Preliminary Economic Assessment Technical Report for the Antilla Copper-Molybdenum Project, Peru

Report Prepared for

Panoro Minerals Ltd.

Report Prepared by

SRK Consulting (Canada) Inc.
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June 16, 2016

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Panoro Minerals Ltd.

Suite 1610, 700 West Pender Street
Vancouver, British Columbia, Canada
V6C 1G8
E-mail: info@panoro.com
Website: www.panoro.com
Tel: +1-604-684-4246
Fax: +1-604-684-4200

SRK Consulting (Canada) Inc.

Suite 1300, 151 Yonge Street
Toronto, Ontario, Canada
M5C 2W7
E-mail: toronto@srk.com
Website: www.srk.com
Tel: +1-416-601-1445
Fax: +1-416-601-9046

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Signature date: June 16, 2016

Authored by:

["Signed and Sealed"]
Paul Daigle, PGeo
(Resource Geology)
Tetra Tech Inc.

["Signed and Sealed"]
Jesse Aarsen, PEng
Senior Associate (Mining)
Moose Mountain Technical Services

["Signed and Sealed"]
Adrian Dance, PEng
Principal Consultant (Metallurgy)
SRK Consulting (Canada) Inc.

["Signed and Sealed"]
Brian Connolly, PEng
Principal Consultant (Mining)
SRK Consulting (Canada) Inc.

["Signed and Sealed"]
Maritz Rykaart, PEng
Principal Geotechnical Engineer
SRK Consulting (Canada) Inc.

["Signed and Sealed"]
Goran Andric, PEng
Principal Consultant (Mining)
SRK Consulting (Canada) Inc.

["Signed and Sealed"]
Luis Vela, CMC
Vice President, Exploration
Panoro Minerals Ltd.

With contributions by:

Aaron Becket - Operations Manager/ Principal Engineer - ATC Williams (Peru)
Antonio Samaniego - Corporate Consultant (Geotechnical Engineering) - SRK Consulting (Peru) Inc.
Carlos Soldi - Principal Consultant (Geotechnical Engineering) - SRK Consulting (Peru) Inc.

Peer Reviewed by:

["Signed and Sealed"]
Jean-Francois Couture, PhD, PGeo
Corporate Consultant (Geology)
SRK Consulting (Canada) Inc.

Cover: Panoro Staff with SRK Staff on Site Visit; Antilla Exploration Camp in the Background

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 *Standards of Disclosure for Mineral Projects* Technical Report for Panoro Minerals Ltd. (Panoro) by SRK Consulting (Canada) Inc. (SRK). The quality of information, conclusions, and estimates contained herein are consistent with the quality of effort involved in SRK's services. The information, conclusions, and estimates contained herein are based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Panoro subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Panoro to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Panoro. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

Executive Summary

Introduction

The Antilla project is an advanced exploration stage copper-molybdenum porphyry project, located in the Apurimac region of southern Peru. The project is wholly owned by Panoro Minerals Ltd. (Panoro), which is a mineral exploration company registered in Canada, based in Vancouver, Canada and Lima, Peru. Its shares are listed on the TSX Venture Exchange (TSX-V) as PML.V. .

In 2014 SRK was commissioned by Panoro to prepare a preliminary economic assessment for the Antilla project in collaboration with Tetra Tech Inc. (Tetra Tech), Moose Mountain Technical Services (Moose Mountain), and ATC Williams (ATC Williams). The study is based on a mineral resource model prepared by Tetra Tech in 2013.

This technical report summarizes the technical information that is relevant to support the disclosure of the preliminary economic assessment for the Antilla project pursuant to Canadian Securities Administrators' National Instrument 43-101. The opinions contained herein, effective May 02, 2016, are based on information collected by the various consultants throughout the course of their investigations. This technical report supersedes all prior technical reports prepared for the Antilla project.

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 2016 and is renewed yearly.

Geology 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 north-west 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 and quartz-arenite 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.

Three main mineralization types are found at Antilla: primary sulphide, secondary sulphides, and a leached cap 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.

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. 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, 96 core boreholes (15,985 metres) have been completed on the project.

In the central and eastern portions of the property, a geochemical rock chip and soil survey has defined a 3 by 5 kilometres area over the known Antilla deposit. This area appears as part of a larger east-west structural trend. The geochemistry has defined several additional exploration targets: Chabuca, West Block and East Block. Geological mapping and geochemical sampling are ongoing on these target areas.

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

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.

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. SRK visited the project between April 20 and 25, 2014 and again from May 5 to 9, 2014. 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. The purpose of SRK's site visits was to ascertain the overall project site, inspect project accessibility, and examine the possible locations for project infrastructure.

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.

Mineral Resource and Mineral Reserve Estimates

The preliminary economical assessment documented in this technical report is based on a mineral resource model prepared by Tetra Tech in conformity with generally accepted CIM *Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines* (May 2014) and reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. 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 considers 88 core boreholes (14,293 metres) drilled from 2003 to 2010 by Panoro and the previous operators of the property. The mineral resources were evaluated using a geostatistical block modelling approach constrained by mineralization wireframes. Block copper and molybdenum values were estimated using ordinary kriging informed from capped composite data. In October 2015, Tetra Tech revised

the block classification in light of a new 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 project is presented in Table i. No mineral reserves have been defined for the Antilla project.

Table i: Mineral Resource Statement*, Antilla Copper-Molybdenum Project, Peru, Tetra Tech Inc., October 19, 2015

Domain	Quantity '000 tonnes	Grade 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.

Mining Methods

A conceptual level mining design, production schedule, and cost model were developed for the Antilla project. A series of pit optimizations were run using the resource block model, applying a range of metal prices and estimated costs for mining and processing. An ultimate pit limit was chosen from the optimized shell and the potentially mineable portion of the resource was estimated within the ultimate pit. The mineralized material contained within the ultimate pit shell and included in the conceptual mine plan consists of 291.1 million tonnes averaging a NSR value of US$16.35/tonne and grading an average of 0.322% copper, 0.0089% molybdenum in the Indicated category, and 59.8 million tonnes averaging a NSR value of US$12.90/tonne and grading an average of 0.249% copper, 0.0071% molybdenum in the Inferred category. The reader is cautioned that Inferred mineralized material are included in the conceptual mine plan and these 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 reserves. Mineral resources that are not mineral reserves have not demonstrated economic viability.

A mill production rate of 40,000 tonnes/day is assumed. The production schedule includes two years of pre-stripping followed by 24 years of production life. Lower grade material mined during the first years of production will be directed to a low-grade stockpile west of the pit. During the last five years of mine life, material from the low-grade stockpile will be reclaimed and used to supply the mill.

Recovery Methods

The proposed project includes a flotation processing plant with a design capacity of 40,000 tonnes/day. According to the conceptual production schedule, Supergene mineralization would be processed during the first half of the 24-year mine life with a transition after Year 11 to predominantly Primary Sulphide feed. Most Leach Cap material will be stockpiled until the end of the mine life for separate processing though the plant.

The run-of-mine mineralized material will feed a gyratory crusher ahead of a conventional SAG and ball mill grinding circuit. A bulk copper-molybdenum concentrate will be recovered in rougher and scavenger flotation stages. Following regrinding, molybdenum will be separated from the bulk concentrate in three or more stages

of cleaner flotation. Both concentrates will be thickened and filtered while the tailings will be thickened prior to being pumped to the tailings storage facility.

Metallurgical testwork was completed in 2013 by Certimin Laboratories S.A. in Lima, Peru on individual samples of Supergene and Primary Sulphide mineralized material. No metallurgical testwork has been conducted on the Cover or Leach Cap domains.

For the Supergene and Primary Sulphide mineralization domains, the copper concentrate will grade 20% to 30% copper, free of penalty elements and with precious metal content below payable levels. The molybdenum concentrate will grade 32% to 40% molybdenum and future testwork will determine if further processing is required to reduce the copper and zinc levels. Both concentrates will be transported off site via truck with the copper concentrate proposed to be shipped out of the port of Marcona in Nazca province.

Project Infrastructure

The property is accessible via 60-kilometre-long gravel road from the village of Santa Rosa on the paved Carretera Interoceánica Highway. The infrastructure to be developed for the project includes roads, water and power supply, mine waste containment and storage facilities, accommodation complex, support buildings, support facilities (i.e., fuel storage, warehousing etc.), and processing facilities.

The majority of the site buildings, including the permanent camp, will be located approximately 1.5 kilometres west of the open pit, immediately adjacent to the plant site. Mine rock from the open pit will be stored in one of eight rock storage facilities in close proximity to the open pit, or hauled to the tailings storage facility and used to construct the embankment. Mill feed will be stockpiled at a location adjacent to the process plant and reclaimed throughout and at the end of the mine life. The copper concentrate will be pipelined to a remote filter plant / truck loadout site southeast of the pit area, where it will be loaded and truck hauled to the final destination.

Flotation tailings will be pumped to the tailings storage facility located in the Huancaspaco River basin southwest from the process plant. A containment dam will be constructed predominantly from waste rock produced from the mining activities and will include a geomembrane liner on the upstream face. Additional zones within the containment dam will be constructed with borrow material. Dam construction will be staged over the life of the mine using the downstream construction method. Reclaim water from the tailings storage facility will be circulated back to the mill. At closure, the tailings surface will be covered with a geosynthetic membrane liner and a growth medium, and the downstream face of the containment dam will be covered with a growth medium.

Surface water will be used to supply all of the water needs of the proposed Antilla project. This water can be obtained through the collection of runoff during the wet season. The process water make-up requirements will be sourced from the contact water collection pond and the fresh water reservoir.

Power will be supplied via a 50-kilometre-long, high voltage transmission line from site to the Cotaruse substation on the Peru national electricity grid.

Market Studies and Contracts

No market studies have been completed. No contractual arrangements for concentrate trucking, port usage, ocean shipping, smelting or refining exist at this time. There are no contracts in place for the sale of copper and molybdenum concentrates. It is assumed that the concentrate produced at the Antilla mine would be marketed to smelters in Asia.

Environmental Studies, Permitting, and Social or Community Impact

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.

Capital and Operating Costs Estimates

All costs are expressed in Q4 2015 United States dollars. The capital costs for the project are summarized in Table ii. The pre-production initial capital costs have been estimated at $603 million and sustaining capital costs at US$324 million. Both estimates include direct cost, indirect costs, and contingency. The accuracy of this estimate is considered to be within -20% to +50%.

Table ii: Capital Cost Estimate Summary

Description	Initial CAPEX (USM$)	Sustaining CAPEX (USM$)	Total CAPEX (USM$)
Mine Equipment	51	111	162
Mine Development	55	-	55
Processing	236	-	236
Tailings Storage Facility	25	145	169
Site Infrastructure and Services	110	9	119
Owner's Costs	28	-	28
Subtotal, before Contingency	**$506**	**$264**	**$770**
Contingency	97	60	157
Total Capital Cost	**$603**	**$324**	**$927**

Total project operating cost is estimated to be US$4,088 million, and includes on-site and off-site costs. On-site operating costs include the operating areas of mining, processing, and administration. Off-site operating costs cover all activities associated with concentrate transport, insurance, smelting, refining, and roasting charges. A summary of life-of-mine operating costs is shown in Table iii. The operating costs are considered to have accuracy of -20% to +50%.

Table iii: Operating Costs by Major Functions

Operating Cost Component	Unit Cost $/t milled	Total OPEX $M	Percent of Total OPEX
Mining	3.57	1,250	31%
Processing	4.60	1,612	39%
Tailings pumping	0.18	64	2%
G&A	0.75	263	6%
Subtotal On-Site Costs	**9.10**	**3,188**	**78%**
Off-Site Costs		900	22%
Total Operating Costs		**$4,088**	**100%**

Economic Analysis

A preliminary economic assessment is a conceptual study of the potential viability of mineral resources. The preliminary economic assessment includes Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. Furthermore, there is no certainty that the conclusions or results as reported in the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves have not demonstrated economic viability.

The Antilla project has been evaluated on a discounted cash flow basis assuming 100% equity project financing. The cash flow analysis has been prepared in constant fourth quarter 2015 US dollars. No inflation or escalation of revenue or costs has been incorporated.

The results of the analysis show the proposed Antilla project as presented herein is potentially viable. At a base case copper price of US$3.00/pound and molybdenum price of US$12/pound, the potential pre-tax present value of the net cash flow at the start of the projected two-year construction period using a 7.5% discount rate ($PVNCF_{7.5\%}$) is estimated at US$491 million, and potential project post-tax $PVNCF_{7.5\%}$ is estimated at US$225 million. Potential internal rates of return (IRR) are respectively 22.2% pre-tax and 15.1% post-tax. The term PVNCF as utilized in this report is also commonly referred to as the project's NPV (net present value).

At base case metal prices the project payback period is estimated to be slightly over four years. Payback period is defined as the time after process plant start-up that is required to recover the initial expenditures incurred developing the Antilla project.

Like most mining projects the key economic indicators of $PVNCF_{7.5\%}$ and IRR are most sensitive to changes in metal prices. The proposed Antilla project is particularly sensitive to the price of copper since copper is the source of 92% of base case revenue. A $0.50/pound reduction in the copper price reduces Antilla's post-tax IRR by 11.3%. A $0.50/pound increase in the copper price increases Antilla's the post-tax IRR by 7.5%.

Conclusions and Recommendations

This technical report provides a summary of the results and findings from each major area of investigation including exploration, geological modelling, mineral resource and plant feed estimations, mine design, metallurgy and process design, infrastructure, environmental management, capital and operating costs, and economic analysis. The level of investigation for each of these areas is considered to be consistent with that normally expected with preliminary economic assessment for resource development projects.

The results of the preliminary economic assessment indicate the proposed Antilla project is potentially economically viable. The favorable economic potential warrants further technical studies to examine the potential viability of the proposed project at a higher level of confidence and the preparation of a preliminary feasibility study.

Exploration activities to date have been completed under the appropriate Peruvian permits. Most of the infrastructure proposed for the conceptual project is within the mineral concession boundaries. However, the fresh water storage reservoir and a portion of the tailings storage facility are located outside the current mineral concession limits.

Panoro holds no surface rights in the area, and any future mining activities will require agreements to be negotiated with both local communities and individual surface rights holders. The inability to purchase or lease the lands on which the proposed Antilla project would be developed poses a significant risk.

In reviewing risks and opportunities presented by the proposed Antilla project, additional exploration, engineering and environmental work is warranted to address risks and opportunities identified during this study. The proposed work program includes infill and stepout exploration drilling, mine design optimization studies, additional metallurgical testing on new samples from the four mineralization types, rock geotechnical and hydrogeological investigations of the quality of the rock mass for mine design, soil geotechnical investigations and geochemical characterization of mine waste product to support waste management design; and environmental baseline studies to provide the information to initiate an environmental impact assessment.

These additional studies are required to allow evaluating the economic viability of the proposed Antilla project and the completion of a preliminary feasibility study.

The total cost for the recommended work program is estimated at US$11.1 million.

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1 Introduction and Terms of Reference

1.1 Introduction

Panoro Minerals Ltd. (Panoro) is a mineral exploration company registered in Canada, based in Vancouver, Canada and in Lima, Peru. Its shares are listed on the TSX Venture Exchange (TSX-V) as PML.V. Panoro is focused on exploring and developing its copper-molybdenum and copper-gold deposits in Peru.

The Antilla project is an advanced exploration stage copper-molybdenum project, located approximately 140 kilometres southwest of Cusco, Peru. Panoro wholly owns the project.

In 2013, Panoro commissioned Tetra Tech Inc. (Tetra Tech) to prepare a Mineral Resource Statement in conformity with generally accepted CIM *Exploration Best Practices Guidelines* and CIM *Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines*.

In 2014 SRK Consulting (Canada) Inc. (SRK) was commissioned by Panoro to prepare a preliminary economic assessment for the Antilla project in collaboration with Moose Mountain Technical Services (Moose Mountain) and ATC Williams, both directly contracted by Panoro.

The purpose of this technical report is to provide the results of a technical and economical evaluation of the Antilla copper-molybdenum project at conceptual level based upon a mineral resource model prepared by Tetra Tech and documented in a technical report dated December 16, 2013. This technical report is a preliminary economical assessment pursuant to Canadian Securities Administrators' National Instrument 43-101and Form 43-101F1. The study is intended to provide Panoro management with a technical and financial basis upon which make strategic decisions on the future direction of the project.

1.2 Scope of Work

The scope of work included:

- Review of the geology and mineral resource model prepared by Tetra Tech
- Geotechnical assessment
- Waste / tailings geotechnical assessment and management
- Open pit mine planning and design
- Metallurgy review and evaluation of the process plant design criteria
- Review and plan infrastructure requirements
- Environmental assessment and permitting requirements
- Economic analysis
- Provide recommendations for the further development of the project
- Compile a technical report describing the preliminary economic assessment for the Antilla project

1.3 Contributing Authors

1.3.1 Management of the Preliminary Economic Assessment

The information reported herein is a collaborative effort between Panoro, SRK, Moose Mountain, and ATC Williams. Goran Andric, PEng of SRK and Luis Vela Arellano, CMC, of Panoro (Vice President, Exploration) were responsible for the overall management of the preliminary economic assessment.

1.3.2 Qualified Persons

The details of the various qualified persons and their respective areas of technical responsibility are presented in Table 1 below.

Table 1: Qualified Persons and Respective Area of Technical Responsibility

Contributing Author	Company	Area of Technical Responsibility	Sections of this Report
Paul Daigle, PGeo	Tetra Tech	Geology and Resource Estimation	11,13, 24, 25
Luis Vela Arellano, CMC	Panoro	Geology and Resource Estimation Review	3-11, 22, 23
Goran Andric, PEng	SRK	Project Management, Infrastructure, Cost Estimation	1, 2, 14, 15.2, 17, 18, 20, 24, 25
Jesse Aarsen, PEng	Moose Mountain	Mining Methods, Cost Estimation	15, 20, 24, 25
Dr. Adrian Dance, PEng	SRK	Process Metallurgy, Cost Estimation	12, 16, 20, 24, 25
Dr. Maritz Rykaart, PEng	SRK	Tailings Design, Cost Estimation, Social and Environmental Aspects	17, 19, 20, 24, 25
Brian Connolly, PEng	SRK	Economic Analysis, Cost Estimation	20, 21, 24, 25
Dr. Jean-François Couture, PGeo	SRK	Overall Project Review	All Sections

1.3.3 Others Contributing to the Preliminary Economic Assessment

Other contributors to the preliminary economic assessment include:

- Aaron Becket, Operations Manager/ Principal Engineer, ATC Williams (Peru)
- Dr. Antonio Samaniego, Corporate Consultant (Geotechnical Engineering), SRK (Peru)
- Carlos Soldi, Principal Consultant (Geotechnical Engineering), SRK (Peru)

Mr. Becket of ATC Williams (Lima) provided specialist information in relation to tailings location selection, design, and cost estimate that were used in Sections 17.3 and 20 of the report.

Dr. Samaniego of SRK Consulting (Lima) provided geotechnical recommendations for the project, presented in Section 15.2 of the report.

Mr. Soldi of SRK Consulting (Lima) provided specialist information in relation to social, permitting, and environmental aspects of the project that were used in Section 19 of the report.

1.3.4 Technical Report Responsibilities

Responsibilities for each report section are listed in Table 2.

Table 2: Responsibility of Technical Report Sections

Section	Title	Responsible
	Executive Summary	SRK / Moose Mountain / Panoro
1	Introduction and Terms of Reference	SRK
2	Reliance on Other Experts	SRK
3	Property Description and Location	SRK / Panoro
4	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	SRK / Panoro
5	History	Tetra Tech / Panoro
6	Geological Setting and Mineralization	Tetra Tech / Panoro
7	Deposit Types	Tetra Tech / Panoro
8	Exploration	Tetra Tech / Panoro
9	Drilling	Tetra Tech / Panoro
10	Sample Preparation, Analyses, and Security	Tetra Tech / Panoro
11	Data Verification	Tetra Tech / Panoro
12	Mineral Processing and Metallurgical Testing	SRK
13	Mineral Resource Estimates	Tetra Tech
14	Mineral Reserve Estimates	Moose Mountain
15	Mining Methods	Moose Mountain
16	Recovery Methods	SRK
17	Project Infrastructure	SRK
18	Market Studies and Contracts	SRK
19	Environmental Studies, Permitting, and Social or Community Impact	SRK
20	Capital and Operating Costs	SRK / Moose Mountain
21	Economic Analysis	SRK
22	Adjacent Properties	SRK
23	Other Relevant Data and Information	SRK / Panoro
24	Interpretation and Conclusions	SRK / Tetra Tech / Moose Mountain / Panoro
25	Recommendations	SRK / Tetra Tech / Moose Mountain / Panoro
26	References	SRK / Tetra Tech / Moose Mountain / Panoro

1.4 Basis of Technical Report

This technical report is based on the following sources of information:

- Previous technical report and mineral resource estimate of the Antilla copper-molybdenum project, prepared by Tetra Tech, dated December 16, 2013
- Mineral resource block model (described in this report) prepared by Tetra Tech, documented in an internal technical memorandum dated March 22, 2016
- Inspection of the Antilla project area during site visits, including outcrop and core and review of exploration data collected by Panoro
- Data, maps, drawings, and other project information provided by Panoro
- Discussions with Panoro management and technical personnel
- Cost information provided by Panoro and its solicited contractors and service providers
- Cost information from SRK, Moose Mountain, ATC Williams in-house databases
- Mining industry cost reference guides
- Assistance from Panoro with application of mining taxes in Peru
- Additional information from public domain sources

1.5 Qualifications of SRK

The SRK Group comprises more than 1,500 professionals, offering expertise in a wide range of resource engineering disciplines. The independence of the SRK Group is ensured because it holds no equity in any project it investigates and that its ownership rests solely with its staff. These facts permit SRK to provide its clients with conflict-free and objective recommendations. SRK has a proven track record in undertaking independent assessments of mineral resources and mineral reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies, and financial institutions worldwide. Through its work with a large number of major international mining companies, the SRK Group has established a reputation for providing valuable consultancy services to the global mining industry.

1.6 Site Visit

In accordance with National Instrument 43-101 guidelines, the following independent qualified persons visited the Antilla project:

- Paul Daigle, PGeo (PEO #1592) conducted a site visit to the Antilla project site from June 3 to 7, 2013. The purpose of the site visit was to review the digitalization of the exploration database and validation procedures, review exploration procedures, define geological modelling procedures, examine core, interview project personnel, and collect all relevant information for the preparation of the mineral resource model and the compilation of a technical report. During the visit, a particular attention was given to the treatment and validation of historical drilling data.
- Goran Andric, PEng (PEO #100103151) visited the Antilla project site from April 20 to 25, 2014 to ascertain the overall project site, inspect project accessibility, and examine the possible locations for project infrastructure. He has inspected core logs on site and in the core storage warehouse in Cusco.
- Dr. Maritz Rykaart, PEng (APEGBC #28531) visited the Antilla project site from May 5 to 9, 2014. He focussed on evaluating topographical and geotechnical conditions for siting and constructing a suitable tailings storage facility.
- Luis Vela Arellano (CMC #0173) has been to site on numerous occasions since he first become involved with the project in 2011, the most recent visit being 2 to 5 November 2015.

1.7 Acknowledgement

SRK would like to acknowledge the support and collaboration provided by Panoro personnel for this assignment. Their collaboration was greatly appreciated and instrumental to the success of this project.

1.8 Previous Technical Reports

Panoro has previously filed the following technical reports for the project:

- Lee, C. Nowak, M. and Wober, H. H., 2007. Independent Technical Report on the Mineral Exploration Properties of the Cordillera de las Minas S.A. Andahuaylas-Yauri Belt, Cusco Region, Peru. Report prepared by SRK Consulting (Canada) Inc. for Panoro, effective date March 9, 2007, 125 p.

- Wrigh, C., Waldo, A, and Vilela, E., 2009. Restated, Amedned Technical Report for the Antilla Property Apurimac, Peru. Report prepared by AMEC for Panoro, effective date June 1, 2009, amended August 23, 2009, 100 p.
- Daigle, Paul and Huang (John) Jianhui, 2013. Technical Report Estimate of the Antilla Copper-Molybdenum Project, Peru. Report prepared by Tetra Tech (Canada) for Panoro, effective date December 16, 2013, 130 p.

1.9 Declaration

SRK's opinion contained herein and effective **May 2, 2016** is based on information collected by SRK throughout the course of SRK's investigations. The information in turn reflects various technical and economic conditions at the time of writing this report. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Panoro, and neither SRK nor any affiliate has acted as advisor to Panoro, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

1.10 Terminology

Metric units of measurement and US dollars are used in this report unless otherwise stated.

2 Reliance on Other Experts

SRK has not performed an independent verification of land title and tenure information as summarized in Section 3 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, but has relied on Humerto Martínez Aponte of Abogados Rosselló, Lima, Peru as expressed in a legal opinion provided to SRK and dated March 10, 2016. A copy of the title opinions is provided in Appendix A. The reliance applies solely to the legal status of the rights disclosed in Sections 3.1 and 3.2 below. On June 2, 2016 Panoro confirmed with SRK that concession payments have been made and that the tenements are in good standing until June 2017.

The qualified persons have relied upon information received from Panoro on taxation applicable to the Antilla project. The information was provided by David W Huber, CA, chief financial officer of Panoro at the time, to Brian Connolly of SRK in personal email communication from November 2014 to January 2015. This information was utilized in Section 21 of the report, and in the Executive Summary, Section 24 Interpretation and Conclusions, and Section 25 Recommendations that relate to economic analysis.

SRK was informed by Panoro that there are no known litigations potentially affecting the Antilla project.

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

Figure 1: Location of the Antilla Project in Apurimac Region of Peru

3.1 Mineral Tenure

The project area comprises 12 mining concessions with a total area of 7,500 hectares (Table 3). The concessions are owned 100% by Panoro Apurimac S.A. a wholly-owned subsidiary of Panoro.

Panoro is currently operating under a Class B exploration permit. The property is subject to annual payments to maintain the concessions in good standing and the concessions are renewed on a yearly basis every June. As the time of writing this report, all concessions were in good standing.

Table 3: Mineral Tenure Information

Concession No.	Concession Name	Area (ha)	Grant Date	Expiry Date	Mineral Resource
10170402	Aluno Cinco 2002	100	31/01/2003	June 2017	
10170302	Aluno Cuatro 2002	800	31/01/2003	June 2017	Yes
10200202	Aluno Quince 2002	900	21/03/2003	June 2017	Yes
10043803	Valeria Quince 2003	1,000	27/01/2004	June 2017	
10043903	Valeria Dieciseis 2003	900	09/09/2003	June 2017	
10329903	Valeria Treintaidos	800	04/03/2004	June 2017	
10344303	Antillana 2003	1,000	26/02/2004	June 2017	
10344203	Antillana Uno 2003	800	16/02/2004	June 2017	
10166404	Valeria Sesentaiuno 2004	400	09/08/2004	June 2017	
10002003	Macla 2003	300	17/06/2003	June 2017	
10313306	Don Marti 1	300	17/11/2006	June 2017	
10059709	Antilla Uno	200	16/07/2009	June 2017	
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 2.

