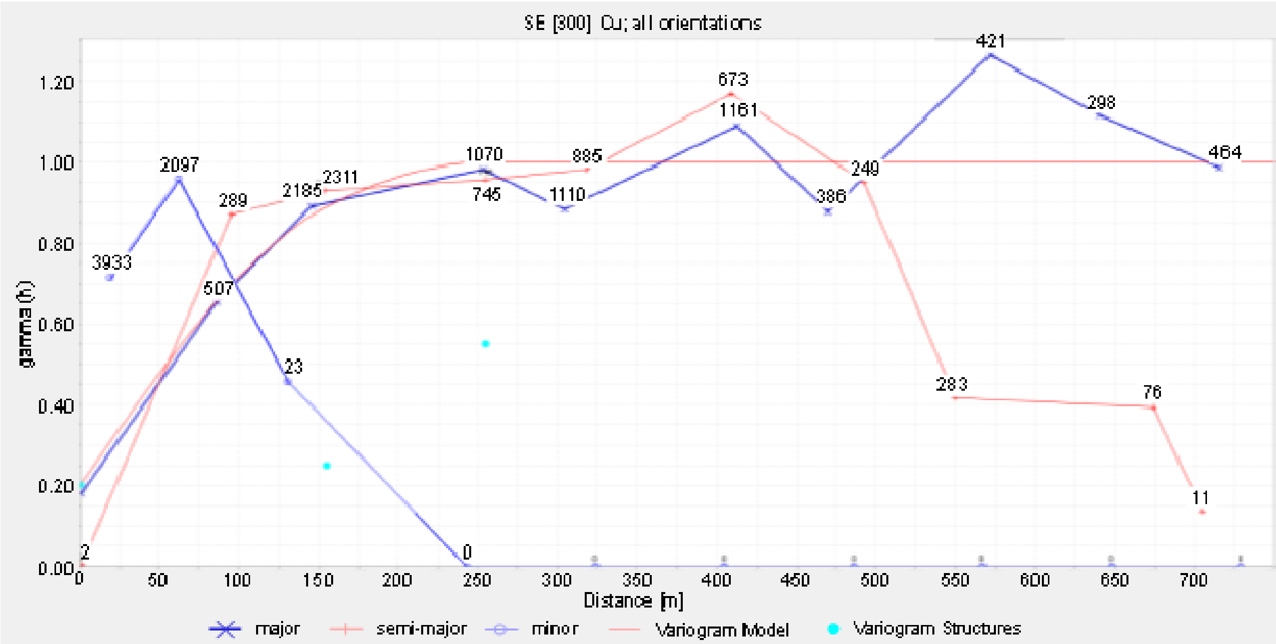
Table 14-6 and Table 14-7 summarize the variography parameters used for interpolation for each domain in the Antilla deposit, while Figure 14-8 and Figure 14-9 illustrate examples of variograms for copper and molybdenum in the Supergene domain.

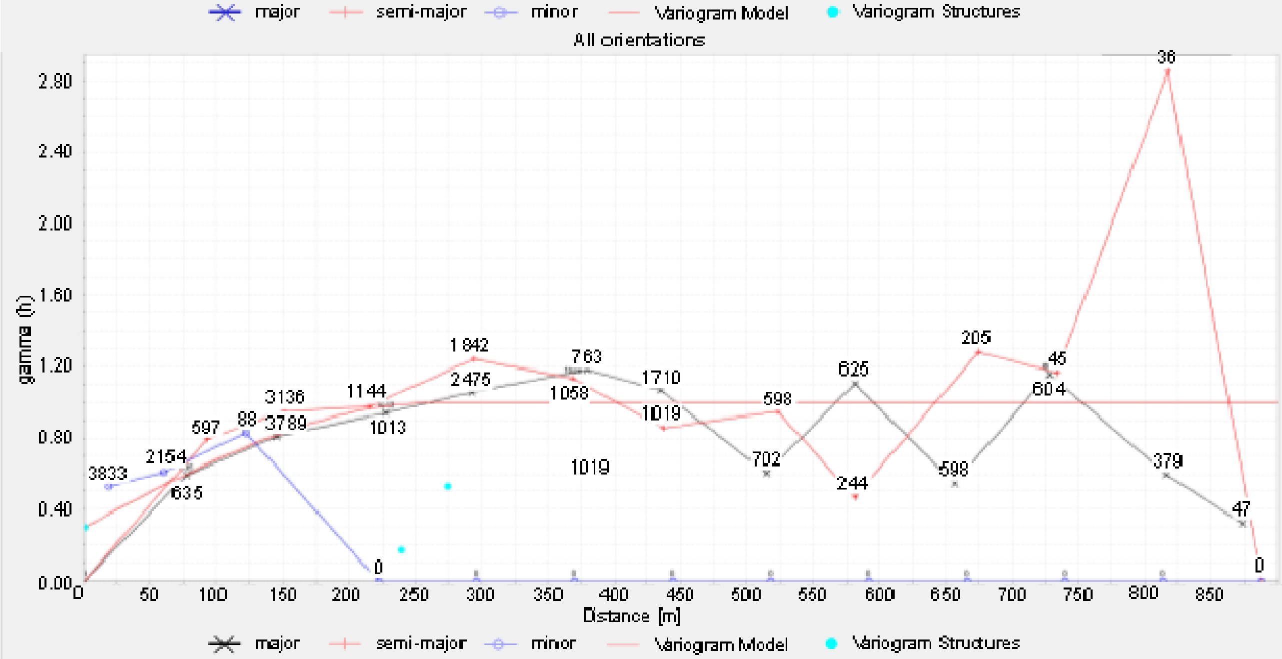
Table 14-6 Variography Parameters for Copper

Profile Name	Sill	Search Anisotrophy-	Azimuth	Dip -	Azimuth-	-X Range	Y Range-	Z Range-	Search Type-
Domains 100 and 200; Sill = 0.0127									
C0 (nugget)	0.00	-	-	-	-	-	-	-	-
C1	0.00	Az, Dip, Az	1	-	48	2	1	13	**Spheric**
C2	0.00	Az, Dip, Az	1	-	48	3	2	20	**Spheric**
Domain 300; Sill = 0.0849									
C0 (nugget)	0.01	-	-	-	-	-	-	-	-
C1	0.02	Az, Dip, Az	1	-	93	1	94	3	**Spheric**
C2	0.04	Az, Dip, Az	1	-	93	2	1	6	**Spheric**
Domain 400; Sill = 0.0345									
C0 (nugget)	0.01	-	-	-	-	-	-	-	-
C1	0.00	Az, Dip, Az	2	-	1	1	80	5	**Spheric**
C2	**0.01**	**Az, Dip, Az**	**2**	**-**	**1**	**2**	**98**	**6**	**Spheric**

Table 14-7 Variography Parameters for Molybdenum

Profile Name	Sill	Search Anisotrophy-	Azimuth	Dip -	Azimuth-	-X Range	Y Range-	Z Range-	Search Type-
Domains 100 and 200; Sill = 0.00004									
C0 (nugget)	0.00000	-	-	-	-	-	-	-	-
C1	0.00001	Az, Dip, Az	23	8	3	1	1	3	**Spheric**
C2	0.00002	Az, Dip, Az	23	8	3	3	3	7	**Spheric**
Domain 300; Sill = 0.000076									
C0 (nugget)	0.00002	-	-	-	-	-	-	-	-
C1	0.00001	Az, Dip, Az	18	-	93	2	1	7	**Spheric**
C2	0.00004	Az, Dip, Az	18	-	93	2	1	9	**Spheric**
Domain 400; Sill = 0.000074									
C0 (nugget)	0.00002	-	-	-	-	-	-	-	-
C1	**0.00005**	**Az, Dip,**	**21**	**-**	**1**	**1**	**1**	**8**	**Spheric**


SE [300] Cu; all orientations
gamma (h)
Distance [m]
major
semi-major
minor
Variogram Model
Variogram Structures


All orientations
gamma (h)
Distance [m]
major
semi-major
minor
Variogram Model
Variogram Structures

14.6 Block Model and Grade Estimation

14.6.1 Block Model

A single block model was created to cover the Antilla deposit. Table 14-8 shows the GEMS coordinates for the block model origin. A block size of 15 by 15 by 6 metres was used for block model and resource estimate. The block size is considered reasonable where distances between boreholes vary between 70 and 100 metres.

Table 14-8 Block Model Parameters

Direction	Origin*	Block Size (metre)	Number of Blocks
East-West	718,400	15.0	**185**
North-South	8,412,500	15.0	**140**
Vertical	4300	6.0	**200**

*** UTM coordinates WGS84 zone 18S**

14.6.2 Interpolation

A copper and molybdenum value was assigned to each block in the model using ordinary kriging (OK), inverse distance squared (ID2) and nearest neighbour (NN) and informed from capped composites. For sensitivity analysis, OK, ID2, and NN interpolations runs were also carried out using uncapped composites. Two estimation passes were employed. Separate interpolation runs were carried out for each of the four domains. A summary of the estimation parameters are described in Table 14-9.

Table 14-9 Description of Estimation Parameters

Domain	Profile Name	Min. Number of Composite	Max. Number of Composites	Max. Samples per Borehole	Max. Number of Boreholes
100	OKxx1_P1	7	16	3	**5**
	OKxx1_P2	3	16	3	**5**
200	OKxx2_P1	7	16	3	**5**
	OKxx2_P2	3	16	3	**5**
300	OKxx3_P1	7	15	3	**5**
	OKxx3_P2	3	15	3	**5**
400	OKxx4_P1	7	15	3	**5**
	OKxx4_P2	3	15	3	**5**
100	NNxx1	1	1	1	**1**
200	NNxx2	1	1	1	**1**
300	NNxx3	1	1	1	**1**
400	NNxx4	1	1	1	**1**
100	IDxx1_P1	7	16	3	**5**
	IDxx1_P2	3	16	3	**5**
200	IDxx1_P1	7	16	3	**5**
	IDxx1_P2	3	16	3	**5**
300	IDxx1_P1	7	15	3	**5**
	IDxx1_P2	3	15	3	**5**
400	IDxx1_P1	7	15	3	**5**
	IDxx1_P2	3	15	3	**5**

Note: xx – denotes metal (copper and molybdenum)

As the transition between the Supergene (300) and Primary Sulphide (400) domains is gradational, a different sample support strategy was employed. Three composite samples on either side of the boundary (approximately 18 metres to either side of the boundary) were coded as Rock Type 350.

During the interpolation, blocks with the Rock Type 300 and 400 were allowed to include Rock Type 350 as part of the sample selection. Therefore, the blocks at the interface of the two domains were allowed to be influenced up to 18 metres into the other domain.

14.6.3 Search Parameters

Search ellipses were generated in GEMS based on orientation of the variograms. Therefore, the search ellipses for the upper domains differ from those of the lower domains. In the lower domains, the first pass search ellipses used half the ranges of the second pass ellipse, constraining data in the core of the deposit over the transition between the supergene and the primary sulphide domain.

A list of parameters for each search ellipse used for each pass is shown in Table 14-10 which illustrates the orientations of the search ellipses used in the interpolation of the Antilla block model.

Table 14-10 Search Ellipse Parameters

Profile Name	Search Anisotropy	Azimuth	Dip	Azimuth	X Range	Y Range	Z Range	Search Type
CU12	Az., Dip, Az.	130	-30	47.6	346	286	208	Ellipsoidal
17CU300_P1	Az., Dip, Az.	188	-19	93	170	103	40	Ellipsoidal
17CU300	Az., Dip, Az.	188	-19	93	255	155	60	Ellipsoidal
17CU400_P1	Az., Dip, Az.	219	-5	124	143	66	41	Ellipsoidal
17CU400	Az., Dip, Az.	219	-5	124	215	98	63	Ellipsoidal
MO12	Az., Dip, Az.	234	8	330	305	305	75	Ellipsoidal
17MO300_P1	Az., Dip, Az.	188	-19	93	183	1247	60	Ellipsoidal
17MO300	Az., Dip, Az.	188	-19	93	275	190	90	Ellipsoidal
17MO400_P1	Az., Dip, Az.	215	-8	116	115	108	54	Ellipsoidal
17MO300	Az., Dip, Az.	215	-8	116	177	163	81	Ellipsoidal

14.6.4 Copper Equivalent Calculation

The mineral resources discussed herein are reported at a copper equivalent cut-off grade, based on the two primary metals of significant economic value. A copper equivalent grade was calculated for each block using a script in the GEMS block model based on the interpolated copper and molybdenum grades. The 2013 metal prices of US$3.25 and US$9.00 per pound of copper and molybdenum, respectively, and recoveries of 90% and 80%, respectively, were used for the equivalency formula. Copper equivalency is calculated as follows: *CuEq%= ([Cu Price * Cu Grade * 22.04622] + [Mo Price * Mo Grade * 22.04622 * Mo Recovery] / Cu Price) / 22.04622*

14.7 Model Validation and Sensitivity

The block model volumes were validated against the solid wireframe volumes and all differences were found to be within a tolerance of less than 1.00%. The results of the comparisons are shown in Table 14-11.

Table 14-11 Summary Block Model Statistics

Domain	Wireframe Volume (m^3)	Block Model Volume (m^3)	%Difference
Cover	38.873,784	38,526,511	0.9
Leach Cap	247,242,243	246,222,755	0.4
Supergene	124,366,168	122,649,882	1.4
Primary	**617,071,460**	**601,283,113**	**2.6**

Swath plots were created for the Antilla block model copper grades by bench, by column (easting), and by row (northing), and compared to each interpolation method as a visual comparison of the precision of the interpolation methods.

Figure 14-10 to Figure 14-12 illustrates the swath plots for copper percent in the Antilla deposit. Variations in block grades estimated using a nearest neighbour estimator, particularly at the ends of the graphs (i.e., the limits of the block model), denote areas where sample populations used for estimation are no longer similar.

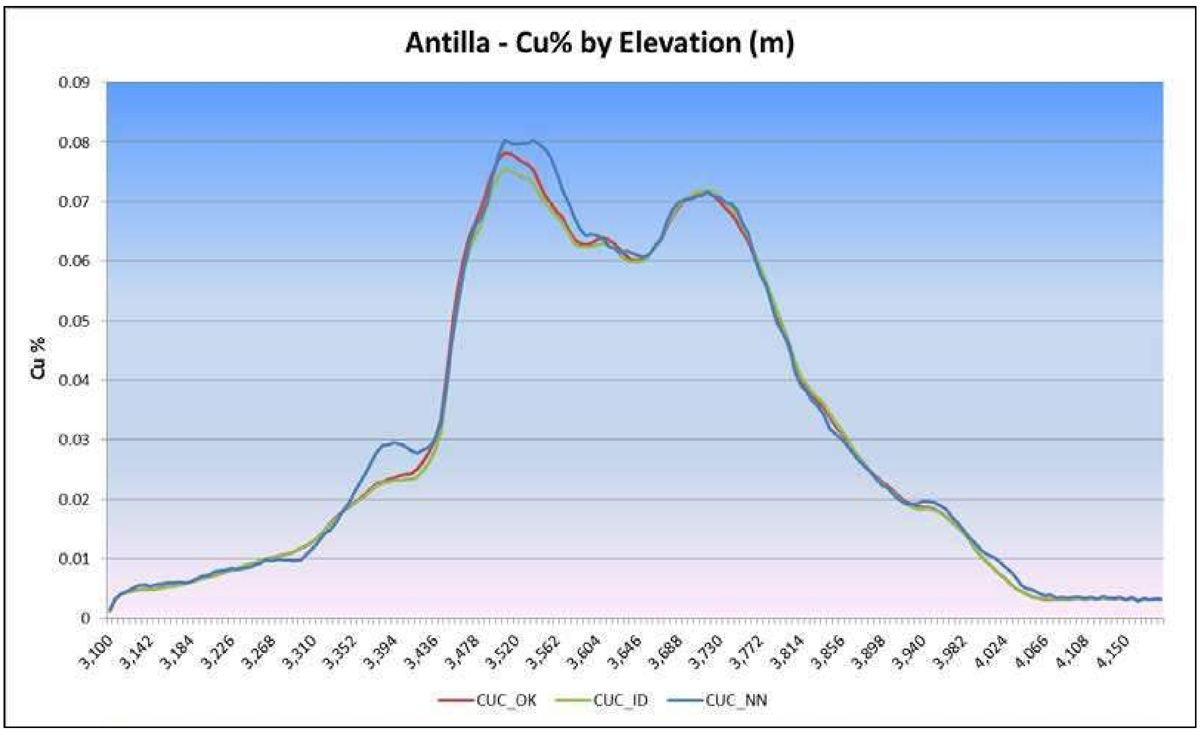


Figure 14-10 Swath Plots for Antilla by Elevation

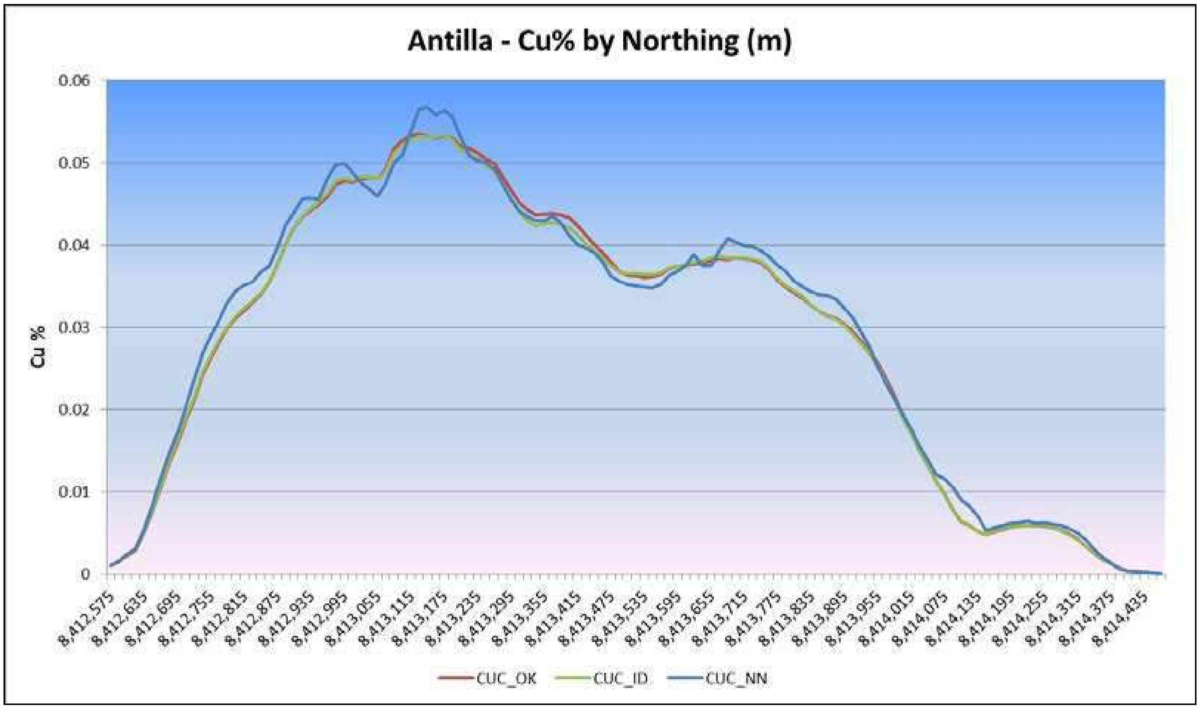


Figure 14-11 Swath Plots for Antilla by Northing

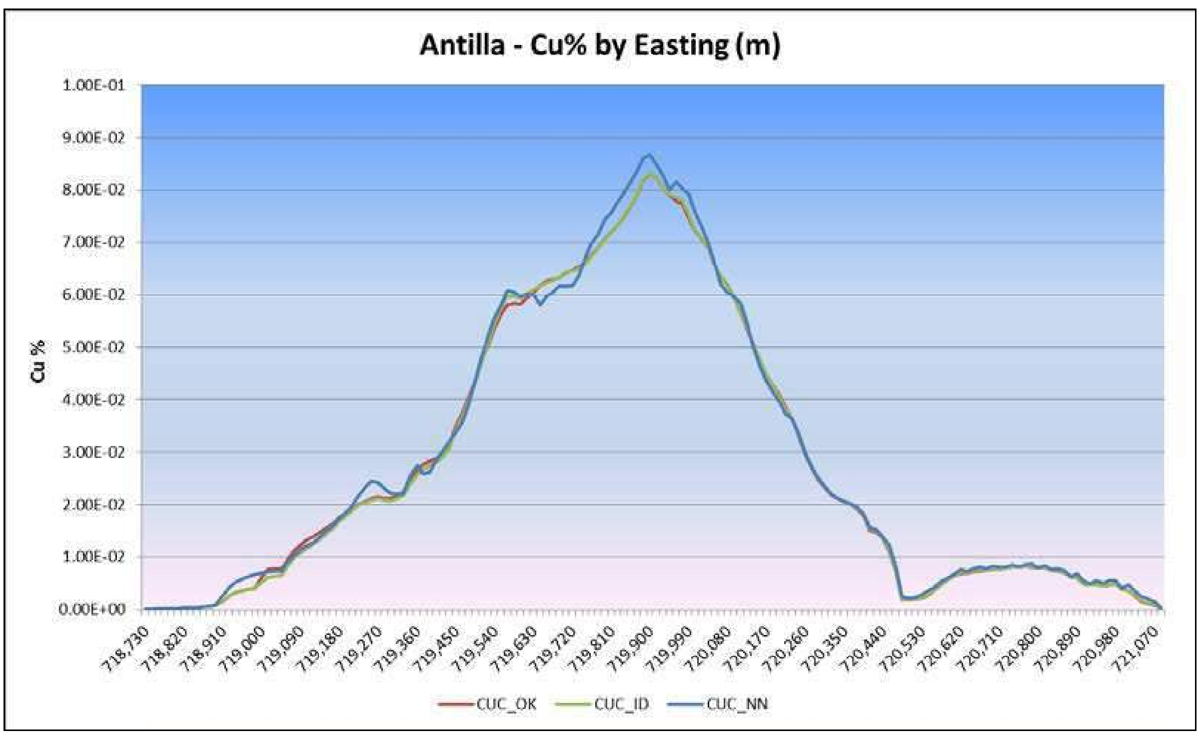


Figure 14-12 Swath Plots for Antilla by Easting

14.8 Mineral Resource Classification

Block model quantities and grade estimates for the Antilla project were classified according to the CIM *Definition Standards for Mineral Resources and Mineral Reserves* (May 2014) by Paul Daigle, PGeo (APGO #1592), an appropriate independent qualified person for the purpose of National Instrument 43-101.

Mineral resource classification is typically a subjective concept. Industry best practices suggest that resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

Tetra Tech is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by core drilling on sections spaced at approximately 100 metres.

The Antilla block model was classified as Indicated and Inferred based on the number of samples and boreholes used to code a block, the borehole spacing, and continuity of the copper-molybdenum mineralization. Nominally, Indicated blocks, (code "2"), are those blocks that are informed by a minimum of three boreholes within a 125 metre radius, while Inferred blocks, (code "3"), are those blocks informed by a minimum of two boreholes within a 225 metre radius.

The classification was a two-step process. The automated classification was reviewed and groomed to remove isolated blocks and delineate regular categories.

14.9 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as:

> *"A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."*

The "reasonable prospects for eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recoveries. In order to meet this requirement, Tetra Tech considers that major portions of the Antilla project are amenable for open pit extraction.

To assess which portions of the Antilla sulphide deposit show "reasonable prospect for eventual economic extraction" a conceptual pit shell was created using Whittle 4.5. The conceptual pit optimization input parameters are shown in Table 14-12.

Table 14-12 Conceptual Pit Optimization Input Parameters

Parameters	Amount	Units
Metal Prices		
Copper	3.00	US$/lb
Molybdenum	9.00	US$/lb
Offsite Costs		
Copper	0.38	US$/lb
Molybdenum	2.75	US$/lb
Metal Recoveries		
Copper / Molybdenum		
Cover	80 / 65	%
Sulphides	85 / 65	%
Mixed	80 / 70	%
Oxides	75 / 65	%
Selling Cost	5	%*
Mining Parameters		
Mining Recovery Rate	97	%
Mining Dilution Rate	3	%
Pit Slope Angle		
Country Rock	52	degrees
Cover	24	degrees
Leach Cap	42	degrees
Supergene	48	degrees
Primary Sulphides	49	degrees
Mining Cost (Phases 1-3)	1.85	US$/t
Mining Cost (Phase 4)	1.00-1.10	US$/t
Total Mining Cost	**1.00-1.85**	**US$/t**
Processing Parameters		
Mill Throughput	30,000	t/d
Mill Throughput	10.5	Mt/a
Mill Costs	7.25	US$/t
Additional Cost for Mineral Resource	0.15	Control blasted size and longer hauling to mill
General and Administration	1.10	US$/t
Ore Handling Cost	0.50	US$/t
Environmental Cost	1.00	US$/t
Total Processing Cost	**10.00**	**US$/t**

**Of selling price includes concentrate transportation, smelter, and refinery charges*

Tetra Tech considers that the blocks located within the conceptual pit envelope show “reasonable prospects for eventual economic extraction” and can be reported as a mineral resource. Table 14-13 presents the Mineral Resource Statement for the Antilla project.

Table 14-13 Mineral Resource Statement*, Antilla Copper-Molybdenum Project, Peru, Tetra Tech, October 19, 2015

Domain	Quantity	Grade		
	'000 tonnes	Cu %	Mo	CuEq%
Indicated				
Overburden/Cover	5,600	0.2	0.0	**0.2**
Leach Cap	13,400	0.2	0.0	**0.2**
Supergene	168,900	0.4	0.0	**0.4**
Primary Sulphide	103,900	0.2	0.0	**0.2**
Total Indicated	**291,800**	**0.3**	**0.0**	0.3
Inferred				
Overburden/Cover	500	0.2	0.00	**0.2**
Leach Cap	13,400	0.2	0.00	**0.2**
Supergene	25,900	0.3	0.00	**0.3**
Primary Sulphide	50,700	0.2	0.00	**0.2**
Total Inferred	90,500	0.2	0.00	0.2

**Mineral resources are not mineral reserves and have not demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. Reported at a cut-off grade of 0.175 CuEq%, assuming an open pit extraction scenario, a copper price of US$3.25 per pound, a molybdenum price of US$ 9.00 per pound, and a metallurgical recovery of 90% for copper and 80% for molybdenum.*

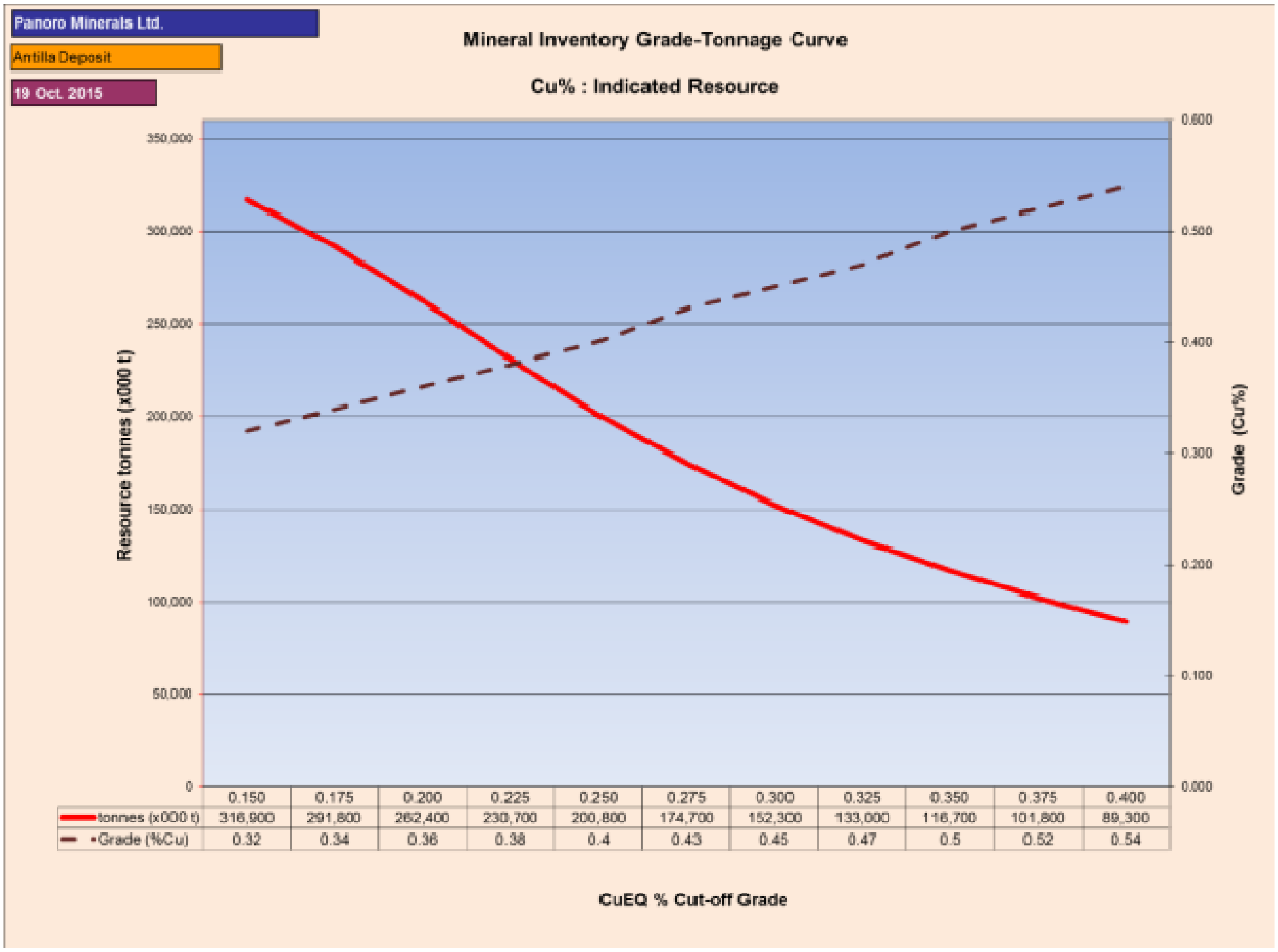
14.10 Grade Sensitivity Analysis

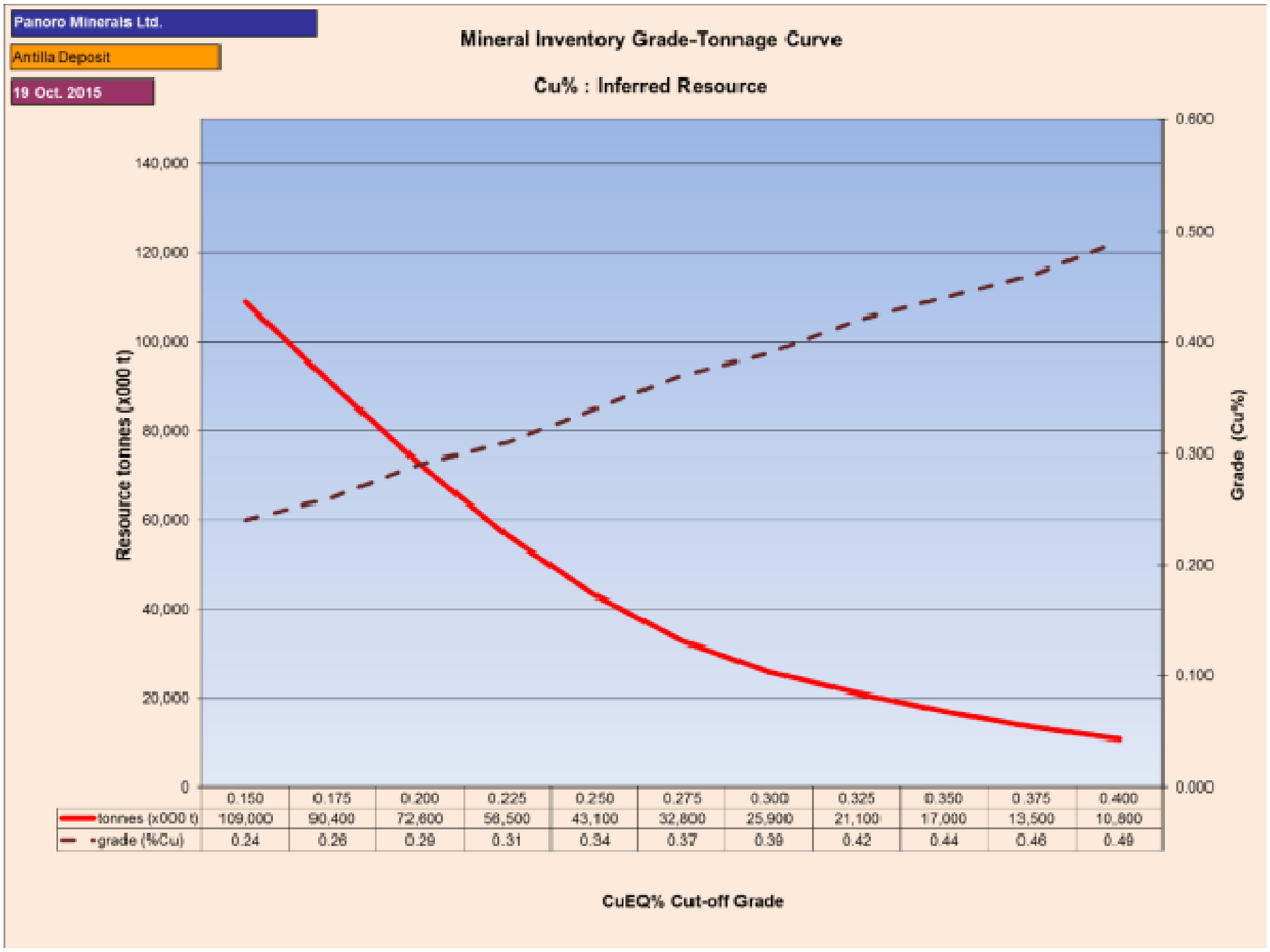
The mineral resources of the Antilla project are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity the block model quantities and grade estimates within the conceptual pit used to constrain the mineral resources are presented in Table 14-14 at different cut-off grades. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade. Figure 14-13 and Figure 14-14 presents this sensitivity as grade tonnage curves.

Table 14-14 Block Model Quantities and Grade Estimates*, Antilla Project at Various Cut-off Grades

Indicated					Inferred			
CuEQ% Cut-off	Tonnes ('000)	Cu (%)	Mo (%)	CuEQ (%)	Tonnes ('000)	Cu (%)	Mo (%)	CuEQ (%)
0.400	89,300	0.54	0.01	0.56	10,800	0.49	0.01	0.51
0.375	101,800	0.52	0.01	0.54	13,500	0.46	0.01	0.48
0.350	116,700	0.50	0.01	0.52	17,000	0.44	0.01	0.46
0.325	133,000	0.47	0.01	0.49	21,100	0.42	0.01	0.44
0.300	152,300	0.45	0.01	0.47	25,900	0.39	0.01	0.41
0.275	174,700	0.43	0.01	0.45	32,800	0.37	0.01	0.39
0.250	200,800	0.40	0.01	0.42	43,100	0.34	0.01	0.36
0.225	230,700	0.38	0.01	0.40	56,500	0.31	0.01	0.33
0.200	262,400	0.36	0.01	0.38	72,600	0.29	0.01	0.30
0.175	291,800	0.34	0.01	0.36	90,500	0.26	0.01	0.28
0.150	316,900	0.32	0.01	0.34	109,200	0.24	0.01	0.26

**The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of a cut-off grade.*


Panoro Minerals Ltd.
Antilla Deposit
19 Oct. 2015
Mineral Inventory Grade-Tonnage Curve
Cu% : Indicated Resource
Resource tonnes (x000 t)
Grade (Cu%)
350,000
300,000
250,000
200,000
150,000
100,000
50,000
0
0.600
0.500
0.400
0.300
0.200
0.100
0.000
0.150 0.175 0.200 0.225 0.250 0.275 0.300 0.325 0.350 0.375 0.400
tonnes (x000 t) 316,900 291,800 262,400 230,700 200,800 174,700 152,300 133,000 116,700 101,800 89,300
Grade (%Cu) 0.32 0.34 0.36 0.38 0.4 0.43 0.45 0.47 0.5 0.52 0.54
CuEQ % Cut-off Grade


Panoro Minerals Ltd.
Antilla Deposit
19 Oct. 2015
Mineral Inventory Grade-Tonnage Curve
Cu% : Inferred Resource
Resource tonnes (x000 t)
Grade (Cu%)
0
20,000
40,000
60,000
80,000
100,000
120,000
140,000
0.000
0.100
0.200
0.300
0.400
0.500
0.600
0.150 0.175 0.200 0.225 0.250 0.275 0.300 0.325 0.350 0.375 0.400
tonnes (x000 t) 109,000 90,400 72,800 56,500 43,100 32,800 25,900 21,100 17,000 13,500 10,800
grade (%Cu) 0.24 0.26 0.29 0.31 0.34 0.37 0.39 0.42 0.44 0.46 0.49
CuEQ% Cut-off Grade

15 Mineral Reserve Estimates

No mineral reserve estimate has been completed on the Antilla project. This PEA study is conceptual in nature and includes Inferred Resources.

The CIM definition of a Mineral Reserve is “the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study.” A PFS has not been completed on the Antilla project and therefore it is not possible to declare a Mineral Reserve of any kind.

16 Mining Method

A scoping level mine design, production schedule, and associated mining cost model is developed for the Antilla project based on an open pit mining method. The production schedule is based on a 20,000 tonne/day leach material rate. All currency values are stated in USD.

16.1 Introduction

The mine planning work is based on the mineral resource model created by Tetra-Tech discussed herein. The mine planning for the Antilla project is based on work completed with MineSight by Hexagon Mining, a suite of software proven in the industry.

Mine planning relies on geological information from the block model and base metal prices. Other information used for mine planning includes: pit slope angles, off-site costs, metallurgical recoveries (Tetra Tech), process costs, and throughput rates (Panoro). The information provided was reviewed by Moose Mountain and is deemed acceptable for use in a scoping level study for this project.

Detailed pit phases are designed using the results of an economic pit limit analysis. The sub-set of the mineral resources contained within the detailed pit design is shown in Table 16-1.

Table 16-1 Summarized In-Pit Resources

	In Situ Undiluted Grades		
	Leach Material	**NSR**	Cu
	kT	**USD/tonne**	%
Indicated	113,300	$21.52	**0.450**
Subtotal of Measured + Indicated	**113,300**	**$21.52**	0.450
Inferred	**5,400**	**$12.27**	**0.259**

**The cut-off grade used is NSR>=$6.10/tonne. NSR is calculated as follows: NSR ($/tonne) = Cu grade (%) * Cu process recovery (%) * 57.76*

The potentially mineable tonnages considered for this preliminary economic analysis include Inferred mineral resources. The reader is cautioned that Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable categorization as mineral reserves. There is no certainty that Inferred mineral resources will ever be upgraded to Mineral reserves. Mineral resources that are not Mineral reserves do not have demonstrated economic viability.

Internal dilution is included in the whole block grades and is considered sufficient for this stage of study. Mining loss is assumed to be equal to mining dilution and since dilution material carries grade they are assumed to cancel each other out.

All classes of Mineral resource are considered as leach material in the mine planning production schedule.

16.2 Mine Planning 3D Block Model (3DBM) and MineSight Project

All mine planning uses whole block grades for copper percent (CUC). The block model also contains a density item (DENS) based on the rock type, a class item (CLASS) showing whether the material is Measured, Indicated or Inferred, and a topography item (TOPO%) representing the percentage of the block below the topographic surface provided.

16.3 Production Rate Consideration

The throughput chosen for the Antilla project is 20,000 tonnes/day. The ultimate pit contains approximately 16.5 years of leach material at the chosen throughput.

16.4 Net Smelter Return (NSR)

A net smelter return (NSR) value, expressed in USD/tonne, is used as a cut-off grade to determine whether a block is leach material or waste. The NSR is calculated for each mineralized block in the mineral resource model using a net smelter price (NSP), heap leach process recoveries, and in situ grades. The NSP for each metal is the market price net of off-site charges such as cathode transportation, freight, and distribution. As such, the NSP represents the metal price available at the mine gate. Any material inside the pit limit with an NSR value greater than the process plus G&A cost is incrementally economic (since total mining and processing costs are accounted for in the ultimate pit limit analysis) and should be sent to the leach pad. Table 16-2 shows the base case metal prices used along with the NSP values. Expected process recoveries vary by rock type and are shown in Table 16-3.

Table 16-2 Metal Prices

	Base Price	NSP
	US$/lb	US$/lb
Copper	**$3.00**	**$2.97**

Table 16-3 Process Recoveries

Rock Type	Process Recoveries - %
	Cu
Cover	**31.1%**
Leach Cap	**38.0%**
Supergene	**72.5%**
Primary Sulphide	**21.2%**

The NSR is calculated as follows: NSR (USD/tonne) = Cu(%) * CuRec(%) * 22.046 * NSPCu
Where:

- Cu(%) = in situ grade of copper expressed as %
- CuRec(%) = process recovery for copper expressed as %
- NSPCu = Net smelter price for copper in USD/lb

16.5 Economic Pit Limits and Pit Designs

Economic pit limits for the Antilla deposit are determined for this study using Lerchs-Grossman pit optimization using MineSight Economic Planner (MS-EP).

16.5.1 Pit Optimization Method

The Pit Optimization method used is a Lerchs-Grossman assessment, carried out by generating a series of pit shells at varying revenue and cost assumptions. This tests the deposits' geometric and topographic sensitivity. For this preliminary economic assessment, Measured, Indicated, and Inferred mineral resources are included in the Lerchs-Grossman economics.

16.5.2 Economic Pit Limit Assessment

The results from earlier scoping studies performed on Antilla were used as inputs for mining and processing costs in the Lerchs-Grossman analysis. These costs are deemed suitable for this level of study and are shown in Table 16-4 below.

Table 16-4 Lerchs-Grossman Unit Costs

	US$/tonne
Mining Costs	$1.65
Processing + G&A Costs	$3.75

16.5.3 Pit Slope Angles

Geotechnical recommendations are derived from earlier scoping studies performed on Antilla, based on the leach cap, supergene, and primary sulphide rock types. Overall Pit Slope Angles are estimated to accommodate the inclusion of ramps in the high wall. The pit slope angles used in the Lerchs-Grossman assessment are summarized in Table 16-5 below.

Table 16-5 Pit Slope Angles Used in Lerchs-Grossman Assessment

Rock Type	Overall Pits
Overburden Cover	24
Leach Cap	42
Supergene	48
Primary Suphide	49
Country Rock	52

16.5.4 Process Recoveries

The Lerchs-Grossman runs use the calculated NSR value for each block. Process recoveries by rock type are shown in Table 16-4.

16.5.5 LG Economic Pit Limits

A series of Lerchs-Grossman shells are created to assess the incremental economics of increasing mining limits. By varying the metal prices from low to high values, the geometry of the mineralized deposit is tested, where low metal prices require high grades and/or low strip ratios and high metal prices can generate incremental revenues to mine lower grade, higher strip ratio areas. The larger pit shells have a greater amount of potentially mineable mineral resources capable of supporting larger capital expenditure, but the extra material has lower economic margins (revenues minus costs). The smaller pit shells have higher margins but create smaller projects and can be more capital sensitive. Note: This is not a price-sensitivity study since the in-pit resource tonnages for all pit shells are calculated at the same NSR cut-off grade.

The ultimate economic pit limit is chosen where an incremental increase in pit size does not significantly increase the pit-delineated resource and therefore has limited potential for a positive economic margin. Alternatively, an ultimate pit shell can be selected which considers other factors such as maximum waste capacity available, maximum pit wall height, etc.

Metal prices are adjusted upwards and downwards in 10% increments from 40% to 130% to generate a series of shells. Potential leach material tonnes inside each shell are calculated using a constant NSR cut-off of US$3.75/tonne. The results are shown graphically in Figure 16-1.

Figure 16-1 inflection points are observed at the 60% shell and the 100% shell. This indicates that the 100% case would be an ideal size for the economic pit limit. However, selecting a shell less than the 100% case ensures that the last pit shell produces positive cash flows at metal prices lower than the base case prices. Therefore, for contingency reasons, the 90% shell is selected for the economic ultimate pit limit. It should be noted that the 90% shell will still generate positive cash flow at metal prices lower than the base case. Figure 16-2 and Figure 16-3 show plan and sectional views of the 90% shell.

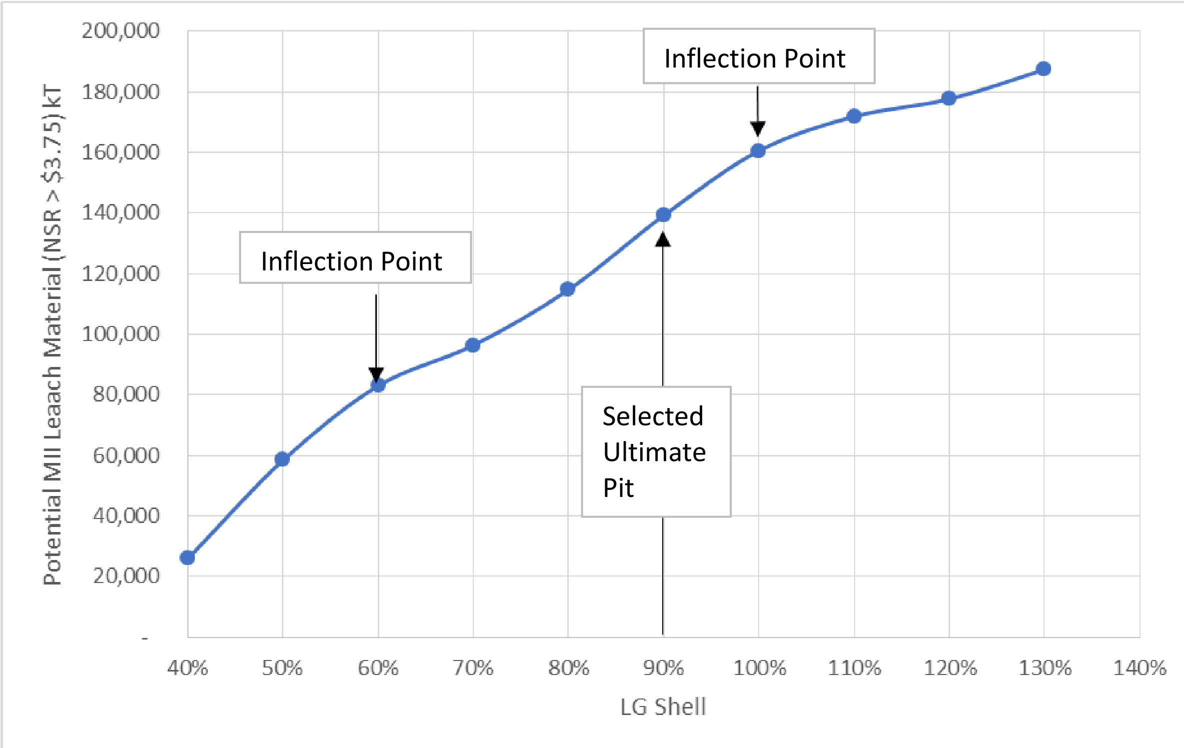


Figure 16-1 Lerchs-Grossman Sensitivity Summary (NSR>$3.75/tonne)