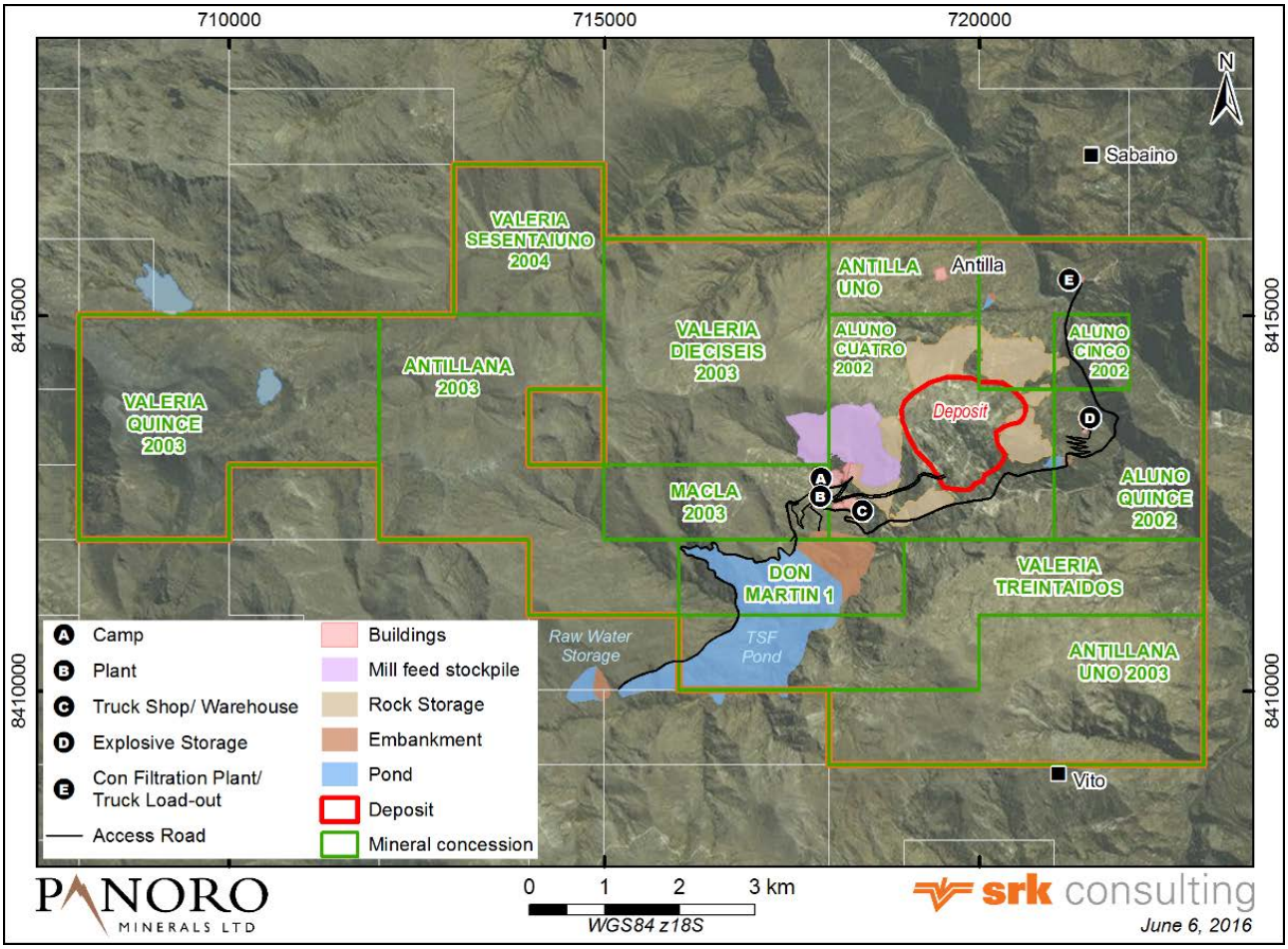


Figure 2: Tenement Map

Panoro has 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 does not include construction or pre-production activities on the property.

SRK is not aware of any other significant factors or risks that may affect access, title, or the right or ability to perform work on the property.

3.2 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.

3.3 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. SRK 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.

3.4 Environmental Considerations

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

3.5 Mining Rights in Peru

This section has been extracted from KPMG (2013) and KPMG (2016) with further sources within.

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

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 called The Institute of Geology, Mining and Metallurgy. Four types of concessions under Peruvian mining law:

- Mining concession: The right to explore and exploit the 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 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 fulfils 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 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. The law is expected to prevent social conflicts, similar to the social unrest that occurred in 2009 that resulted in the deaths of 24 police officers and 10 indigenous people. The president also signed a new mining law in 2011 that is expected to generate US$1 billion every year. This money will be directed toward social improvement for the poorest sections of the country.

On October 31, 2012, Peru’s congressional committee on the economy approved the creation of a new oversight institution, known as the national environmental certification service (SEANCE), 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 onward, 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 the fiscal year 2015-2016 is 28%. In mid-2014 Peru introduced a series of tax reforms that aim to reduce the corporate income tax rate to 26% by 2019. The tax rate on dividends is 6.8%, and on interest is 4.99% under specific conditions. Mining companies pay the standard corporate tax on income. In addition to the corporate income tax, the mining companies pay royalties. Under the revised version of the 2004 Mining Royalties Law, mining companies must pay on a quarterly basis a royalty to the state for exploitation of resources based on operating profit. Under the revised fiscal system, the new royalty is applied to operational profit of the mining companies – at a marginal-increment rate – rising from 1% to 12%, depending on operational margin, in 17 separate brackets. The state takes a percentage of the value of the concentrates at prevailing international market prices. The law also defines the distribution of royalties to the provinces in the following manner:

- Local governments of the districts in which the mining exploitation occurs receive 20% of royalties, of which 50% goes to the local communities located near the mine.
- The government of the province in which mining takes place receives 20%.

Concession rights are granted by the Peruvian government for mining, exploration, production, refining, and transport of minerals by individuals or companies. The following benefits 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.
- Taxation exemption: Investors who undertake large mining operations may enter into tax law stability agreements with the government for periods of ten and fifteen years. Tax stability would include taxes known as *impuestos*. No taxes created subsequently to the date of execution of the stability agreement shall be applicable.

Apart from these taxes, there are two industry-specific taxes levied on the mining sector by the central government:

- Special Mining Tax: A tax that is applied as a marginal rate, from 2% to 8.4%, on operational profit, also in successive brackets for different operational margins.
- Special Mining Lien: Mining companies that have taxation stability agreements previously signed with the government pay this contribution in the form of lien as the marginal rate rising from 4% to 13.2%.

The special mining tax and the special mining lien are assessed by multiplying an effective rate specifically assessed for each charge by the quarterly operational profits.

4 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

4.1 Accessibility

The property is situated 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.

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.

4.2 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. Weatherhaven 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. This power line is privately owned.

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.

4.3 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 4 millimetres to 150 millimetres 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.

4.4 Physiography

The property is located in the high altitudes of the Andean Cordillera where elevations vary between 2,500 to 4,500 metres above sea level. Relief on the property varies 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 village of Antilla (Wright 2009).

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.

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 3.

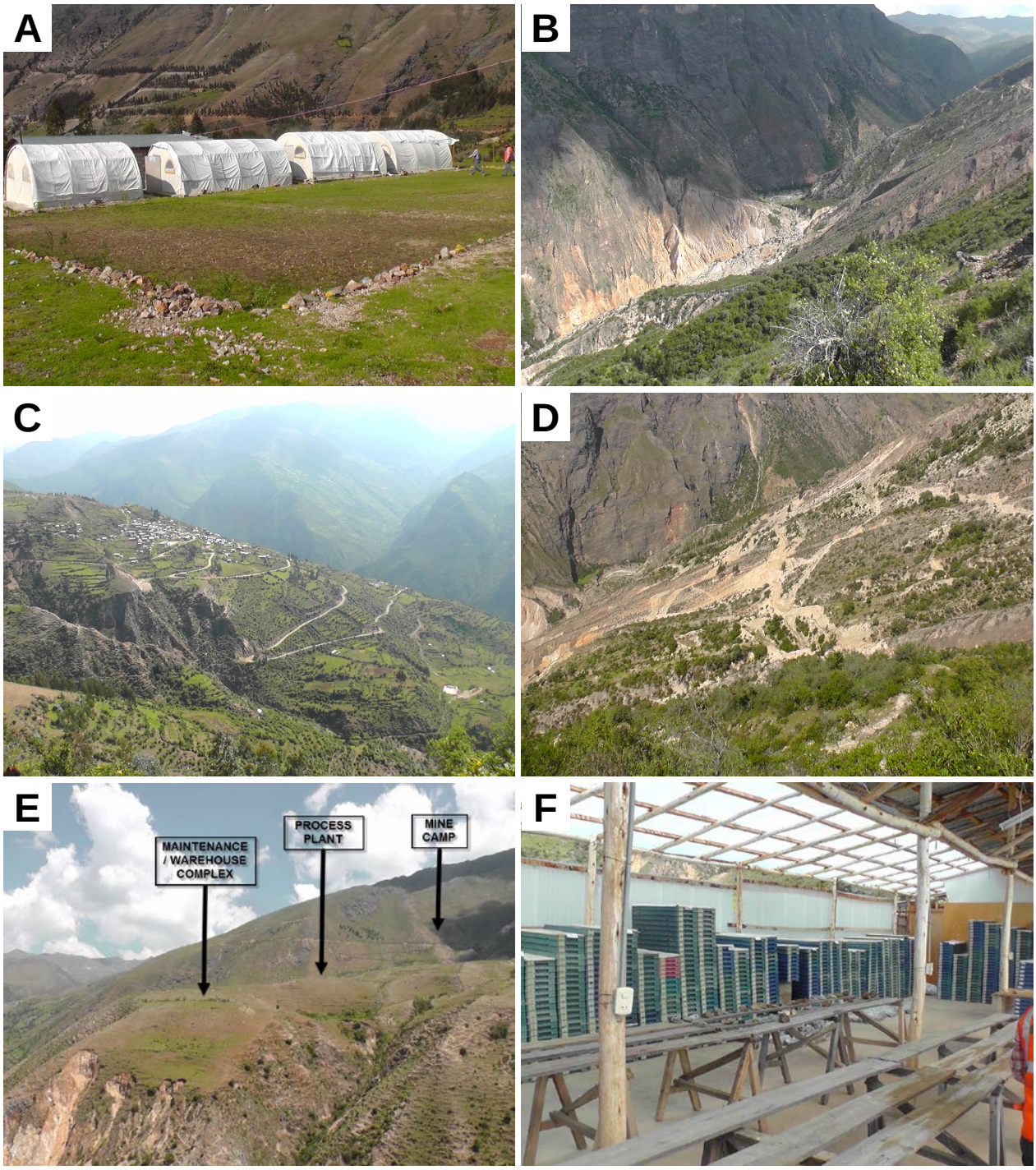


Figure 3: Typical Landscape in the Project Area

A. Exploration camp

B. Deeply incised valley of the Quebrada Huancaspaco

C. Village of Antilla

D. Difficult access to drill pads along mountain flank

E. Proposed plant/mine camp location

F. Core logging and storage facility

5 History

The following is modified from Wright (2009).

5.1 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.

5.2 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, re-collared, 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.

6 Geological Setting and Mineralization

6.1 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 4). 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 Palaeozoic basement rocks which are exposed to the northeast. The basement sequence culminates in Permian to Early Triassic age Mitu Group volcaniclastic and clastic rocks.

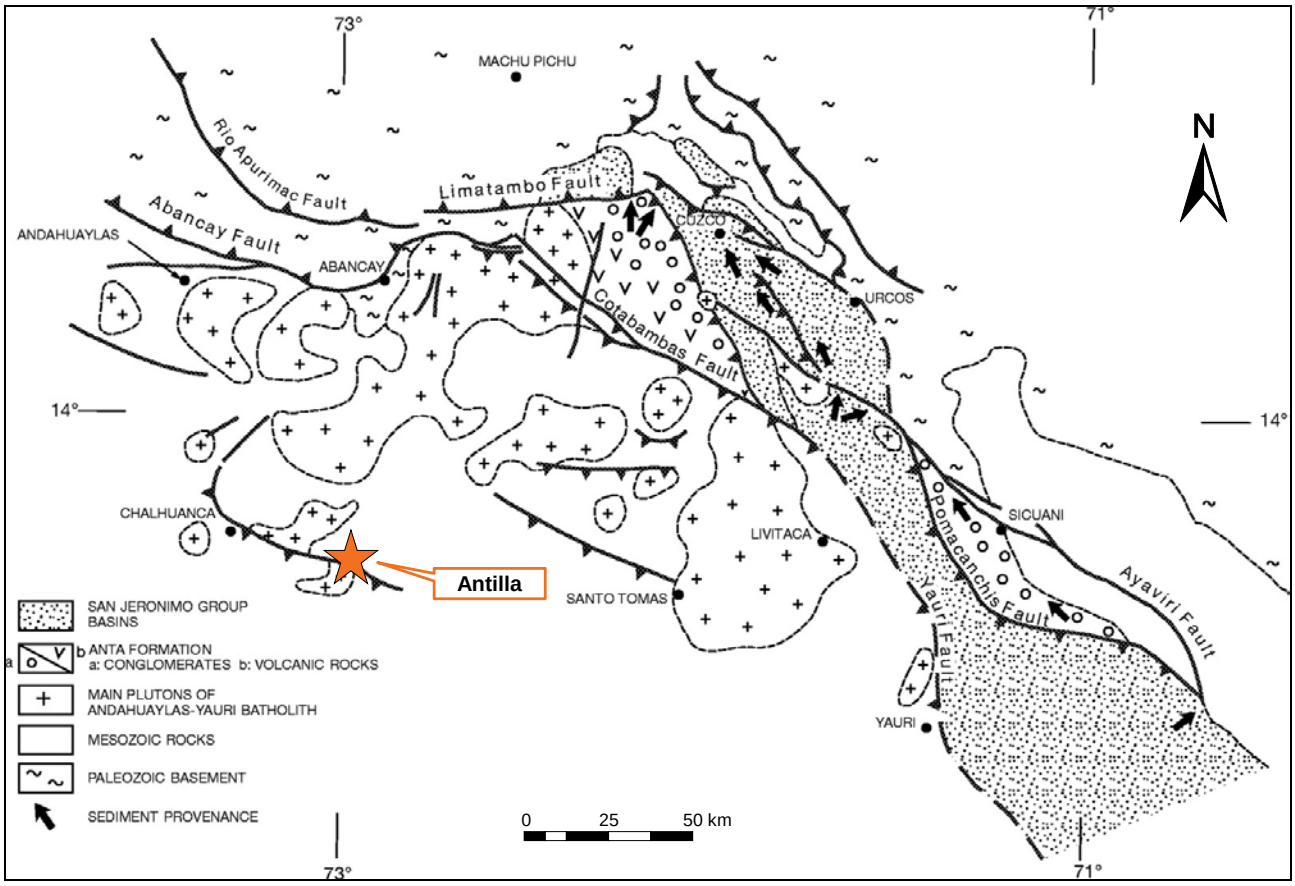


Figure 4: Regional Structural Geology of the Andahuaylas-Yauri Belt
(modified from Perelló et al., 2003)

The basement is overlain by Mesozoic and Cenozoic sedimentary rock deposited in the Eastern and 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 Lagunilla and Yura Groups, made up of middle to late Jurassic quartz-arenite, 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 arenite, quartz arenite, and quartzite, which hosts the Antilla sulphide deposit.

Eocene and Oligocene stratigraphy is dominated by the sedimentary San Jerónimo Group and the dominantly volcanic 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 5).

6.2 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 6). 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.

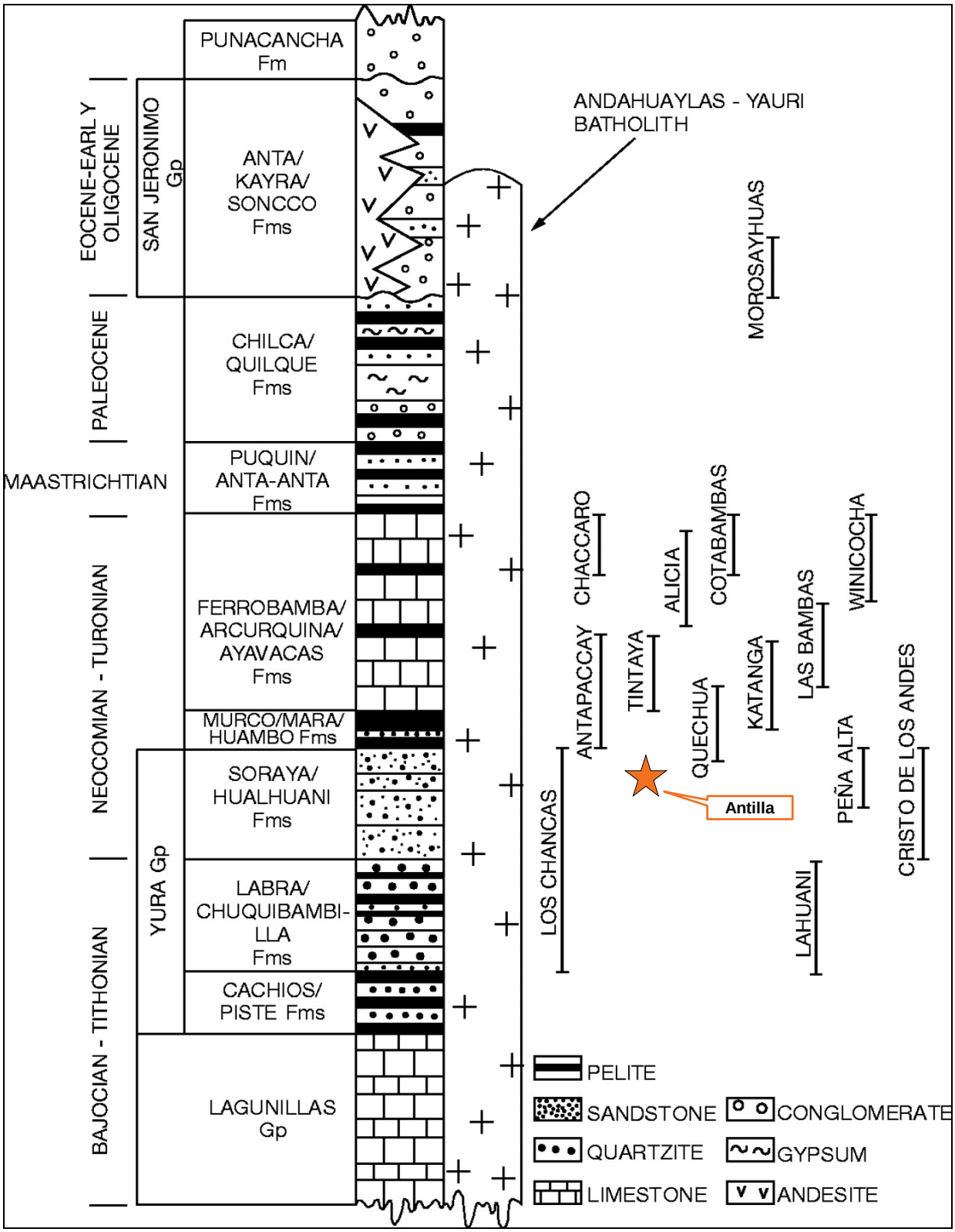


Figure 5: Regional Stratigraphic Position of the Antilla Deposit
(modified from Perelló et al., 2003)

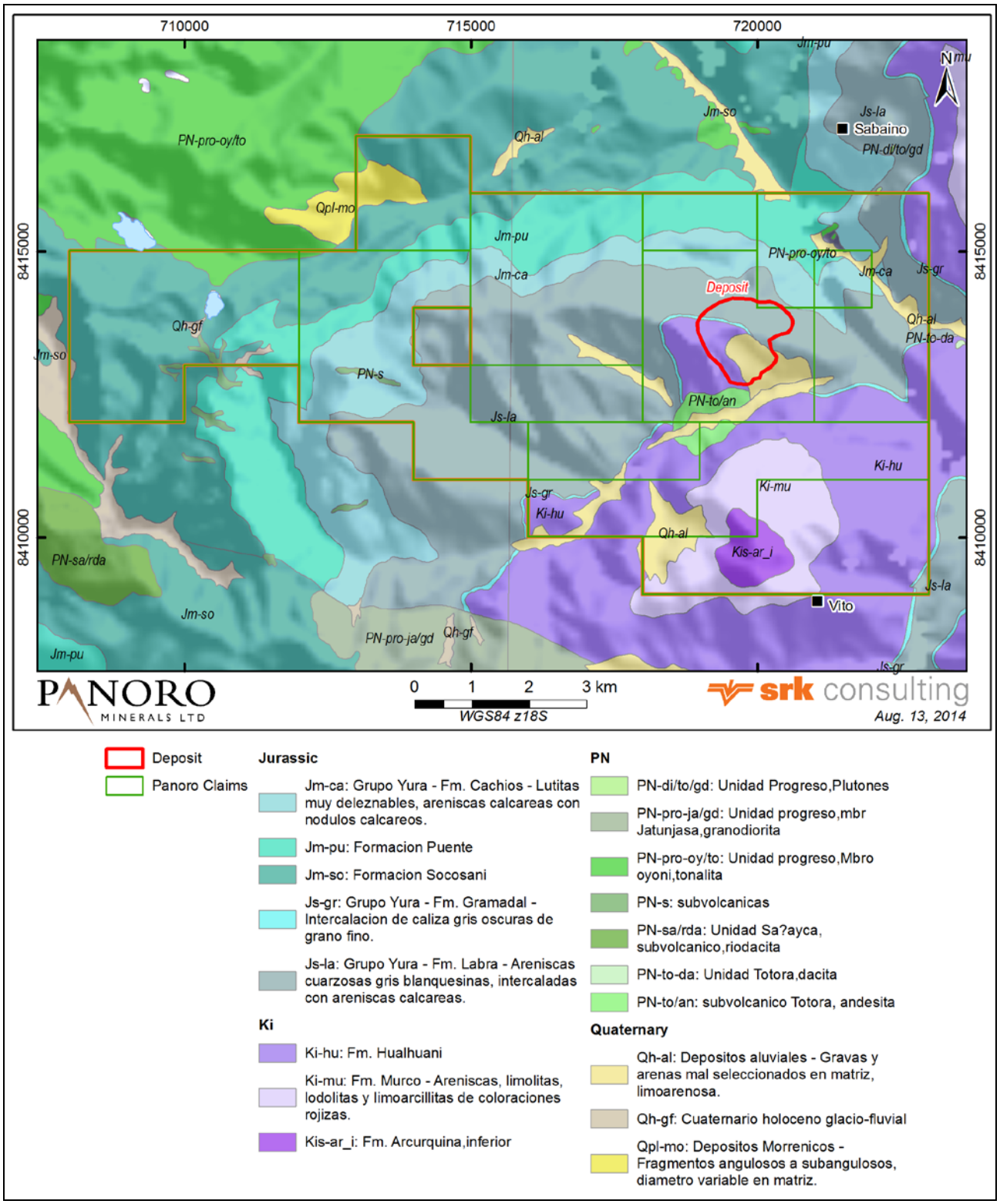


Figure 6: Local Geology Setting

The Late Porphyry is fine grained, with fine- to medium-grained porphyroblasts and a dark grey glassy groundmass (Figure 7). Plagioclase porphyroblasts constitute approximately 25% of the volume of 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 un-mineralized monzonitic sills are exposed to the north east of the mineralized zone.

6.3 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 5). 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 north-northeast-striking normal faults with dextral offsets have been interpreted from outcrop mapping (Figure 7 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.

6.4 Mineralization

The most important mineralization encountered to date on the property is 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. 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.

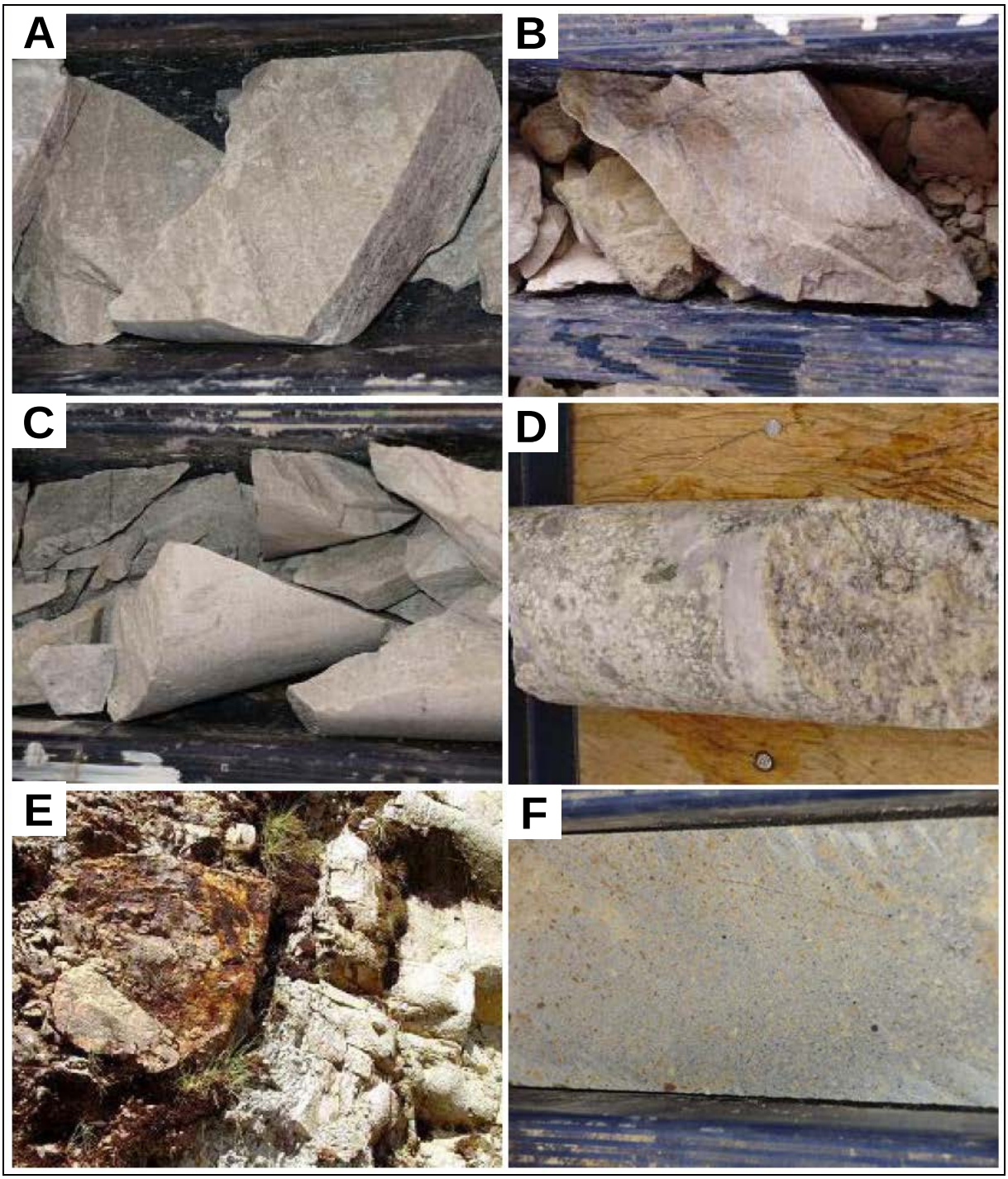


Figure 7: Main Rock Types at Antilla

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

6.4.1 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-centimetre width quartz veinlets
- Occasionally as disseminated grains or coating disseminated grains of primary chalcopyrite in zones of more intense fracturing and silicification (Figure 8)

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.

6.4.2 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.

6.4.3 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 9).