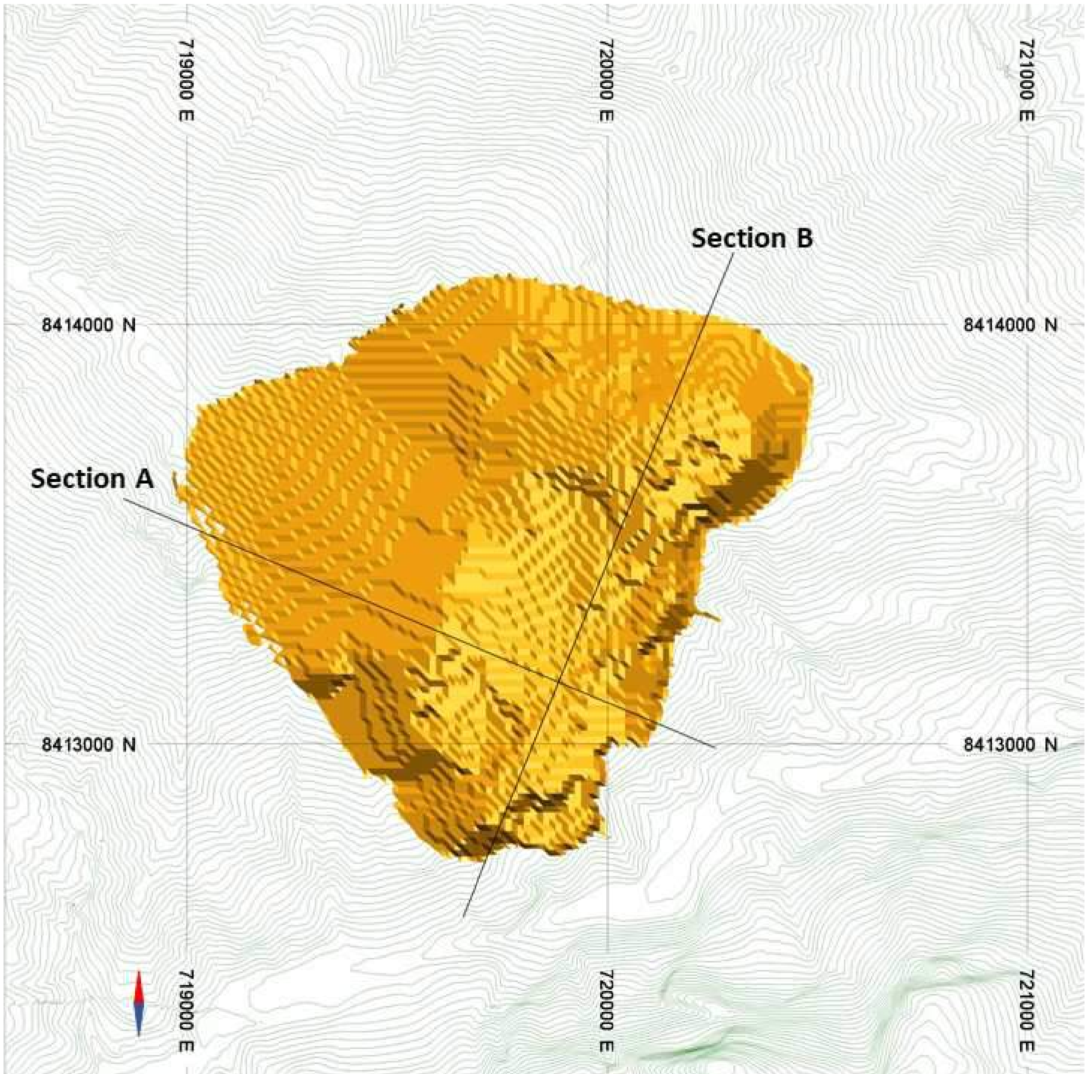
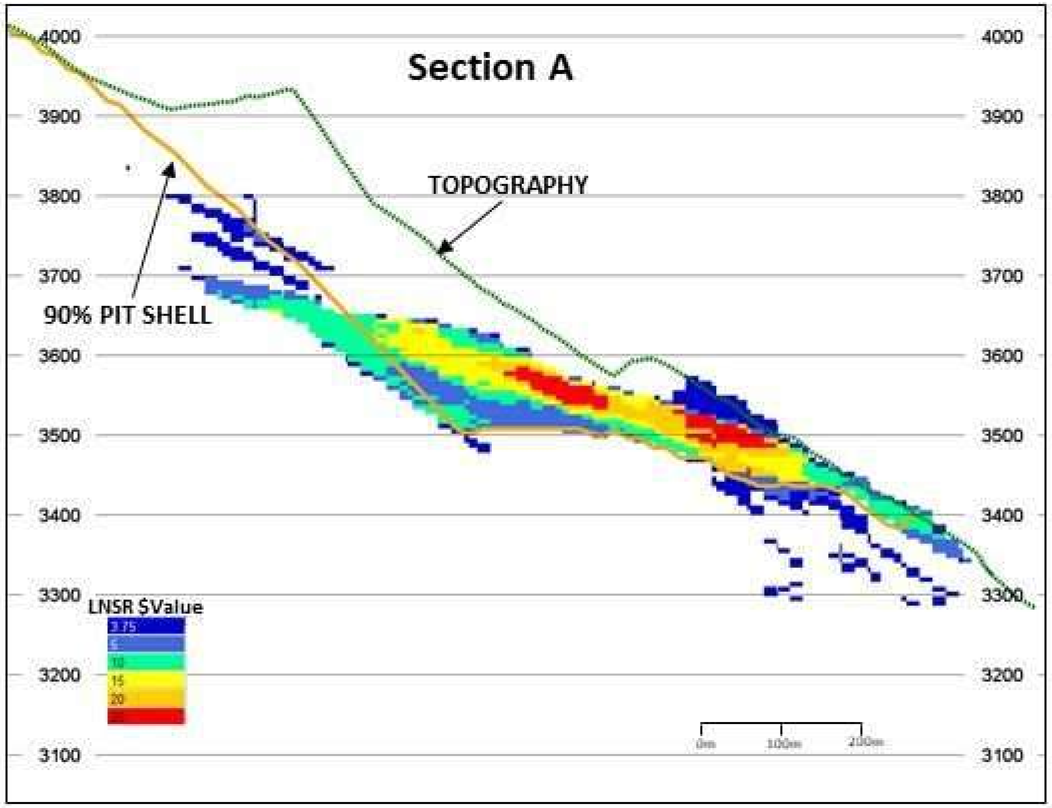


Figure 16-2 Plan View of 90% Lerchs-Grossman Shell



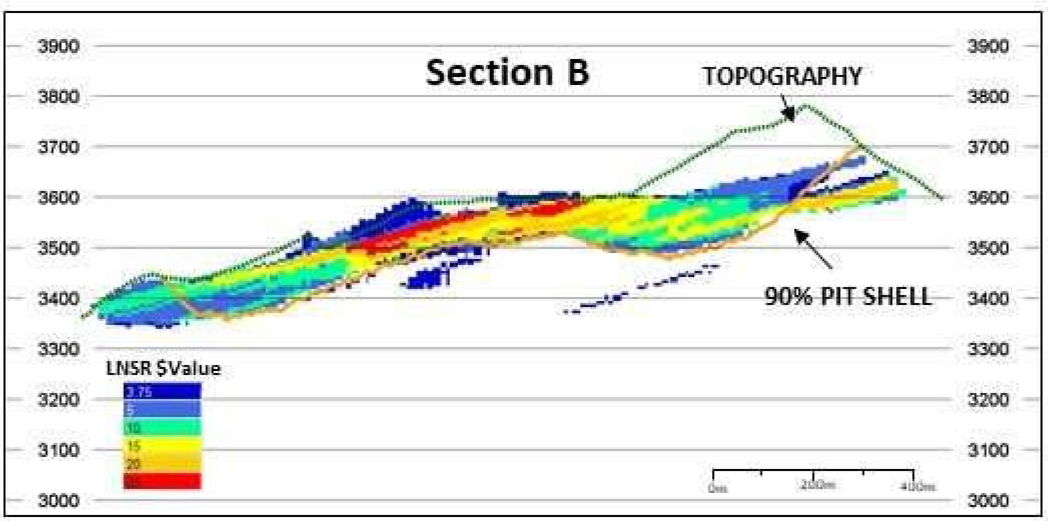


Figure 16-3 Top: Section A of 90% Lerchs-Grossman Shell. Bottom: Section B of 80% Lerchs-Grossman Shell (Measured, Indicated, and Inferred blocks shown with their corresponding NSR value)

Economic pit limits are determined early in the design cycle and are based on the best available data and parameters at the time. Detailed mine planning including haulage costs has indicated that an equivalent constant average LOM cut-off grade of $8.10/tonne is more appropriate. To check the validity of the early results of the economic pit limit analysis, LG pit shells were run again at the end of the design cycle analysis with NSR cut-off of US$8.10/tonne. The selected ultimate economic pit limit using the lower cut-off grade is less than the 100% case using the higher cut-off grade. Therefore, it is determined that the selected ultimate pit limit remains valid, and will still generate positive economics, even at a higher cut-off grade.

16.5.6 Detailed Pit Design

The detailed pit designs were developed with MineSight software and follow the economic pit limits determined in Section 16.5.5. Bench heights used for pit designs are based on the block height of the 3DBM and do not necessarily match the optimal digging bench heights for the mobile mining fleet. However, drilling and blasting can be done on 12m benches to follow the recommended pit designs.

The parameters used for the detailed designs are shown in Table 16-6 and Table 16-7.

Table 16-6 Pit Design Parameters

Parameter	Value	Un
Bench height	12	m
Berm width	13	m
Vertical spacing between berms	24	m
Minimum mining width between phases	100	m
Minimum mining width operational (i.e., at pit bottoms)	30	m
Maximum ramp grade	**10**	%

Table 16-7 Pit Slope Parameters

Rock Type	Face Angle (°)	Inter-Ramp Angle (°)
Overburden Cover	55	40
Leach Cap	65	45
Supergene	75	51
Primary Sulphides	78	52
Country Rock	78	52

It is assumed that ramps will be incorporated in the final pit walls. However, where ramps are not required, it is recommended to design a 20-metre-wide "geotechnical berm" every 100 metres in elevation. Detailed design of geotechnical berms in final pit walls is not included at a scoping level. Instead an allowance for future designs is implemented, by designing all berm widths at 13m which is slightly more conservative than designing 10m berms with a 20m geotechnical berm every 100 meters.

The ultimate pit, shown in Figure 16-7, ranges from an elevation of 4,184 metres down to 3,364 metres. The maximum wall height is 820 metres. Most of the final high wall does not require ramps, as mining operations will tie into external haul roads that progress down the mountain side outside the pit boundary.

The detailed pit design is comprised of four mining phases, designed to even out waste stripping and leach material tonnes mined throughout the schedule.

Phase 1, shown in Figure 16-4, targets areas of higher economic return to increase the early economics of the project. Mining begins at an elevation of 3,819 metres, with narrow benches requiring cast blasting and/or dozer push mining, allowing waste to be pushed down to a lower elevation where it can be loaded and hauled more efficiently. Phase 1 mines down to an elevation of 3,448 metres.

Phase 2, shown in Figure 16-5, is a push back to the southwest side of Phase 1 that eventually mines out the bottom of Phase 1. Due to the terrain in the higher elevations of Phase 2, it requires extensive cast blasting and/or dozer push mining before an operable truck mining bench can be developed. Mining in Phase 2 ranges from 4,164 metres down to 3,364 metres.

Phase 3, shown in Figure 16-6, is a push back to the north side of Phase 1, developing the northern half of the final highwall of the ultimate pit. Mining in Phase 3 ranges from 4,108 metres down to 3,484 metres, establishing the northern ultimate pit bottom.

Phase 4, shown in Figure 16-7, is a push back to the southwest of the other phases, developing the southern half of the final highwall of the ultimate pit. The Phase 4 pit bottom expands the pit bottom established by phase 2 but does not mine deeper. Mining in Phase 4 ranges from 4,175 metres down to 3,364 metres.

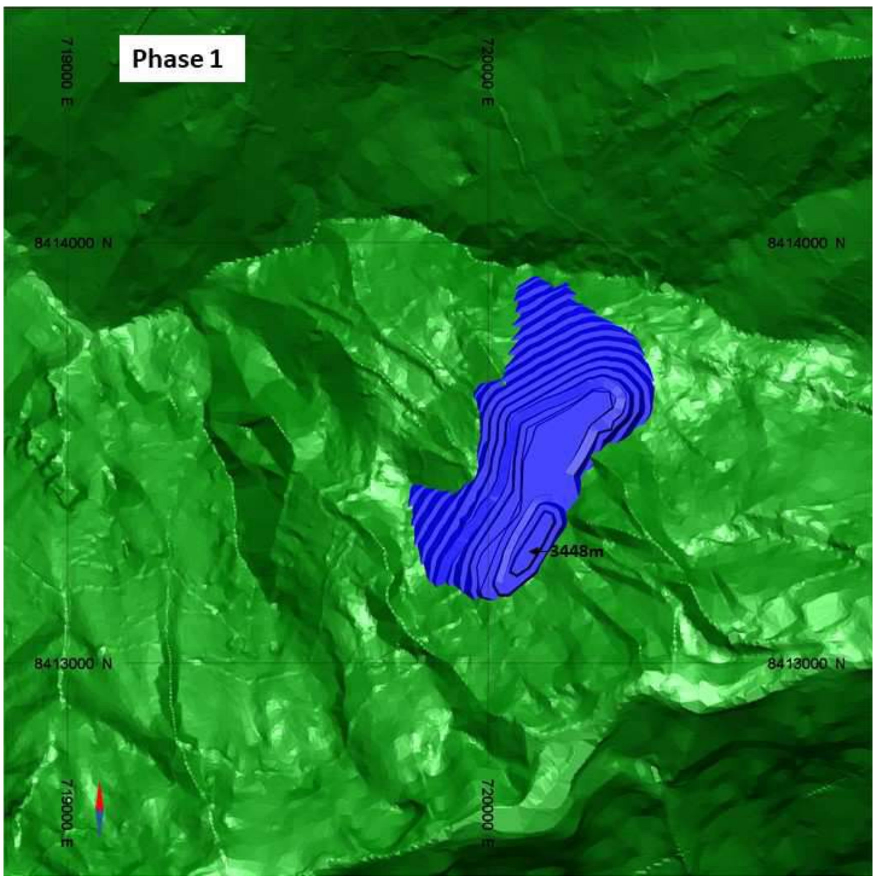


Figure 16-4 Detailed Pit Design Phase 1

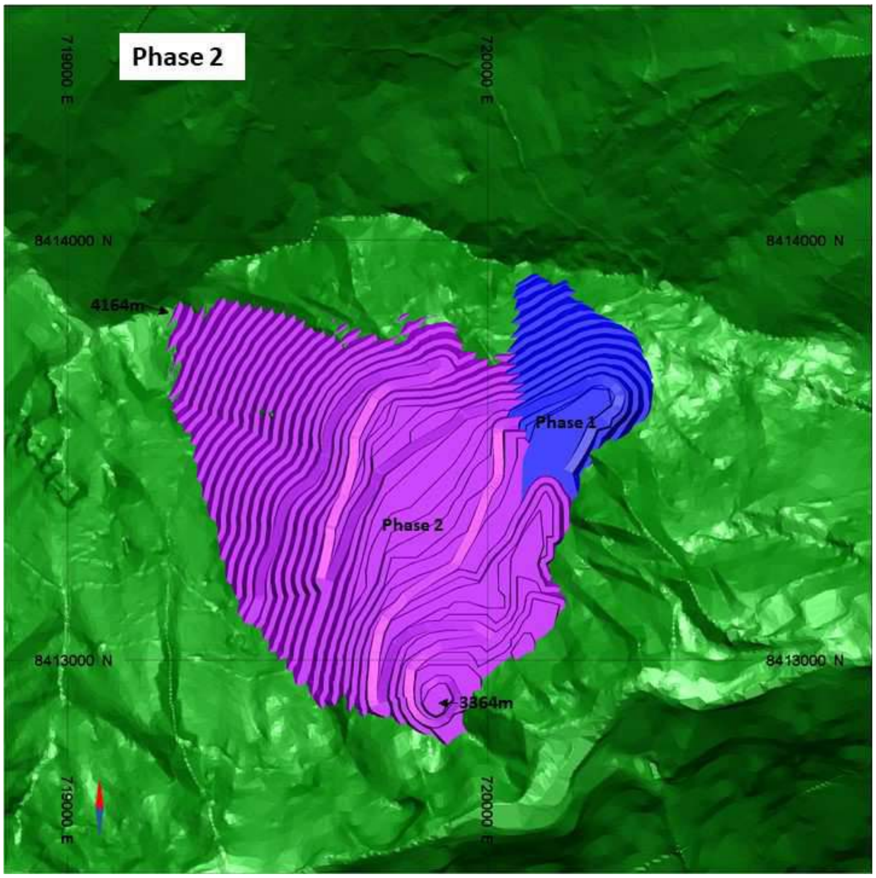


Figure 16-5 Detailed Pit Design Phase 2

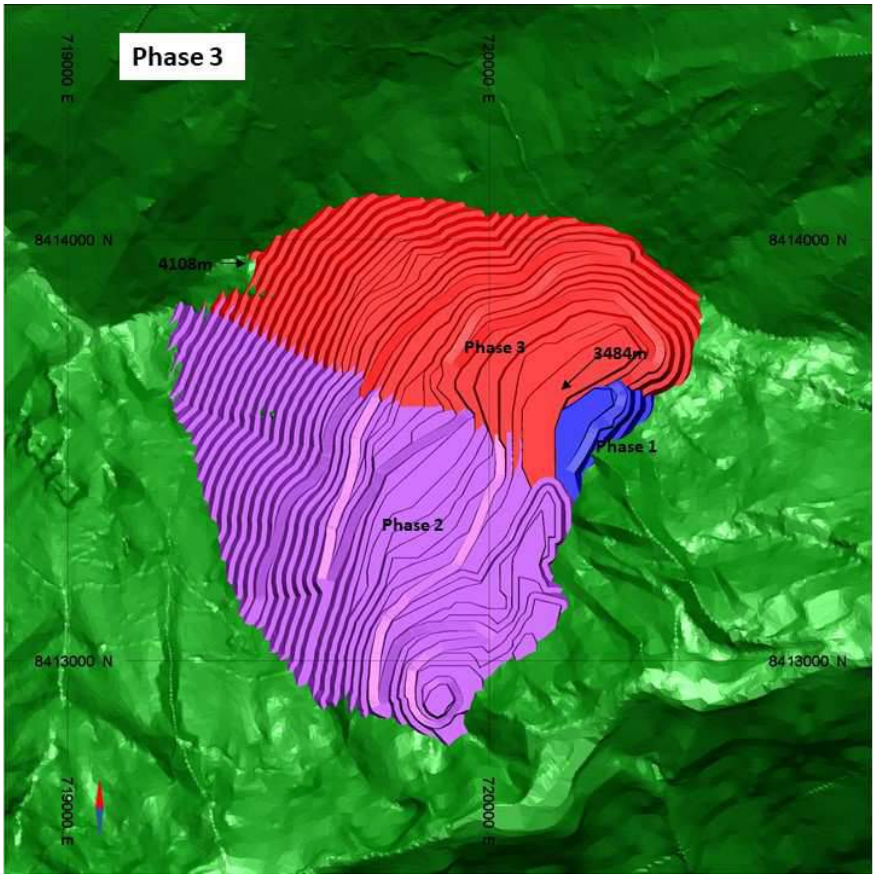


Figure 16-6 Detailed Pit Design Phase 3

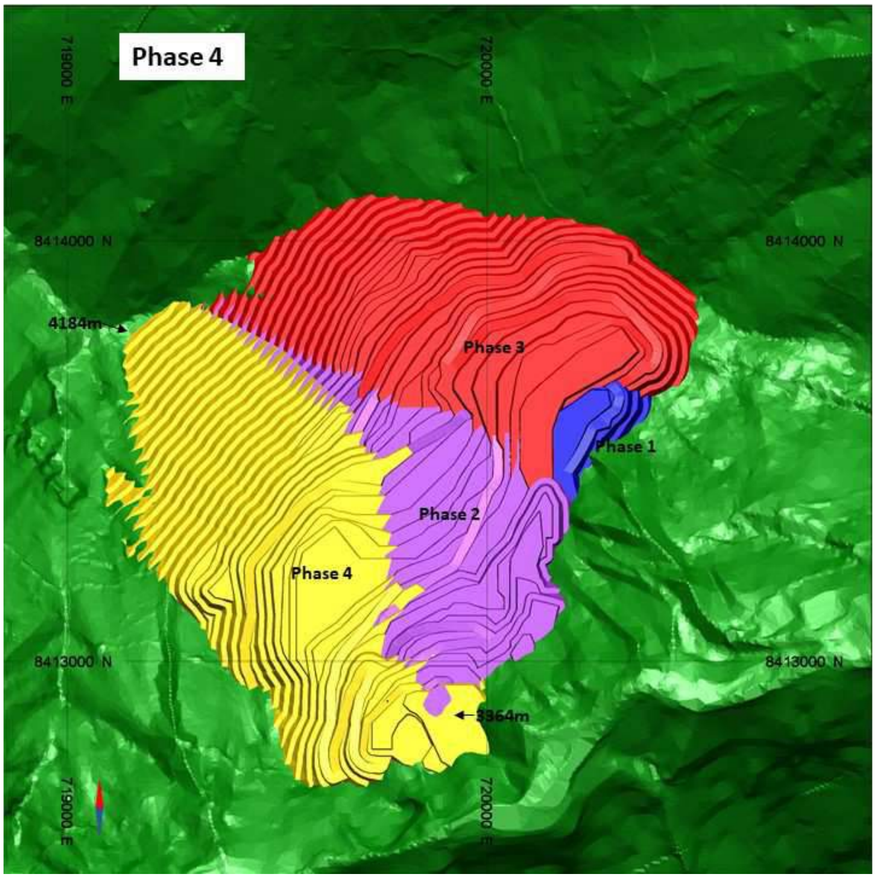


Figure 16-7 Detailed Pit Design Phase 4 (Ultimate Pit)

16.6 In-Pit Leach Material

The NSR value is used to determine whether a block inside the economic ultimate pit is waste or leach material. A variable NSR cut-off of is used to select leach material for the production schedule on an annual basis. The NSR cut-off must be greater than the processing + G&A costs for the block to be sent to the leach pad. Material sent to stockpile must also include rehandle costs to be considered economic. Processing costs increase with time as the overhaul cost required to move leach material from the crusher to the top of the leach pad becomes larger.

The average life-of-mine (LOM) NSR cut-off is calculated as $8.10/tonne.

A summary of the waste/leach material by pit phase is shown in Table 16-8 below.

Table 16-8 Summary of Waste/Leach Material by Pit Phase

	Leach Material	NSR	Cu	Waste	
	kT	US$/tonne	%	kT	S/R (tonnes waste : tonnes ore)
Phase 1	8,300	$21.79	0.462	13,900	1.67
Phase 2	39,200	$25.34	0.529	49,000	1.25
Phase 3	39,900	$18.62	0.391	51,400	1.29
Phase 4	31,200	$18.78	0.392	58,200	1.87
Total	**118,700**	**$21.10**	**0.442**	**172,400**	**1.45**

16.7 Mine Plan

The Antilla mine plan includes 1 ½ years of pre-production and 16 ½ years of standard open pit mining operations, for a total mine life of 18 years.

The mine plan utilizes open pit mining methods, with mine rock being used to construct the base of the leach pad well as reporting to the Rock Storage Facility (RSF). Potential leach material mined from the pits reports either directly to the leach pad or is stored in a stockpile.

As seen in Table 16-3, the Supergene rock type has a process recovery that is considerably higher than all other rock types. After analysis of different mining scenarios, the selected mine plan focused on sending only Supergene material to the leach pad for the first 10 years of production. All other rock types with an economic cut-off grade are sent to stockpile. This improves the project cash flow, while preventing economic material from being wasted. Stockpiled material is reclaimed as needed in the later years of the mine schedule.

16.7.1 Conceptual Mine Production Schedule

The conceptual mine production schedule is based on 7.3 million annual tonnes of leach material (20,000 tonnes/day) with with earlier, higher grades accomplished through a stockpiling strategy. The stockpiling strategy results in higher material movement per year but improved cash flows. The stockpile is only reclaimed throughout the mine life if it is economically beneficial to do so.

The conceptual mine production schedule is based on a 12-metre high operating bench rather than the 18-metre bench proposed by the geotechnical design criteria in Table 16-9. Geotechnical criteria were determined prior to the selection of a mine fleet, and therefore do not represent an optimal digging face for the selected fleet. This does not affect the validity of the detailed pit designs or the mine production schedule since the overall pit slope angles are followed in the detailed pit designs.

The conceptual mine production schedule is summarized in Figure 16-8 and Table 16-9.

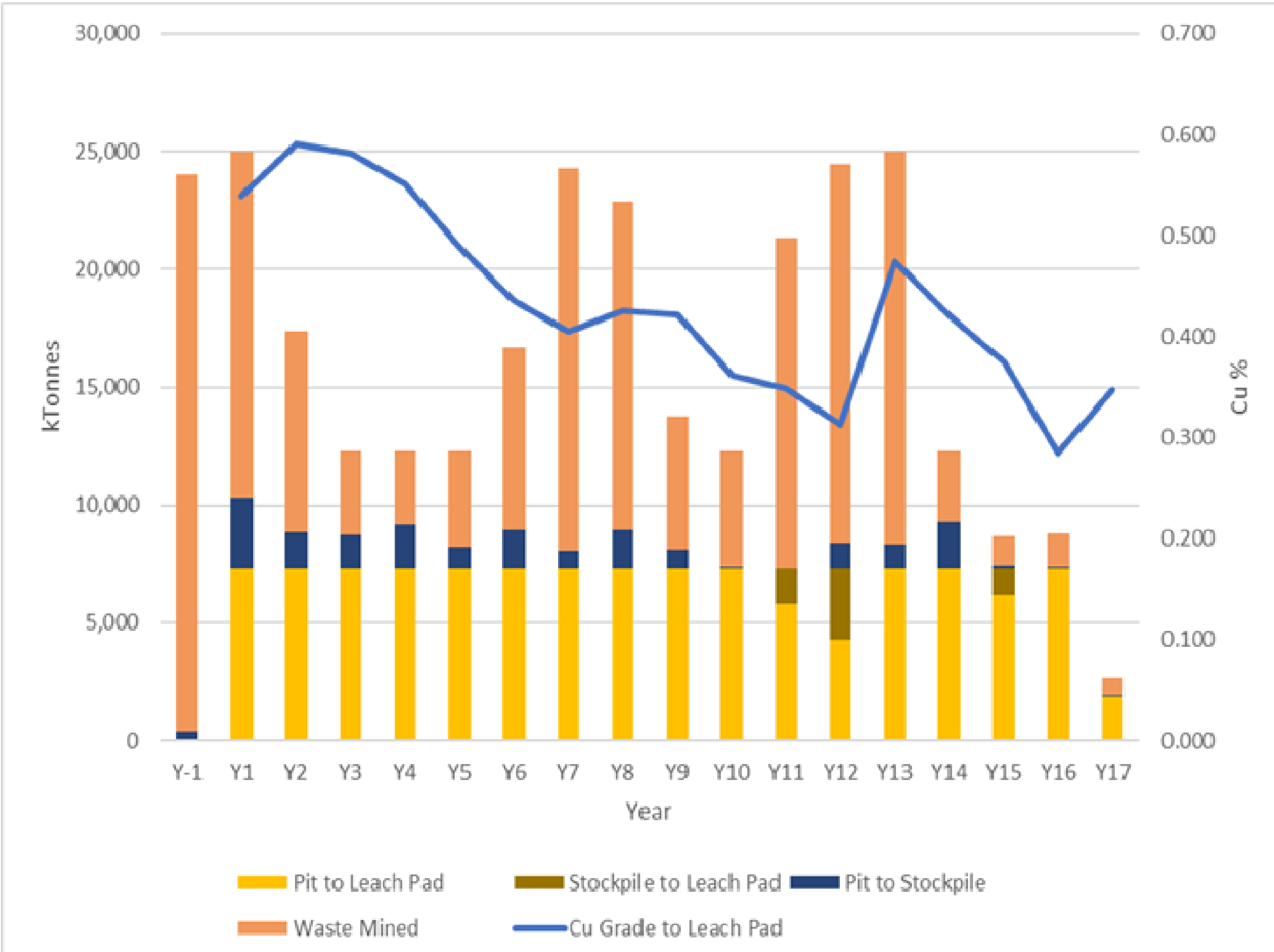


Figure 16-8

Table 16-9 Conceptual Mine Production Schedule Summary

	YR	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	LOM
Leach Material Total	**kT**			**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**1,866**	**118,667**
NSR	USD/t			25.65	28.04	28.45	26.88	24.04	20.67	19.21	20.23	20.31	17.16	16.28	13.25	23.31	20.09	18.37	13.51	16.51	20.90
Cu Grade	%			0.540	0.591	0.581	0.551	0.488	0.435	0.405	0.426	0.421	0.362	0.349	0.314	0.475	0.420	0.375	0.285	0.348	0.437
Pit to Leach Pad	**kT**			**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**5,818**	**4,300**	**7,300**	**7,300**	**6,225**	**7,300**	**1,866**	**113,110**
NSR	USD/t			25.65	28.04	28.45	26.88	24.04	21	19	20.23	20.31	17.16	18.23	16.55	23	20	18.14	13.51	16.51	21.40
Cu Grade	%			0.540	0.591	0.581	0.551	0.488	0	0	0.426	0.421	0.362	0.384	0.348	0	0	0.370	0.285	0.348	0.445
Pit to Stockpile	**kT**		**371**	**3,028**	**1,526**	**1,409**	**1,853**	**915**	**1,635**	**764**	**1,631**	**840**	**44**	**7**	**1,093**	**1,020**	**1,952**	**92**	**66**	**72**	**18,319**
NSR	USD/t		7.45	10.02	7.03	6.74	7.79	6.43	9.09	7.82	9.24	7.73	6.09	6.89	6.99	7.01	14.26	5.62	5.77	6.51	8.75
Cu Grade	%		0.288	0.311	0.296	0.301	0.273	0.327	0.280	0.235	0.254	0.187	0.437	0.496	0.284	0.275	0.298	0.129	0.175	0.306	0.283
Stockpile to Leach Pad	**kT**													**1,482**	**3,000**			**1,075**			**5,557**
NSR	USD/t													8.65	8.52			19.75			10.73
Cu Grade	%													0.212	0.264			0.400			0.276
Stockpile Balance, End of Year	kT		371	3,400	4,926	6,334	8,188	9,102	10,737	11,501	13,132	13,972	14,016	12,541	10,634	11,654	13,606	12,623	12,689	12,762	12,762
Waste Mined	kT	4,000	23,625	14,672	8,568	3,591	3,147	4,085	7,731	16,219	13,948	5,620	4,956	14,036	16,072	16,680	3,048	1,260	1,435	747	159,439
Total Material Mined	**kT**	**4,000**	**23,996**	**25,000**	**17,394**	**12,300**	**12,300**	**12,300**	**16,666**	**24,283**	**22,879**	**13,761**	**12,300**	**19,861**	**21,465**	**25,000**	**12,300**	**7,577**	**8,801**	**2,685**	**290,868**
Total Material Moved	**kT**	**4,000**	**23,996**	**25,000**	**17,394**	**12,300**	**12,300**	**12,300**	**16,666**	**24,283**	**22,879**	**13,761**	**12,300**	**21,343**	**24,465**	**25,000**	**12,300**	**8,652**	**8,801**	**2,685**	**296,425**

16.7.2 Rock Storage Facility (RSF)

Initial mine rock from the open pit is used to construct the leach pad base and catch berm. Where required it is also used as fill for construction of infrastructure pads. The remainder of mine rock is stored in one of three areas of the RSF (Upper, Middle, Lower).

The RSF is placed either as top-down or wrap-around lifts, at angle of repose (37 degrees). Some parts of the RSF include haul roads built into the face which reduces the overall face angles. A swell factor of 1.3 is applied to all material placed in the RSFs. The capacities of the different areas of the RSF is outlined in Table 16-10 and its location is shown in Figure 16-9.

Table 16-10 Rock Storage Facility Capacity Summary

Location	Final Elevation (m)	Total ('000 m^3)
Leach Pad Base	3600	**4,700**
Upper RSF	3950	**29,800**
Middle RSF	3800	**40,200**
Lower RSF	3630	**12,600**
Total RSF		**87,300**

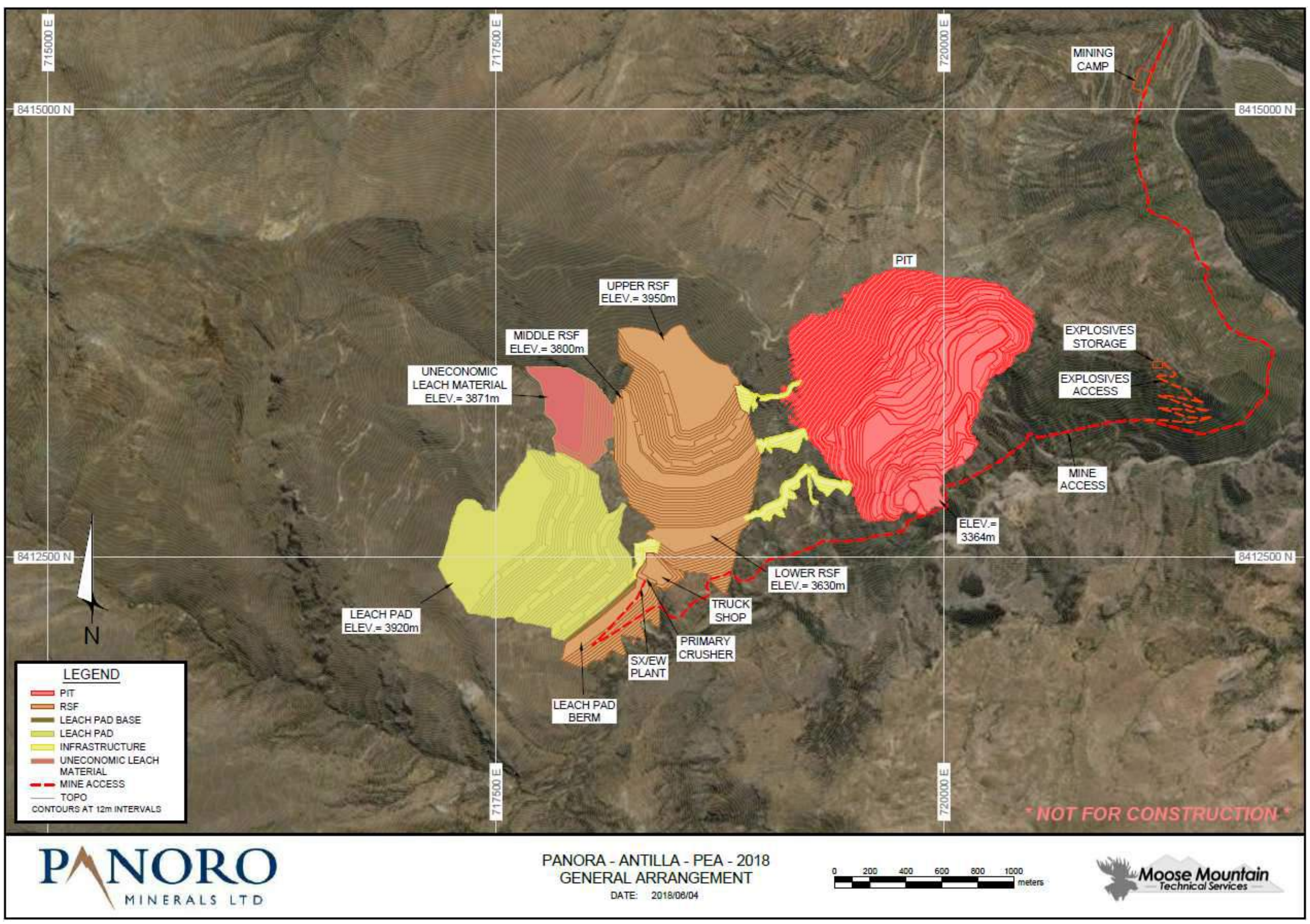


Figure 16-9 Project General Arrangement showing RSF

16.7.3 Leach Pad and Stockpile

The leach pad is constructed on a lined earth fill pad made from mine waste rock at an elevation of 3600 metres. An 80-90 metre wide berm is included in the base of the leach pad for stability and collection of run off. The leach pad is constructed at an overall slope angle of approximately 30 degree, and is accessed by a 22.8 metre wide 8% grade haul road that crosses the face of the leach pad at even intervals The maximum height of the leach pad is 320 metres, reaching a final elevation of 3,920 metres and containing 118.7 million tonnes of crushed leach material.

The leach material stockpile is located in a natural drainage between the open pit and leach pad. It is constructed top-down at angle of repose (37 degrees), with a maximum height of approximately 200 metres. The stockpile is accessed by haul roads from the middle RSF and the top of the leach pad. By the end of the mine life some of the stockpile becomes uneconomic due to increasing processing costs as a function of leach pad height. The maximum size of the stockpile is approximately 14.0 million tonnes and occurs in Year 10 of the schedule. By the end of the mine life the stockpile contains 12.7 million tonnes of leach material that is no longer economic.

The leach pad and stockpile location is shown in Figure 16-9.

16.7.4 Details for the Conceptual Mine Production Schedule

The details of the conceptual mine production schedule is provided for three sample periods from the production schedule: End of Preproduction (Figure 16-10), End of Year 5 (Figure 16-11), and Life of Mine (Figure 16-12).

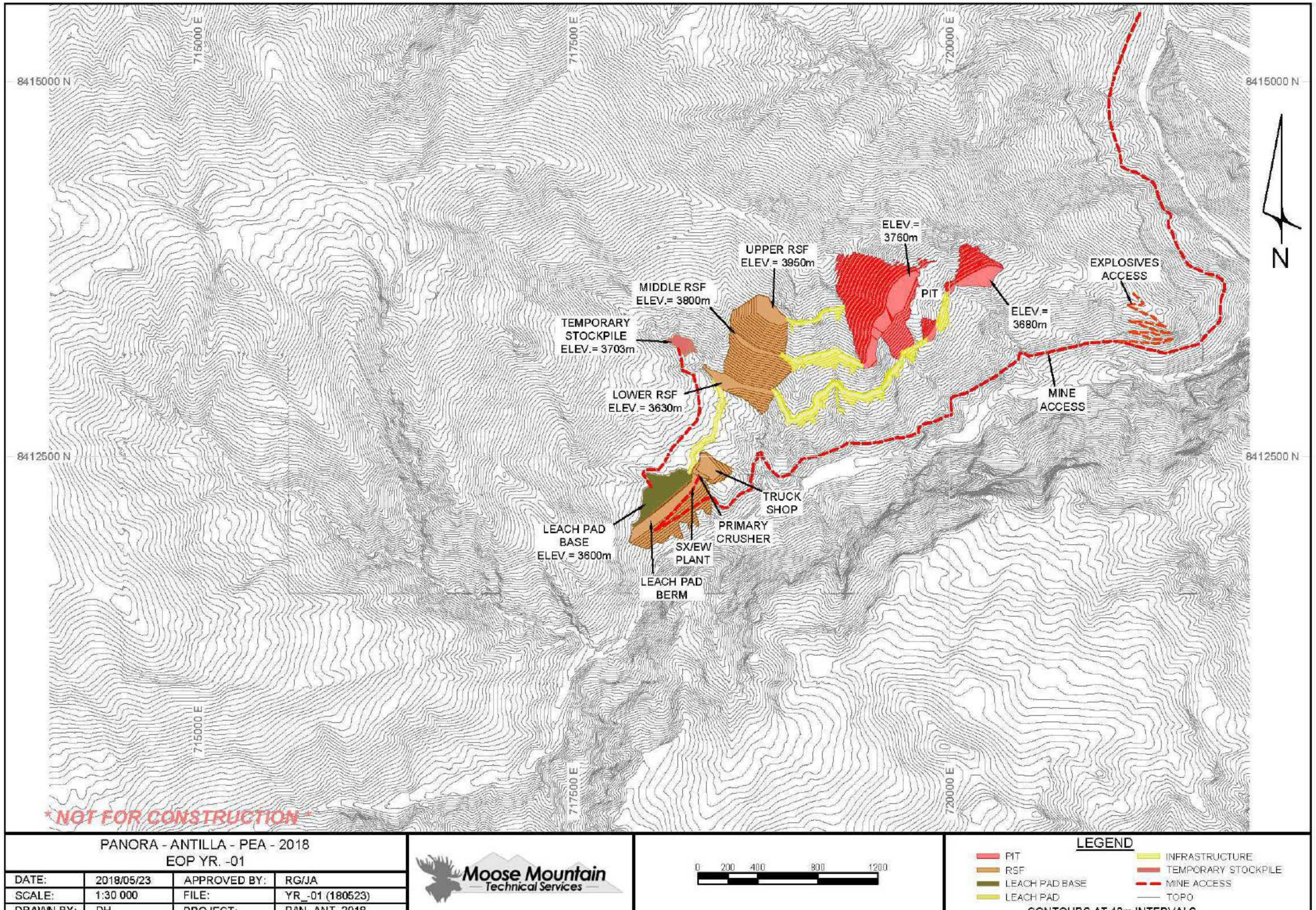


Figure 16-10 End of Pre-production (Time Zero) Detail

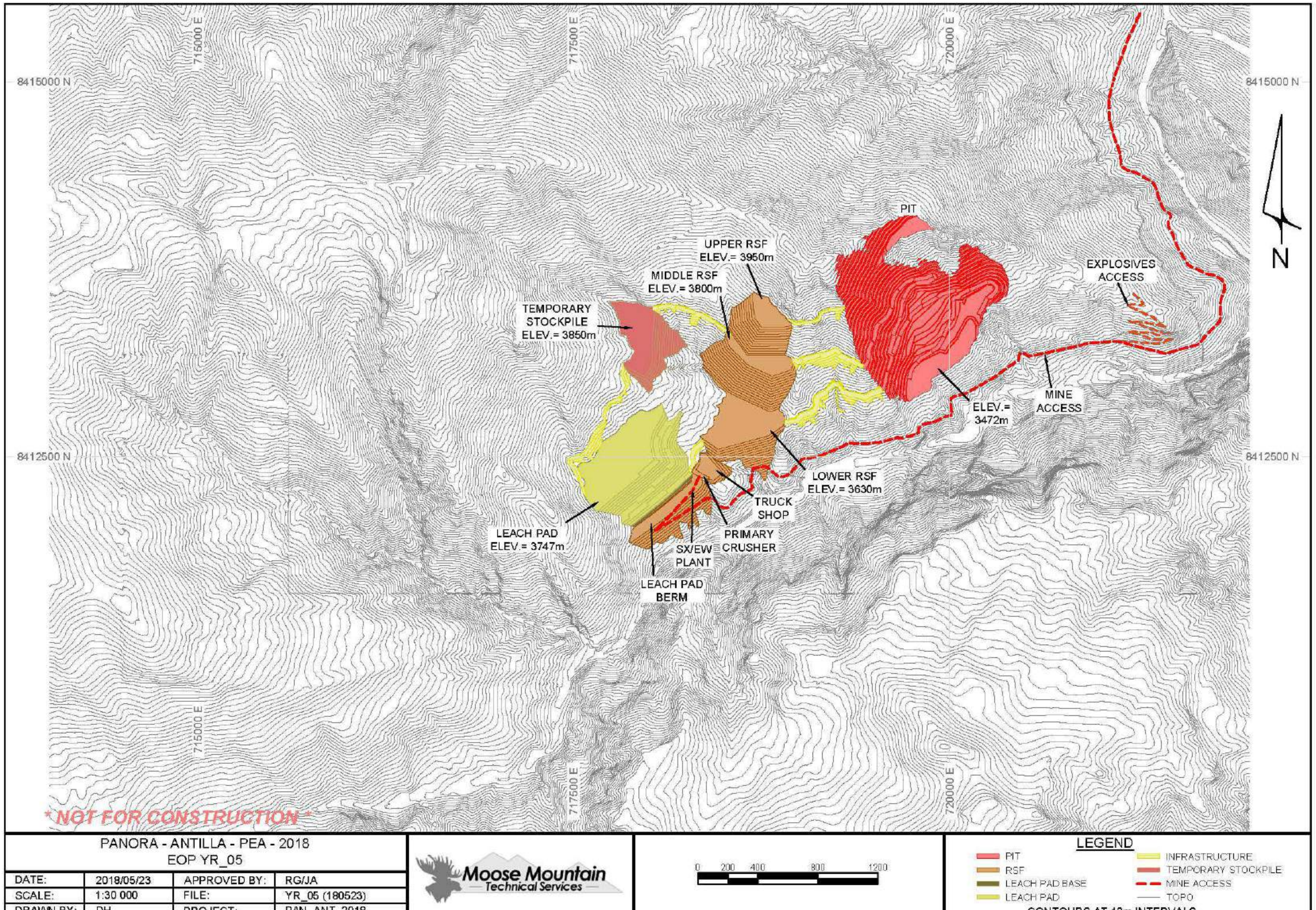


Figure 16-11 End of Year 5 Detail

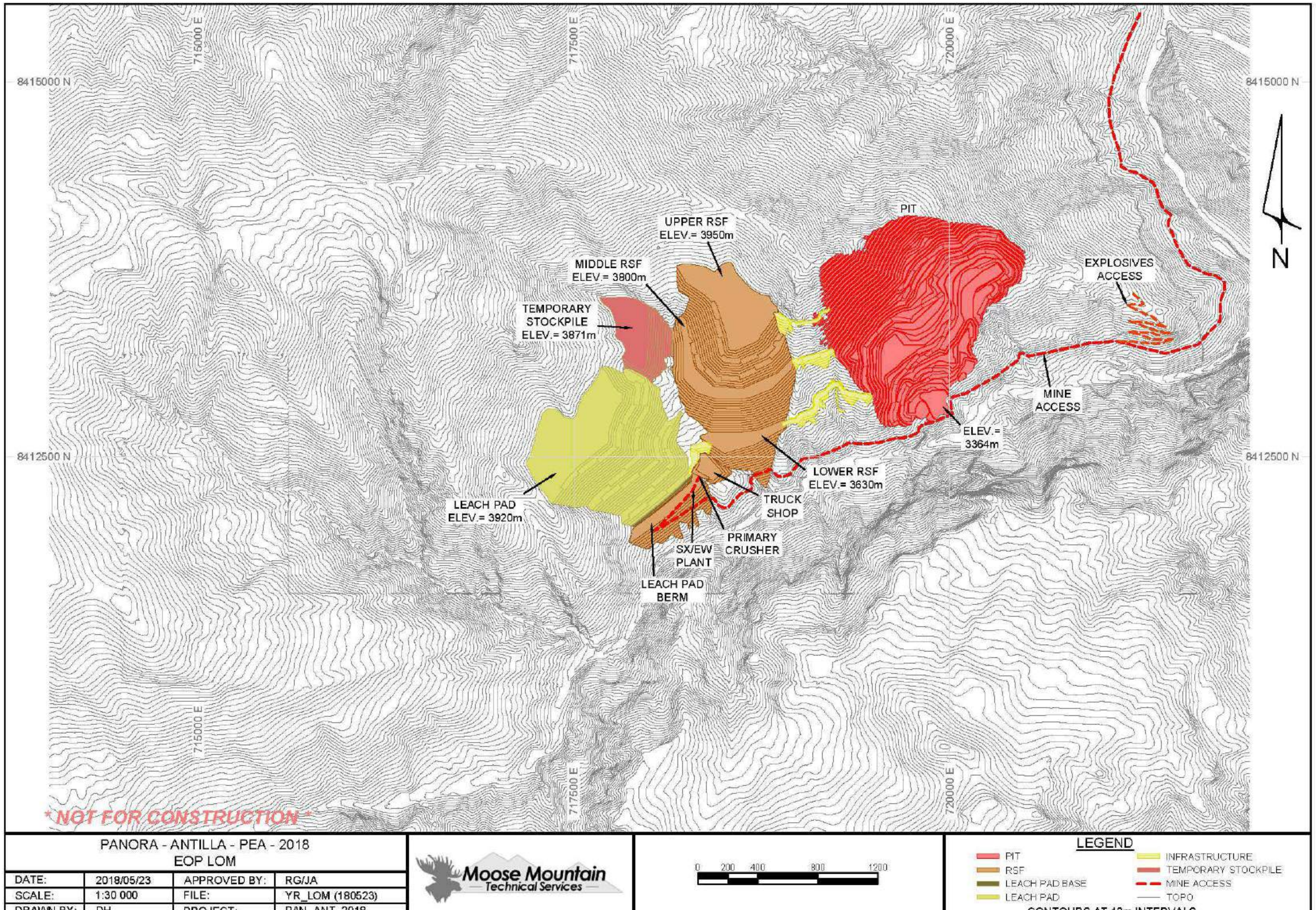


Figure 16-12 Life of Mine Detail

The end of the pre-production period, referred to as time zero, is signified by the start-up of the crushing and leaching facilities. The pre-production periods are used to blast and dozer push narrow development benches of phase 2 down slope to an approximate elevation of 3964 metres. This provides an operable working bench to begin conventional truck and excavator mining to further develop the pits in preparation for crushing and leaching start-up. At time zero, there will be two active pit phases. Phase 1 is mined down to the 3,680 metre bench while Phase 2 is mined to the 3,760 metre bench. Approximately 7.1 million tonnes of mine waste rock from these phases is used to build the leach pad base and berm. Any remaining mine waste rock is placed in the lower, middle, or upper RSF, depending on the source elevation. All leach material mined during pre-production is placed in the temporary stockpile.