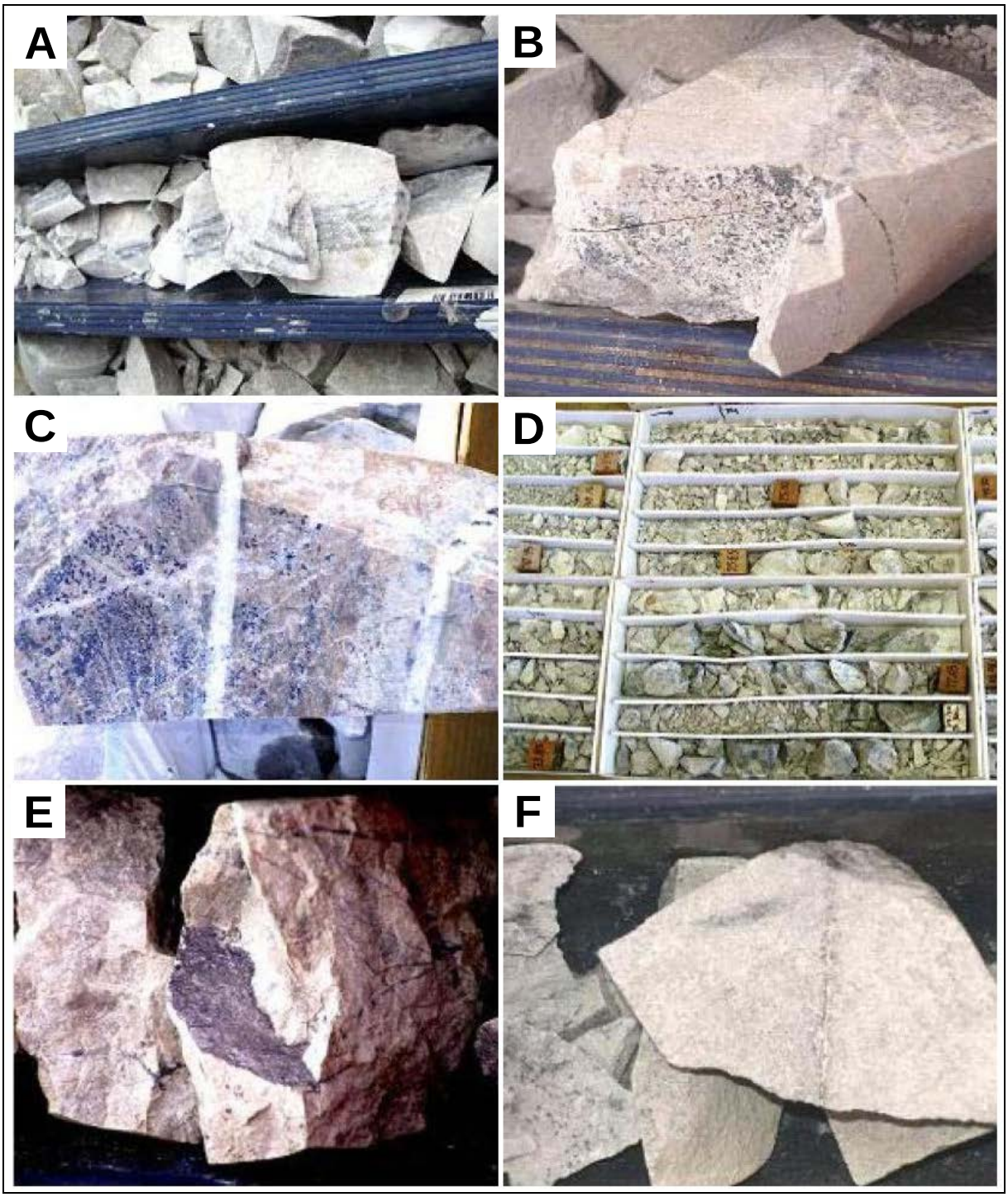


Figure 8: Mineralization at Antilla

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

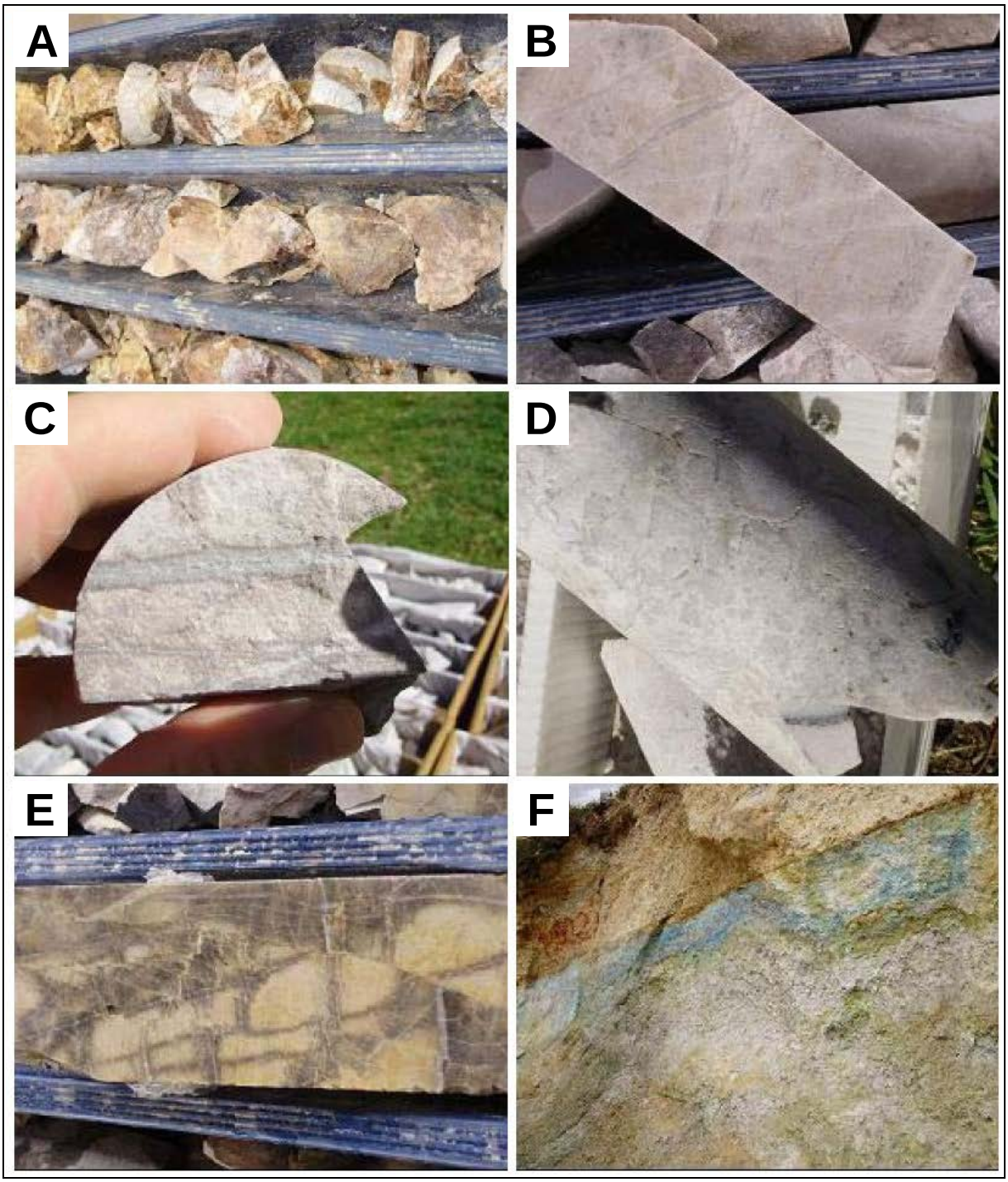


Figure 9: Hydrothermal Alteration

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

6.4.4 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 4). 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.

6.4.5 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 4.

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 4: Mineralization Domains

Zone	Name	Alteration	Cu Grades	Mo Grades	Characteristics
1	Primary Sulphide	Silicification, biotitization, 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	Silification, sericitization, 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

7 Deposit Types

The mineralization identified to date on the property is consistent with a supergene enrichment blanket 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 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 11).

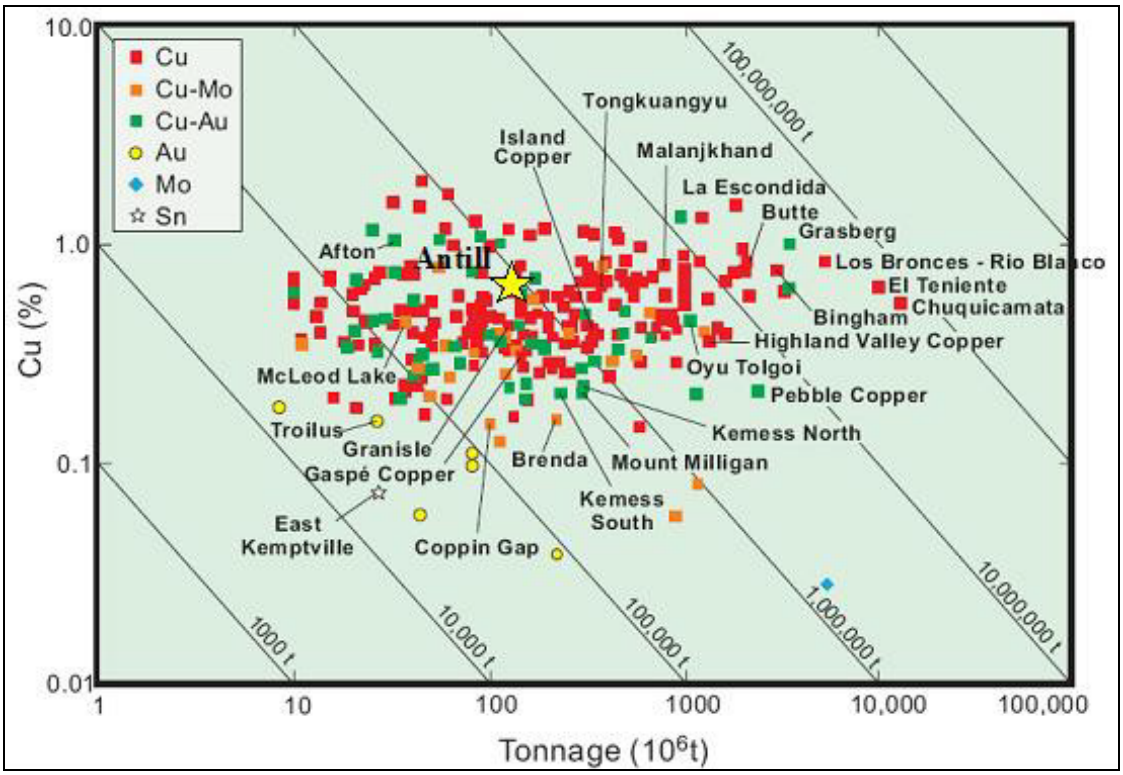


Figure 10: Grade-Tonnage Profile of Selected Porphyry Copper Deposits (Sinclair 2007)

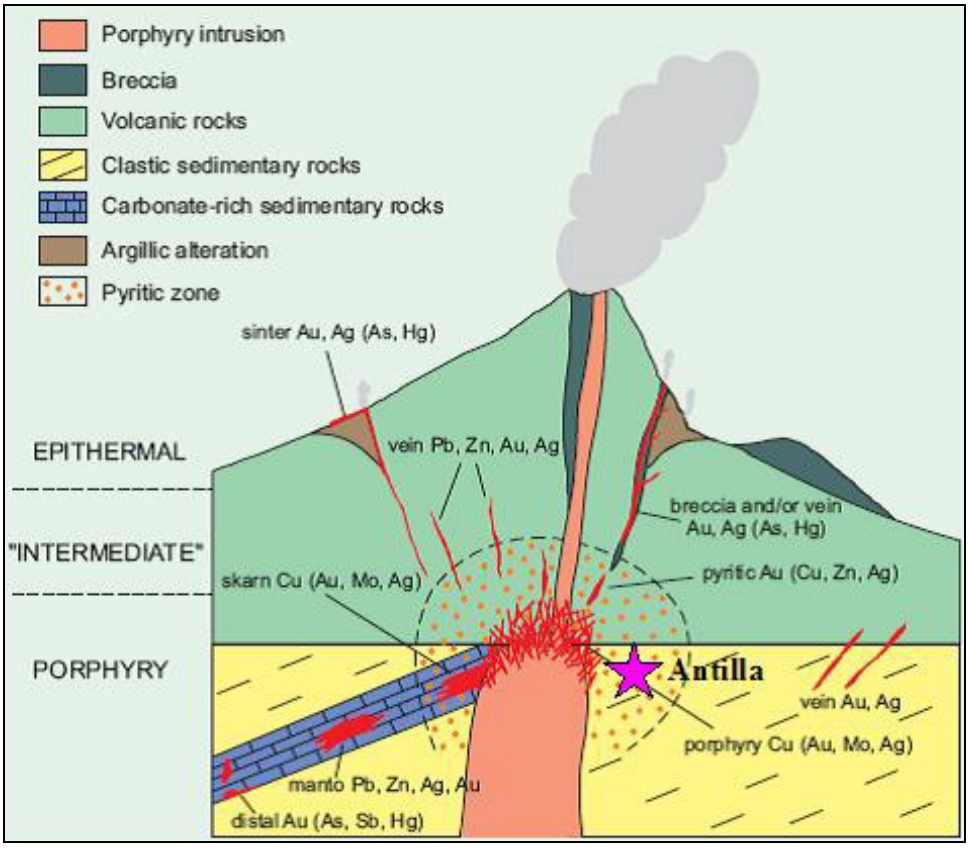


Figure 11: Geological Environment of Porphyry Copper Deposits (Wright 2009)

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.

8 Exploration

In September 2006, Panoro requested John Fox of Laurion Consulting Ltd., Vancouver, Canada, to undertake a review of assay data from the Cordillera de las Minas S.A. (Cordillera) exploration programs, and consider process options for the property. The potential amenability of the mineralization to acid leach and flotation was reviewed and some initial operating costs, capital costs, and smelter returns were discussed for the concentrator and heap leach scenarios.

In 2008, Eagle Mapping Peru S.A.C. was contracted to prepare a topographic map for Cordillera from a series of 1:45,000 scale ortho-photos from the Carta Nacional (Peru). A digital elevation surface with 1-metre resolution was created from the data.

In 2013, Panoro commissioned Seggistem S.R.L. of Trujillo, Peru 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 the east area of the property and subsequently was used for the mineral resource estimation, geologic mapping, geochemical survey, and infrastructure design for this study.

8.1 Geological Mapping

Geological mapping at 1:5,000 scale has been concentrated on the central and eastern 4,000 hectares of the property. Mapping was completed by Cordillera between 2002 and 2004 and was updated in 2008 by Panoro. Additional mapping was completed in 2013 and 2014. Outcrop is reasonably good and exposures of the porphyritic intrusions and the Soraya Group sedimentary rock are common. 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 early 2014 Panoro commissioned Seggistem S.R.L. of Trujillo, Peru to carry out a ground-based topographic survey of the Antilla project area using differential GPS receivers with a base station for post processing of the data.

8.2 Geochemistry and Mineralogy

In 2002 and 2003, systematic rock and soil sampling was carried out on a 100 by 50 metres grid across the western part of the property. A total of 2,772 samples were taken, including 1,130 rock samples and 1,727 soil samples. The results of the soil and rock samples were not promising and failed to generate additional targets for testing.

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).

Between 2013 and 2015 Panoro carried out systematic rock chip 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 the economic potential of the project area. A summary of the samples collected by Panoro is presented in Table 5. Results of the sampling led to the identification of six anomalies (Figure 12).

Table 5: Summary of Panoro Sampling Since 2018

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

8.3 Geophysics

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

8.4 Other Exploration Targets 2013 and 2014

In mid-2013, Panoro restarted their exploration activities primarily on the western portion of the property. Exploration included geological mapping and soil and rock geochemical sampling (Figure 12). The exploration focused on Antilla West and Piste, situated to the west of the known Antilla deposit on the Aluno Cuatro 2002 and Valeria Dieciseis 2002 concessions, respectively.

The Antilla West Block, Chabuca, and Piste exploration targets are not subject to this report and have been included here for completeness only.

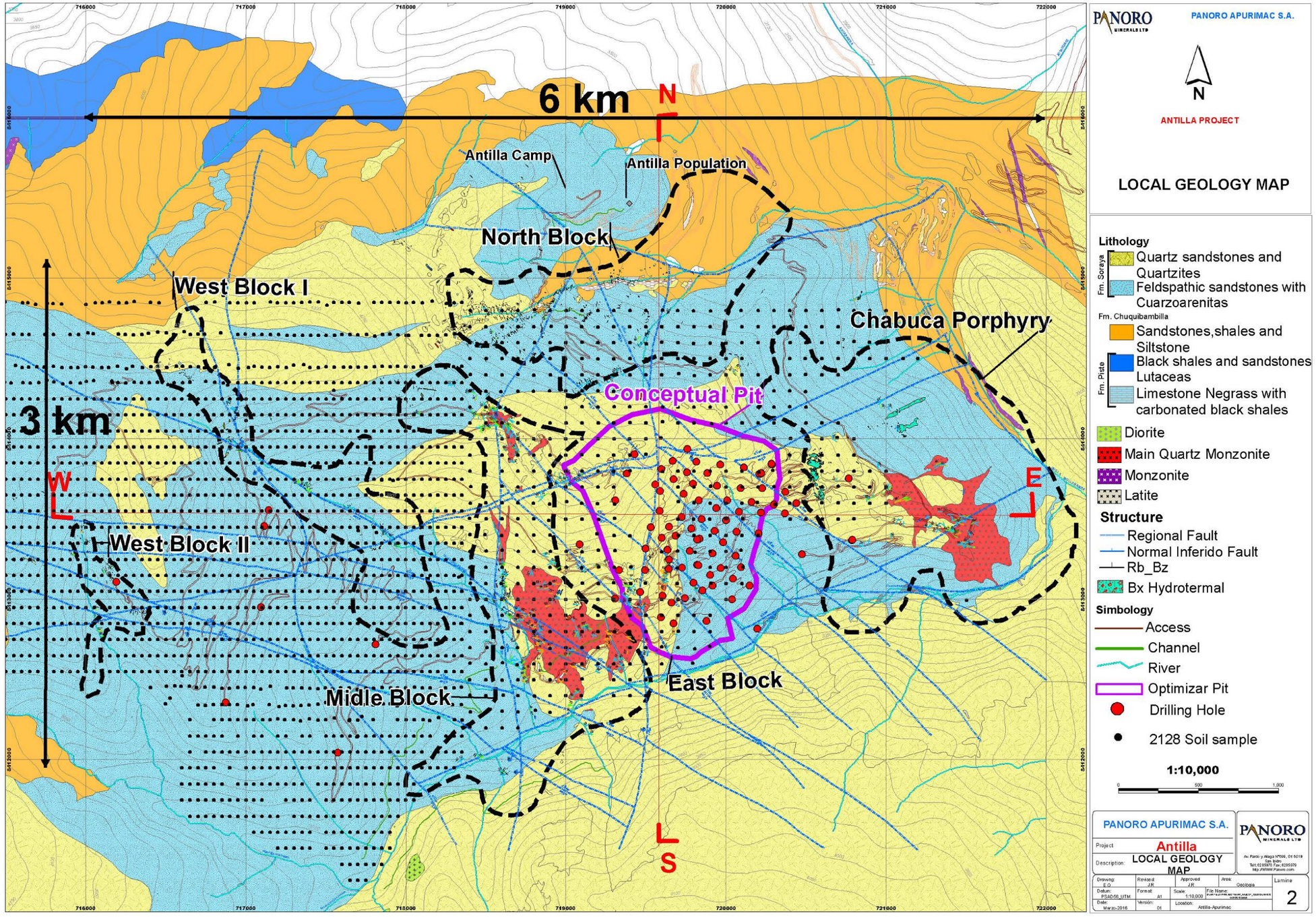


Figure 12: Exploration Targets and Extent of 2013-2014 Soil Sampling

9 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 metres) had been drilled on the property (Table 6). Between 2003 and 2010 a total of five drilling programs were completed on the property. Borehole locations are shown in Figure 13.

Table 6: Summary of Drilling Programs on the Property

Year	No. Holes	Metres	Targets
2003	12	1,983.1	Reconnaissance of main mineralized zone, holes collared 500 m apart
2004	12	1,378.9	Reconnaissance of Punkuccasa, Carachara, Cayarani, and Hualhuani areas, 3 km west of the main mineralized zone
2005	4	650.1	Reconnaissance holes at the edge of the zone defined in 2003
2008	49	9,130.6	Drilling on 100 m centres to define mineralization in the main zone
2010	19	2,242.8	Infill drilling on the Antilla deposit
Total	**96**	**15,385.5**	

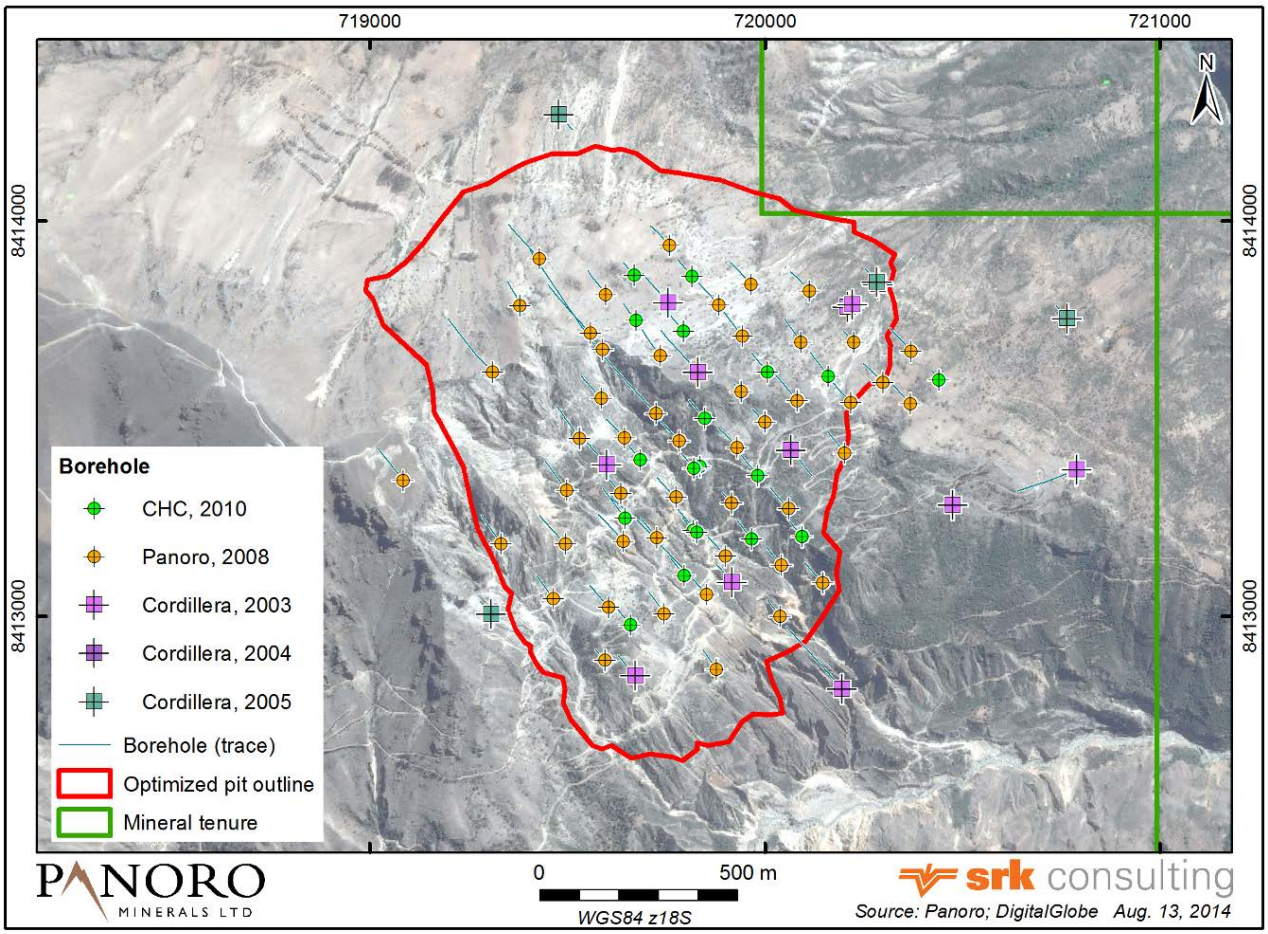


Figure 13: Map Showing the Distribution of Drilling in Relation to Conceptual Pit Outline

Only the drilling programs 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 in Table 7.

Table 7: Selected Borehole Sulphide Mineralization Intersections

Borehole	From (m)	To (m)	Length* (m)	Cu (%)	Mo (%)	Au (ppm)	Ag (ppm)	Mineralization Type
Cordillera Campaigns								
ANT-01-03	2.00	54.00	52.00	0.79	0.003	0.01	1.00	Secondary Sulphide
ANT-05-03	18.00	84.00	66.00	0.67	0.008	0.01	0.90	Secondary Sulphide
ANT-06-03	10.00	76.00	66.00	0.89	0.014	0.01	0.70	Secondary Sulphide
ANT-06-03	150.00	165.00	15.00	0.26	0.027	0.01	0.40	Primary Sulphide
ANT-07-03	18.00	98.00	80.00	0.68	0.008	0.01	0.90	Secondary Sulphide
ANT-09E-05	139.00	166.00	30.00	0.72	0.001	0.01	0.80	Secondary Sulphide
ANT-10-03	20.00	48.00	28.00	0.39	0.023	0.00	0.90	Primary Sulphide
Panoro 2008 Campaign								
ANT-20-08	60.00	94.00	34.00	0.75	0.007	0.01	0.60	Secondary Sulphide
ANT-22-08	56.00	98.00	42.00	0.80	0.005	0.01	1.20	Secondary Sulphide
ANT-24-08	18.00	74.00	56.00	0.70	0.017	0.01	0.60	Secondary Sulphide
ANT-25-08	26.00	64.00	38.00	0.56	0.004	0.01	0.90	Secondary Sulphide
ANT-26-08	6.00	68.00	62.00	0.63	0.012	0.01	0.70	Secondary Sulphide
ANT-28-08	26.00	86.00	60.00	0.85	0.023	0.01	3.20	Secondary Sulphide
ANT-30-08	32.00	96.00	64.00	0.75	0.002	0.01	0.50	Secondary Sulphide
ANT-34-08	60.00	82.00	22.00	0.51	0.039	0.01	0.50	Secondary Sulphide
ANT-37-08	62.00	136.00	74.00	0.54	0.001	0.01	0.50	Secondary Sulphide
ANT-38A-08	37.00	67.00	30.00	0.75	0.029	0.01	1.10	Primary Sulphide
ANT-38C-08	34.00	130.00	96.00	0.72	0.029	0.01	0.80	Secondary Sulphide
ANT-39-08	6.00	82.00	76.00	0.57	0.002	0.01	0.90	Secondary Sulphide
ANT-41-08	58.00	149.00	91.00	0.56	0.013	0.01	0.50	Secondary Sulphide
ANT-43-08	44.00	62.00	18.00	0.67	0.011	0.01	1.10	Secondary Sulphide
ANT-46-08	76.00	102.00	26.00	0.63	0.006	0.01	0.50	Secondary Sulphide
ANT-49-08	28.00	70.00	42.00	0.93	0.017	0.01	0.80	Secondary Sulphide
ANT-51-80	66.00	99.00	33.00	0.49	0.016	0.01	0.60	Secondary Sulphide
ANT-61-08	196.00	226.00	30.00	0.65	0.009	0.01	1.00	Secondary Sulphide
ANT-62-08	60.00	102.00	42.00	0.52	0.012	0.01	1.00	Secondary Sulphide
ANT-65-08	46.00	90.00	44.00	0.51	0.006	0.01	1.10	Secondary Sulphide
ANT-66-08	9.00	64.00	55.00	0.42	0.033	0.01	0.50	Secondary Sulphide

* Reported core length intervals

9.1 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.

9.2 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 receivers 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 metres in length. However, boreholes ANT-62-08 and ANT-66-08 were drilled to just over 750 metre 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.

9.3 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 metres). Borehole lengths varied from 25 to 169 metres with an average length of 116 metres. The aim of the program was to confirm high grade intersections encountered by previous drilling and to reduce the borehole spacing to 100 metres. 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.

9.4 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 8). 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 8: 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	100.0	42.8
Late Porphyry	95.3	42.1
Total	**93.2**	**36.0**

9.5 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, brecciation 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.

9.6 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.

10 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. SRK 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. SRK 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). SRK 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.

10.1 Sample Preparation and Analyses

10.1.1 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.

10.1.2 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 millimetres. 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).

10.1.3 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.

10.1.4 Sampling by Panoro (2013 to 2015)

For the 2013 to 2015 soil and rock chip sampling, Panora maintained a chain-of-custody of all samples 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).

10.2 Specific Gravity Data

Between 2003 and 2005, Cordillera measured specific gravity on core. Two consecutive measurements were taken from each 2-metre core interval. SRK 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 9).

Table 9: Summary of Specific Gravity Determinations

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	2.40	2.44	2.38
Late Dike	8	2.41	2.53	2.28
Total	**262**	**2.43**	**2.83**	**2.07**

10.3 Quality Assurance and Quality Control Programs

10.3.1 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.

10.3.2 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.

10.3.3 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.

10.4 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.

11 Data Verification

11.1 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 reassay. Panoro inserted standard reference material samples with this sample stream. According to Panoro, results compared well with the original assay results.

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

11.2 Verifications by Tetra Tech

Upon receipt of Panoro's 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.

11.2.1 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.

11.2.2 Lithology Data

Lithological data were provided in CSV file format. The lithology of the 23 boreholes chosen for review represent 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, relogging, and consolidation of lithological data. Once these changes were considered, 100% of the data verified matched the originals.

11.2.3 Assay 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.

11.2.4 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.

11.2.5 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 Weatherhaven 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 14 A and B illustrate the core logging and storage facility.

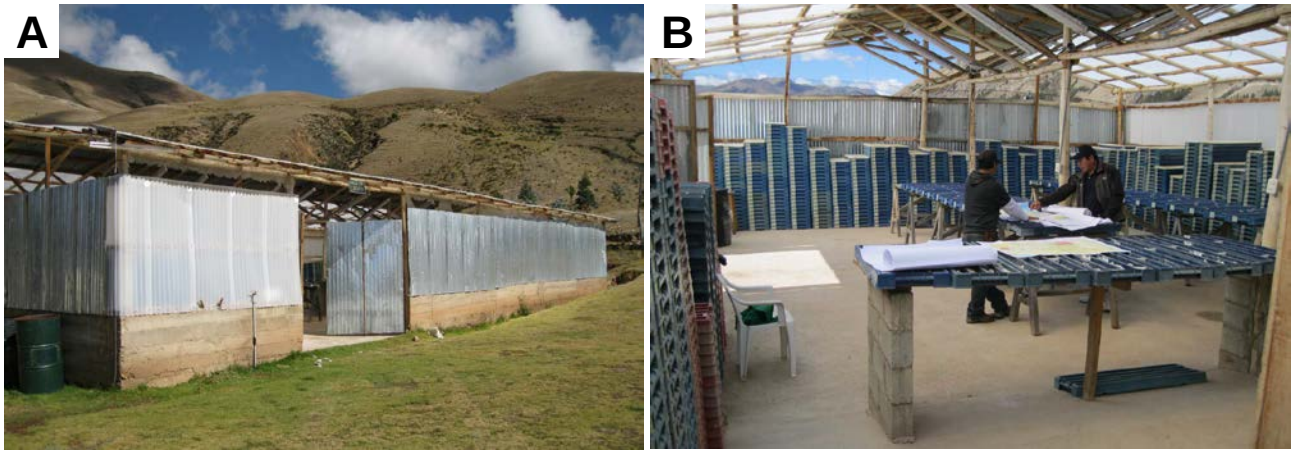


Figure 14: Project Site Facilities

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.