By the end of Year 5 of the mine schedule mining of Phase 1 and 2 is completed down to an elevation of 3,472 metres. Phase 3 is completed down to an elevation of 3,916 metres while mining of Phase 4 has not begun. Mine waste rock from these phases is placed in the lower, middle, or upper RSF, depending on the source elevation. All three RSF locations have excess capacity available, however the Lower RSF is more than half filled at this point. By the end of Year 5 approximately 9.1 million tonnes of leach material is sent to the stockpile and 36.5 million tonnes of leach material has been crushed and sent to the leach pad.

By the end of life of mine operations, the ultimate pit has multiple pit bottoms established by the completion of all four Phases, with a final pit bottom at 3,364 metres. All three areas of the RSF are filled. 18.3 million tonnes of leach material has been sent to the stockpile over the life of mine, with only 5.6 million tonnes reclaimed, leaving a balance of 12.7 million tonnes of potential leach material in the stockpile. 118.7 million tonnes of leach material is crushed and placed in the leach pad by the end of the mine life.

16.7.5 Mine Operations

The mining operations proposed for the Antilla project are typical of open pit, truck-and-shovel heap leach mining methods for year-round operations in mountainous terrain. Truck-and-shovel mining operations are assumed to be completed by a contract miner utilizing their own mining fleet.

The mine operations are organized into three departments: mine maintenance, technical services, and direct mining. Mine maintenance accounts for the supervision and planning of the mine maintenance activities. Technical services account for the technical support from mine engineering, planning, geology, and surveying functions. Direct mining accounts for the drilling, blasting, loading, hauling, and pit maintenance activities in the mine. Other areas of the organization are dealt with elsewhere in this report.

Operations are assumed to run 365 days per year, 24 hours per day, seven days a week, on a 12-hour shift rotation.

16.8 Mine Equipment

The Antilla mining operations will be undertaken by contract miners who will supply and maintain their own fleet of conventional diesel powered equipment on a $/tonne contract.

The contractor miners will size their fleet accordingly to match the production schedule outlined in Section 15.8, including a peak material movement capacity of approximately 25 million tonnes per year and a life-of-mine average material movement capacity of approximately 17.3 million tonnes per year.

16.8.1 Drilling and Blasting

Drilling and blasting will be undertaken by the contract miners, based on the production schedule. It is assumed that in addition to production drilling, the contractor will also be responsible for high wall and pre-shear drilling using a secondary drill rig.

16.8.2 Loading & Hauling

Contractor miners will use truck and shovel/excavator sizes as per their own fleet availabilities; however, based on the mining schedule, the primary loading units for the Antilla project should be sized at approximately 12-15 cubic metre loaders/excavators matched with 90-tonne haul trucks. This truck and shovel/excavator pairing is assumed for both leach material and mine waste rock. Detailed pit designs include road widths designed for 90-tonne haul trucks. If a contractor uses different equipment, the pit designs should be updated accordingly.

Representative cycle times (assuming 90-tonne haul trucks) are used to approximate variable haulage costs, incorporated into the production schedule so that the economics of the mining schedule consider longer hauls and the NPV is maximized where possible. Overhauling costs (for leach material) are estimated on an increasing basis as the height of the leach pad increases.

16.9 Open Pit Personnel

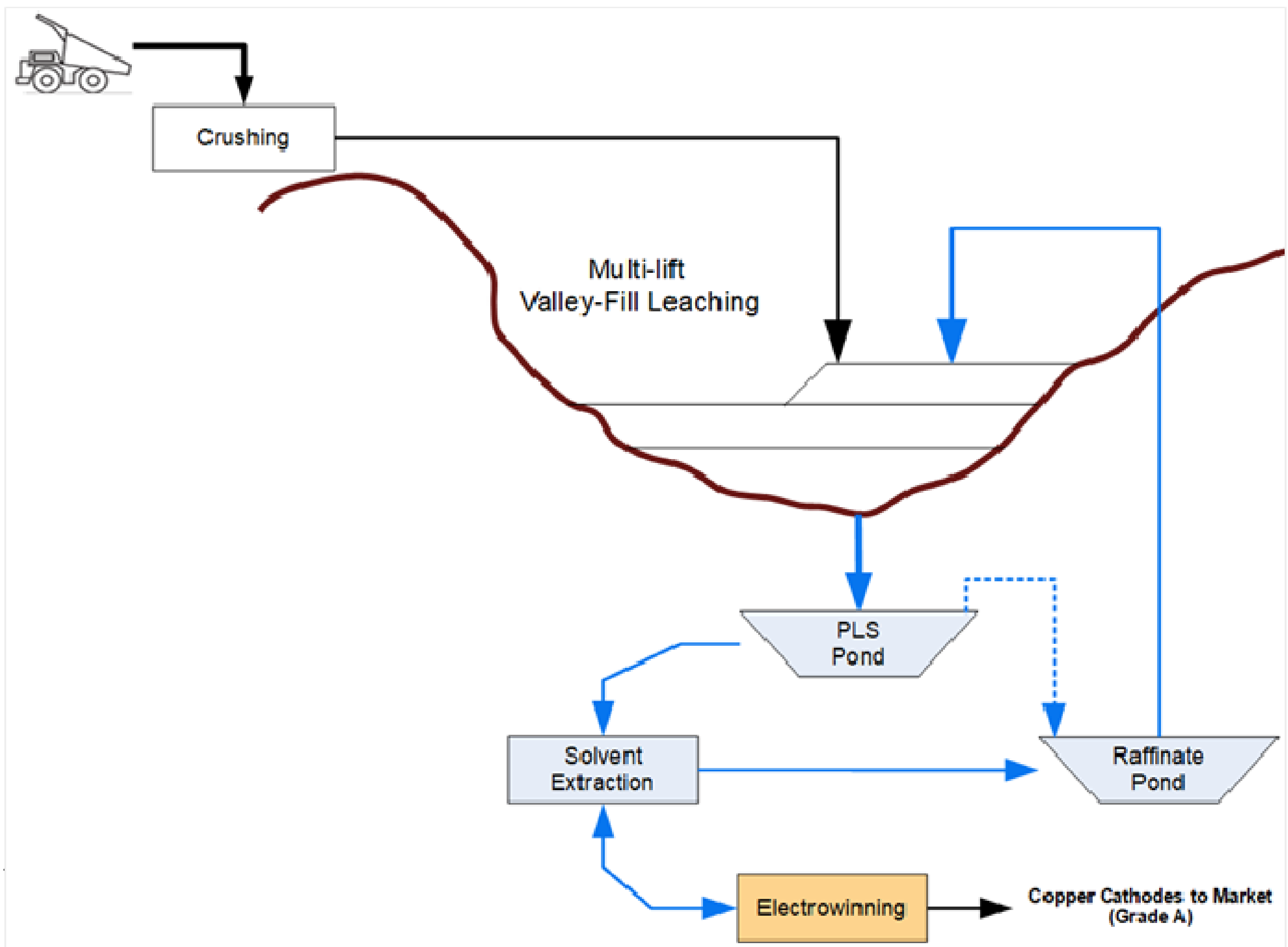
The mine plan assumes that contract miners will staff operators to match their fleets, with four crews rotating to fill the mine roster of 12 hours per shift (two shifts per day). Technical staff and mine operations supervision will be provided by the owner.

17 Recovery Methods

The Antilla Project will operate a conventional hydrometallurgical process consisting of a crushing plant, a valley-fill leaching plant, solvent extraction plant, and Electrowinning plant to produce grade-A copper cathodes, see simplified block flow diagram in Figure 17-1.

The run-of-mine ore (ROM) is trucked from the open pit to the crushing plant, the crushed ore will be trucked to the heap leaching area to form the ore lifts. Each lift will be irrigated with acid solution. The percolating solutions (PLS) will be collected at the bottom of the valley on the pregnant solution pond.

The valley's bottom will be make impermeable with synthetic liner to ensure all percolating solutions will report to the PLS pond.



recirculated to the leaching circuit after conditioning with acid and make up water to irrigate the ore. The copper cathodes are trucked off site to an ocean port for sale to markets.

The projected operational performance of Antilla Project is shown in Figure 17-2. The total processed ore will reach 7.3 million tonnes. The ore's head grade will vary from 0.54%Cu and 0.59%Cu for Years 1 and Year 2 respectively. Ore head's grade shows a downward trend reaching 0.3% by Year 12. Year 13 is expected to process ore with 0.48%Cu head grade then continue its trend to 0.3%Cu.

The copper production trend is estimated based on the metallurgical parameters presented in Section 13 of this report. The projected copper production will be consistent with the variable head grade at approximately 30,000 tonnes of copper cathodes from Year 1 to 4, then showing a lower trend towards 15,000 tonnes by Year 12. Copper production in Year 13 is expected to reach 25,000 tonnes of cathodes, after then it will decrease to 15,000 tonnes of cathodes by Year 16. The last year of the life of mine plan, Year 17, will expected to process 1.9 million tonnes bearing 0.35%Cu to produce 4,700 tonnes of cathodes.

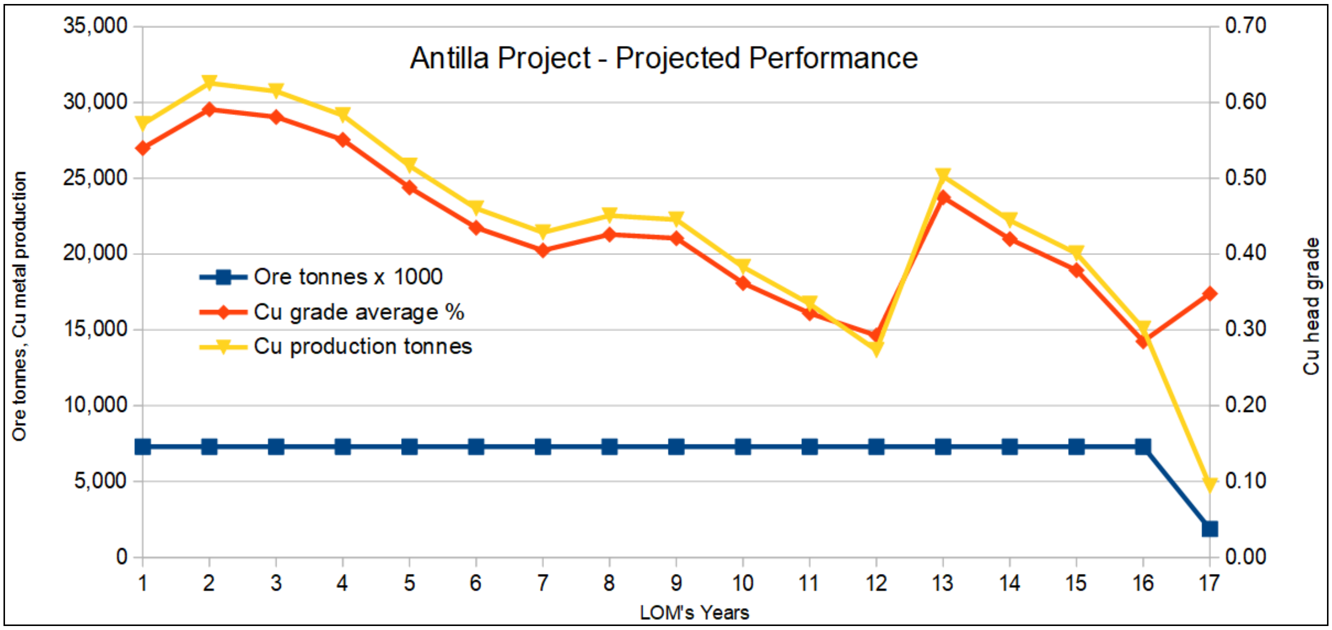


Figure 17-2 **PEA Antilla Project, Projected Performance**

17.1 Design and Sizing of the Processing Facilities

The key parameter used for design and sizing the processing facilities for Antilla Project is presented in Table 17-1.

Table 17-1 Key Design and Sizing of Processing Facilities

Item	Value	Units
Design & Sizing Criteria		
Total ore	118,667,452	tonnes
Copper grade	0.434	%Cu
Total Copper Fed	515,570	tonnes
Global Recovery [1]	72.1%	%Cu
Total Copper Produced	371,639	tonnes
Project LOM	17	years
Days/y	350	days
Total days	5,950	days
hours/day	24	hours
Density (In situ)	1.7	t/m3
Irrigation		
Raffinate Concentration	0.5	g/l Cu+2
FO/FA Flowrate Ratio	1	Organic / Solution
Extraction Efficiency	95	%
Bank (High)	10	m
Irrigation Specific Rate	4.5	l/m2/h
Leaching time [1]	200	days
Sizing		
Throughput	19,944	t/d
Throughput	11,732	m3/day
Solutions and Concentrations		
Solution Flowrate	1,055,865	l/h
Solution Flowrate	1,056	m3/h
Solution Flowrate	293	l/s
Ore/Cycle	2,346,366	m3
Area/Cycle	234,637	m2
Cycles	30	#
Ore	69,804,384	m3
LOM Irrigated Surface	6,980,438	m2
Copper Production	371,639	tonnes
Copper Production	21,861	t/y
Copper Production	60	t/d
Average Cu Concentration in PLS	2.86	g/l Cu+2

Note: (1) figures by TetraTech

17.1.1 Crushing Plant

The crushing plant considers a multi-stage circuit to reduce the ROM ore to approximately 100% minus 1” inch. The crushed ore will be stockpile, then loaded onto dump trucks and transferred to the heap leach plant using a front-end loader and 20 tonne dump trucks.

17.1.2 Heap Leaching

The valley-fill leaching operation will have capacity to receive approximately 120 million tonnes of ore over the 17 years of life of mine. The ore placed on the leach pad will be irrigated with diluted acid solution to transfer copper metal into the PLS. The bottom of the valley will be impermeabilized with a synthetic liner to collect all percolating solutions.

17.1.3 Solvent Extraction and Electrowinning

A conventional multi-stage solvent extraction plant will extract the copper from the PLS and transfer it to the Electrowinning plant where it will be deposited as copper cathodes. At this PEA level, the SX plant is estimated to process approximately 650 m3/h of PLS.

Copper cathodes production will be trucked off site at a rate of initially approximately 30,000 tonnes per year that will progressively decrease to 5,000 tonnes by year 17 of the life of mine plan.

18 Project Infrastructure

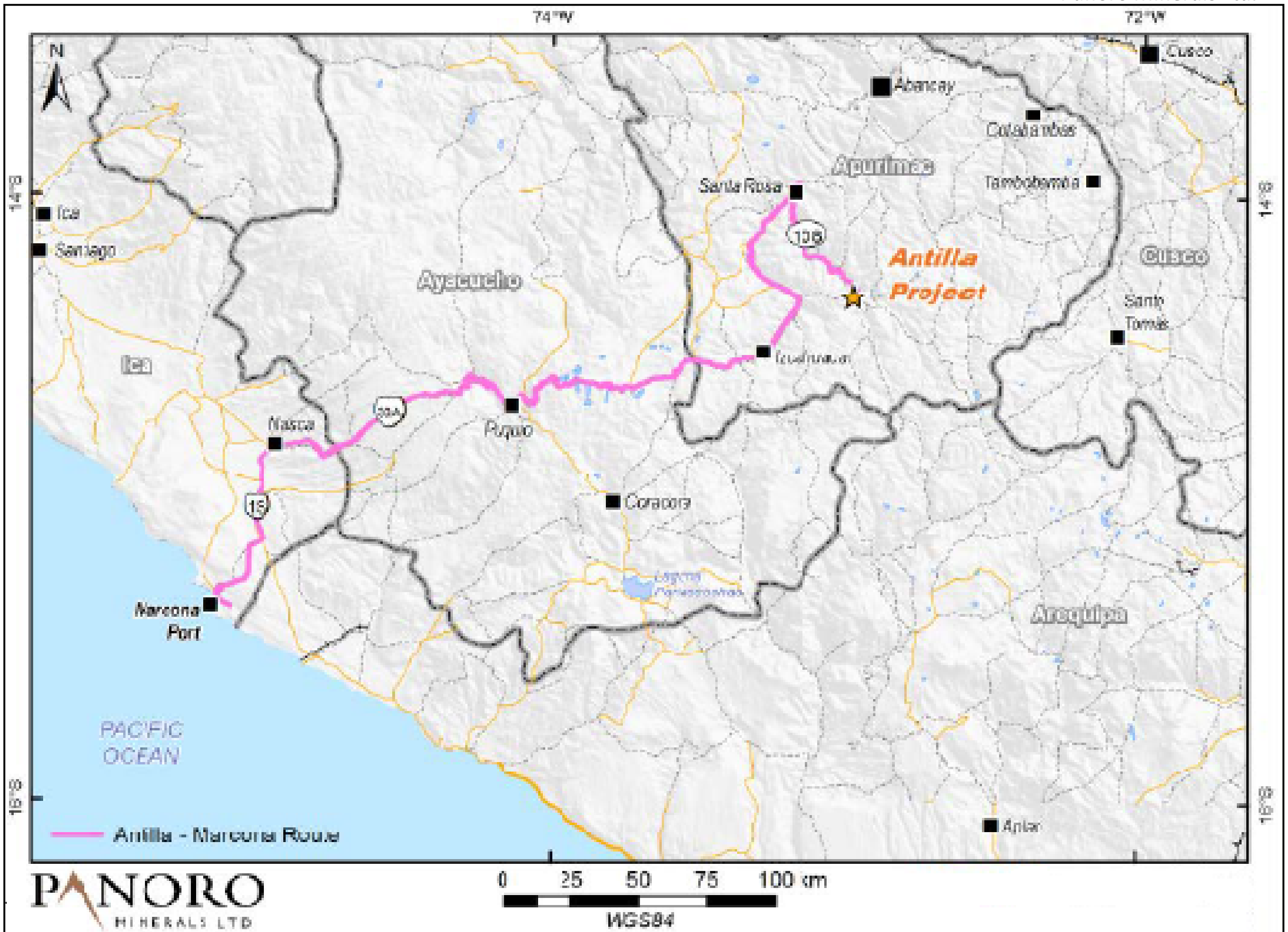
The property is accessible via 60-kilometre-long gravel road from the village of Santa Rosa on the paved Carretera Interoceánica Highway. General infrastructure required for the project includes the following:

- Internal Roads
- Power transmission line and sub-stations
- Water Supply and Management
- General Site Infrastructure Buildings
- Rock Storage Facilities
- Process Facilities
- Heap Leach Pad

18.1 Roads

Copper cathodes will be transported via truck from the SX/EW plant over the mine access road, site access road and then onwards to the port of Marcona where it will be offloaded and shipped to the end user. The total length of this transport option is just 554 kilometres and is illustrated in Figure 18-1.

The Antilla project is accessed from Highway 26 by Santa Rosa by a 60km long gravel and dirt site access road. This is a province public road. The mine access road is an 8km road that connects the site access road to the mine site area and project infrastructure. The mine access road follows an existing road for part of its duration while the other sections require upgrading.



The 220kV line tension with installed capacity of 255 MW (two triple lines of 150 MVA each) was constructed by Abengoa Peru S.A., a power generator company who operates and maintains the concession on this line. An allowance has been made to add a 10km transmission line and substation to connect the mine site to the local grid. The estimated power requirements for the project are 13 MW.

18.3 Water Supply and Management

The Antilla project will collect runoff water during the wet season to supply the water needs. Principal sources are represented by the Huancaspato River passing by a side of the operational area and the Ticlia lagoons located 7 kms to the northwest, both into the Antilla's property. Surface water management will include collection ponds, settling ponds and water diversion channels around the leach pad area. There is no allowance for a water treatment plant in this study. A potable water supply system is included.

18.4 General Site Infrastructure Buildings

There are a variety of smaller buildings that will be required for the Antilla project including: 1mining camp, administrative and mine offices, laboratory, truck shop and warehouse, explosives storage magazine and fuel station. The location of these buildings is shown in Figure 18-2.

18.4.1 Mine Camp

The mine camp will be located along the mine access road near the entrance to the project site.

18.4.2 Truck shop and Warehouse

The truck shop and warehouse will be located between the Rock Storage Facility and the leach pad berm. Major maintenance on haul trucks and ancillary equipment such as dozers and graders will be done here. The wash bays and tire bay will also be located here.

18.4.3 Explosives Storage

Silos for storing ANFO and emulsion will be located to the east of the pit area at a safe distance from the mine access road and mine camp as determined by the Quantity-Distance table for an estimated 90-tonne capacity. Explosives accessories such as booster, caps, delays and det cord will be stored in magazines at the explosives storage site. Access to this site will be controlled.

18.5 Rock Storage Facilities (RSF)

The rock storage facilities are discussed in Section 16.7.2.

18.6 Process Facilities

The process facilities include an SX/EW plant, laboratory and crusher. These facilities will be located between the RSF and the Heap Leach Pad.

18.7 Heap Leach Pad

A leach pad base will be built using material mined during the pre-stripping period. The very bottom of the leach pad will be lined prior to placement of leach material onto the base.

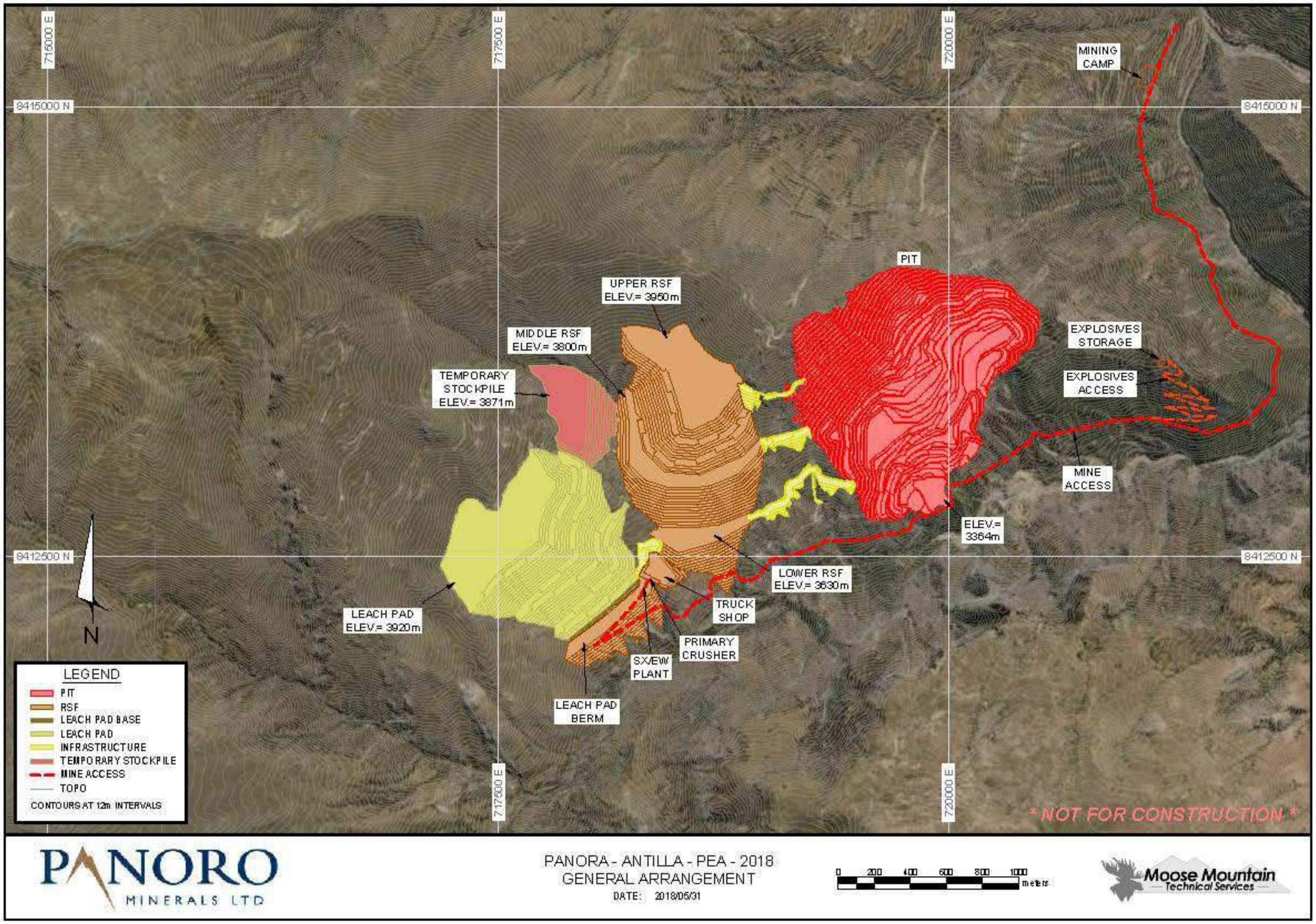


Figure 18-2 General Arrangement for Antilla

19 Market Studies and Contracts

19.1 Market Studies

The Antilla project is not currently in production and has no operational sales contracts in place. No market studies were undertaken in conjunction to the preparation of this technical report. As the project progresses through the next phases of development, it is recommended that further review be made of market conditions and obtain more accurate estimates related to copper cathode pricing, payment timing, metal accountability, and other contract terms, as well as transportation, port, and shipping costs.