11.2.6 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 15 displays the blank performance.

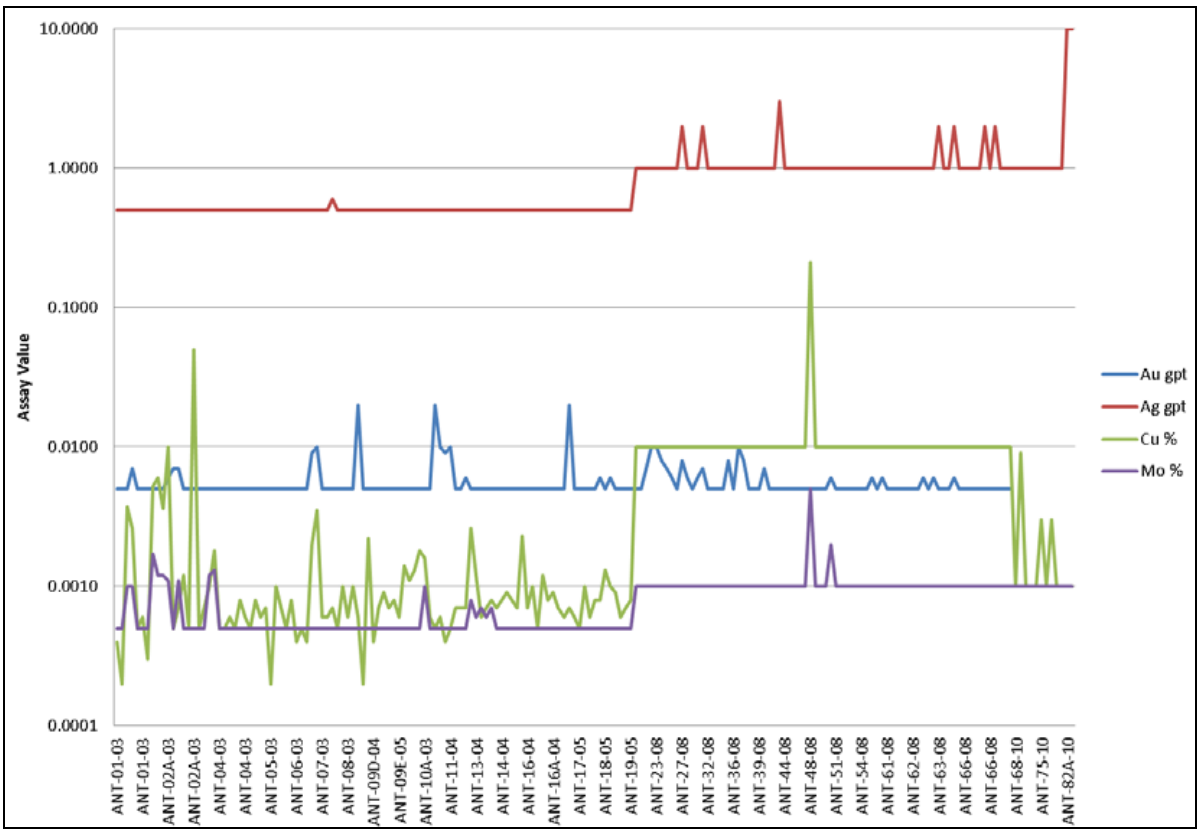


Figure 15: Blank Performance for Various Metals

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 16 through Figure 18 display the duplicate control graphs.

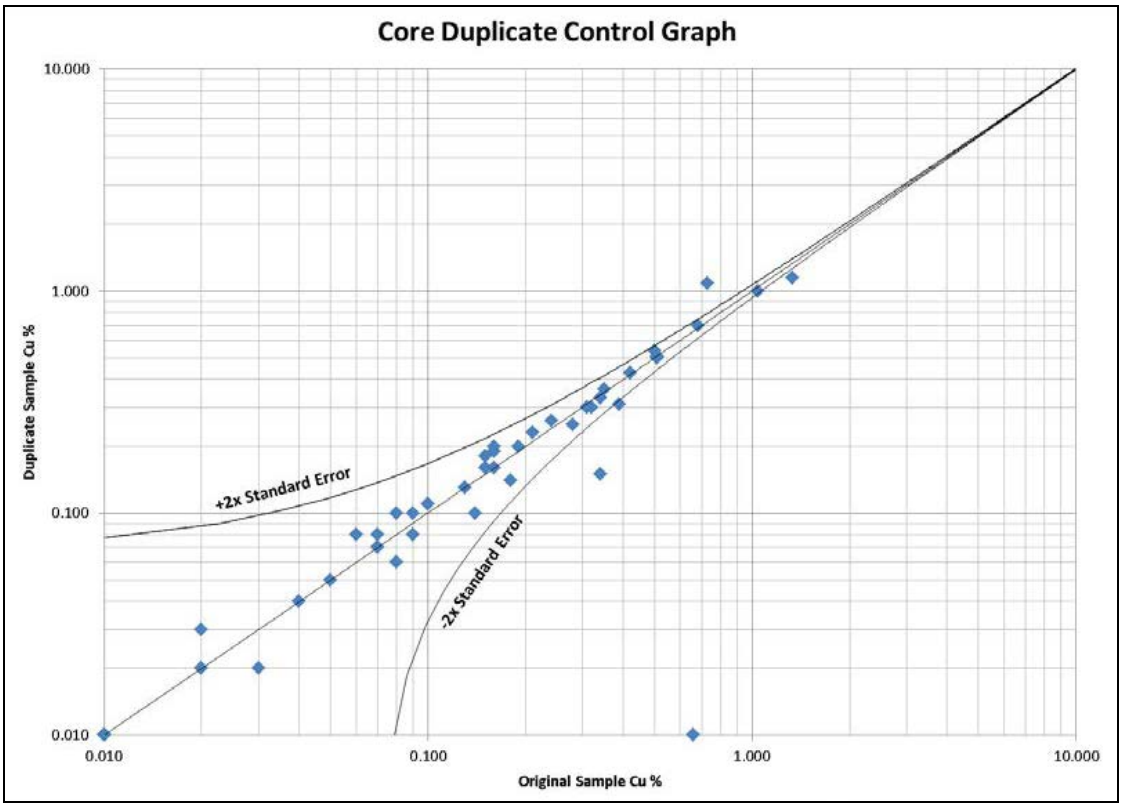


Figure 16: Performance of Core Duplicate Samples

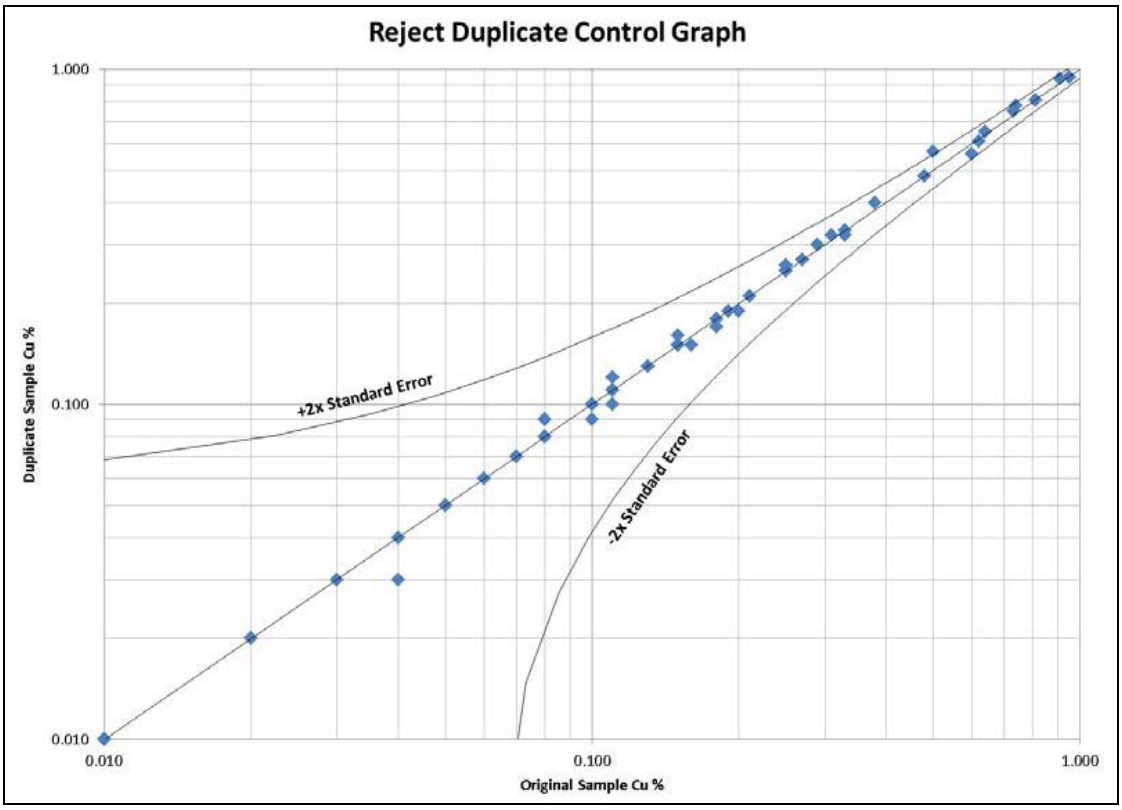


Figure 17: Performance of Reject Duplicate Samples

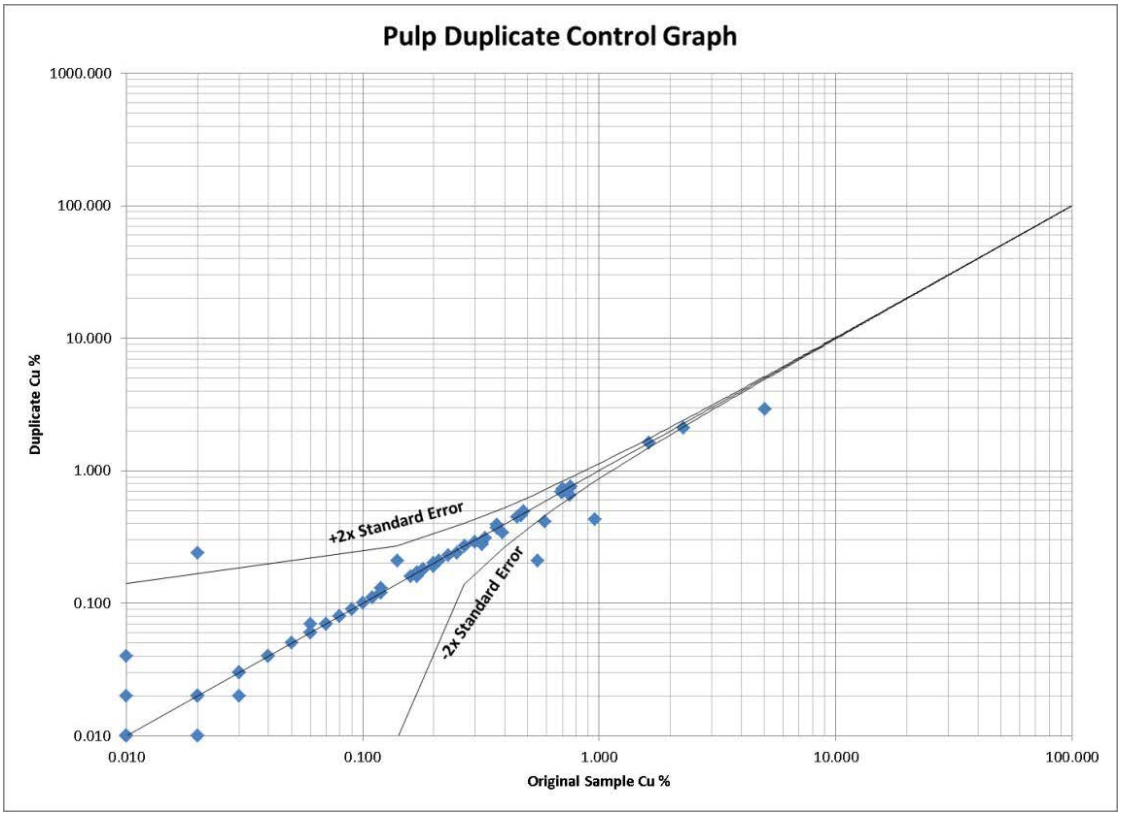


Figure 18: Performance of Pulp Duplicate Samples

11.2.7 Independent Verification Sampling

Independent verification samples were collected during the site visit by Tetra Tech. Three samples were collected from the available archived core.

The verification sample intervals were selected randomly from the same sample intervals sampled by Panoro. As no core saw was available, Tetra Tech selected alternating pieces of half core. The samples were collected, placed in labeled sample bags and sealed. Sample tags were inserted in the core box and in the sample bag. The samples were kept with the author at all times during the site visit. Upon returning to Toronto, the author shipped the samples to Activation Laboratories Ltd. (Actlabs) for analysis.

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

The purpose of the verification samples was to confirm indications of mineralization and they were not intended to replicate the results obtained by Panoro for these intervals. Assay results for the Tetra Tech samples compare well with the original Panoro assay results as shown in Table 10 and Table 11.

Table 10: Comparative Summary of Verification Samples Collected by Tetra Tech

Tetra Tech Sample No.	Panoro Sample No.	Borehole	Sample Interval (m)	Mineralization
626478	66456	ANT-01	15.4 – 16.0	Supergene
626479	66492	ANT-01	80.0 – 82.0	Primary Sulphides
626480	69580	ANT-05	56.0 – 58.0	Supergene

Table 11: Summary of Verification Sample Results Collected by Tetra Tech

	Borehole	Cu%	Mo%	Au (g/t)	Ag (g/t)
Tetra Tech Samples					
626478	ANT-01	0.303	0.003	0.004	<3
626479	ANT-01	0.188	<0.003	<0.001	<3
626480	ANT-05	0.707	0.004	0.007	<3
Panoro Samples					
66457	ANT-01	0.390	0.001	0.008	0.5
66492	ANT-01	0.446	0.001	0.005	0.6
69580	ANT-05	0.907	0.008	0.013	0.7
Difference					
	-	-0.087	0.002	-0.004	-
	-	-0.258	-	-	-
	-	-0.200	-0.004	-0.006	-

12 Mineral Processing and Metallurgical Testing

A summary of the metallurgical testwork programs and results was included in the technical report prepared by Tetra Tech in 2013 (Tetra Tech 2013). No additional metallurgical testwork has been completed since.

A preliminary assessment of flotation and heap leaching was carried out by Laurion Consulting in 2005/2006 and is briefly summarized in Fox (2006). Subsequently, Panoro completed metallurgical testwork on a single sample in 2011, conducted by Bureau Veritas Inspectorate of Vancouver, Canada, and in 2013 on two composite samples of the main mineralization domains conducted by Certimin S.A. (Centimin) of Lima, Peru.

12.1 Inspectorate (2011)

No formal report exist for the test work completed by Inspectorate, only the laboratory test result sheets are available. A single sample referred to as master composite was tested, of unknown origin and 0.44% copper grade. It is not known if this sample is representative of the sulphide mineralization from the Antilla project. No molybdenum assays were quoted in the testwork results.

A range of flotation tests were completed at different grind sizes, with/without re-grinding of the rougher concentrate and with different reagent additions, including sodium cyanide for pyrite depression.

As the mineralization domain was not reported, these results are not included in the predictions of metallurgical performance in Section 16.

12.2 Certimin (2013)

Two composite samples were tested by Certimin from HQ diameter core sample rejects. Sample A was of Primary Sulphide and Sample B was of Supergene. The samples are representative of two of the four main mineralization domains fond at Antilla. The testwork conditions and results are reported in Certimin (2013).

The samples were composited from seven boreholes shown in Figure 19 as black circle icons, all from the East Block zone and within the pit shell limit identified by Tetra Tech in their 2013 report (magenta line in Figure 19). Of the seven boreholes sampled, two were from the 2003 program, four from the 2008 program, and one from the 2010 program. The intervals sampled were from 0 to 172 metres down each borehole.

No comments were included in the Tetra Tech summary on how the samples were collected or the condition of the core, in particular, the 2003 intervals which were ten years of age when they were sampled.

Table 12 summarizes the two sample head grades including the copper speciation assays.

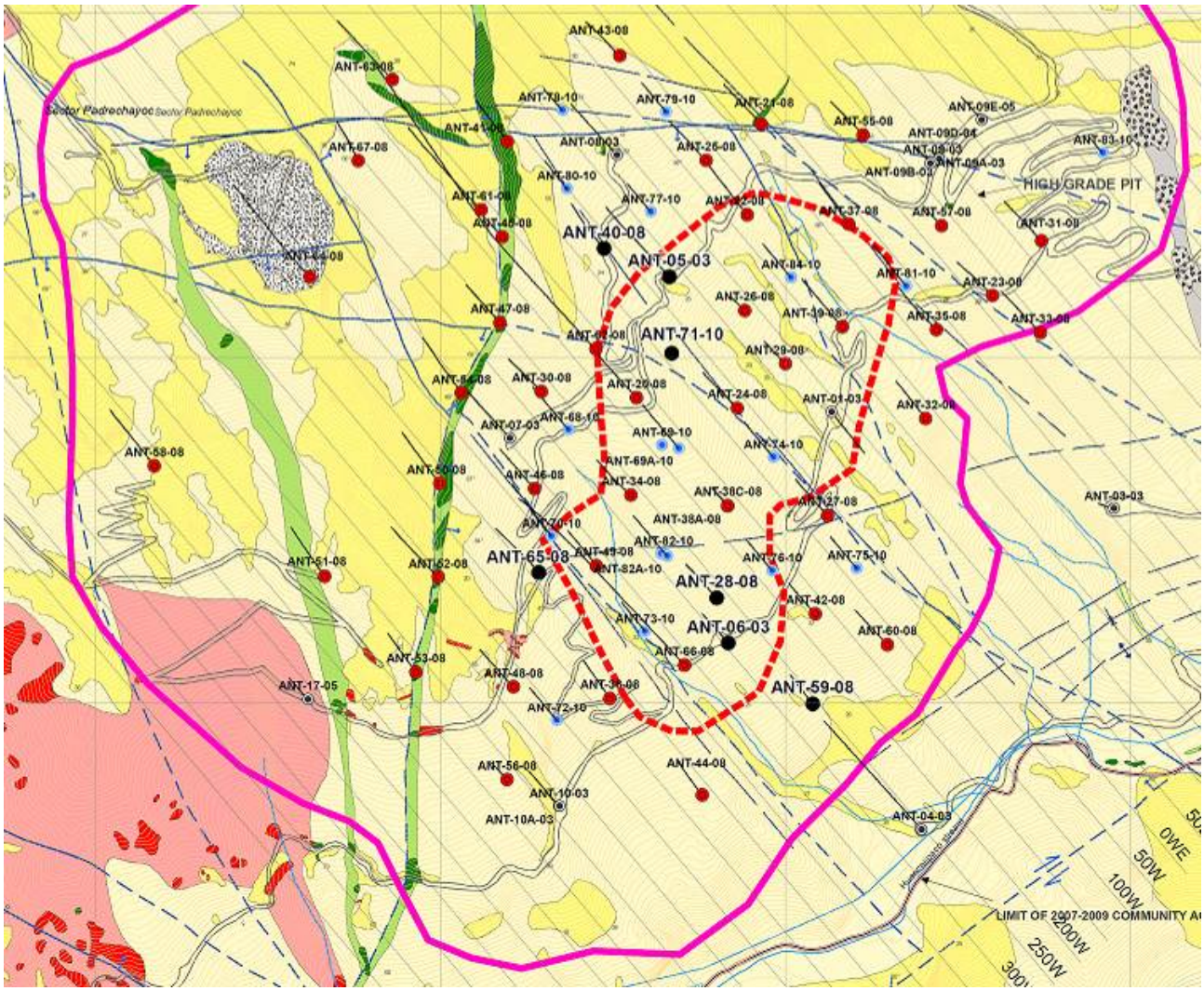


Figure 19: Locations for Metallurgical Borehole Composite Samples (black circles)

Table 12: Summary Certimin 2013 Composite Sample Head Grades

Sample	Zone	Cu (%)	CuCN- (%)	CuRes (%)	CuSol H+ (%)	Mo (ppm)	Ag (g/t)	Au (g/t)	Fe (%)	Stotal (%)
Sample A	Primary Sulphide	0.29	0.05	0.21	0.03	115	0.60	0.02	1.08	0.73
Sample B	Supergene	0.56	0.32	0.10	0.14	97	0.97	<0.01	0.98	0.73

Note: CuCN- = cyanide-soluble copper; CuRes = residual copper; CuSolH+ = acid-soluble copper

The Supergene and Primary Sulphide mineralization domains represent most of the Antilla deposit. In 2013, Tetra Tech stated that 70% of the Indicated mineral resource tonnage is within the Supergene domain and 23% of the tonnage is within the Primary Sulphide domain. The remaining 7% are in the Overburden/Cover and Leach Cap domains.

The average copper grade of the 2013 Indicated mineral resources was 0.45% and 0.30% for Supergene and Primary Sulphide, respectively. Therefore, compared to the deposit average grades, Sample A was very similar, while Sample B was higher in copper.

In the conceptual production plan discussed herein the majority of mill feed for the first 10 years would come from Supergene mineralization, with a transition to Primary Sulphide material until the end of the conceptual life of mine when stockpiled Leach Cap material would be finally processed.

The conceptual production plan shows that the head grade would fall below 0.3% copper while processing Primary feed and as low as 0.15% copper for the last few years. Lower copper grades have not been evaluated in the metallurgical testwork program to date.

The Primary Sulphide domain is characterized by an absence of chalcocite with minor chalcopyrite while Sample A contains a high residual copper assay at 0.21%, or 72% of the copper was present as chalcopyrite. Sample B reported a high cyanide-soluble copper assay as Supergene consists entirely of secondary sulphide minerals.

Beyond the copper speciation, no mineralogical analysis was undertaken by Certimin to quantify the copper minerals present, their relative levels of liberation and association with pyrite and molybdenite.

In summary, the Supergene sample tested by Certimin was higher in copper grade than the deposit average but was similar in mineralogy. Sample A had a high residual copper assay, indicative of chalcopyrite but was similar in grade to the deposit as a whole.

12.2.1 Comminution Testwork

Bond Ball Mill Work Index (BWi) tests were conducted at a closing screen size of 150 micrometres (µm) on both Sample A and Sample B. With a product size of around 120 µm, Sample A reported a BWi of 10.4 kilowatt-hours per tonne (kWh/t) while Sample B reported 8.9 kWh/t.

Both of these results are considered relatively soft and agree with the geotechnical summary in the Tetra Tech report that showed estimated uniaxial compressive strength (UCS) values of only 32 to 43 megapascal (MPa).

Additional comminution testing is warranted but early indications are that the two principal mineralization domains are relatively soft, with low crushing and grinding power requirements.

12.2.2 Copper-Molybdenum Bulk Flotation

The two samples were tested for flotation response under bench scale, open circuit as well as locked cycle tests for the optimal flowsheet conditions. Primary grind was targeted as an 80% passing (P80) size of 100 µm.

Two flowsheets were tested for locked cycle testwork – one involving a single tailings stream and the other with two tailings streams. As the two stream option produced lower copper and molybdenum recoveries, only the one stream option results are presented here. For more information on the testwork results, Certimin 2013 provided full details.

Table 13 summarises the locked cycle test results for Sample A (Primary Sulphide) while Table 14 is a summary of the results for Sample B (Supergene).

Table 13: Sample A (Primary Sulphide) Locked Cycle Test 1 (One Tailings Stream)

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	34.30	1.24	85.30	77.60	73.10	46.30
Tailings	0.04	22	0.01	0.50	98.76	14.70	22.40	26.90	53.70
Calculated Head	0.29	97	0.02	0.90	100.00	100.00	100.00	100.00	100.00

The Primary Sulphide sample generated a final concentrate (copper-molybdenum) of 20% copper with 85% copper recovery and 78% molybdenum recovery. Gold recovery to bulk concentrate was also good at 73%. Considering the copper mineralogy was predominantly chalcopyrite (by speciation assay), the results are to be expected.

Table 14: Sample B (Supergene) Locked Cycle Test 1 (One Tailings Stream)

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	37.90	1.25	79.40	83.30	54.20	49.00
Tailings	0.12	17	0.01	0.50	98.75	20.60	16.70	45.80	51.00
Calculated Head	0.57	98	0.01	1.00	100.00	100.00	100.00	100.00	100.00

The Supergene sample results produced a 36% copper bulk concentrate due to secondary copper minerals with an almost 80% copper recovery assisted by the higher copper head grade. Molybdenum recovery was also good at 83%, but only 54% of the gold reported to the final concentrate.

Based on the very low precious metal head grade of these two samples, the final copper concentrate (after molybdenum separation) does not appear to contain payable gold or silver grades.

Included in the Tetra Tech 2013 report was a full analysis of the locked cycle bulk concentrate. Table 15 is an extract from the report, showing a selected set of minor elements from the single tailings flowsheet option. No elements are present at levels which would invoke smelting penalties based on typical copper concentrate contract terms.

Table 15: Concentration* of Selected Minor Elements in the Bulk Concentrate

Element	Units	Primary Sulphide	Supergene
Ag	ppm	33.4	36.8
Al	%	2.74	1.64
As	ppm	366	465
Bi	ppm	<5	<5
Ca	%	0.7	0.57
Ni	ppm	250	410
Pb	ppm	177	125
Sb	ppm	65	36
Zn	ppm	1,200	1,880

* Determined by inductively coupled plasma spectrometry

12.2.3 Copper-Molybdenum Separation

The bulk copper-molybdenum concentrate generated from the open circuit tests was used for preliminary copper-molybdenum separation flotation testing (Table 16).

Table 16: Copper-Molybdenum Separation Test Results

Sample	Test ID	Product	Grade (%)		Recovery (%)	
			Cu	Mo	Cu	Mo
Primary Sulphide	1	Mo Cleaner Concentrate	3.6	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
	2	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
Supergene	3	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
	4	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

The separation process used sodium hydrosulphide (NaHS) to depress the copper minerals and allow the molybdenite (MoS_2) to be recovered with three to four stages of flotation cleaning. The test results showed a 32% molybdenum concentrate for Sample A and a 40% molybdenum concentrate for Sample B.

However, these preliminary results also show a relatively high copper content in the molybdenum concentrate after four cleaning stages. Additional testwork is warranted to investigate the trade-off between molybdenum cleaning and molybdenum losses through additional cleaning stages. In addition, the option of ferric chloride leaching of the final molybdenum concentrate to reduce the copper content should be investigated.

12.2.4 Other Testwork

Tetra Tech (2013) also reported that acid-base accounting (ABA) tests were completed on the flotation tailing samples from the locked cycle testwork. The results indicated that the two samples tested had the potential to be acid generating.

Additional geochemical and ABA testwork is warranted to understand the water treatment requirements and potential for acid generation.

No testwork has been undertaken on samples of the Cover/Overburden or Leach Cap mineralization domains, which are to be included in the mine production plan as 9% of the total mill feed and not in significant quantities until Year 19 of operation and the last year of the current mine plan.

A variability testing program should be undertaken to assess the impact of copper head grade and mineralogy on bulk copper-molybdenum flotation performance, in particular, samples of Primary Sulphide with a head grade of 0.2% copper and lower.

Mineralogical studies of the four mineralization domains are recommended to understand the variability in copper minerals and their possible effect on flotation performance. In particular, the presence of acid-soluble copper minerals, of which some are not recoverable by flotation.

No thickening or filtering testwork has been conducted to date to confirm conventional equipment and flocculant dosage will achieve expected tailing density and concentrate moisture contents.

13 Mineral Resource Estimates

13.1 Introduction

This section discloses a new mineral resource statement for the Antilla copper-molybdenum deposit, prepared by Paul Daigle, PGeo of 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 there region. Tetra Tech considers this CuEq% cut-off to be reasonable for this deposit.

13.2 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 17 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 17: Summary of Boreholes

Company	Year	No. of Boreholes	Total Length (m)
Cordillera	2003-2005	20	2,919.2
Panoro	2008	49	9,130.6
Chancadora	2010	19	2,242.8
Total		**88**	**14,292.6**

13.2.1 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 18 summarizes the specific gravities used for the various lithologies and rock type domains.