19.2 Principal Assumptions

The Antilla project will produce a copper cathode.

For this marketing assessment, assumptions are based on metallurgical data with respect to the characteristics of the copper cathodes, current market understanding and data available in the public domain. Therefore, all estimates of costs used in this study have been benchmarked against prevailing industry rates.

Copper prices are affected by worldwide trends in supply and demand, and determined by trading on the major metals exchanges, including the New York Mercantile Exchange (COMEX) and the London Metals Exchange (LME). As of the date of this report, over the last five years, the price of copper has ranged from US$1.94 to US$3.30 per pound, with an average of US$2.65 per pound. Figure 19-1 shows the trend in copper prices since 2013.

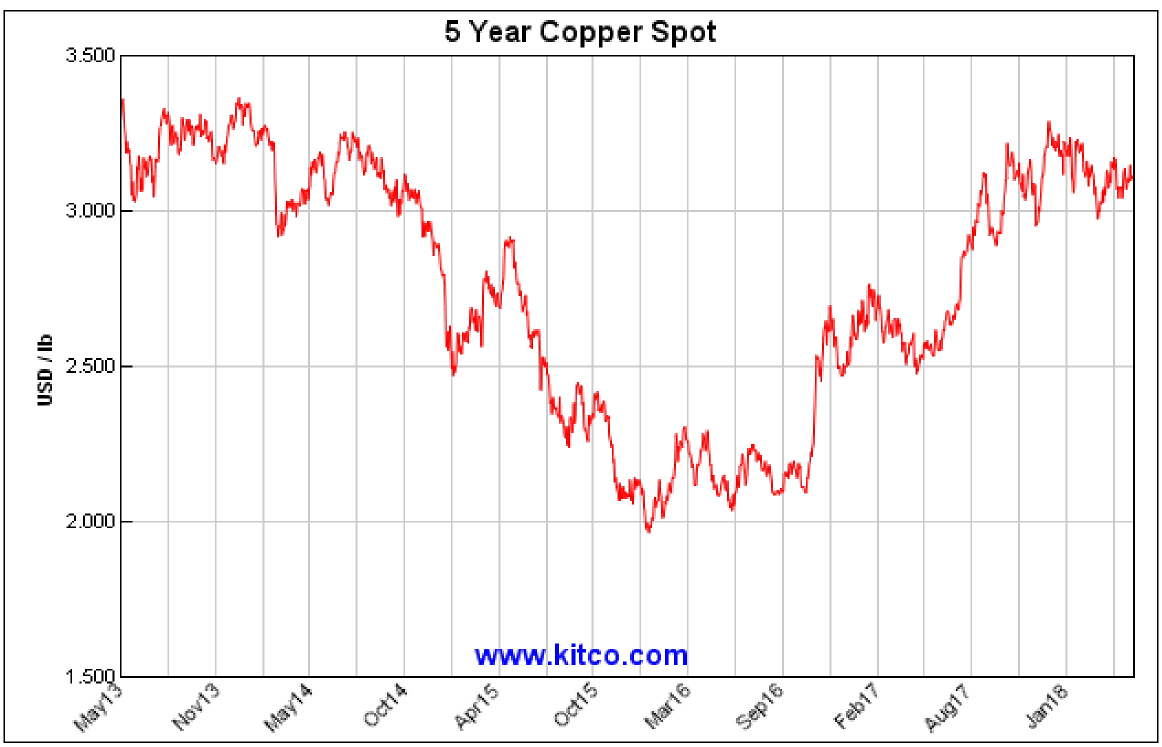


Figure 19-1 Five Year Copper Monthly Price (Source: KITCO.com 2018)

The global copper market is entering a new cycle where the lack of recent investment into new production is estimated to result in a growing deficit in copper supply. As a result, the estimates for long-term copper price have been increasing. As an example, Figure 19-2 presents the May 2018 estimates for copper prices from BMO Capital Markets for 2017 to 2021. The long-term copper price is estimated by BMO Capital Markets as $US 3.25/lb.

		2017	2018E	2019E	2020E	2021E	LT
Copper	US$/lb	2.80	3.03	3.25	[illegible]	3.10	3.25
Nickel	US$/lb	4.72	5.48	5.00	5.00	5.75	7.50
Zinc	US$/lb	1.31	1.44	1.35	1.15	1.15	1.10
Lead	US$/lb	1.05	[illegible]	1.10	1.10	1.00	1.00
Met Coal	US$/	[illegible]	204	150	135	133	133
Gold	US$/oz	1,257	1,327	1,275	1,250	1,200	1,200
Silver	US$/oz	17.07	17.32	17.00	17.90	18.50	20.00
USD/CAD		0.77	0.79	0.81	0.82	0.82	0.82
GBP/USD		3.19	3.20	3.20	3.33	3.43	3.44

Source: BMO Capital Markets

Figure 19-2 Annual Copper Price Forecast (Source: BMO Capital Markets)

A review of commodity analysts' forecasts for copper price was carried out in early 2018 for the preparation of this report. The results are summarized in Figure 19-3.

	2018	2019	2020	LT
MACQUARIE	$3.14	$3.02	$3.18	$2.68
CORMARK	$3.25	$3.00	-	$3.00
BMO Capital Markets	$2.94	$3.19		$3.19
CIBC	$3.23	$3.30	$3.30	$3.30
PARADIGM CAPITAL	$3.50	-	-	$3.50
GMP Securities	$2.89	$2.85	-	$2.85
HAYWOOD	$3.10	$3.20	-	$3.20
VIII EIGHT CAPITAL	$2.95	$3.25	-	$3.25
CREDIT SUISSE	$2.65	$2.30	-	$2.30
Scotiabank	$3.05	$3.25	$3.40	$3.50
NATIONAL BANK	$3.00	$3.00	-	$3.00
CANACCORD Genuity	$3.00	$3.00	$3.00	$3.00
Average	**$3.06**	**$3.03**	**$3.22**	**$3.06**

Figure 19-3 Consensus Commodity Analysts Copper Price Forecast (Source: Analysts' Consensus and Macquarie Equity Research)

A long-term copper price of US$3.05 per pound was selected for the project life and represents the consensus long-term price forecasts. Considering higher copper price forecasts in the nearer term, the copper price used in this report is summarized in Table 19-1.

Table 19-1 Copper Price Estimates Used in Analyses

Year of Mine Life	Copper Price ($US/lb)
1	3.20
2	3.15
3	3.10
4 to end of life of mine	3.05

A more precise marketing plan and terms of sale of final products is recommended during subsequent, more detailed, technical studies of the project. Transport firms, ports, and smelters should be contacted to obtain firmer estimates of freight, treatment and refining charges.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Setting

Limited environmental baseline data has been collected for the site (2007). Baseline studies have focused on environmental permitting in support of exploration activities.

The project area is located between 3,400 and 4,100 metres above sea level on the eastern flank of the Andes Mountain Range. The terrain is dominated by rugged topography with intersecting mountain ranges and deep valleys; low vegetation or fauna in the area. The project is located in the watershed of the Huancaspaco River with minor watercourses, such as the Chalhuani, Huarajoni, Hantarajo and Lloclla, contributing to the overall flow. The Huancaspaco River discharges to the Antabamba River. All of the rivers and watersheds are steep, fast flowing watersheds with no wetland areas or lagoons present.

Surface water samples collected from natural springs immediately downstream of the mineralized zone of the project area show elevated acidity (pH of 2.96 to 4.01), and copper and iron significate concentrations.

20.2 Social Setting

The closest community to the project site is the village of Antilla, and therefore likely to be directly impacted by the project. The villagers lead a subsistence lifestyle focused around agriculture and livestock. The main water source for the Antilla village is the Ocramayo Creek located outside the project area. Water in this creek is not impacted by any of the proposed activities and is suitable for potable use.

Soils observed in the project area are generally acidic, have low clay content, are highly erodible, and are subsequently not well suited for commercial agricultural or livestock use. Notwithstanding, there are select areas on the project site where crops are being cultivated.

Panoro´s Geological works were done under Community Social agreements (“Convenios”). From 2003 to 2014, Panoro signed two Convenios related to Drilling Exploration and another one related to Superficial geological works. All the commitments were accomplished and the relationship with Antilla Community and other Local institutions were set with high social satisfaction.

In April 2012, Peru´s Government approved a Prior Consultation Law that requires prior consultation with indigenous communities before any infrastructure or projects are developed in their territories.

Regarding as the Prior Consultation Law, no formal consultation with any of the local communities has been completed yet. However, based on the current relationship with the Antilla village and other surrounding communities, local people are generally supportive of the exploration activities undertaken by Panoro.

20.2.1 Social and Community Considerations

The socio-economic baseline studies will:

- identify communities that may potentially be affected by the development of the Project
- identify potential positive and adverse effects of the Project on local communities
- advise on further study requirements.

20.2.2 Consultation with Communities

The implementation of an effective community engagement program is fundamental to the successful permitting of mining projects. As part of a comprehensive community engagement program, consultation should be initiated at an early stage of the Project.

Consultation will include addressing concerns on the environmental and social-economic impacts of the Project activities during its different stages, according to current Peruvian laws.

As part of the Panoro Geological works in the Area, Panoro maintains a permanent dialogue with the public and local authorities as part of the stakeholder participation process, which involved the management of communications and the dissemination of project information.

Consultation and the development of a working relationship with local communities typically involve the development of a series of agreements that lay the groundwork for ongoing discussion. These include:

- memorandums of understanding
- protocol agreements
- community consultation/participation agreements.

As project exploration and development proceeds, other agreements will become necessary, including:

- socio-economic benefits agreements
- environmental monitoring agreements
- training agreements
- accommodation and impact benefit agreements
- Communities participation plan in the development of the project

20.2.3 Community Relations Plan

Since the beginning of the Project exploration activities, Panoro has been able to establish and maintain an effective relationship with communities in the area of influence of the Project by keeping them informed about project activities, having them participate in environmental monitoring programs, hiring temporary local labour and supporting social development initiatives to improve educational infrastructure and programs, local infrastructure and to promote small businesses. The Project has been widely accepted by the public.

The Community Relations Plan (CRP) for the Project addresses the actions necessary to establish and maintain relations with the key authorities, institutions and local organizations to keep them informed about the Project activities, mitigate exploration impacts, and support community development programs.

The main CRP is designed to reinforce local capacity building and promote active participation of local and provincial authorities and engage leaders and representatives from Antabamba Valley organizations in activities aimed at improving transparency of Panoro's activities related to the Project. These actions are being carried out according to the program structure of the CRP, which integrates community development, community relations, public relations and communications. The main activities selected for the CRP supported by Panoro include the following:

- consultation and disclosure
- education, health and nutrition

- environmental management
- local employment and local economy
- participatory monitoring program
- infrastructure
- development and capacity building of local institutions
- promotion of local culture and customs.

20.3 Regulatory Requirements

Prior to the initiation of mine development and operation, the Peruvian Environmental Regulations, require the proponent to conduct a comprehensive environmental impact assessment. The environmental impact assessment must be approved by the National Environmental Certification Service (SENACE) of the Ministry of the Environment before mining activities may commence.

The environmental impact assessment has to comply with the requirements set out in the Ministerial Resolution N° 092-2014-MEM-DM and Supreme Decret N° 005-2016-MINAM, which includes a very detailed description of all aspects of the project that should be considered to ensure that the environment is adequately protected. This includes a public consultation process involving all interested and affected parties and communities.

The project description used as the basis for the environmental impact assessment should be developed to at least feasibility level, and must include all aspects of the project including off-site facilities such as electric power source, cathodes transport and shipping.

Construction of the project must be initiated within three years after the approval of the environmental impact assessment; otherwise it is deemed invalid and will need to be redone (Peruvian Law No. 27446 and Supreme Decree No. 019-2009-MINAM). Twelve months following the approval of the environmental impact assessment, a detailed closure plan must be submitted for approval and a closure bond must be surrendered within one year of the approval of the closure plan.

Following the approval of the environmental impact assessment, the proponent must demonstrate they have secured the surface rights required to carry out the project. At the same time, the proponent must also apply for all other regulatory permits and authorizations as summarized in Table 20-1. The cost of conducting the environmental impact assessment and obtaining all of the necessary permits and approvals is estimated to be approximately between US$1 to 2 million. This excludes development of the project description and execution of baseline studies.

Table 20-1 List of Regulatory Permits and Approvals Required

Authority	Mining Project	Transport of Concentrates	Transmission Line	CIR A	Closure Plan	Wate r Use	Discharges	Construction & Operations
DGAAM	X	X	X	X	X	X	X	
DGM					X	X		X
DGAAE			X	X		X		
DREM	X	X	X	X	X			
MTC		X		X				
MINAG	X	X	X	X		X		
MINAM	X	X	X	X		X	X	
Ministry of Culture				X				
ANA/ALA	X					X	X	
DIGESA	X					X	X	
Provincial/Local Council	X	X	X		X			
Rural Community	X	X	X		X			

20.4 Archaeology

To date, nine archeological sites have been identified on the project site as part of the exploration phase of the environmental baseline studies. None of these sites contain human remains. Subsequently, a Certificate of Non Existence of Archaeological Remains is required from the Ministry of Culture.

20.5 Mine Waste Geochemistry

Detailed geochemical characterization studies have not been carried out on mine waste materials for the Antilla project. In 2015 four waste rock samples were submitted by Panoro for acid base accounting (ABA) to establish a screening basis for acid rock drainage (ARD) potential. Geochemical testing has not been carried out on the tailings.

Table 20-2 summarizes the four composite waste rock samples which were selected by Panoro to represent the primary mineralogical domains observed. Each approximate 3-kilogram sample was made up from core samples that had a copper grade of less than 0.15%. A complete description of each sample, as provided by Panoro, including the sample locations are provided in Appendix B.

Table 20-2 Samples Submitted for Acid Base Accounting Testing Sample ID Mineralogical Domain Description

Sample ID	Mineralogical Domain	Description
COV-ANT-1A	Surface Cover	Colluvium with quartzite clasts and some claystone clasts.
EL-ANT-2A	Leach Cap	Quartzites (whitish grey, beige), with slight sericite alteration and some late porphyry and sandstone. Hematite, jarosite, goetite and some pyrite in fractures. Some pyrite horizons in sandstone. Traces of calcocite and calcopyrite (disseminated and in fractures).
SE-ANT-3A	Secondary Enrichment	Quartzites (whitish grey), alternated with sandstones (yellowish brown) and some principal porphyry. Sporadic pyrite, calcocite, chalcopyrite (disseminated and in fractures); molybdenite (disseminated); jarosite, goetite (disseminated and in fractures).
SP-ANT-4A	Primary Sulphides	Intercalation of quartzites (whitish grey) and sandstones (yellowish brown); and some clayey siltstone and principal porphyry. Sporadic pyrite, calcocite, chalcopyrite (disseminated and in fractures); molybdenite (disseminated); less jarosite, goetite (disseminated and in fractures).

All samples were tested at the ALS Laboratory in Peru using the Modified Sobek Method (ASTM E195-97 and MEND 1.16.3). The results are summarized in Table 20-3 and the full report is included in Appendix C.

Table 20-3 Acid-Base Account Analysis Results

Parameter	Unit	Detection Limit	Sample			
ALS ID			144675/2014 -1.0	144676/2014 -1.0	144677/2014 -1.0	144678/2014 -1.0
Panoro ID			COV-ANT-1A	EL-ANT-2A	SE-ANT-3A	SP-ANT-4A
Total sulphur	%S	0.01	0.02	0.02	0.36	0.54
Inorganic carbon	%	0.01	0.01	0.02	<0.01	0.06
Fizz rate	-	n/a	None	None	None	None
Paste pH	-	n/a	6.96	7.78	6.61	6.06
Maximum acid potential (MAP)	$\frac{t\ CaCO_3}{1000t}$	0.5	0.6	< 0.5	10.3	15.3
Net neutralization potential (NNP)	$\frac{t\ CaCO_3}{1000t}$	n/a	-2.6	-2.0	-14.3	-18.3
Neutralization potential (NP)	$\frac{t\ CaCO_3}{1000t}$	n/a	-2	-2	-4	-3
NP/MAP	-	n/a	-3.33	n/a	-0.39	-0.20
Sulphate leachable in carbonate	%S	0.01	< 0.01	0.01	0.04	0.06
Sulphate leachable in HCl	%S	0.01	< 0.01	0.01	0.03	0.05
Sulphur	%S	0.01	0.02	0.01	0.33	0.49

The test results have been interpreted by SRK (2016) using two different classification systems: MEND (MEND 1.20.1, 2009) and the Ministry of Energy and Mines of Peru (DGAAM-MEM, 1995). After testing the four samples, the following conclusions were reached:

COV-ANT-1A: Sulphur content of 0.02%, MAP = 0.6 at near neutral paste pH. Sample unlikely to produce ARD due to low sulphur content. EL-ANT-2A: Sulphur content of 0.01%, MAP < 0.5 at slightly alkaline paste pH. Sample unlikely to produce ARD due to low sulphur content.

SE-ANT-3A: Sulphur content of 0.33%, MAP = 10.3 and NP/MAP = 0.39 at near neutral paste pH. Likely, though uncertain to produce ARD based on sulphur content and NP/MAP ratio.

SP-ANT-4A: Sulphur content of 0.49%, MAP = 15.33 and NP/MAP = 0.20 at near neutral paste pH. Likely, though uncertain to produce ARD based on sulphur content and NP/MAP ratio.

20.6 Mine Closure

20.6.1 Regulatory Requirements

Peruvian Mine Closure Law #28090 (enacted in October 2003 and effective since August 2005) requires a mine closure plan to be submitted within one year following the approval of a mine's environmental impact assessment. Construction and operation of the mine cannot start until this plan has been approved by the General Environmental Directorate of the Ministry of Energy and Mines.

This law also requires that the mine owner provide financial guarantee for the estimated closure costs. The guarantee may be provided in the form of insurance, cash, a trust agreement, or another form approved by the regulator. Should at any time the mine owner not fulfill his commitments as outlined in their closure plan, the regulator may draw on the guarantee to complete closure activities as required.

The financial guarantee is provided in annual installments, starting one year after approval of the closure plan. Installments are prorated based on the life of mine.

20.6.2 Closure Objectives

A conceptual closure plan has been developed as part of study discussed herein. Since an environmental impact assessment has not yet been completed, there are no stated closure objectives for the project. However, using a standard best practice approach, the closure plan must ensure physical, hydric and chemical stability of the site such that there would be no long-term effects associated with the mining activities after operations cease.

20.6.3 Leach Pad Storage Facility Closure

The leach pad (LP) is assumed to be acid generating; therefore, a low infiltration cover is required. At closure, the LP surface will be allowed to consolidate for a period of time such that the surface would be trafficable. The surface will then be covered with a geomembrane and a geotextile before placing a three-layer soil cover on the geotextile. This will include 0.2 metres of granular material, overlain by 0.2 metres of inorganic soil, and finally 0.3 metres of organic soil that would support revegetation. An initial hydro seeding campaign will be used to facilitate the revegetation process. The seed mix will mimic local vegetation. All of the soil material will be sourced from local borrow sites, including stockpiled salvaged materials from pre-stripping of the containment dam, and waste rock pile foundations.

The downstream face of the containment dam will be closed similar to the rock storage facilities. The slope will be covered in situ, i.e., with no re-grading of the slope, using the same three-layer soil cover placed over the geomembrane layer of the tailings surface. Surface water runoff from the downstream face of the dam will be managed through the construction of a series of riprap lined drainage channels.

A small pond will be maintained on the surface of the LP storage facility to attenuate and direct water to a permanent overflow closure spillway on the containment dam. Also, a study of quarries in the area could be necessary.

20.6.4 Waste Rock Dump Closure

Although preliminary data suggest the waste rock may be acid generating, it has been assumed that long-term leachate generation from the RSF would be sufficiently low that long-term environmental degradation would not be of a concern. The closure strategy for the rock storage facilities is therefore simple soil covers with the primary objective of allowing revegetation.

The cover system for the rock storage facilities will be similar to the downstream face of the LP storage facility containment dam. The slopes will be covered in situ, i.e., no re-grading of the slopes assumed, using a three-layer soil cover. The first layer will be a 0.2 metres thick granular material, overlain by 0.2 metres of inorganic soil, and finally 0.3 metres of organic soil that would support revegetation. An initial hydro seeding campaign will be used to facilitate the revegetation process using local species. All of the soil materials will be sourced from local borrow sites, including stockpiled salvaged materials from pre-stripping of the containment dam, and waste rock dump foundations.

No specific allowance has been made for runoff management via engineered conveyance channels.

The final designs will be development in the closure plan.

20.6.5 Open Pit Mine Closure

The walls of the open pit are assumed to be stable in the long term and similar to the waste rock. Leachate from the pit wall slopes is assumed to not result in long-term environmental degradation, notwithstanding that the waste rock appears to be potentially acid generating. It is assumed that a pit lake would develop over time, and that a natural outflow will discharge water to the downstream river valley; a geochemistry detailed study will be necessary in order to define the management of the outflows before the discharge. No active pit infill is planned, with only direct precipitation allowed to contribute.

The only closure activity planned for the open pit mine is to construct a safety berm around the perimeter to restrict access. The berm will be constructed from compacted local borrow material and will be approximately 1.5 metres high with angle of repose slopes.

20.6.6 SX/EW and Crushing Plant Closure

At closure, all hazardous waste material from the SX/EW plant and ancillary facilities will be gathered and disposed of at an off-site licensed hazardous waste disposal facility. The process plant, crusher, and ancillary facilities will be dismantled with any salvageable material sold as scrap (although the closure cost estimate is not credited with any salvage value). Concrete footings and foundation slabs will be demolished. Concrete and other demolition debris deemed non-hazardous will be disposed of in a non-hazardous landfill site located within 3 kilometres from the process plant site.

Once all facilities have been removed, the area will be regraded to restore natural surface drainage and covered with 0.2 metres of organic soil from local borrow areas. The soil cover will be hydro seeded with a seed mix comprised of local species.

20.6.7 Roads Closure

Culverts and other water crossings along all site roads will be removed. All site roads will be scarified and the area regraded to restore natural surface drainage. The disturbed areas will be covered with 0.15 metres of organic soil sourced from local borrow sources. Hydro seeding will be used to facilitate growth of local vegetation.

20.6.8 Ancillary Facilities Closure

Hazardous material from the site ancillary facilities such as the water treatment plant, mine camp, workshops, warehousing facilities, administration offices, laboratories power substation, fuel storage area, and explosives magazines will be collected and disposed of at a licensed off-site hazardous waste disposal facility. The facilities will subsequently be dismantled and all concrete foundation slabs will be demolished. Concrete and other demolition debris deemed non-hazardous will be disposed of in a non-hazardous landfill site together with all other similar site debris.

Prior to the demolition of all fuel tanks, they will be rinsed with all wash water cleaned using hydrocarbon separators. Excess explosives will be transported off site and sold or disposed of in accordance with appropriate safe handling regulations. Transmission lines will be dismantled and disposed of as any other site demolition debris, contingent on not being classified as hazardous waste.

Once all of the site ancillary facilities have been removed and any hydrocarbon contaminated areas have been cleaned up, the areas will be regraded to restore natural surface drainage and covered with 0.2 metres of organic soil from local borrow areas. The soil cover will be hydro seeded with a seed mix comprised of local species.