Table 18: 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	32	2.69	2.36	2.96	0.13
Primary Sulphides [PS]	400	2.70	13	2.70	2.42	2.80	0.11
Country Rock [CR]	99	2.68	47	2.68	2.36	2.96	0.13

13.3 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 20 to Figure 25 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)

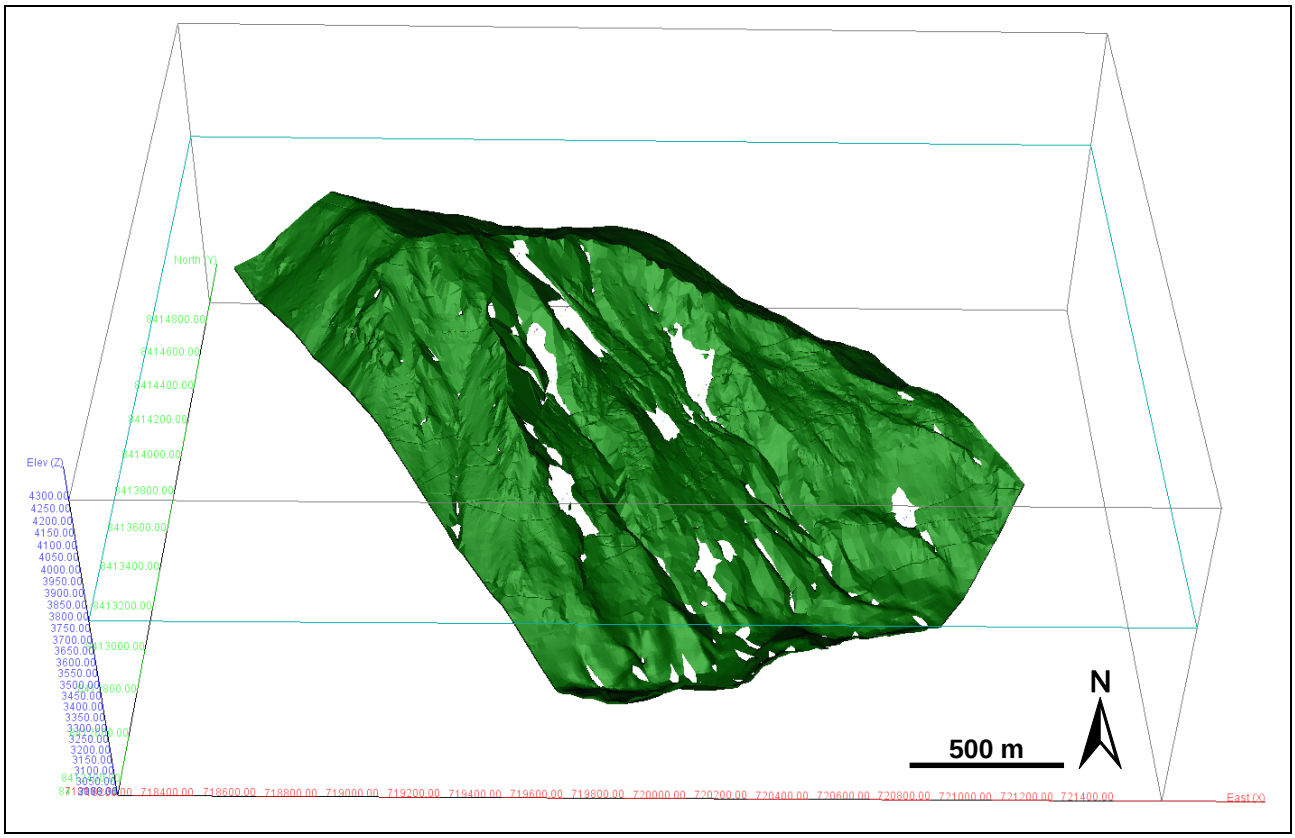


Figure 20: Solid Wireframe for the Covertura (100) Domain

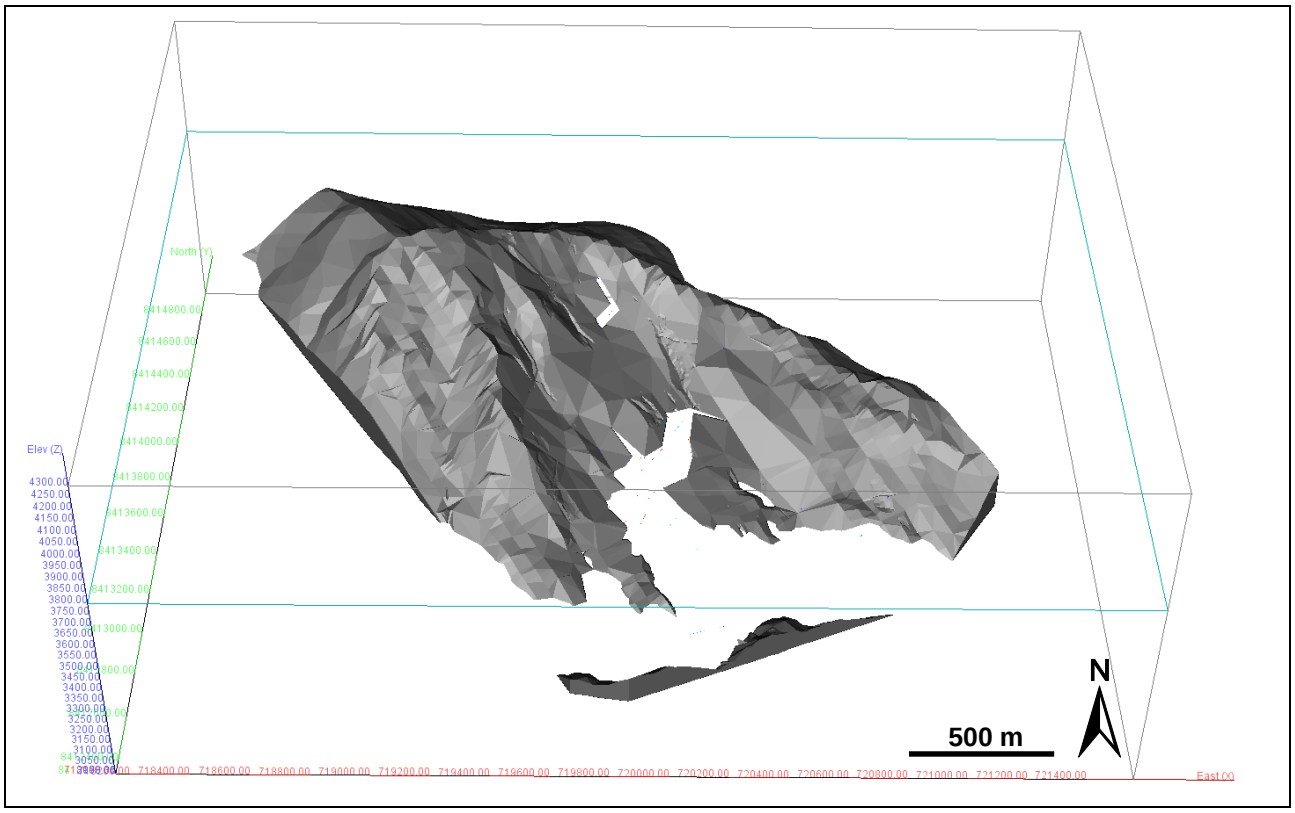


Figure 21: Solid Wireframe for the Leach Cap (200) Domain

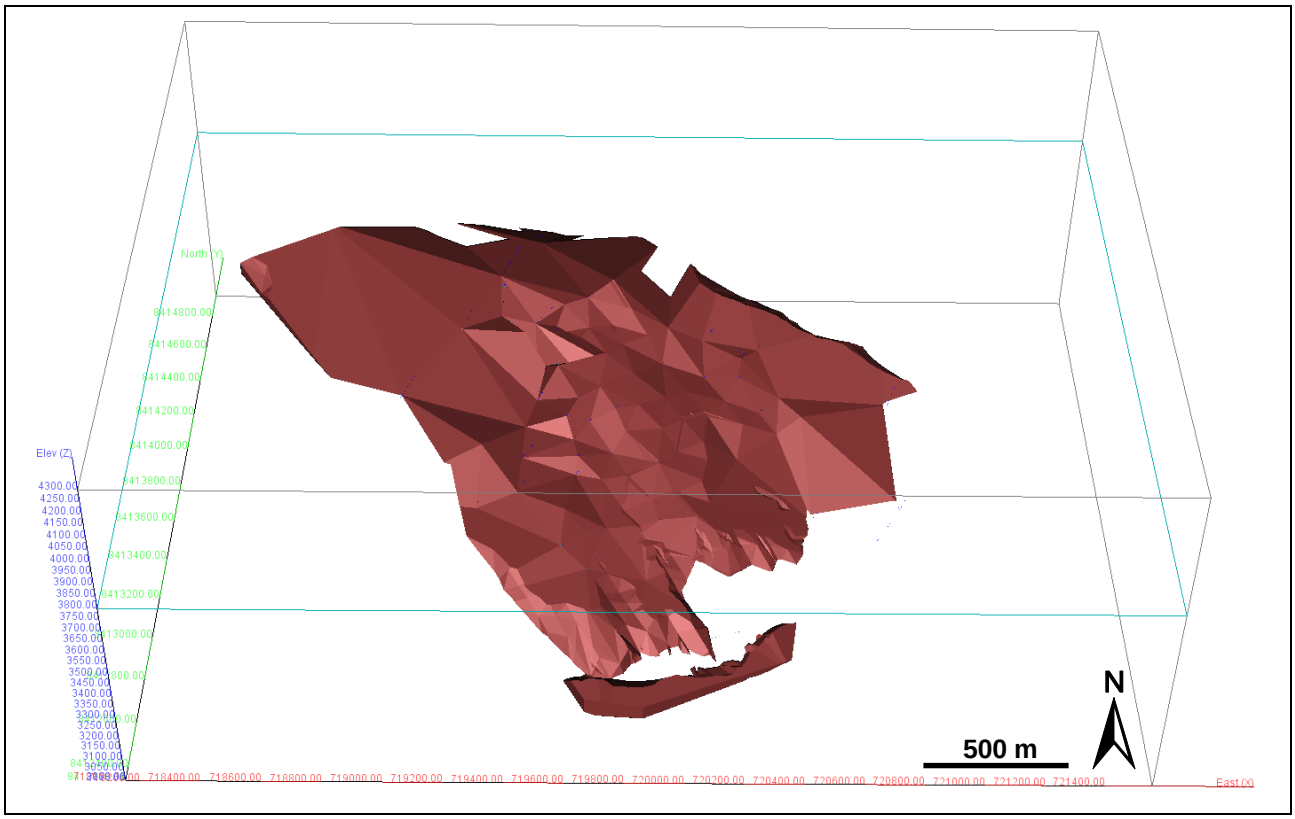


Figure 22: Solid Wireframe for the Supergene (300) Domain

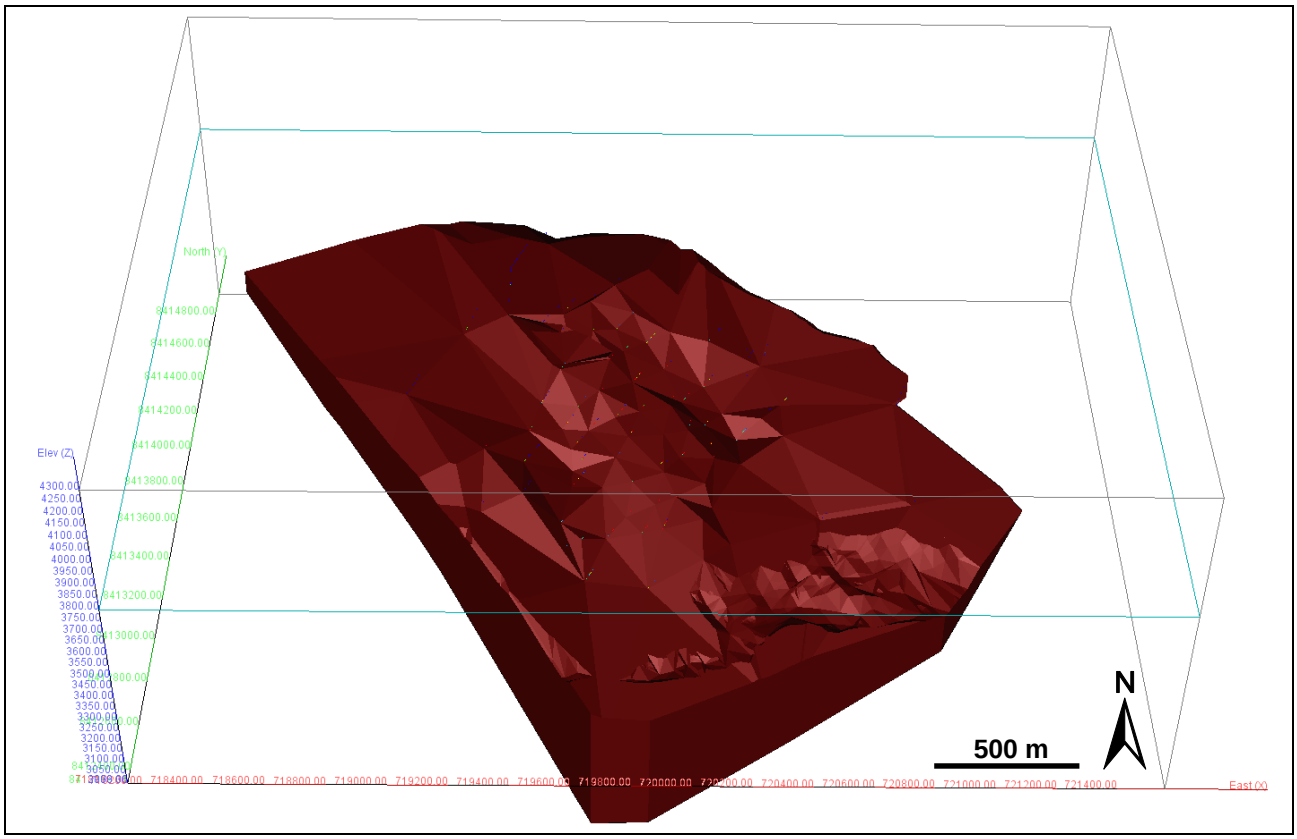


Figure 23: Solid Wireframe for the Primary Sulphide (400) Domain

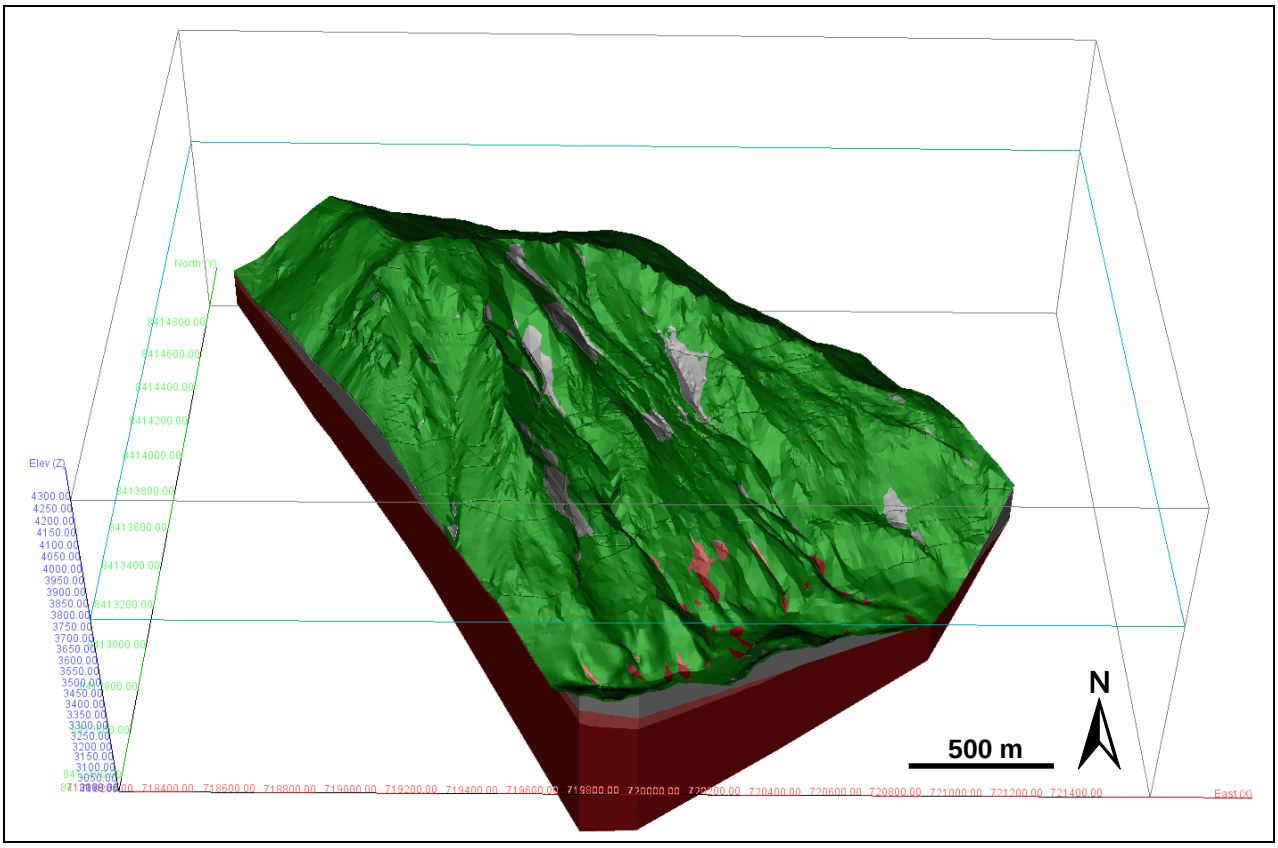


Figure 24: Solid Wireframe for all Mineralized Domains

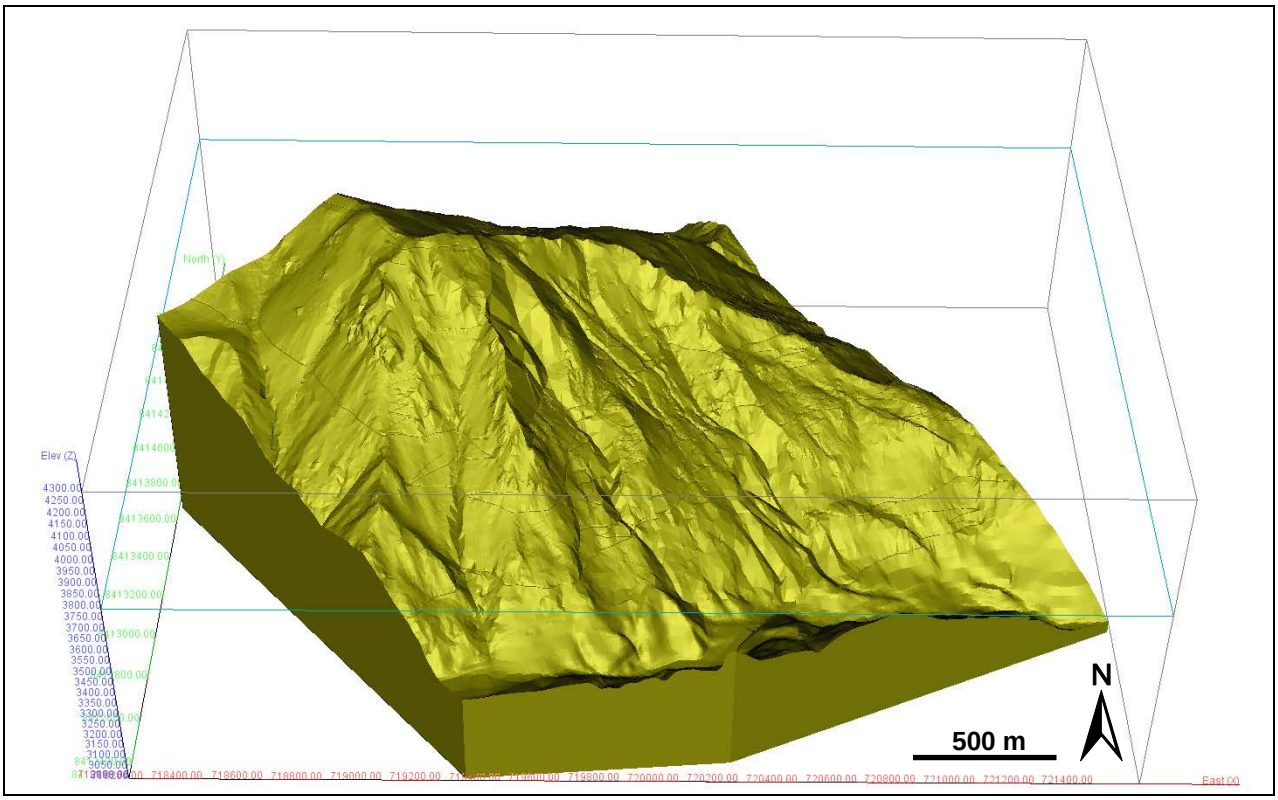


Figure 25: Solid Wireframe for the Country Rock (99) Domain

13.4 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.

13.4.1 Raw Assays

Raw assay statistics for copper, molybdenum, gold and silver in the modelled domains are shown in Table 19. 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 19: 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

* Std. Dev. = standard deviation; COV = coefficient of variation

13.4.2 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 26). 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.

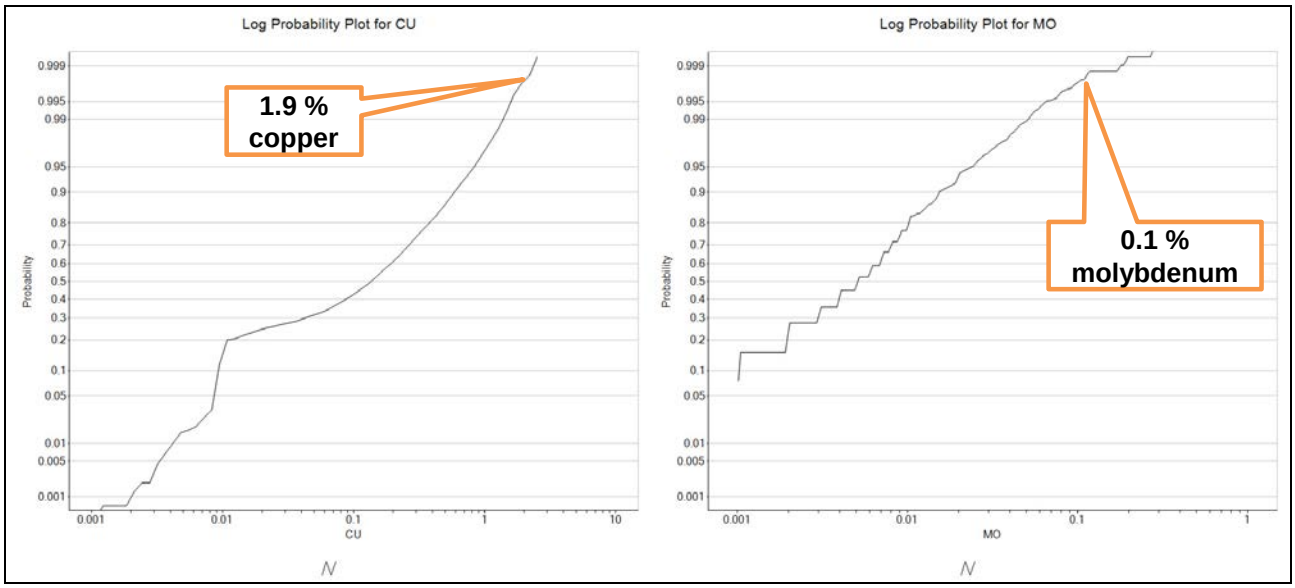


Figure 26: Logarithmic Cumulative Probability Plots Used for the Capping Analyses (copper on the left, molybdenum on the right)

Tetra Tech found that capping of raw data was deemed necessary. Capping values selected are shown in Table 20.

Table 20: Summary of Capping Values

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

13.4.3 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 21 presents the comparison between the capped data 6-metre composite data (no zeroes) for the mineralized domains.

Table 21: Statistics for Capped 6-metre Composites

	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

* Std. Dev. = standard deviation; COV = coefficient of variation

13.4.4 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 27 presents the contact plots for each of these domains using copper percent grades.

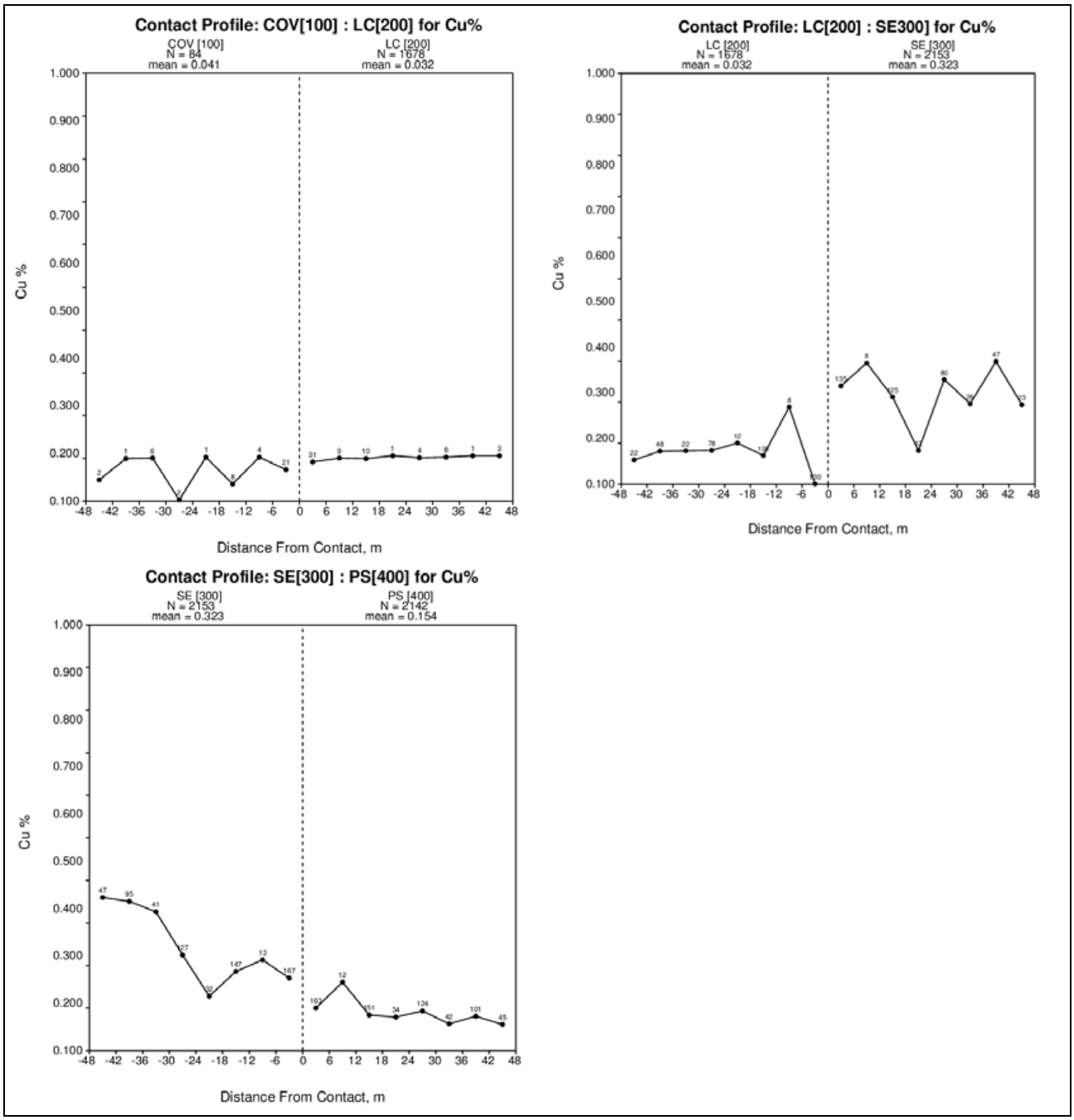


Figure 27: Contact Plots for Domain Boundaries for Copper

COV = Coverture, LC = Leach Cap, SE = Supergene and PS = Primary Sulphide

13.5 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 22 and Table 23 summarize the variography parameters used for interpolation for each domain in the Antilla deposit, while Figure 28 and Figure 29 illustrate examples of variograms for copper and molybdenum in the Supergene domain.