20.6.9 Settling Ponds Closure

Settling ponds will be drained and the geomembranes removed and disposed of in the on-site non-hazardous landfill site or at an off-site licensed hazardous landfill site if necessary. The ponds will be backfilled using local borrow materials before placing a 0.25-metre thick organic soil cover sourced from a local borrow source and hydro seeding with a seed mix of local species.

20.6.10 Closure Schedule

Progressive reclamation opportunities for the project are limited as currently designed. RSF can be closed as they are constructed and the downstream face of the LP storage facility containment dam can be closed one year prior to cease of operations, but all other closure activities can only occur later. Placement of the LP cover may have to occur many years after closure to allow for the surface to be trafficable.

20.6.11 Closure Cost Estimate

The closure cost estimate for the conceptual closure plan has been developed using benchmarking information from other similar projects as opposed to a first principles cost estimate. It comprises a combination of scaled numbers and equivalent unit rates. Where unit rates are applied, quantities are based on preliminary conceptual design drawings developed as part this study. The closure cost estimate at US$ 23 million, as presented in
Table 20-4 Table 20-4 .

Table 20-4 Closure Cost Estimate

Component	Cost (USD)
LP Storage	8,000,000
Rock Storage Facilities	2,645,935
Open Pit Mine	154,866
SX/EW Plant	3,500,000
Roads	554,531
Water Pipelines	15,000
Substation and Transmission Lines	376,764
Mine Camp	1,041,962
Workshop and Storage Facilities	359,371
Settling Ponds	175,062
Crushing Plant	70,000
Explosives Magazine	300,000
Offices and Laboratories	90,000
Fuel Station	75,000
Subtotal Direct Closure Costs	$17,358,491
Indirect Costs (25% of Direct Costs)	$4,339,623
Subtotal	**$21,698,114**
Contingency (30% of Direct Costs)	$ 1,301,887
Total	**$23,000,000**

21 Capital and Operating Costs

This preliminary economic assessment 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. There is no certainty that the results of this preliminary economic assessment will be realized. Mineral resources that are not mineral reserves have no demonstrated economic viability.

All currencies shown in this Section are expressed in USD.

21.1 Capital Cost Estimate

The expected accuracy of the capital cost estimate is +/-50% which is suitable for a PEA-level study.

21.1.1 Capital Cost Summary

The four main areas of the capital cost estimate are listed below along with the contributing parties for each area:

- Mining Capital (Moose Mountain)
- Mineral processing (Moose Mountain)
- Infrastructure (Moose Mountain)
- Owner's costs (Panoro)

The pre-production initial capital costs have been estimated at $250.4 million USD including contingency.

The capital cost estimate is expressed in USD for the first quarter of 2018. A summary of the project capital costs is shown in Table 21-1.

Table 21-1 Capital Cost Estimate Summary

Item	Cost (Million USD)
Mine Equipment	$1.3
Mine Development	$41.1
Crushing, SX, and EW plants	$94.7
Infrastructure	$42.4
Sub-total	$179.5
Owner's Cost	$7.8
Indirect Costs	$13.7
Sub-total	$201.0
Contingencies	$49.4
Total Initial Capital Cost	$250.4

Items such as sunk costs, exploration costs, continuing studies, and working capital are not included in the initial capital cost estimate.

Indirect capital costs are calculated as 10% of the Mineral Processing and Infrastructure direct costs.

Contingency costs are estimated by applying various contingency factors against each of the four main components of the capital cost estimate. Contingency factors range between 20-30%. The initial capital contingency allowance of $49.4 million USD is 28% of initial capital costs (excluding owner's costs and Indirects).

21.1.2 Mining Capital Cost Estimate

The Mining capital cost estimate includes Mine Equipment and Mine Development (pre-stripping).

The Mine Equipment capital cost estimate is based on mobilization cost estimates provided by contract miners. Moose Mountain compared the estimates to benchmarks from similar operations at a PEA-level and deemed them sufficient for use.

Mine Development capital costs is composed of the mining operating costs during the pre-stripping period. This includes drilling, blasting, loading, hauling and GME costs related to mining activities in the pre-stripping period. Mining haul road construction and leach pad base preparation is also included.

A 20% contingency factor is applied to Mine Equipment and Mine Development capital costs.

21.1.3 Processing Capital Cost Estimate

The initial capital expenditure for the processing facilities has been estimated by a benchmark with comparable projects in the industry in the same Andean region of Antilla Project.

The crushing facilities' initial capex for a 20,000 tonnes per day plant is estimated at USD37.1 million.

The leaching facilities initial capex consider the preparation of the base and installation of a synthetic liner for the first (1) month of operation. The expenditure to continue impermeabilizing the base of the leaching operation will be considered as part of the operating expenditure. This initial expenditure has been estimated at USD5.6 million.

Initial capital expenditure for the solvent extraction, Electrowinning, solutions ponds (PLS and Raffinate), piping and pumps to irrigate the ore, collect solutions the percolating solutions, and transfer solution to and from the SX-EW facilities and solution ponds has been estimated at USD51.9 million.

The A 30% contingency factor is applied to Processing capital costs (see Table 21-2).

Table 21-2 Processing Capital Costs

Processing Area Initial Capital Expenditure	USD million
Crushing facilities	37.1
Leach Pad Lining	5.6
Solvent Extraction & Electrowinning (including ponds, piping, pumps)	51.9
Total	**94.6***

**Numbers may not add due to rounding*

21.1.4 Infrastructure Capital Cost Estimate

A description of the infrastructure associated with the project is listed in Section 18 of the report. Infrastructure capital cost estimates are based on scaled factors or 3rd party estimates (where possible). A 25% contingency factor is applied to the Infrastructure capital costs.

Table 21-3 Infrastructure Capital Costs

Description	Million USD
Contractor Mining Infrastructure (truck shop, warehouse, offices, rest rooms)	$1.9
Mine Camp	$8.0
Laboratory	$1.0
Explosives Storage Magazine	$0.5
Fuel Station	$0.5
10km power transmission line	$4.5
Power substations	$13.5
Communication systems	$1.0
Mine Area Access Road (8km)	$4.4
Potable water supply	$0.2
Water diversion channels	$3.6
Water Management structures	$1.4
Site preparation	$2.0
Total Infrastructure	**$42.4***

**numbers may not add due to rounding*

21.1.5 Owner's Costs Capital Estimate

The estimated total owner's costs are $7.8 million USD. These costs include land acquisition, light equipment, permit applications, local community projects, and personnel. A 25% contingency factor is applied to the Owner's Cost Capital. Costs normally attributed to G&A in pre-production Years -2 and -1 have been included in the initial capital cost.

The owner's costs are summarized in Table 21-4 below:

Table 21-4 Owner's Costs Direct Capital

Description	Million USD
Land Acquisition (650 ha)	$1.3
Light Equipment	$2.4
Permit applications	$0.9
Local Community Projects	$2.0
Personnel	$1.2
Total Owner's Costs Capital	**$7.8**

21.2 Operating Costs Estimate

The total life of mine operating costs for the Antilla project is $7.04/tonne of leach material. Operating costs are summarized in the table below:

Table 21-5 Antilla On-site Operating Costs

Item	Cost (USD/tonne)
Mining Costs	$1.63
Processing Costs (including crushing)	$3.85
Average Leach Material Haulage Costs	$0.81
G&A Costs	$0.75
Total Onsite Operating Costs	**$7.04**

The operating costs are considered to have accuracy of +/-35% which is suitable for PEA-level study.

21.2.1 Mining Cost

The mine operating costs are based on averages of cost estimates provided from three contract miners based in Peru. The mine operating costs include all costs related to drilling, blasting, loading, hauling, auxiliary equipment operations, equipment maintenance, mine operations supervision and contractor mark-up. Contractors were provided the mine schedule tonnes and associated haul distances for various source/destination combinations. The contractors used this information to provide estimated unit mine operating costs. The average mining cost is $1.63/tonne.

21.2.2 Processing Cost

The operating expenditure for the processing facilities has been benchmark with comparable facilities to Antilla Project that are operating in the Andean region, see Table 21-6.

The operating expenditure in the crushing plant, the ore's rehandle with a front-end loader to load it onto dump trucks, and hauling ore to the leach pad (variable height and distance) is estimated in average at USD0.81/tonne of ore.

The leaching, solvent extraction, and Electrowinning operation's average expenditure is estimated at USD3.85/tonne of ore that includes the increase in elevation of the leaching facilities over the life of mine.

Table 21-6 Antilla Project, Operating Expenditure Estimate

Item	USD	USD/lb Cu	USD/tonne Ore
Leach Material - Hauling	95,875,153	0.12	0.81
Leaching – SX – EW (incl. crushing)	456,869,690	0.56	3.85
Total	552,744,843	0.67	4.66

21.2.3 General and Administrative Cost

G&A cost estimates are based on scaled estimates and benchmarks to similar projects. The estimated G&A cost for the Antilla heap-leach project is $0.75/tonne leach material. G&A cost include infrastructure supervision, maintenance truck shop and facilities, roads maintenance, company personnel, etc.

21.2.4 Off-Site Costs

Off-site costs include transport costs (assumed as ~$38/tonne) to the port as well as royalties. The total off-site costs are $0.03/lb.

22 Economic Analysis

22.1 Peru Taxation

Peru taxes and royalties included in the economic model are described below. MMTS does not provide expert advice on taxation matters. Antilla taxes were determined based on information provided by Panoro that was supported by public domain documentation.

The Peru value added tax (VAT) of 18% is excluded from the economic model. It is understood that VAT payments are reimbursed for exporters, and that there are mechanisms in place for early recovery of VAT payments made in the pre-production period for companies that have entered into investment contracts with the Peruvian government.

The Antilla economic analysis is prepared on a 100% equity project basis and does not consider financing scenarios. Financing related costs such as interest expense, and Peru withholding taxes on dividends and interest income, are excluded from the economic model.

22.2 Depreciation and Amortization

Asset classes and depreciation rates assumed in the economic analysis in order to estimate depreciation of initial and sustaining capital assets include:

- Mine Development and Land Improvements: Expensed in year incurred or amortized over a period of up to three years
- Buildings: Depreciated at 5% per year
- Machinery and Equipment: Depreciated at 20% per year
- Other Assets: Depreciated at 20%/year during the TSA term and at 10%/year after expiry of the TSA.

Sunken exploration and other eligible project costs are carried forward and amortized over the mine life.

22.3 Mining Royalty and Special Mining Tax

Mining projects are subject to both a mining royalty and special mining tax. Royalty rates and special mining tax rates are a function of operating income and sales revenue.

The mining royalty is applied to operating income at marginal-incremental rates rising from 1% to 12% in 16 separate tax brackets that are dependent on the ratio of operating income to sales.

Amortization and depreciation are deductible from operating income subject to the mining royalty.

The minimum mining royalty is 1% of sale revenue.

The special mining tax is applied to operating income at marginal-incremental rates ranging from 2% to 8.4% in 17 successive tax brackets that are dependent on the ratio of operating income to sales. Amortization and depreciation, and worker profit sharing (described below) are deductible from operating income subject to the special mining tax.

22.4 Regulatory Fees

Two Peru government agencies, i.e., Energy and Mining Investment Regulatory Agency (OSINERGMIN) and Environmental Regulatory Agency (OEFA), assess regulatory fees or contributions based on a percentage of mining project sales revenue. Regulatory fees totaling 0.24% of sale revenue were assumed for the Antilla project's economic analysis.

22.5 Worker Profit Sharing

Mine employees are entitled to a statutory worker profit sharing of 8% of taxable income.

Deductions from operating margin (EBITDA) for the purposes of estimating income subject to worker profit sharing include depreciation and amortization, mining royalty, special mining tax, regulatory fees, and retirement fund contributions.

22.6 Corporate Income Tax

The Peru corporate tax rate is 29.5%.

Deductions from operating margin for the purpose of estimating income subject to tax include:

- Depreciation and amortization
- Mining royalty
- Special mining tax
- Contributions OEFA and Osinergmin
- Worker profit sharing

22.7 Sensitivity Analysis

A sensitivity analysis is done considering lower and higher copper prices, recoveries, capex and opex, mineral mill feed, and taxes along with variances in the discount rates. The before and after tax results of the copper sensitivity analysis is shown in Table 22-1. The before and after tax results of the other sensitivity analyses are shown in Table 22-2 and Figure 22-1.

Table 22-1 Copper Price Sensitivity Analysis

		Before Tax					After Tax				
Scenarios	Long Term Cu Price (*) From Y4 to LOM ($/lb)	NPV 5% (million USD)	NPV 7.5% (million USD)	NPV 10% (million USD)	IRR	Payback	NPV 5% (million USD)	NPV 7.5% (million USD)	NPV 10% (million USD)	IRR	Payback
Min	**2.75**	487	383	301	28.8%	2.9	232	169	118	18.7%	3.6
Base	**3.05**	648	**520**	418	34.7%	2.6	393	**305**	236	25.9%	3.0
Max	**3.25**	755	611	497	38.4%	2.5	501	397	314	30.3%	2.7

**Base Case: Year1= US$ 3.20 / Year2= US$ 3.15 / Year3= US$ 3.10*

**Min Case: Year1= US$ 2.90 / Year2= US$ 2.85 / Year3= US$ 2.80*

**Max Case: Year1= US$ 3.40 / Year2= US$ 3.35 / Year3= US$ 3.30*

Table 22-2 Other Sensitivity Analysis

Variant	Range	NPV 7.5% Post Tax (US$Mill.)		
		Min	Base	Max
Capex	±10%	288	305	334
Opex	±10%	268	305	343
S.S. Recovery	±2.5%	276	305	335
Mineral Mill Feed	±10%	248	305	363
Taxes	±10%	284	305	327

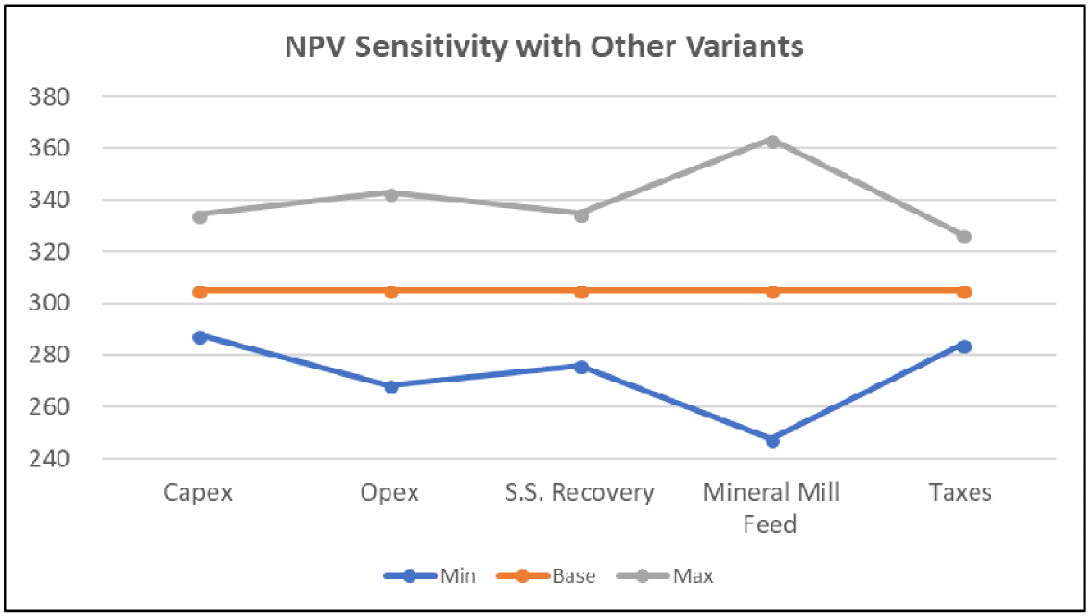


Figure 22-1 Other Sensitivity Analysis

22.8 Economic Results

The results of the analysis show the Antilla project mineral resources to be potentially viable, warranting further study. At a base case long term copper price of US$3.05/pound, the potential pre-tax present value of the net cash flow at the start of the projected two-year construction period using a 7.5% discount rate ($PVNCF_{7.5\%}$) is estimated at US$520 million, and potential project post-tax $PVNCF_{7.5\%}$ is estimated at US$305 million. Potential internal rates of return (IRR) are respectively 34.7% pre-tax and 25.9% post-tax. The payback period is estimated to be less than three years.

For the base case, Years 1 to 3 of the mine of life used estimated copper prices of US$ 3.20, us$ 3.15 and US$ 3.10, respectively. Molybdenum is not included in the proposed process recovery and not included in the project economics.

Annual cash flows by year are shown in Table 22-3.

The term PVNCF as utilized in this report is also commonly referred to as the project's NPV (net present value).

Table 22-3 Base Case Annual Cash Flows

			Project Year																			
	Unit	TOTAL	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Cover/OB Leach Material	kT	330			0	0	0	0	0	0	0	0	0	0	0	330	0	0	0	0	0	0
Leach Cap Leach Material	kT	753			0	0	0	0	0	0	0	0	0	0	91	662	0	0	0	0	0	0
Primary Sulphide Leach Material	kT	512			0	0	0	0	0	0	0	0	0	0	131	381	0	0	0	0	0	0
Supergene Leach Material	kT	117,072			7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,079	5,927	7,300	7,300	7,300	7,300	1,866	0
Total Leach Material	**kT**	**118,667**			**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**7,300**	**1,866**	**0**
Leach Copper grade	%	**0.434**			0.540	0.591	0.581	0.551	0.488	0.435	0.405	0.426	0.421	0.362	0.322	0.293	0.475	0.420	0.379	0.285	0.348	0.000
Waste Mined	kT	**163,439**	4,000	23,625	14,672	8,568	3,591	3,147	4,085	7,731	16,219	13,948	5,620	4,956	14,036	16,072	16,680	3,048	1,260	1,435	747	0
Material to Stockpile	kT	**18,319**	0	371	3,028	1,526	1,409	1,853	915	1,635	764	1,631	840	44	7	1,093	1,020	1,952	92	66	72	0
Stockpile Reclaim	kT	**5,557**	0	0	0	0	0	0	0	0	0	0	0	0	1,482	3,000	0	0	1,075	0	0	0
Total Material Mined	kT	**294,868**	**4,000**	**23,996**	**25,000**	**17,394**	**12,300**	**12,300**	**12,300**	**16,666**	**24,283**	**22,879**	**13,761**	**12,300**	**19,861**	**21,465**	**25,000**	**12,300**	**7,577**	**8,801**	**2,685**	**0**
Total Material Moved	kT	**300,425**	**4,000**	**23,996**	**25,000**	**17,394**	**12,300**	**12,300**	**12,300**	**16,666**	**24,283**	**22,879**	**13,761**	**12,300**	**21,343**	**24,465**	**25,000**	**12,300**	**8,652**	**8,801**	**2,685**	**0**
Recovered lbs of Cu	**'000 lbs**	**819,314**	**0**	**0**	**63,007**	**68,958**	**67,791**	**64,291**	**56,940**	**50,756**	**47,255**	**49,706**	**49,122**	**42,238**	**36,858**	**30,130**	**55,423**	**49,006**	**44,200**	**33,254**	**10,379**	**0**
Payable Cu lbs (96% payable)	**'000 lbs**	**786,542**	0	0	60,487	66,200	65,079	61,719	54,662	48,726	45,365	47,717	47,157	40,549	35,384	28,925	53,206	47,045	42,432	31,924	9,964	0
Copper Price	$/lb				$3.20	$3.15	$3.10	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05	$3.05
Gross Cu Revenue	$'000	**$2,417,900**	0	0	$193,558	$208,529	$201,746	$188,243	$166,720	$148,613	$138,364	$145,538	$143,830	$123,673	$107,922	$88,222	$162,279	$143,488	$129,417	$97,367	$30,391	$0
Off-site Costs	$'000	**$23,596**	0	0	$1,815	$1,986	$1,952	$1,852	$1,640	$1,462	$1,361	$1,432	$1,415	$1,216	$1,062	$868	$1,596	$1,411	$1,273	$958	$299	$0
Gross Mine Cu Revnue	$'000	**$2,394,304**	**$0**	**$0**	**$86,679**	**$198,434**	**$203,492**	**$193,735**	**$176,758**	**$156,975**	**$142,564**	**$140,214**	**$143,342**	**$133,393**	**$115,406**	**$98,042**	**$120,502**	**$152,272**	**$135,778**	**$113,798**	**$66,430**	**$16,489**
Mining Capital Cost	$'000	**$41,114**	$2,000	$39,114																		
Indirects Capital Cost	$'000	**$13,710**	$585	$13,125																		
Mine Equipment Capital Cost	$'000	**$1,300**	$50	$1,250																		
Process Capital Cost	$'000	**$94,662**	$0	$94,662																		
Infrastructure Capital Costs	$'000	**$42,440**	$5,850	$36,590																		
Owner's Cost Capital Cost	$'000	**$7,760**	$3,060	$4,700																		
Contingency Capital Cost	$'000	**$49,431**	$2,638	$46,794																		
Total Capital Cost	$'000	**$250,418**	**$14,183**	**$236,235**																		
Mining Operating Cost	$'000	**$445,359**	**$0**	$0	$40,750	$28,353	$20,049	$20,049	$20,049	$27,166	$39,581	$37,293	$22,430	$20,049	$34,789	$39,878	$40,750	$20,049	$14,102	$14,345	$4,377	$1,300
Rehandle Cost	$'000	**$4,168**	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$1,112	$2,250	$0	$0	$806	$0	$0	$0
G&A cost	$'000	**$89,001**	**$0**	$0	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$5,475	$1,400	$0
Processing Cost	$'000	**$552,745**	**$0**	$0	$29,930	$29,930	$29,930	$31,755	$31,755	$31,755	$33,580	$33,580	$33,580	$35,405	$35,405	$35,405	$37,231	$37,231	$37,231	$39,056	$9,983	$0
Closure Cost	$'000	**$23,000**	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$23,000
Total Operating Costs	**$'000**	**$1,114,272**	**$0**	**$0**	**$76,156**	**$63,758**	**$55,455**	**$57,280**	**$57,280**	**$64,396**	**$78,636**	**$76,349**	**$61,486**	**$60,930**	**$76,781**	**$83,008**	**$83,456**	**$62,755**	**$57,614**	**$58,876**	**$15,759**	**$24,300**
Pre-Tax NET cashflow	$'000	**$1,029,614**	-$14,183	-$236,235	$10,523	$134,675	$148,037	$136,456	$119,478	$92,579	$63,928	$63,866	$81,857	$72,463	$38,626	$15,034	$37,047	$89,517	$78,164	$54,922	$50,671	-$7,811
Pre-Tax Payback	years	**2.6**																				
Pre-Tax NPV (7.5% discount rate)	**$'000**	**$519,774**																				
Pre-Tax IRR	%	34.70%																				
Gross Revenue	$'000	**-$2,312,835**	$0	$0	-$88,493	-$208,529	-$201,746	-$188,243	-$166,720	-$148,613	-$138,364	-$145,538	-$143,830	-$123,673	-$107,922	-$88,222	-$162,279	-$143,488	-$129,417	-$97,367	-$30,391	$0
Costs and Expenses	$'000	**$1,137,868**	$0	$0	$77,970	$65,744	$57,407	$59,131	$58,920	$65,858	$79,997	$77,780	$62,900	$62,146	$77,842	$83,875	$85,052	$64,166	$58,887	$59,833	$16,058	$24,300
(-) Depreciaciones	$'000	**$255,998**	$0	$0	$57,686	$50,986	$50,986	$26,737	$26,737	$3,269	$3,269	$3,269	$3,269	$3,269	$2,653	$2,653	$2,653	$2,653	$2,653	$2,653	$2,653	$7,958
Operation Profit	$'000	**-$918,969**	$0	$0	$47,163	-$91,799	-$93,354	-$102,375	-$81,064	-$79,487	-$55,098	-$64,490	-$77,661	-$58,259	-$27,427	-$1,694	-$74,574	-$76,670	-$67,877	-$34,881	-$11,680	$32,258
(Simplified) Royalty	$'000	**$37,670**	$0	$0	$885	$3,000	$3,206	$4,122	$2,924	$3,151	$1,629	$2,120	$3,106	$2,037	$1,079	$882	$2,543	$3,036	$2,640	$974	$335	$0
Special Mining Tax	$'000	**$335**	$0	$0	$0	$29	$31	$37	$27	$28	$17	$21	$28	$19	$7	$0	$24	$27	$24	$10	$3	$0
OSINERGMIN	$'000	**$3,007**	$0	$0	$115	$271	$262	$245	$217	$193	$180	$189	$187	$161	$140	$115	$211	$187	$168	$127	$40	$0
OEFA	$'000	**$2,544**	$0	$0	$97	$229	$222	$207	$183	$163	$152	$160	$158	$136	$119	$97	$179	$158	$142	$107	$33	$0
Worker Profit Sharing	$'000	**$75,430**	$0	$0	$0	$3,201	$7,171	$8,190	$6,485	$6,359	$4,408	$5,159	$6,213	$4,661	$2,194	$135	$5,966	$6,134	$5,430	$2,791	$934	$0
Income Tax (net deprec.)	$'000	**$255,897**	$0	$0	$0	$10,858	$24,326	$27,785	$22,001	$21,573	$14,954	$17,503	$21,077	$15,811	$7,444	$460	$20,239	$20,808	$18,422	$9,467	$3,170	$0
Total Taxes	**$'000**	**$374,883**	**$0**	**$0**	**$1,097**	**$17,589**	**$35,218**	**$40,586**	**$31,837**	**$31,468**	**$21,340**	**$25,152**	**$30,770**	**$22,826**	**$10,983**	**$1,689**	**$29,162**	**$30,350**	**$26,826**	**$13,475**	**$4,515**	**$0**
After-Tax NET cashflow	$'000	**$654,731**	-$14,183	-$236,235	$9,426	$117,086	$112,819	$95,870	$87,641	$61,112	$42,588	$38,714	$51,087	$49,638	$27,643	$13,345	$7,885	$59,167	$51,338	$41,448	$46,155	-$7,811
After-Tax Payback	years	**3.0**																				
After-Tax NPV (7.5% discount rate)	**$'000**	**$305,432**																				
After-Tax IRR	%	25.90%																				

23 Adjacent Properties

Los Chancas Cu/Mo project and Trapiche Cu project represent the principal adjacent properties around Antilla. Los Chancas is 100% own by Southern Peru Corp, located 28 Km to the northwest of Antilla, and Trapiche is 100% own by Cia Minera Buenaventura, located 24 km to the southeast of Antilla. Both projects have advanced exploration and engineering studies.