Table 22: Variography Parameters for Copper

Profile Name	Sill	Search Anisotropy	Azimuth	Dip	Azimuth	X Range (m)	Y Range (m)	Z Range (m)	Search Type
Domains 100 and 200; Sill = 0.0127									
C0 (nugget)	0.0038	-	-	-	-	-	-	-	-
C1	0.0027	Az, Dip, Az	130	-30	48	228	188	137	Spherical
C2	0.0062	Az, Dip, Az	130	-30	48	346	286	208	Spherical
Domain 300; Sill = 0.0849									
C0 (nugget)	0.0170	-	-	-	-	-	-	-	-
C1	0.0212	Az, Dip, Az	188	-19	93	155	94	37	Spherical
C2	0.0467	Az, Dip, Az	188	-19	93	255	155	61	Spherical
Domain 400; Sill = 0.0345									
C0 (nugget)	0.0104	-	-	-	-	-	-	-	-
C1	0.0090	Az, Dip, Az	219	-5	124	175	80	51	Spherical
C2	0.0152	Az, Dip, Az	219	-5	124	215	98	63	Spherical

Table 23: Variography Parameters for Molybdenum

Profile Name	Sill	Search Anisotropy	Azimuth	Dip	Azimuth	X Range (m)	Y Range (m)	Z Range (m)	Search Type
Domains 100 and 200; Sill = 0.00004									
C0 (nugget)	0.000002	-	-	-	-	-	-	-	-
C1	0.000011	Az, Dip, Az	234	8	330	147	147	37	Spherical
C2	0.000027	Az, Dip, Az	234	8	330	305	305	75	Spherical
Domain 300; Sill = 0.000076									
C0 (nugget)	0.000023	-	-	-	-	-	-	-	-
C1	0.000013	Az, Dip, Az	188	-19	93	240	166	79	Spherical
C2	0.000040	Az, Dip, Az	188	-19	93	275	190	90	Spherical
Domain 400; Sill = 0.000074									
C0 (nugget)	0.000022	-	-	-	-	-	-	-	-
C1	0.000052	Az, Dip, Az	215	-8	116	177	163	81	Spherical

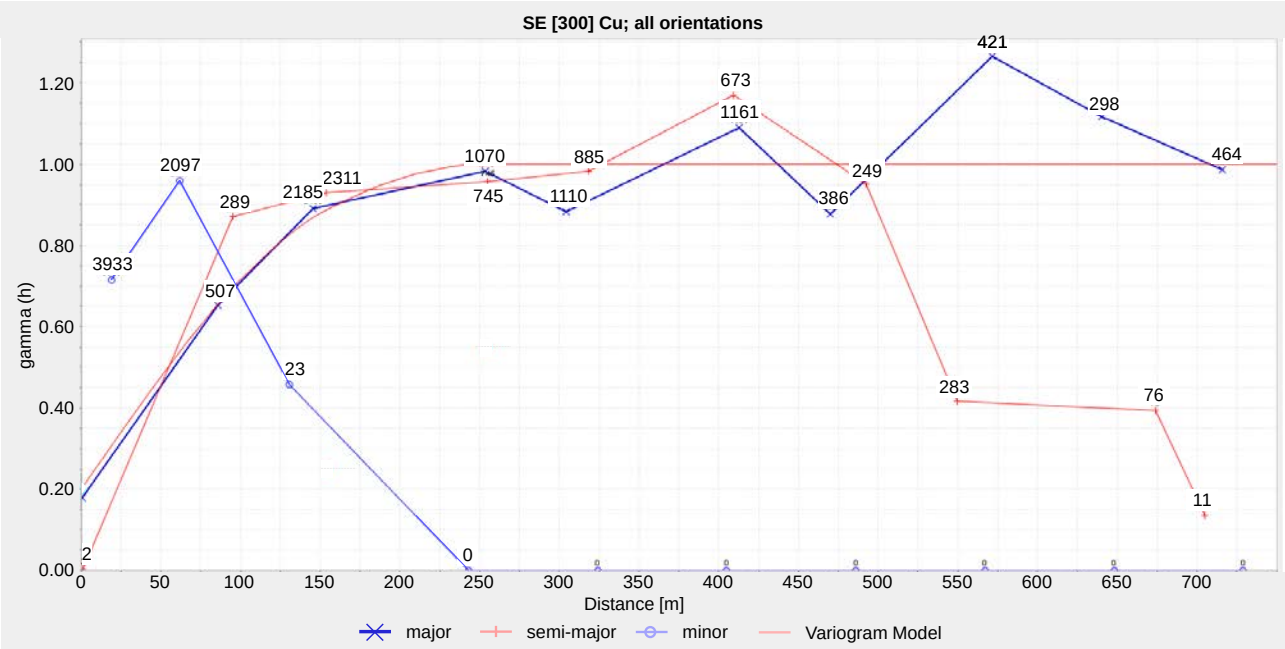


Figure 28: Variogram for Copper in the Supergene (300) Domain

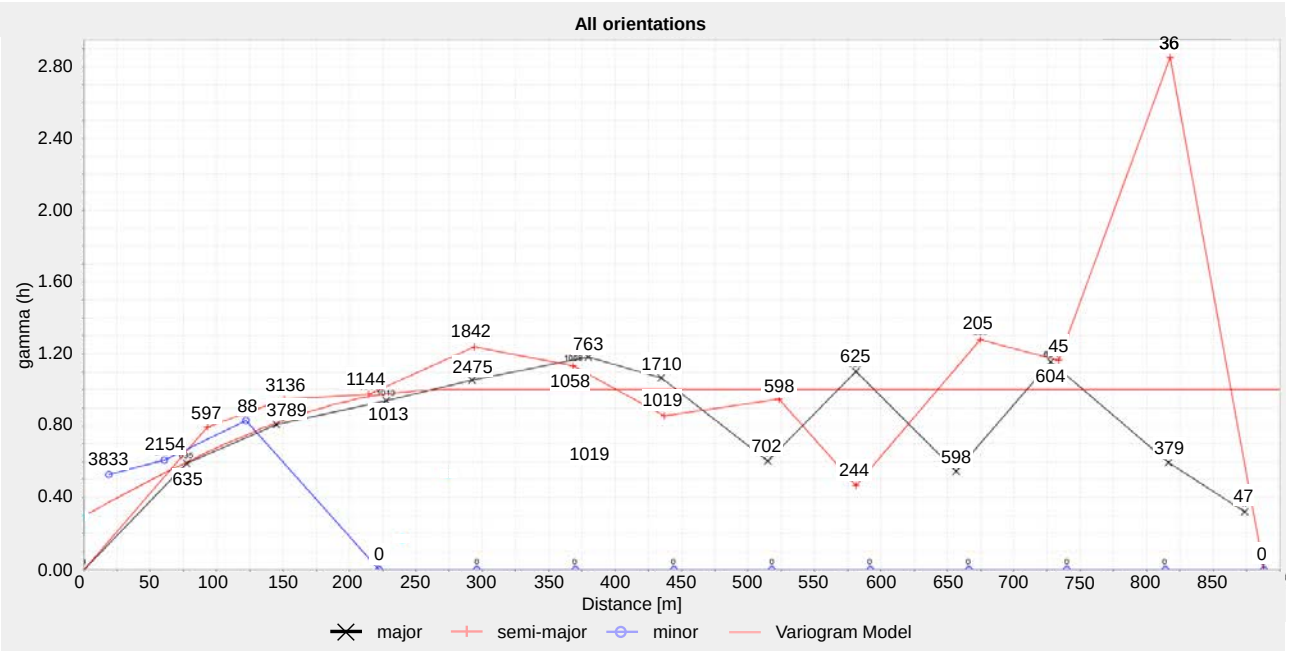


Figure 29: Variogram for Molybdenum in the Supergene (300) Domain

13.6 Block Model and Grade Estimation

13.6.1 Block Model

A single block model was created to cover the Antilla deposit. Table 24 show 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 24: 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

13.6.2 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 25.

Table 25: 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.

13.6.3 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 26, which illustrates the orientations of the search ellipses used in the interpolation of the Antilla block model.

Table 26: 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

13.6.4 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.

13.7 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 27.

Table 27: 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 Sulphide	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 30 to Figure 32 illustrate 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.

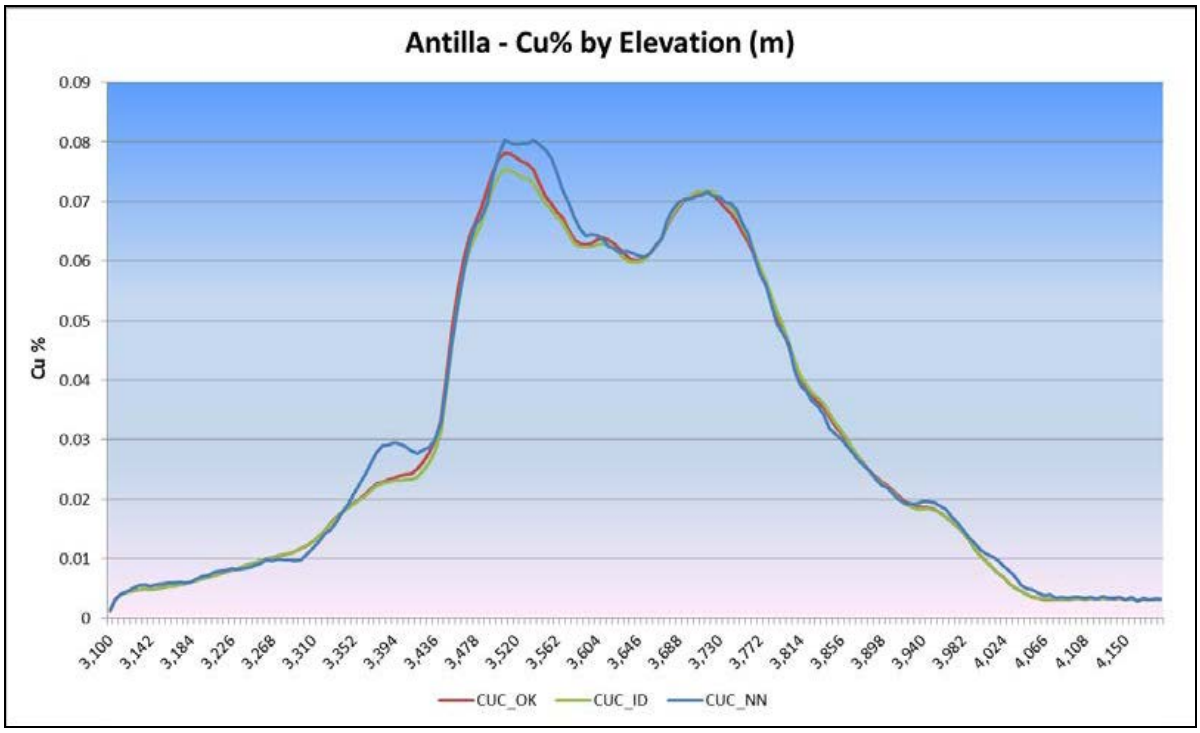


Figure 30: Swath Plots for Antilla by Elevation

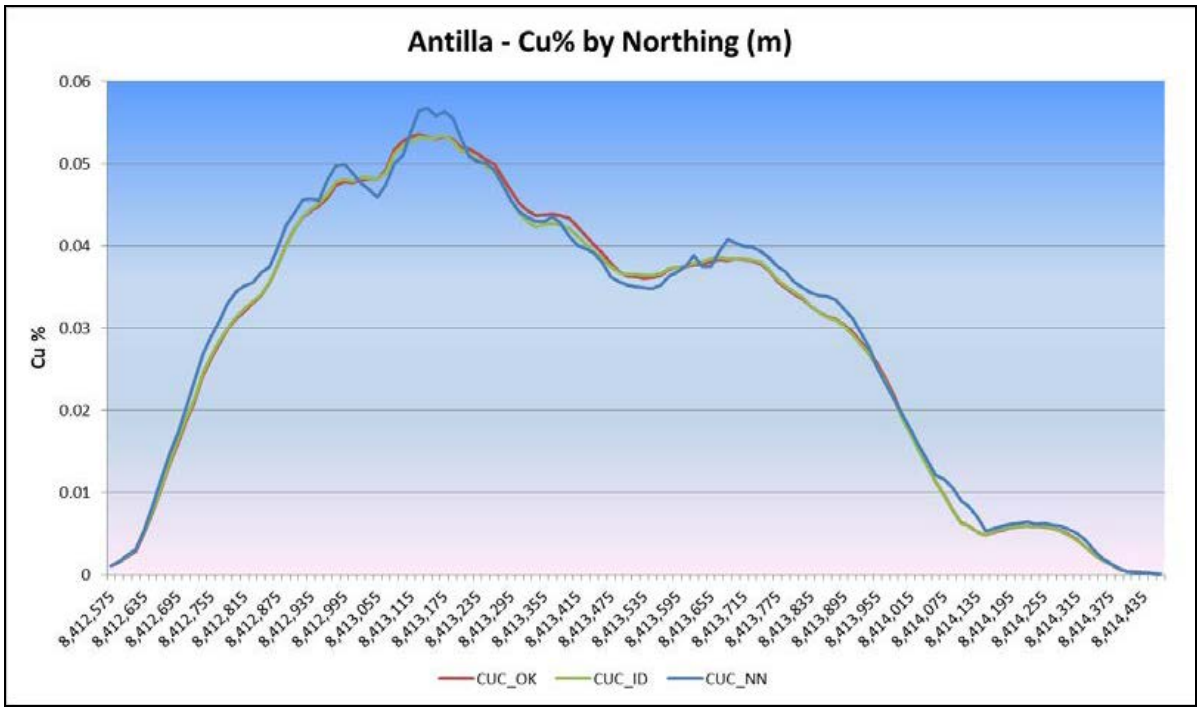


Figure 31: Swath Plots for Antilla by Northing

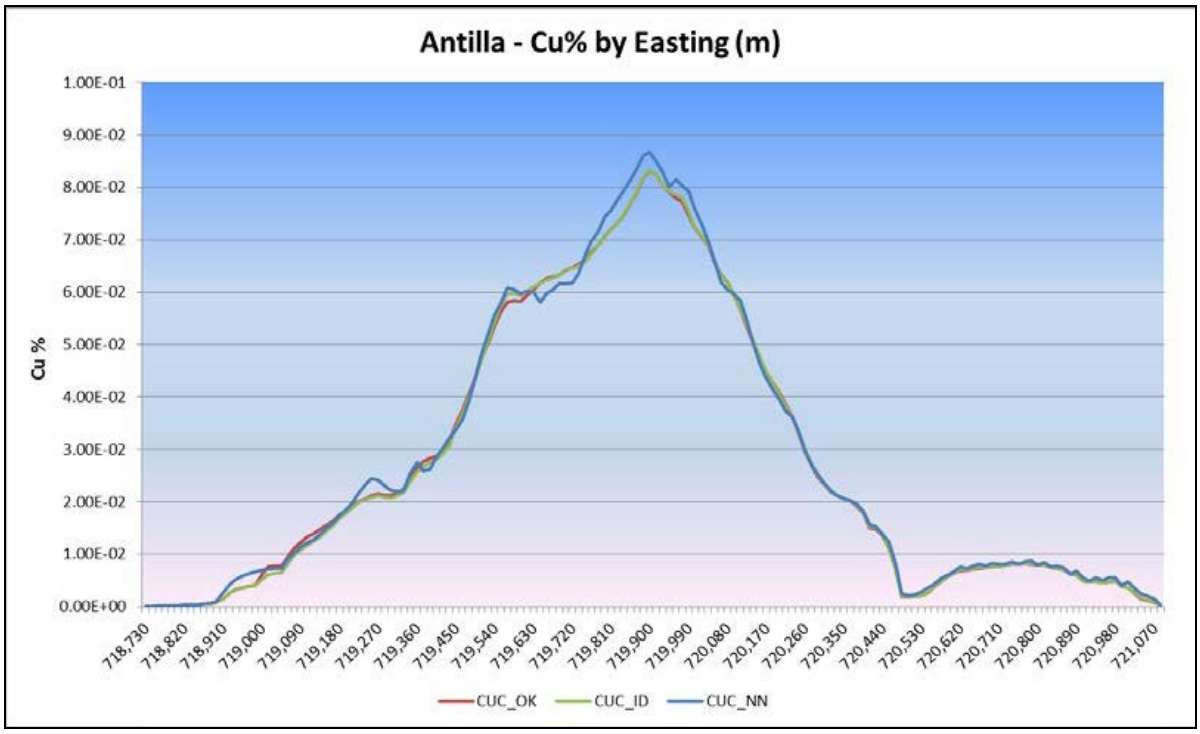


Figure 32: Swath Plots for Antilla by Easting

13.8 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.

13.9 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 28.

Table 28: 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	%
Sulphide	85 / 65	%
Mixed	80 / 70	%
Oxide	75 / 65	%
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 Sulphide	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**

* 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 29 presents the Mineral Resource Statement for the Antilla project.

Table 29: Mineral Resource Statement*, Antilla Copper-Molybdenum Project, Peru, Tetra Tech, October 19, 2015

Domain	Quantity '000 tonnes	Grade 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 Sulphide	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 Sulphide	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$3.25 per pound, a molybdenum price of US$ 9.00 per pound, and a metallurgical recovery of 90% for copper and 80% for molybdenum.

13.10 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 30 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 33 presents this sensitivity as grade tonnage curves.

Table 30: 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.

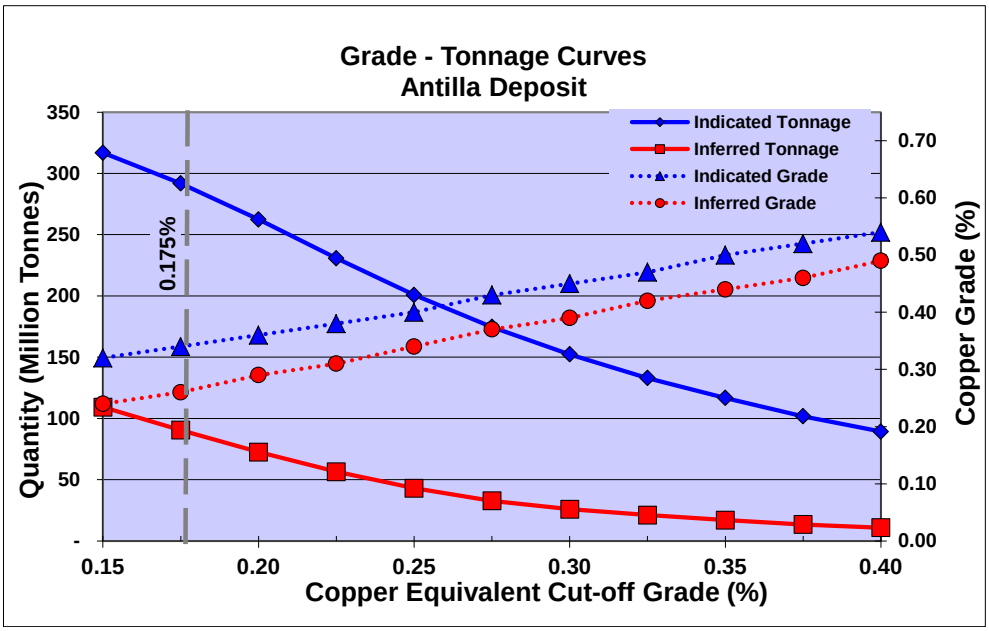


Figure 33: Grade-Tonnage Curves of Indicated and Inferred Blocks Inside the Conceptual Pit Shell

13.11 Previous Mineral Resource Estimates

In 2009, Panoro retained AMEC (now AMEC Foster Wheeler Plc) to prepare a Mineral Resource Statement in compliance with National Instrument 43-101 on the Antilla project. AMEC reported 154,400 tonnes grading 0.47% copper and 0.009% molybdenum at a cut-off grade of 0.25% copper in the inferred category. The effective date of this Mineral Resource Statement is unknown. This mineral resource has been superseded by the current Mineral Resource Statement reported herein.

In 2013 Tetra Tech a mineral resource model for the Antilla project. This mineral resource evaluation was based on the same database as the mineral resources reported herein (88 core boreholes drilled between 2003 and 2010). The mineral resources reported at a cut-off grade of 0.2% copper equivalent were estimated at 188.8 million tonnes grading 0.4% copper and 0.009% molybdenum in the Indicated category and 145.9 million tonnes grading 0.28% copper and 0.009% molybdenum in the Inferred category. The mineral resources were constrained by two conceptual pit shells; material between the two pit shells was assigned an inferred category, while blocks located in the smaller of the two model pit shells were assigned to the Indicated category. The 2013 mineral resource statement is superseded by the mineral resource reported herein.

Table 31 shows the variance between the 2013 and 2015 Mineral Resource Statements, which highlights the net increase in Indicated mineral resources at the expense of Inferred mineral resources.

The 2015 Mineral Resource Statement show a significant net redistribution of material from Inferred to Indicated category. The primary reasons for the significant increase in Indicated mineral resources include the revised pit shell, and a drop in reporting cut-off grade from 0.20 CuEq% to 0.175 CuEq%.

Table 31: Comparison Between the 2013 and 2015 Mineral Resource Statements

Domain	Quantity	Grade		
	'000 tonnes	Cu %	Mo %	CuEq%
Indicated				
Overburden/Cover	22%	-7%	0%	-7%
Leach Cap	54%	-11%	10%	-10%
Supergene	27%	-9%	-20%	-9%
Primary Sulphide	144%	-20%	-10%	-19%
Total Indicated	**55%**	**-16%**	**-15%**	**-15%**
Inferred				
Overburden/Cover	101%	-8%	-10%	-8%
Leach Cap	59%	-5%	-20%	-8%
Supergene	-47%	3%	-20%	6%
Primary Sulphide	-42%	-8%	-30%	-7%
Total Inferred	**-38%**	**-6%**	**-26%**	**-5%**

* Reconciliation: 22% = (2015-2013)/2013

14 Mineral Reserve Estimates

There are no mineral reserves on the Antilla project.

15 Mining Methods

A scoping level mine design, production schedule, and associated mining cost model was developed for the Antilla project based on an open pit mining method. The production schedule is based on a 40,000 tonne/day mill feed rate.

15.1 Introduction

The mine planning work is based on the updated mineral resource model created by Tetra-Tech discussed herein. The mine planning for the Antilla project is based on work done with Mintec Inc.'s Minesight, 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: equipment cost data (Panoro), pit slope angles (SRK), off-site and concentrate transportation costs (SRK), metallurgical recoveries (SRK), process costs (SRK), throughput rates (Panoro) and tailings dam design (ATC Williams). The information provided was reviewed by Moose Mountain and is deemed acceptable for use at a scoping level study for this project.

Detailed pit phases were 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 32.

Table 32: Summarized In-Pit Mineral Resources

		In Situ Undiluted Grades			
	Mill Feed*	**NSR**	**Cu**	**Mo**	**CuEq**
	kT	**US$/tonne**	**%**	**%**	**%**
Indicated	291,093	$16.35	0.322	0.0089	0.342
Subtotal of Measured + Indicated	**291,093**	**$16.35**	**0.322**	**0.0089**	**0.342**
Inferred	59,797	$12.90	0.249	0.0071	0.265

* The cut-off grade used is NSR>=US$6.10/tonne. NSR is calculated as follows:
NSR (US$/tonne) = [Cu grade (%) * Cu process recovery (%) * 57.76] + [Mo grade (%) * Mo process recovery (%) * 203.93]

The mineral resources 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 mill feed in the mine planning production schedule.

15.2 Geotechnical Recommendations

Several geological studies have been conducted by Panoro on the Antilla deposit. SRK reviewed these to understand the geological settings of the mineral deposit and the geotechnical characteristics of the rock located within the pit area.

Between 2003 and 2010, a total of 93 core boreholes (15,386 metres) were completed. Geotechnical logging was carried out on core from 20 boreholes, while RQD measurements were recorded on 73 f them.

A total of 573 point-load tests have been carried out on core recovered from different boreholes. Point-load index values obtained in these tests ranged between 7.79 and 8.12 MPa.

The geomechanical characterization of the rock-mass was carried out using the RMR classification system.

15.2.1 Site Geological Assessment

Geomorphology and Structural Geology

The Antilla deposit is located within the regional morpho-structural domain identified as the Eastern Cordillera, the drainage water of which discharges into the Atlantic Ocean. The area where the site is located shows rough morphological features due to the presence of numerous narrow steep creeks and canyons. The Antilla deposit occurs along the Huancaspaco Creek, a feature with an asymmetrical cross-section and steep sides. The left side of the creek where the deposit is located, typical local relief associated with ancient landslides can be observed. Steep cliffs are found along the upper part of the creek, whilst at the foot of the slopes, accumulations of alluvial material can be observed, as shown in Figure 6.

Rocks in the project area are folded and faulted, and some major structures can be identified within the project area. The sulphide mineralization of interest is located on the northern flank of a syncline, where the strata dip 25 to 30° to the southeast, a dip that practically coincides with the slope of the hillside. The orientation of the layers subparallel to the slope leads to large landslides whose traces can be seen as scarps on the upper parts of the slope and accumulation of materials along the bottom and the sides of the hills. In subsequent studies, the influence of the dip of the strata and the lithology of the area in the stability of the pit walls should be assessed.

A number of faults have been identified within the Antilla project area, most of them showing a predominantly northwest-southeast strike and a steep dip. There is a major N030°E/80°SE fault that intersects the central part of the mineralization, dividing it into an eastern and a western block.

15.2.2 Geological Model of the Deposit

The lithology of the pit area is the same for the different domains that were used in the mineral resource model, which are quartzite interbedded with thin sandstone units.

The domains are more related to the presence of mineralization and alteration level. For this reason, Leach cap was split into two types, Leach Cap 1 and Leach Cap 2, as shown in Figure 34.

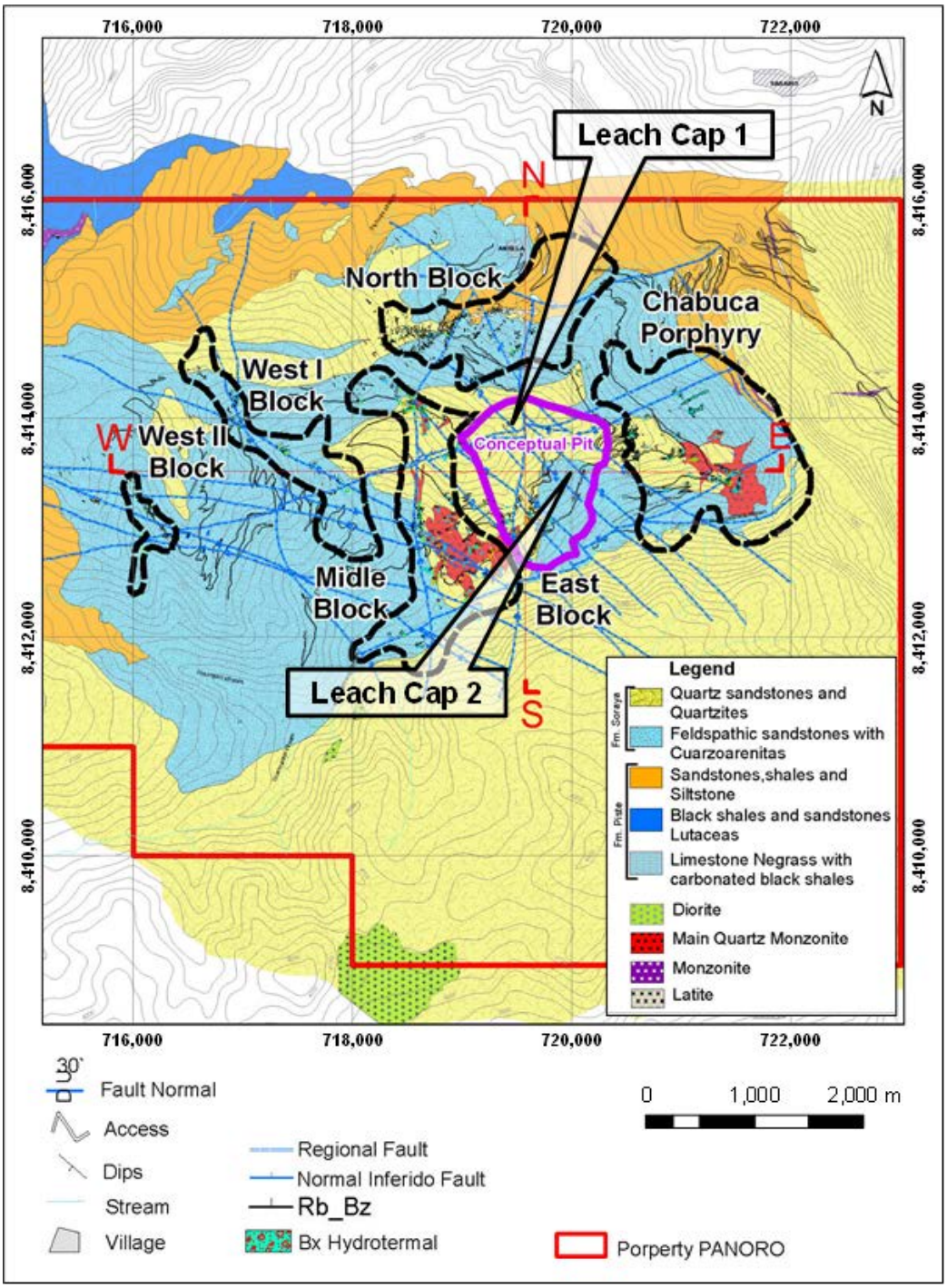


Figure 34: Leach Cap 1 and Leach Cap 2

Panoro has prepared a geo-economic model of the deposit, which identifies five distinct mineralized zones: overburden (COV), Leach Cap 1(LC1), Leach Cap 2 (LC2), Chalcocite Zone (SE) and Sulphides Primary (SP), each of which are represented in the geological section shown in Figure 35. Leach Cap 1 has similar geomechanical characteristics as the Primary Sulphide domain, because they are both quartzites and are less altered than Leach Cap 2. It was observed in several boreholes that the Supergene and Primary Sulphide domains have similar geotechnical characteristics to Leach Cap 1, such as RQD and alteration.

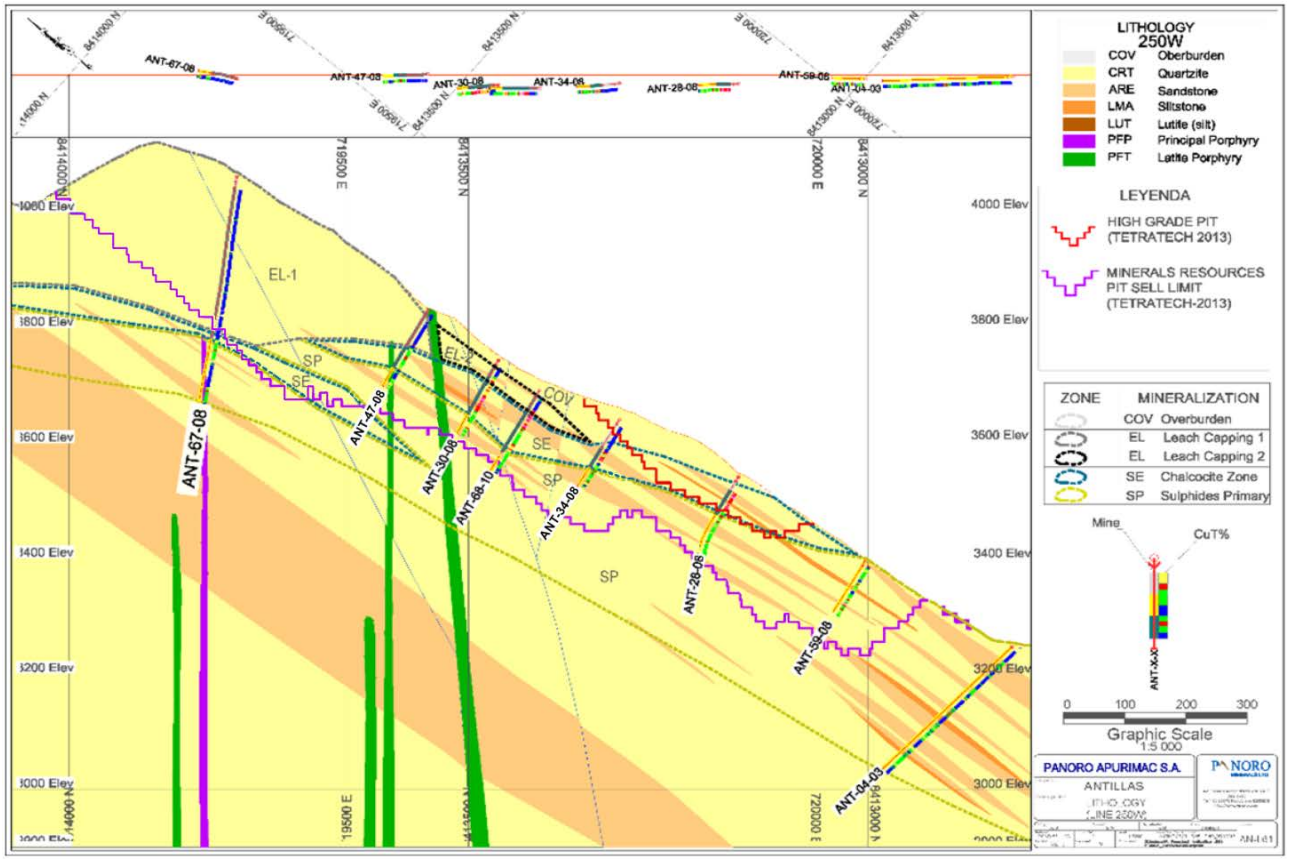


Figure 35: Geological Section Showing Leach Cap 1 and Leach Cap 2 Locations

15.2.3 Geomechanical Assessment and Pit Slope Recommendations

The geological and geotechnical information available for the project area allows the estimation of the geomechanical parameters indicated in Table 33.

Table 33: Geomechanical Parameters for the Geological Domains of the Antilla Pit Area

Structural Domain	RQD	RMR89	GSI	UCS (MPa)	C (kPa)	φ°(phi)
Cover (CO) - Soil	-	-	-	-		30
Leach Capping 1 (LC1)	11.1	34	29	28.3	155	27
Leach Capping 2 (LC2)	16.7	39	35	33.1	180	30
Supergene Enriched (SE)	21.9	39	35	38.2	187	31
Primary Sulphide (PS)	18.8	38	36	37.8	196	32
Country Rock (CR)	46.7	45	41	42.45	220	36

Pit slope geotechnical parameters, i.e., bench-berm configuration and inter ramp slopes for the life of mine plan pit designs are summarized in Table 34.

Table 34: Pit Slope Design Parameters

Geological Domain	Bench Face Angle (°)	Bench Height (m)	Berm Width (m)	Inter-ramp Angle (°)
Cover	30	18	10	24
Leach Cap 1	65	18	10	44
Leach Cap 2	75	18	10	51
Supergene Enriched	75	18	10	51
Primary Sulphide	78	18	10	52
Country Rock	78	18	10	52

It is assumed that pit ramps will be incorporated in the final pit walls. However, where the ramps are not required, it is recommended to design a minimum of 20 metres wide “geotechnical berm” between a 100-metre bench stacks. The 20 metres minimum width includes the normal 10 metres berm width.

15.3 Mine Planning 3D Block Model (3DBM) and Minesight Project

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

The mine planning block model dimensions are provided in Table 24. The block model is not rotated.

15.4 Production Rate Consideration

The throughput chosen for the Antilla project is 40,000 tonnes/day. The ultimate pit contains approximately 24 years of mill feed at the chosen throughput.

15.5 Net Smelter Return (NSR)

A net smelter return (NSR, US$/tonne) value is used as a cut-off grade to determine whether material is mill feed or waste. The NSR is calculated for each mineralized block in the mineral resource model using a net smelter price (NSP), mill process recoveries, and in situ grades. The NSP for each metal is the market price net of smelting, refining, and other off-site charges such as concentrate 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 mill. Table 35 shows the base case metal prices used along with the NSP values. Expected process recoveries vary by rock type and are shown in Table 36.

Table 35: Metal Prices

	Base Price US$/lb	NSP US$/lb
Copper	$3.00	$2.62
Molybdenum	$12.00	$9.25

Table 36: Process Recoveries

Rock Type	Process Recoveries - %	
	Cu	Mo
Cover	80%	65%
Leach Cap	75%	65%
Supergene	80%	70%
Primary Sulphide	85%	65%

The NSR is calculated as follows: NSR (US$/tonne) = [Cu(%) * CuRec(%) * 22.046 * NSPCu] + [Mo(%) * MoRec(%) * 22.046 * NSPMo]
Where:

- Cu(%) = in situ grade of copper expressed as %
- CuRec(%) = process recovery for copper expressed as %
- NSPCu = Net smelter price for copper in US$/lb
- Mo(%) = in situ grade of molybdenum expressed as %
- MoREc(%) = process recovery for molybdenum expressed as %
- NSPMo = Net smelter price for molybdenum in US$/lb

15.6 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).

15.6.1 Pit Optimization Method

A Lerchs-Grossman assessment was 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.

15.6.2 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. These costs are shown in Table 37 below.

Table 37: Lerchs-Grossman Unit Costs

	US$/tonne
Mining Costs	$1.85
Processing + G&A Costs	$5.48

15.6.3 Pit Slope Angles

Geotechnical recommendations for bench face angles and inter-ramp angles by rock type are provided is section 15.4.2. The provided inter-ramp angles in the leach cap, supergene, and primary sulphide rock types are reduced to accommodate the inclusion of ramps in the high wall. The pit slope angles used in the Lerchs-Grossman assessment are summarized in Table 38 below.

Table 38: Pit Slope Angles Used in Lerchs-Grossman Assessment

Rock Type	Overall Pit Slope Angle (°)
Overburden Cover	21
Leach Cap	42
Supergene	48
Primary Sulphides	49
Country Rock	52

15.6.4 Process Recoveries

The Lerchs-Grossman runs use the calculated NSR value for each block. Recommended process recoveries by rock type are shown in Table 36.

15.6.5 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 all in-pit mineral resource tonnages 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.

Metal prices are adjusted upwards and downwards in 10% increments from 50% to 150% to generate a series of shells. Potential mill feed tonnes inside each shell are calculated using a constant NSR cut-off of US$5.48/tonne. The results are shown graphically in Figure 36.

In Figure 36 it can be seen that there is an inflection point at the 80% shell. Figure 37 and Figure 38 show plan and sectional views of the 80% shell.

The 80% shell is selected as a guide for the ultimate pit design. It should be noted that the 80% shell will still generate positive cash flow at metal prices lower than the base case. Selecting a shell less than 100% case ensures that incremental pit shells produce positive cash flows at metal prices lower than the base case prices.

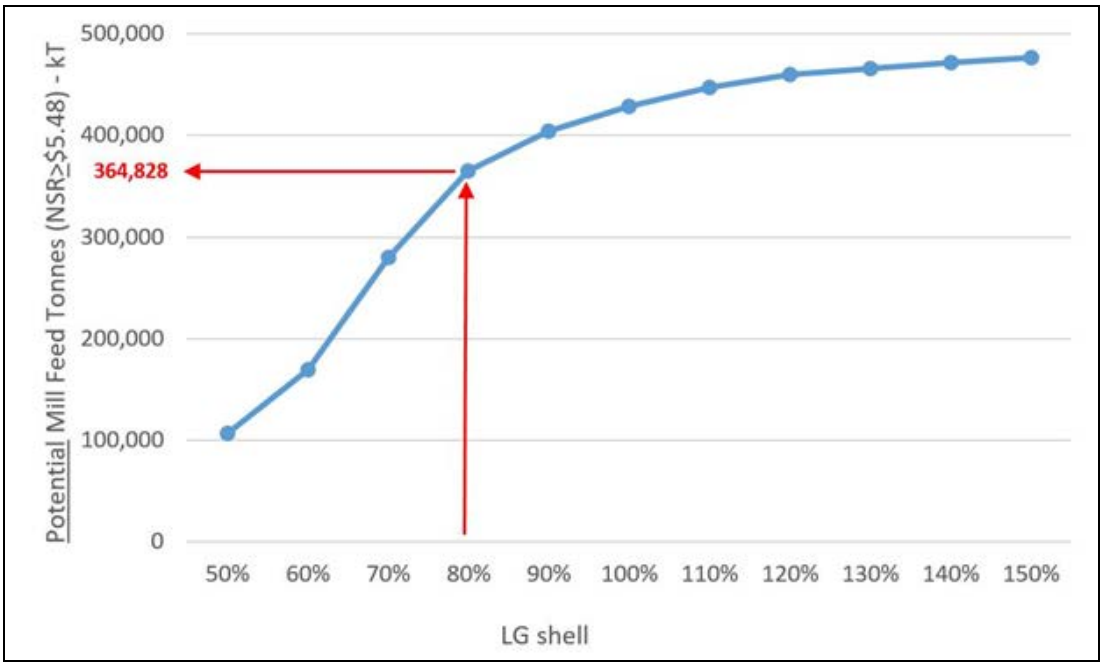


Figure 36: Lerchs-Grossman Sensitivity Summary (NSR>$5.48/tonne)

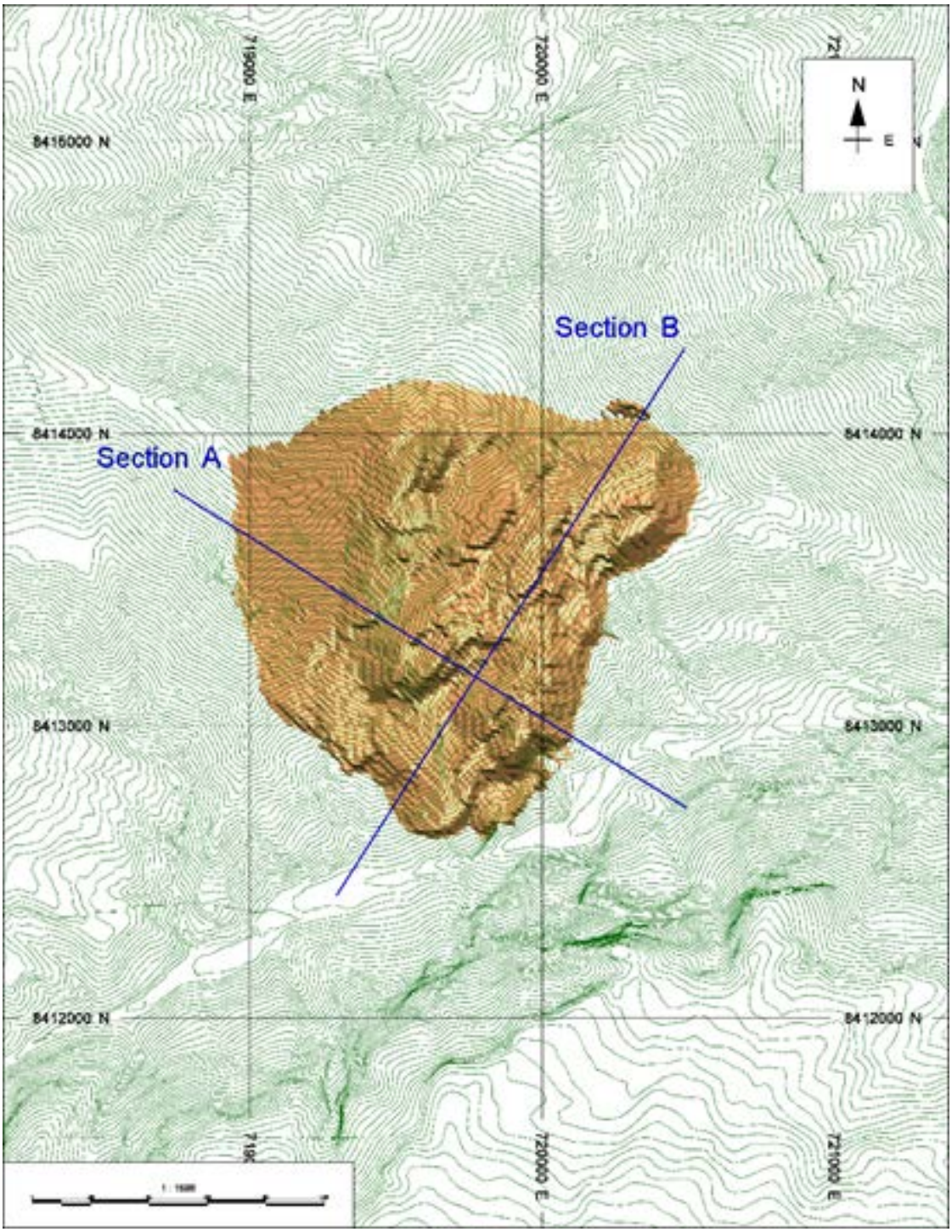


Figure 37: Plan View of 80% Lerchs-Grossman Shell

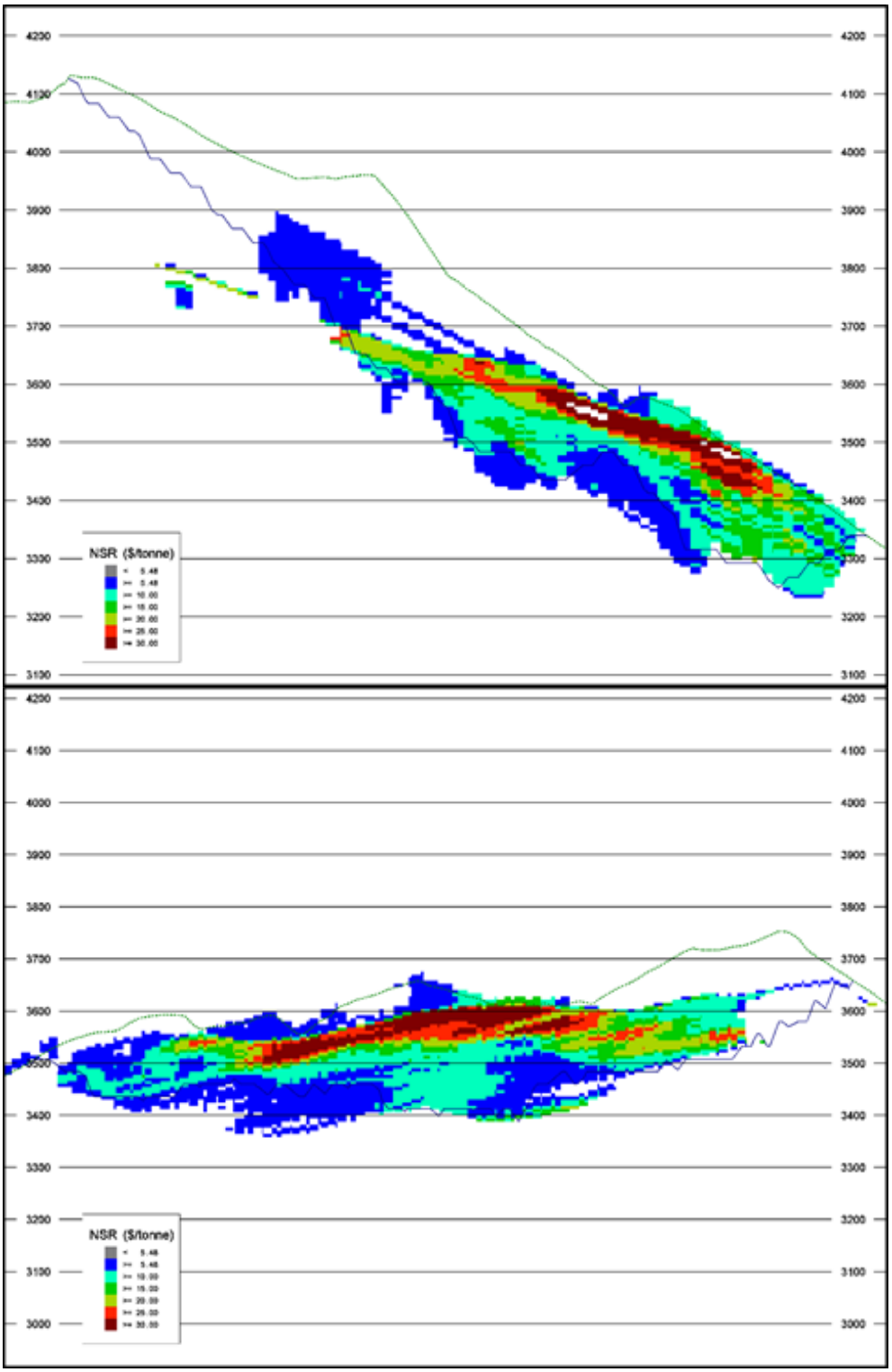


Figure 38: Top: Section A of 80% Lerchs-Grossman Shell. Bottom: Section B of 80% Lerchs-Grossman Shell

(Measured, Indicated, and Inferred blocks shown with their corresponding NSR value)

15.6.6 Detailed Pit Design

The detailed pit designs were developed with MineSight software. Bench heights used for pit designs are based on the geotechnical parameters provided and do not necessarily match the optimal operating bench height for the mobile mining fleet.

The parameters used for the detailed designs are shown in Table 39.

Table 39: Pit Design Parameters

Parameter	Value	Unit
Bench height	9	m
Safety berm width	10	m
Safety berm vertical spacing	18	m
Geotechnical berm* width	20	m
Geotechnical berm* vertical spacing	108	m
Minimum mining width between phases	100	m
Minimum mining width operational (i.e., at pit bottoms)	30	m
Ramp grade	10	%

The ultimate pit, shown in Figure 39, ranges from an elevation of 4,184 metres down to 3,238 metres. The maximum wall height is 946 metres. The majority 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 mill feed tonnes mined throughout the schedule.

Phase 1, shown in Figure 40, targets areas of higher economic return to increase the early economics of the project. Mining begins at an elevation of 4,007 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 3,304 metres.

Phase 2, shown in Figure 40, is a push back on the north side of Phase 1 with its own independent pit bottom. Mining in Phase 2 ranges from 4,181 metres down to 3,370 metres.

Phase 3, shown in Figure 40, is a pushback to the south side of Phase 1 that also eventually mines out the bottom of Phase 1. Mining in Phase 3 ranges from 4,148 metres down to 3,238 metres, establishing the ultimate pit bottom.

Phase 4, shown in Figure 40, is a push back to the northwest of the other phases. Phase 4 has two pit bottoms. However, both are at higher elevations than the ultimate pit bottom established by Phase 3. Mining in Phase 4 ranges from 4,148 metres down to 3,412 metres.

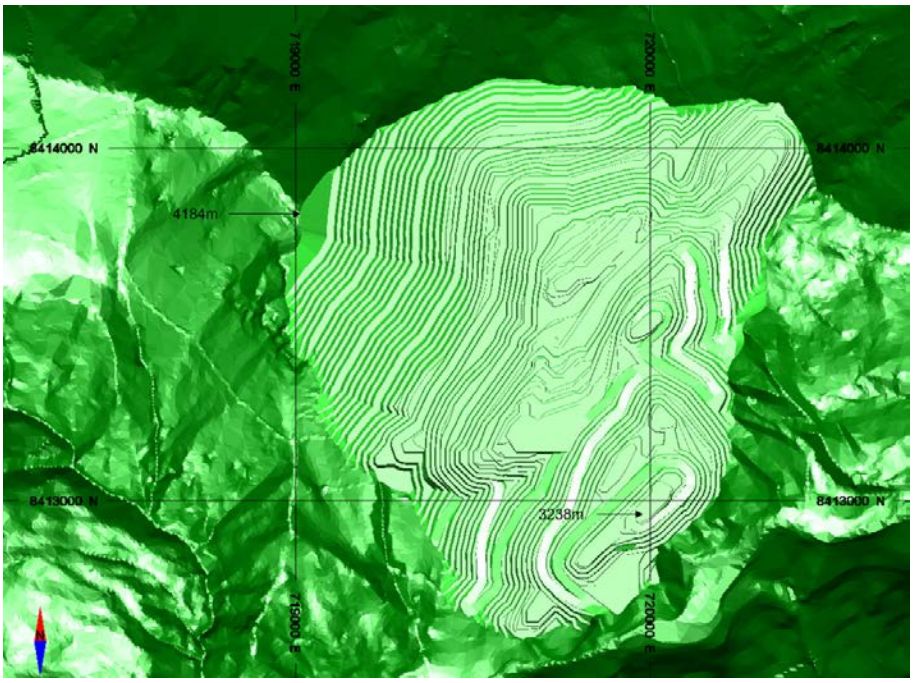


Figure 39: Ultimate Pit Design

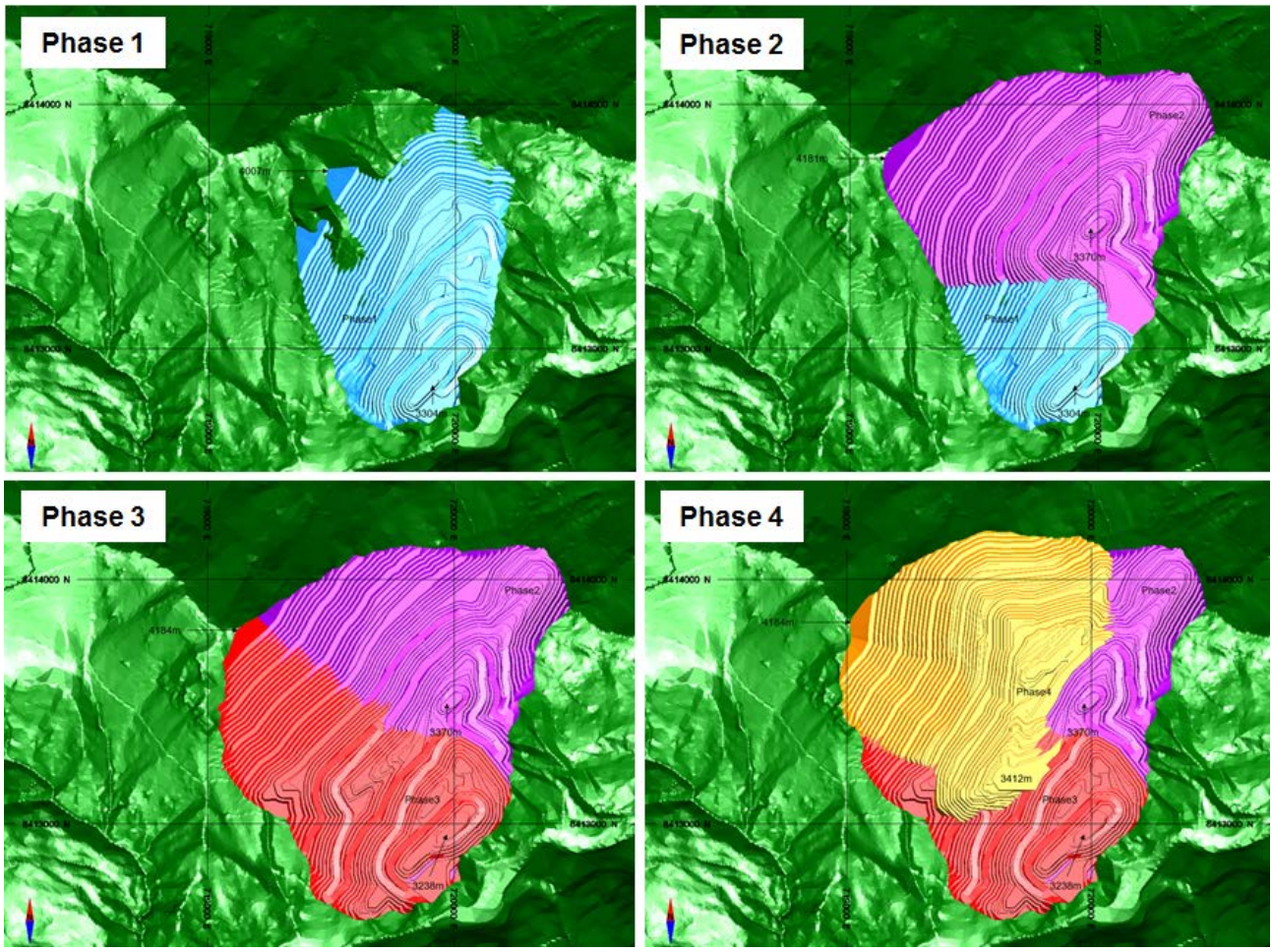


Figure 40: Detailed Pit Design Phase 1 to Phase 4

15.7 Pit Mineral Resource

The NSR value is used to determine whether a block is waste or mill feed. A NSR cut-off of US$6.10/tonne is used to select mill feed for the production schedule. The chosen cut-off value accounts for processing + G&A costs as well as expected re-handle costs for stockpiled material. A summary of the waste/mill feed by pit phase is shown in Table 40 below.

Table 40: Summary of Waste/Mill Feed by Pit Phase

		In situ Undiluted Grades					
	Mill Feed	NSR	Cu	Mo	CuEq	Waste	
	kT	US$/tonne	%	%	%	kT	S/R
Phase 1	63,852	$21.70	0.434	0.0104	0.457	29,152	0.46
Phase 2	97,906	$16.03	0.323	0.0061	0.337	105,612	1.08
Phase 3	94,594	$13.88	0.263	0.0108	0.287	60,696	0.64
Phase 4	94,538	$13.35	0.258	0.0077	0.275	104,144	1.1
Total	**350,890**	**$15.76**	**0.310**	**0.0086**	**0.329**	**299,604**	**0.85**

15.8 Mine Plan

The Antilla mine plan includes 2 years of pre-production, 19 years of standard open pit truck and shovel mining operations, and 5 years of stockpile reclaim at the end of the mine plan, for a total life of mine of 26 years.