Jalaoca Cu/Mo/Au project is a grassroots prospect into an ANAP (Area de No Admisión de Petitorios) covering 6,500 Ha. The project is located 11km to the south of Antilla and is reserved by the Peruvian Government to promote the private investment. The sale process is scheduled for Q4 2018 by Proinversion, a government entity. See Figure 23-1 below.

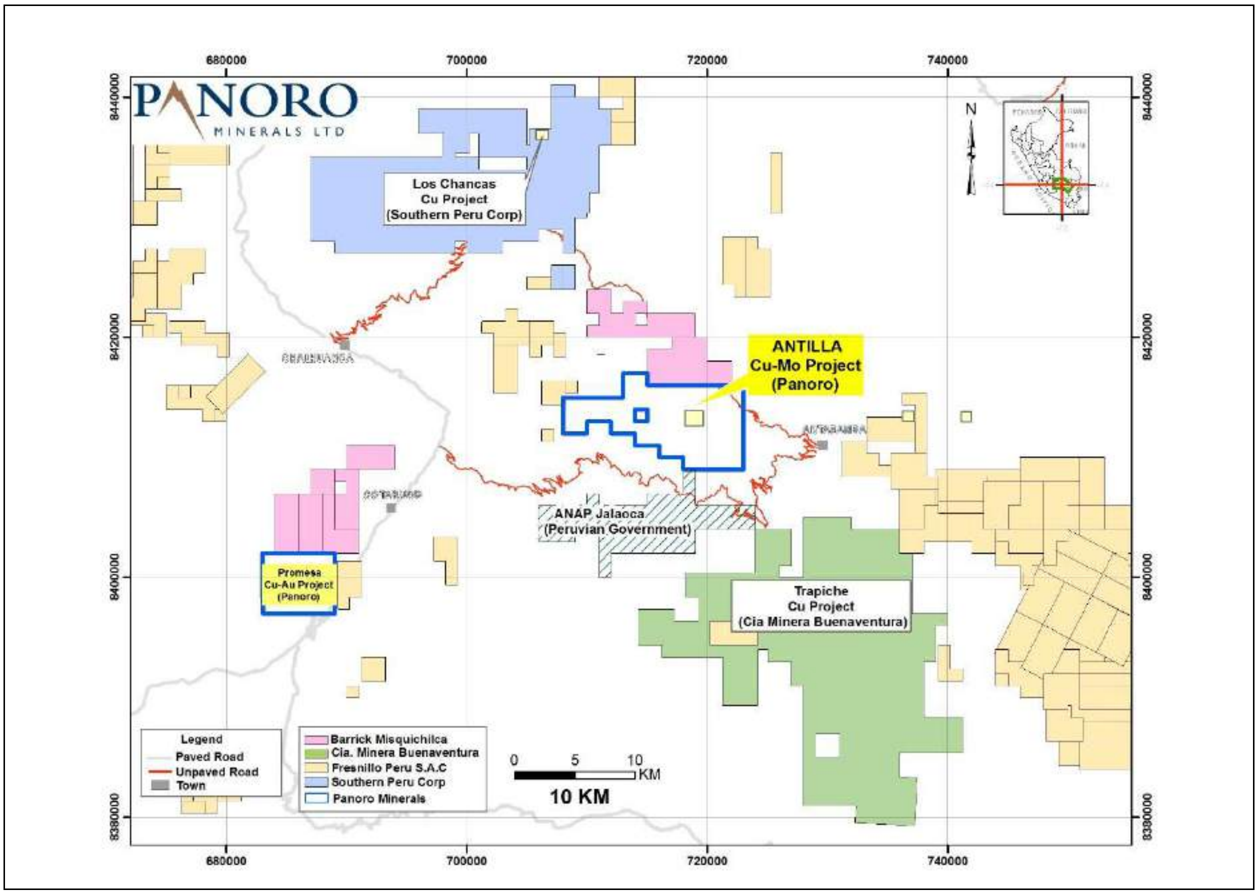


Figure 23-1 **Adjacent Properties to Antilla**

Antilla project is located into a mining region where new bulk-tonnage operations were constructed in the last 5 years, developing new infrastructure for future projects. They are located between 75 to 190 km to the east of Antilla, such as Las Bambas Cu-Mo mine (MMG), Constancia Cu-Mo mine (Hudbay Minerals) and Antapaccay Cu-Mo mine (Glencore).

24 Other Relevant Data and Information

There is a column testwork in progress developed in Aminpro Laboratories in Lima, Peru. The leaching test started in March 15th 2018 and final results are hoped for September 2018.

The authors are not aware of any other relevant information with respect to the Antilla Property that is not disclosed in this report.

25 Interpretation and Conclusions

A PEA open pit mine plan has been developed for the Antilla project incorporating a heap leach recovery method. The results of the PEA show improved economics, a lower capital cost and lower operating costs (when compared to a flotation recovery method).

The following conclusions are observed by area.

25.1 Geology and Mineral Resources

- The secondary sulphides blanket is hosted by the sandstones layers in the lower levels of the Soraya formation. The main mineralogy is supergene chalcocite, covellite and digenite.
- The PEA was based on a Mineral Resources model prepared by Tetra Tech, which is documented in a technical report filed on Sedar, dated of 2013 and updated in 2015.
- There is a high geological potential for discovery additional supergene mineralization next and around the current mineral resources area. Five copper anomalies were identified with mapping and rock sampling covering an influence area of 3km x 6km.

25.2 Mining

- The total heap leach material to mill feed is 119 Million tonnes averaging 0.43%Cu, where a 95% is in indicated and 5% in inferred resources category.
- Mine production is scheduled to provide 20kt/d to the heap leach pad at a life-of-mine average strip ratio of 1.38:1 (tonnes waste : tonnes leach material) The operational life of the mine is 17 years
- Mining is done using typical excavator/truck mining methods, utilizing a contract miner
- The first 10 years of mine production include only supergene (secondary sulphide) material due to the higher recovery rate for this material

25.3 Metallurgical Testing

- Both primary and secondary sulphide materials respond well to conventional treatment by milling and flotation. A conventional treatment process may be considered as part of a phased development for primary ores.
- Conventional treatment also provides for the recovery of by-product molybdenum.
- Secondary sulphides respond well to a simulated bio-leach ferric leach cycle. Based on interim results for the Aminpro bulk sample, copper leach extractions of circa 75 % are indicated with cycle times of between 150 days and 175 days for 3/8" and 1" crush sizes respectively. Future trade-off studies will determine the economic crush size. These predictions remain to be confirmed in the laboratory.

25.4 Mineral Processing

- The processing method considers a throughput of 7.3 million tonnes per year of ore through a conventional crushing, followed by leaching with sulfuric acid in a valley-fill type operation.
- The total ore expected to be processed will be approximately 120 million tonnes total over the 17 years of mine life.
- The pregnant leach solutions (PLS) percolating from the leaching operation at an approximate rate of 650 m3/h will be subject to solvent extraction in a conventional multi-stage SX plant.

- A final Electrowinning stage will produce Grade A copper cathodes at an initial rate of 30,000 tonnes per year that will progressively decrease to 5,000 tonnes per year by year 17 of the life of mine. The life of mine cathodes production will total approximately 372,000 tonnes.

25.5 Economics

- The initial capital has been reduced through incorporation of a heap leach processing method
- The after-tax NPV for the base case scenario offers a good multiplier on initial capital
- After-tax payback is achieved in less than 3 years
- The economics are sensitive to changes in copper recovery

The following risks are observed for the Antilla project

a) Pyrite as a principal accessory mineral in the deposit may represent an acidity source to be managed in the waste dams. However, it may also represent a principal source for ferric solution to use in the leaching process.

26 Recommendations

The PEA for the Antilla project indicates its potential as an economically viable mining operation. The Qualified Persons recommend that the Project should proceed to a pre-feasibility study (PFS). A description of the activities to support a PFS, along with costs in Canadian dollars is shown below.

26.1 Geology Recommendations

- A geotechnical and geomechanical program to support a PFS is recommended.
- Significant volume of primary sulphides is drilled and left below the economic PEA pit. This material may represent an opportunity to expand the throughput scale of the project with new metallurgical testwork for leaching. It is also recommended explore a flotation process that could extract copper from the primary sulphide material more efficiently than a leaching process.
- Two drilling programs are proposed; an infill drilling program to upgrade indicated to measured resources inside the PEA Pit and an expansion program to explore and estimate inferred resources over the copper anomalies recognized in areas located 3 km around the PEA Pit.
- The ***infill drilling program*** is focus on the first three years production mineral of the mine plan, into the secondary sulphides blanket. The program includes 4,800m distributed in 53 drillholes, reducing the holes spacing to 50m x 50m. Considering a global cost of 400 US$/meter of drilling the cost of this program could be around US$ 1.9 Million. The Figure 26-1 shows the infill drillholes location.

Table 26-1 Summary of Budget for Infill

ITEM	
Drilling (4,800m at $200/m)	$ 960,000
Assays (2,400Samples at $50/sample)	$ 120,000
Access (10kms at $ 3,000/km)	$ 30,000
Company Personal	$ 150,000
Administration	$ 80,000
Logistics	$ 100,000
Technical Studies	$ 50,000
Environment permit	$ 100,000
Safety & Security	$ 30,000
Social Permit (~20%)	**$ 300,000**
Total	**$ 1,920,000M**

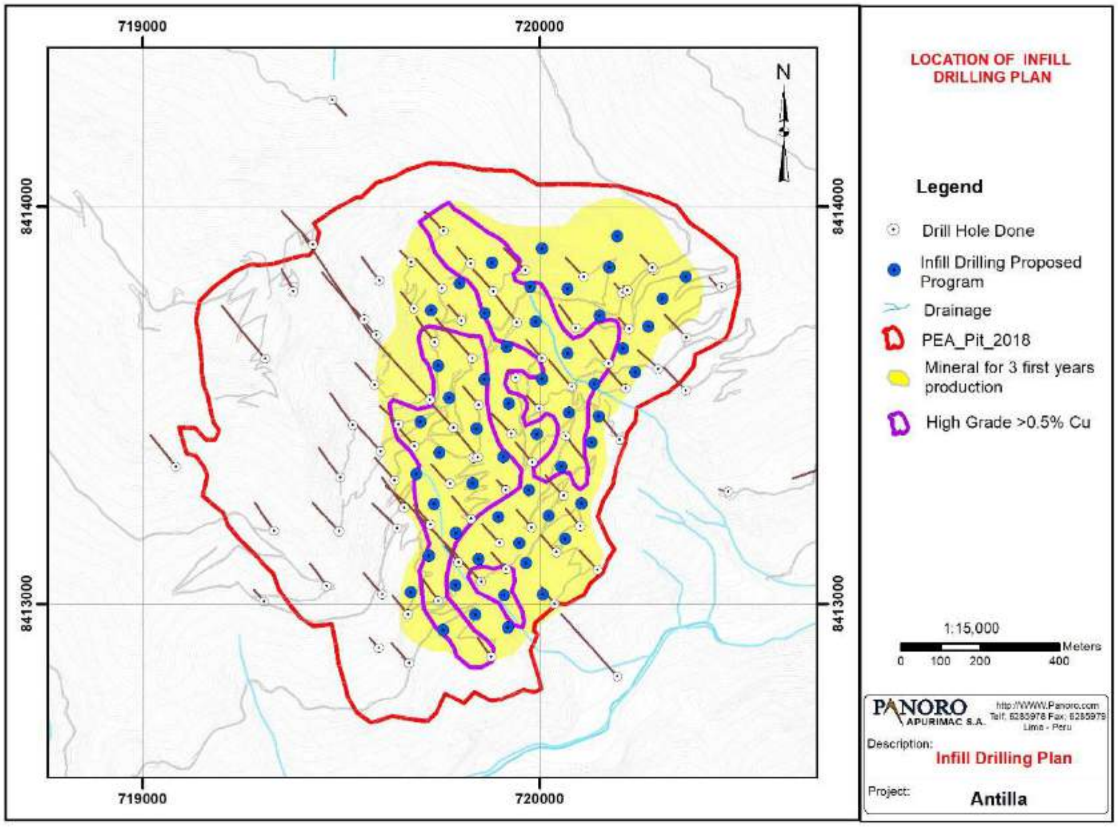


Figure 26-1 Proposed Infill Drilling Program inside the PEA Pit

- The expansion drilling program will explore and step out the mineral resources area towards the high copper anomalies with a spacing of 200m to 150m approximately, with 150m as average depth each hole. The program sum 12,800 meters and assuming a unit cost of 450 US$/meter of drilling the global cost will be around US$ 5.8 Million. The Table 26-2 show the number of drillholes and meters amount distributed by target. The Figure 26-2 show drillholes location.

Table 26-2 Expansion Drilling Program distributed by Exploration Target

Anomaly	N° Hole	Total Meters
Chabuca	18	3,600
North Block	17	3,600
Middle Block	16	3,200
West Block I	9	1,800
West Block II	3	600
Total	**63**	**12,800**

Table 26-3 Summary of Budget for Expansion Drillholes around the PEA Pit

ITEM	
Drilling (12,800m at $200/m)	$ 2,560,000
Assays (6,400Samples at $50/sample)	$ 320,000
Access (35kms at $ 3,000/km)	$ 105,000
Company Personal	$ 500,000
Administration	$ 200,000
Logistics	$ 350,000
Technical Studies	$ 100,000
Environment permit	$ 120,000
Safety & Security	$ 150,000
Social Permit (30%)	**$ 1,350,000**
Total	**$ 5,755,000M**

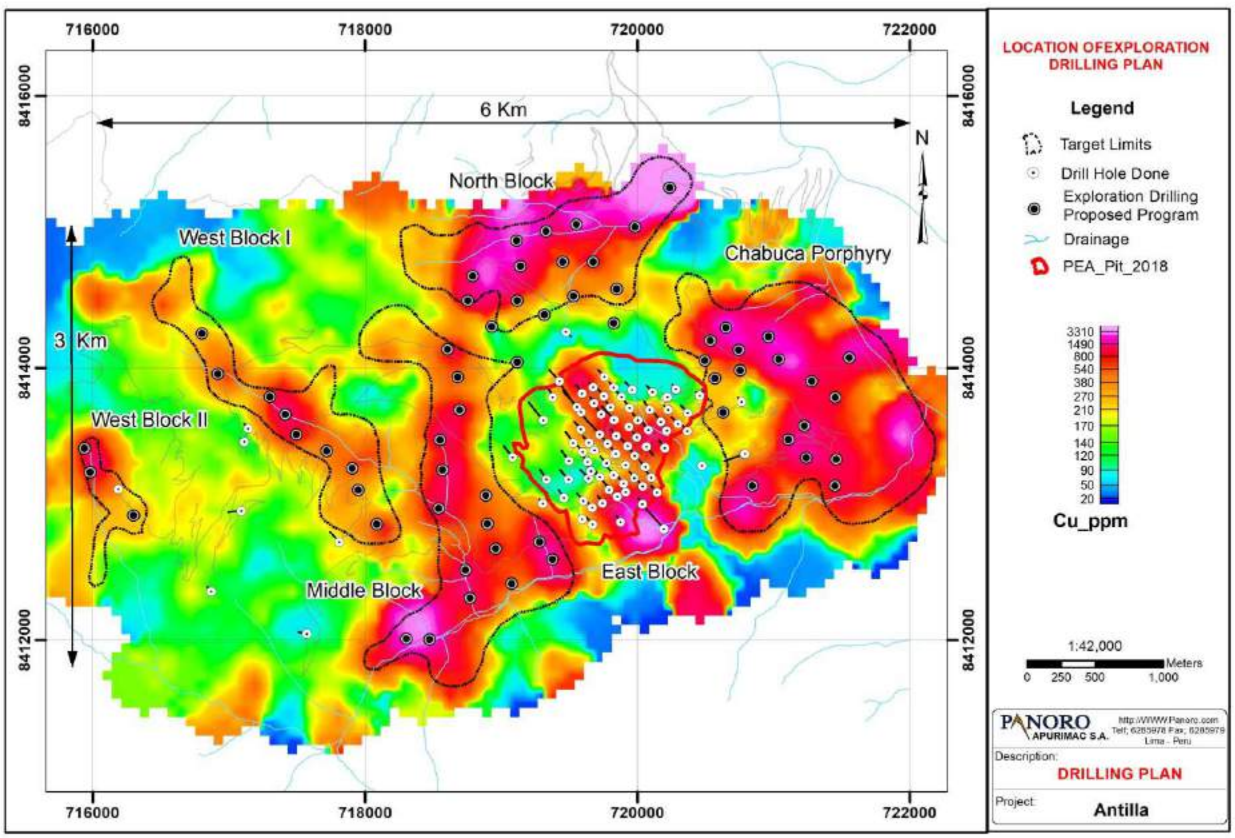


Figure 26-2 Location and Distribution of Exploratory Drillholes around the PEA Pit

A total cost estimated for both drilling programs is $7,675,000.

26.2 Mining Recommendations

The pit limit, phase designs, mining method/equipment and production schedule will be further optimized and detailed at a design level to support a PFS. These recommendations are not necessarily contingent on positive results from previous phases but reflect ongoing level of detail required to advance the project.

Activities involved in updating the mining section include (but are not limited to):

- Updating the ultimate economic pit limits using Measured and Indicated Resources only from the updated block model based on the latest in-fill drilling
- Examine potential backfill strategies to reduce waste haul costs
- Optimize the pit phase progression
- Optimize the production schedule through examination of different waste placement locations and strategies
- Update the operating cost estimates using budgetary quotes from mining contractors
- Update the leach pad design based on geotechnical study
- Examine expansion opportunities utilizing a flotation recovery method in the primary sulphides
- Develop a detailed reclamation plan

Total costs estimated between $200,000 and $300,000.

26.3 Metallurgical Testing Recommendations

The copper mineralization of the Antilla primary and secondary sulphide zones responds well to conventional treatment by milling and flotation. Molybdenum from both mineralized zones also responded well to treatment. The impurity levels of the copper concentrates are not expected to attract smelting penalties as defined by most smelters. If a mill and concentrator were to be considered as part of a later phase development or part of an integrated treatment facility a comprehensive testwork and development program on primary zone material will be required. This will likely include,

- Definition of more detailed lithological and mineralogical domains
- Identification and selection of metallurgical variability samples
- Advanced grinding tests, Bond, JK, SPI, SMC, MacPherson
- Grinding variability testwork
- Flotation optimisation testwork
- Flotation variability testwork
- Locked cycle flotation testwork
- Concentrate and tailings filtration tests
- Concentrate and tailings thickening tests
- Flotation pilot testwork
- Development of geometallurgical mine development plan

The amenability of Antilla supergene material to conventional acid heap leaching is poor. It is unlikely that a project can be successfully implemented based on an acid leaching of secondary sulphides. However, the supergene material is amenable to a ferric leach cycle and acceptable leach extractions have been demonstrated in the laboratory. Although column tests are still underway cycle times

comparable with existing commercial operations are anticipated. A process based on a conventional copper bioheap leach is indicated for supergene materials.
Total estimated costs between \$250,000 and \$500,000.

Future process development work should focus on,

- Identification, isolation and adaptation of suitable bacterial consortia
- Bioleach amenability tests using, mesophile, moderate thermophile and thermophilic bacteria
- Bioleach column optimisation tests using the selected bacteria
- Potential of using thermophiles for leaching primary ore
- Locked cycle column tests
- Iron and acid balances and internal process requirements
- Understanding arsenic mineralogy, arsenic generation, stabilisation and control
- Neutralisation and residue stabilisation tests
- Supporting environmental testwork
- Ore variability testing
- Process modelling
- Development of geometallurgical based mine model, acid/base accounting,
- Crushing and agglomeration and "engineered" feed material
- Potential for an integrated process facility comprising bioheap and conventional mill and concentrator technologies for both secondary and primary ores
- Potential large-scale column or site trial heap testwork

Total estimated costs between \$500,000 and \$750,000.

The mine plan also indicates the availability of significant quantities of cover and cap materials containing copper values these are inferred as 0.3 Mt and 0.7 Mt grading 0.28 and 0.26%Cu, respectively. Little work has been undertaken to date regarding recovery of copper from these resources and testing of these materials should also be incorporated in any future development plans.

Total estimated costs \$200,000.

26.4 Process Recovery Recommendations

It is recommended to perform a detailed calculation of the capacity of the valley-fill leaching operation. This calculation should include at least the following key elements:

- Stability analyses of the heap leach under the local conditions of seismicity and pluviosity.
- Study the ground conditions of the heap leaching area to determine the most cost-effective method to impermeabilized the ground and ensure collecting all percolating solutions through the leaching operation.
- Develop a detail loading plan of the leaching operation that will guarantee to meet the key metallurgical conditions necessary to ensure the optimum copper extractions are achieve. This detailed ore loading plan should also confirm that the valley selected to host the leaching operation has enough or exceeds the ore capacity required by Antilla Project.

Additional recommended work for the next engineering phase includes:

- Test the ore under the target particle size conditions to ensure the required percolating rates can be achieved or can be exceeded when loaded onto the leaching pad.
- Develop a site-wide water balance using the local climate and processing conditions to verify sizing of the solution management system.
- Given the likely site conditions, it is recommended to develop detailed layout of the operating facilities to ensure proper sizing, particularly of the solution management systems, and to estimate the energy necessary to transfer acid solutions.

Total estimated costs around $100,000.

26.5 Environmental Recommendations

As the project advances the followings recommendations should be considered:

- A comprehensive baseline data collection campaign should be initiated as soon as possible, as the successful completion of an environmental impact assessment will be contingent on this information. This includes not only the project site, but all transport corridors to and from the site.
- A plan for completion of the environmental impact assessment should be developed as it requires considerable time to undertake. As part of this plan, early consideration of public consultation should be undertaken such that as far as practical community concerns can be addressed through the project development stage of the project.

Total estimated cost between $100,000 and $200,000.

26.6 Infrastructure Recommendations

It is recommended to proceed with onsite and offsite infrastructure details to support a PFS.

Total estimated cost between $60,000 and $75,000.

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