The mine plan utilizes open pit mining methods, with mine rock being used to construct the tailings storage facility embankment as well as reporting to several mine rock storage facilities. Potential mill feed material mined from the pits reports either directly to the mill or is stored in a temporary stockpile.

15.8.1 Conceptual Mine Production Schedule

The conceptual mine production schedule is based on an annual mill feed of 14.6 million tonnes (40,000 tonnes/day) with increased mill feed grades during the first years of production by stockpiling marginally economic material (results in higher material movement in the first years but an improved cash flow). Portions of the stockpile are reclaimed throughout the mine life in order to increase grade or smooth out fleet requirements. However, the majority of the stockpile is reclaimed at the end of the mine life.

The conceptual mine production schedule is based on a 12-metre high operating bench rather than the 9-metre bench proposed by the geotechnical design criteria in Table 39. 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 geotechnical parameters were followed in designing the pits.

The conceptual mine production schedule is summarized in Figure 41 and Table 41.

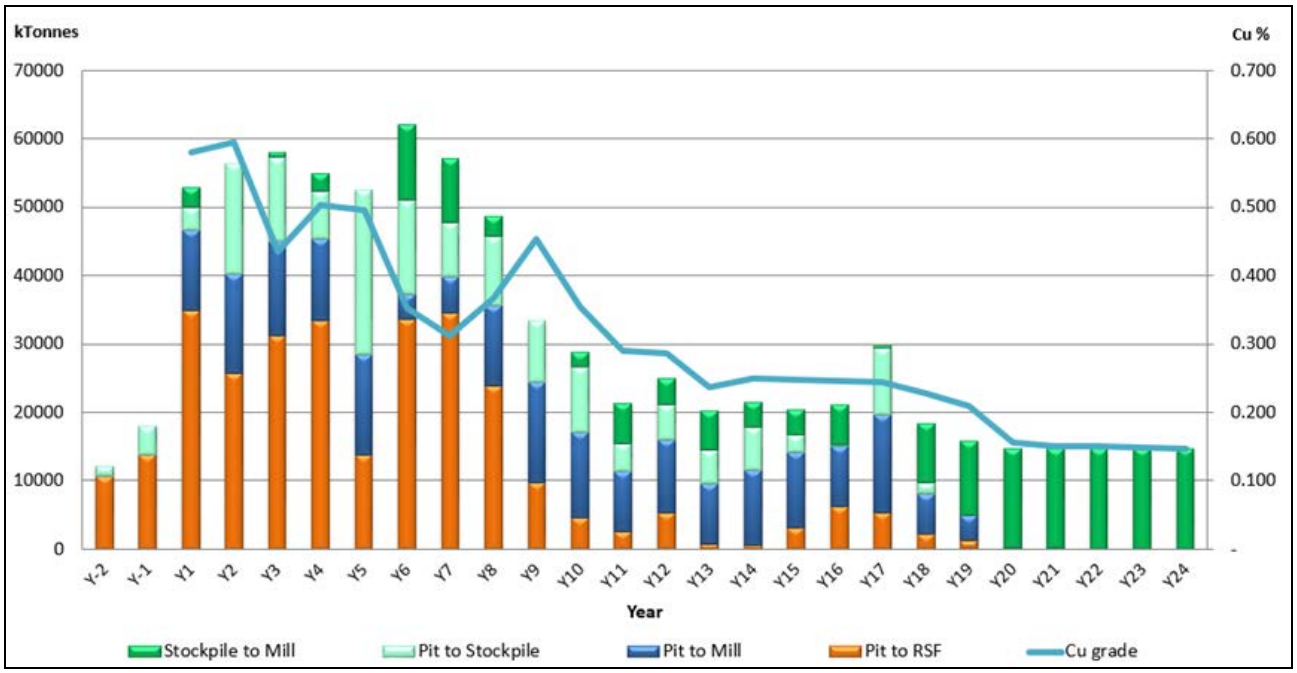


Figure 41: Conceptual Mine Production Schedule Summary

Table 41: Conceptual Mine Production Schedule Summary

	YR	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Mill Feed Total	**kT**	**-**	**-**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**
Cu Grade	%	-	-	0.579	0.594	0.435	0.504	0.496	0.352	0.312
Mo Grade	%	-	-	0.008	0.013	0.009	0.008	0.005	0.008	0.008
NSR	US$/t	-	-	$27.99	$29.40	$21.64	$24.74	$23.67	$17.76	$15.69
Pit to Mill Direct	**kT**	**-**	**-**	**11,632**	**14,600**	**13,870**	**11,942**	**14,600**	**3,667**	**5,318**
Cu Grade	%	-	-	0.588	0.594	0.429	0.513	0.496	0.384	0.349
Mo Grade	%	-	-	0.009	0.013	0.009	0.007	0.005	0.008	0.011
NSR	US$/t	-	-	$28.48	$29.40	$21.31	$24.82	$23.67	$18.69	$17.67
Pit to Stockpile	**kT**	**1,458**	**4,226**	**3,419**	**16,127**	**12,233**	**6,884**	**24,170**	**13,821**	**7,864**
Cu Grade	%	0.239	0.425	0.182	0.31	0.244	0.245	0.255	0.201	0.181
Mo Grade	%	0.006	0.006	0.008	0.011	0.007	0.004	0.007	0.008	0.008
NSR	US$/t	$11.68	$20.38	$9.35	$16.16	$12.37	$11.93	$13.04	$10.63	$9.84
Stockpile to Mill	**kT**	**-**	**-**	**2,968**	**-**	**730**	**2,658**	**-**	**10,933**	**9,282**
Cu Grade	%	-	-	0.545	-	0.55	0.463	-	0.341	0.29
Mo Grade	%	-	-	0.006	-	0.013	0.013	-	0.008	0.006
NSR	US$/t	-	-	$26.05	-	$27.75	$24.42	-	$17.45	$14.55
Stockpile Balance, End of Year	kT	1,458	5,684	6,134	22,261	33,764	37,990	62,160	65,047	63,629
Waste Mined	kT	10,609	13,822	34,928	25,682	31,112	33,489	13,786	33,585	34,548
Total Material Mined	**kT**	**12,068**	**18,048**	**49,978**	**56,410**	**57,215**	**52,315**	**52,556**	**51,072**	**47,730**
Total Material Moved	**kT**	**12,068**	**18,048**	**52,947**	**56,410**	**57,945**	**54,973**	**52,556**	**62,005**	**57,012**
:	**YR**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**	**Y13**	**Y14**	**Y15**	**Y16**
Mill Feed Total	**kT**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**
Cu Grade	%	0.367	0.453	0.354	0.289	0.286	0.238	0.25	0.248	0.246
Mo Grade	%	0.008	0.01	0.009	0.007	0.009	0.012	0.011	0.009	0.007
NSR	US$/t	$18.13	$22.41	$17.86	$14.59	$14.76	$13.01	$13.51	$13.27	$13.09
Pit to Mill Direct	**kT**	**11,664**	**14,600**	**12,540**	**8,715**	**10,820**	**8,760**	**10,950**	**10,950**	**8,760**
Cu Grade	%	0.393	0.453	0.37	0.308	0.3	0.246	0.259	0.256	0.261
Mo Grade	%	0.009	0.01	0.01	0.008	0.009	0.014	0.012	0.009	0.006
NSR	US$/t	$19.51	$22.41	$18.71	$15.84	$15.56	$13.55	$13.95	$13.63	$13.67
Pit to Stockpile	**kT**	**10,074**	**8,982**	**9,589**	**4,118**	**5,146**	**4,845**	**6,328**	**2,629**	**311**
Cu Grade	%	0.164	0.204	0.183	0.145	0.138	0.152	0.142	0.157	0.171
Mo Grade	%	0.01	0.008	0.007	0.007	0.007	0.011	0.01	0.006	0.003
NSR	US$/t	$9.10	$10.61	$9.85	$7.91	$7.68	$8.73	$8.18	$8.47	$8.72
Stockpile to Mill	**kT**	**2,937**	**0**	**2,060**	**5,885**	**3,780**	**5,840**	**3,650**	**3,650**	**5,840**
Cu Grade	%	0.263	0	0.261	0.261	0.247	0.225	0.224	0.224	0.224
Mo Grade	%	0.003	0	0.005	0.005	0.008	0.009	0.009	0.009	0.009
NSR	US$/t	$12.64	$0.00	$12.72	$12.72	$12.47	$12.22	$12.21	$12.21	$12.21
Stockpile Balance, End of Year	kT	70,767	79,749	87,278	85,511	86,877	85,882	88,560	87,539	82,011
Waste Mined - kT	kT	23,881	9,774	4,549	2,631	5,224	795	562	3,171	6,196
Total Material Mined	**kT**	**45,619**	**33,357**	**26,678**	**15,463**	**21,189**	**14,400**	**17,841**	**16,750**	**15,267**
Total Material Moved	**kT**	**48,555**	**33,357**	**28,738**	**21,348**	**24,970**	**20,240**	**21,491**	**20,400**	**21,107**
	YR	**Y17**	**Y18**	**Y19**	**Y20**	**Y21**	**Y22**	**Y23**	**Y24**	**Totals**
Mill Feed Total	**kT**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**14,600**	**350,400**
Cu Grade	%	0.244	0.228	0.209	0.156	0.15	0.15	0.149	0.147	**0.31**
Mo Grade	%	0.008	0.009	0.009	0.008	0.007	0.007	0.007	0.009	**0.009**
NSR	US$/t	$12.92	$12.40	$10.85	$8.39	$8.34	$8.33	$8.19	$7.55	**$15.77**
Pit to Mill Direct	**kT**	**14,217**	**5,840**	**3,690**	**120**	**63**	**0**	**0**	**0**	**197,317**
Cu Grade	%	0.245	0.235	0.179	0.216	0.214	0	0	0	**0.41**
Mo Grade	%	0.008	0.009	0.008	0.002	0.002	0	0	0	**0.009**
NSR	US$/t	$12.94	$12.68	$9.83	$10.86	$10.77	$0.00	$0.00	$0.00	**$20.51**
Pit to Stockpile	**kT**	**9,794**	**1,555**	**1**	**0**	**0**	**0**	**0**	**0**	**153,574**
Cu Grade	%	0.15	0.161	0.119	0	0	0	0	0	**0.22**
Mo Grade	%	0.007	0.007	0.008	0	0	0	0	0	**0.008**
NSR	US$/t	$8.34	$8.76	$6.65	$0.00	$0.00	$0.00	$0.00	$0.00	**$11.31**
Stockpile to Mill	**kT**	**383**	**8,760**	**10,910**	**14,480**	**14,537**	**14,600**	**14,600**	**14,600**	**153,083**
Cu Grade	%	0.224	0.224	0.219	0.156	0.15	0.15	0.149	0.147	**0.21**
Mo Grade	%	0.009	0.009	0.009	0.008	0.007	0.007	0.007	0.009	**0.005**
NSR	US$/t	$12.21	$12.21	$11.20	$8.37	$8.33	$8.33	$8.19	$7.55	**$11.32**
Stockpile Balance, End of Year	kT	91,422	84,217	73,308	58,828	44,291	29,691	15,091	491	-
Waste Mined	kT	5,324	2,272	1,193	73	36	0	0	0	**297,242**
Total Material Mined	**kT**	**29,335**	**9,667**	**4,883**	**193**	**99**	**0**	**0**	**0**	**648,133**
Total Material Moved	**kT**	**29,718**	**18,427**	**15,793**	**14,673**	**14,636**	**14,600**	**14,600**	**14,600**	**801,216**

15.8.2 Rock Storage Facilities

Mine rock from the open pit is stored in one of eight rock storage facilities or hauled to the tailings storage facility and used to construct its embankment.

Rock storage facilities are placed either as top-down or wrap-arounds, at angle of repose (37 degrees). Some include safety/geotechnical berms to reduce the overall rock storage facilities face angles. A swell factor of 1.34 is applied to all material placed in the RSFrock storage facilities. The rock storage facilities schedule is outlined in Table 42 and their location is shown in Figure 42.

Table 42: Rock Storage Facility Schedule Summary

Location	Totals (kt)	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6-10	Y11-15	Y16-20	Y21-24
TSF	**143,164**	4,747	4,747	6,796	7,101	11,208	13,385	3,501	64,203	12,383	15,057	36
RSF1	**14,569**	-	-	-	1,010	3,788	-	-	9,771	-	-	-
RSF2	**8,952**	5,862	-	2,944	-	-	-	146	-	-	-	-
RSF3	**21,948**	-	9,075	12,873	-	-	-	-	-	-	-	-
RSF4	**33,100**	-	-	9,578	15,737	7,785	-	-	-	-	-	-
RSF5	**30,146**	-	-	-	-	8,331	20,104	1,711	-	-	-	-
RSF6	**13,000**	-	-	2,738	1,834	-	-	8,428	-	-	-	-
RSF7	**25,963**	-	-	-	-	-	-	-	25,963	-	-	-
RSF8	**6,400**	-	-	-	-	-	-	-	6,400	-	-	-
All	**297,242**	**10,609**	**13,822**	**34,928**	**25,682**	**31,112**	**33,489**	**13,786**	**106,337**	**12,383**	**15,057**	**36**

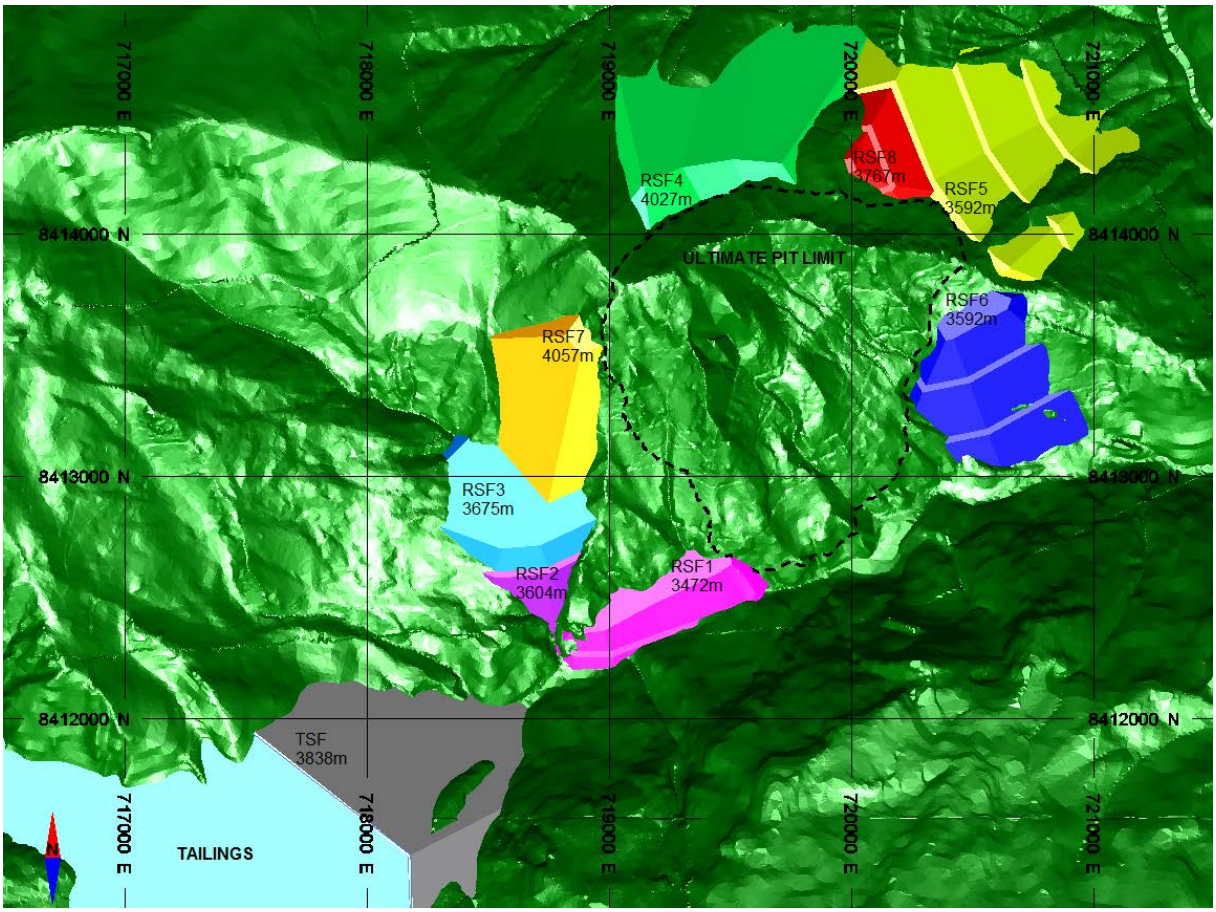


Figure 42: Location of Rock Storage Facilities

15.8.3 Mill Feed Stockpile

The mill feed stockpile is located adjacent to the mill and is constructed in 50-metre lifts with a 27 metres wide berm between each subsequent lift. This results in an overall stockpile slope of approximately 28°. The stockpile is accessed by using a 10% haul road built along the outer edge.

The maximum size of the stockpile is approximately 91 million tonnes and occurs in Year 17 of the schedule. The stockpile is almost completely reclaimed by the end of year 24, with 491 kilo tonnes left forming the base of the stockpile pad.

The stockpile location is shown in Figure 43.

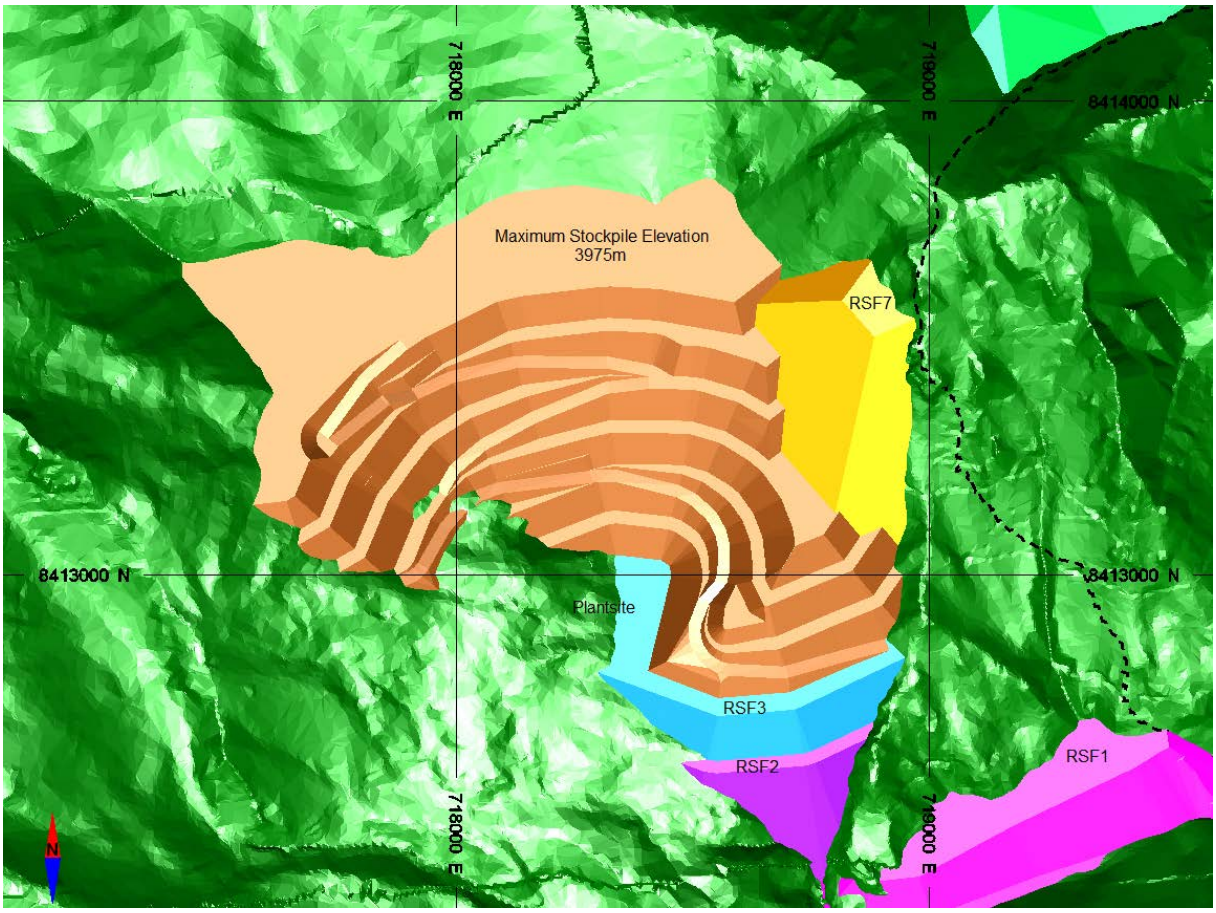


Figure 43: Location of Stockpile

15.8.4 Details for the Conceptual Mine Production Schedule

The details of the conceptual mine production schedule is provided for three samples periods from the production schedule: End of Preproduction (Figure 44), End of Year 10 (Figure 45), and Life of Mine (Figure 46).

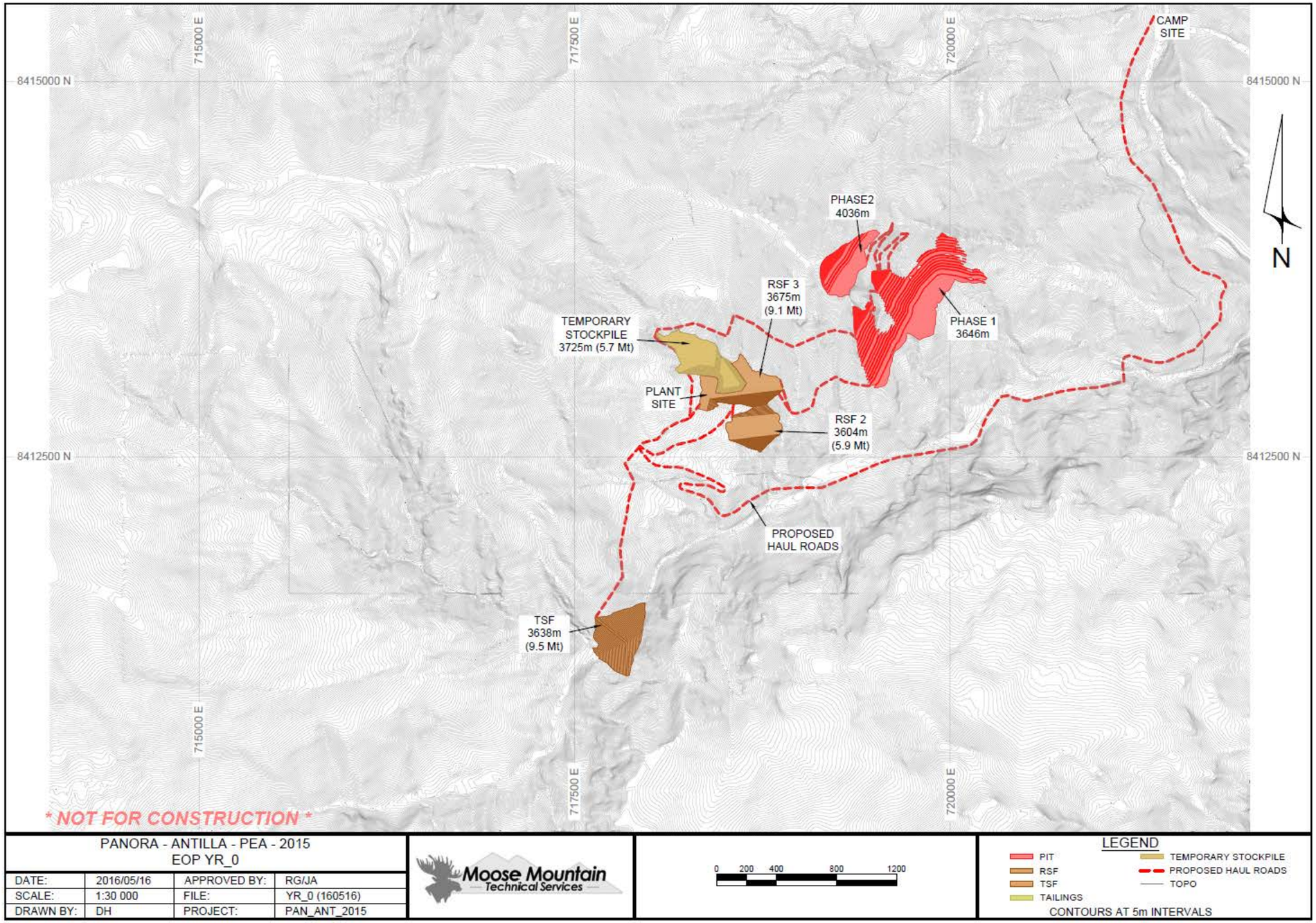


Figure 44: End of Pre-production (Time Zero) Detail

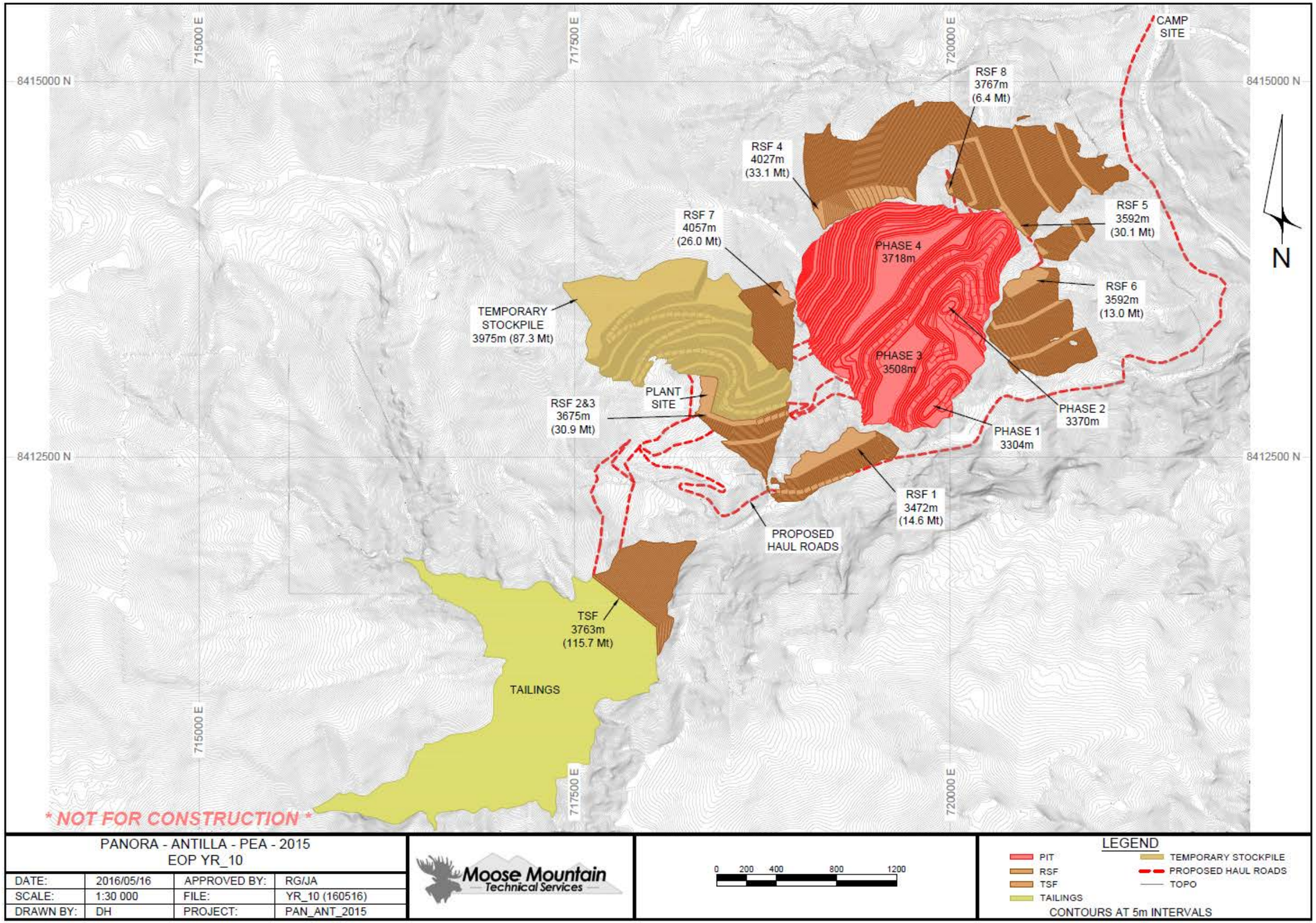


Figure 45: End of Year 10 Detail

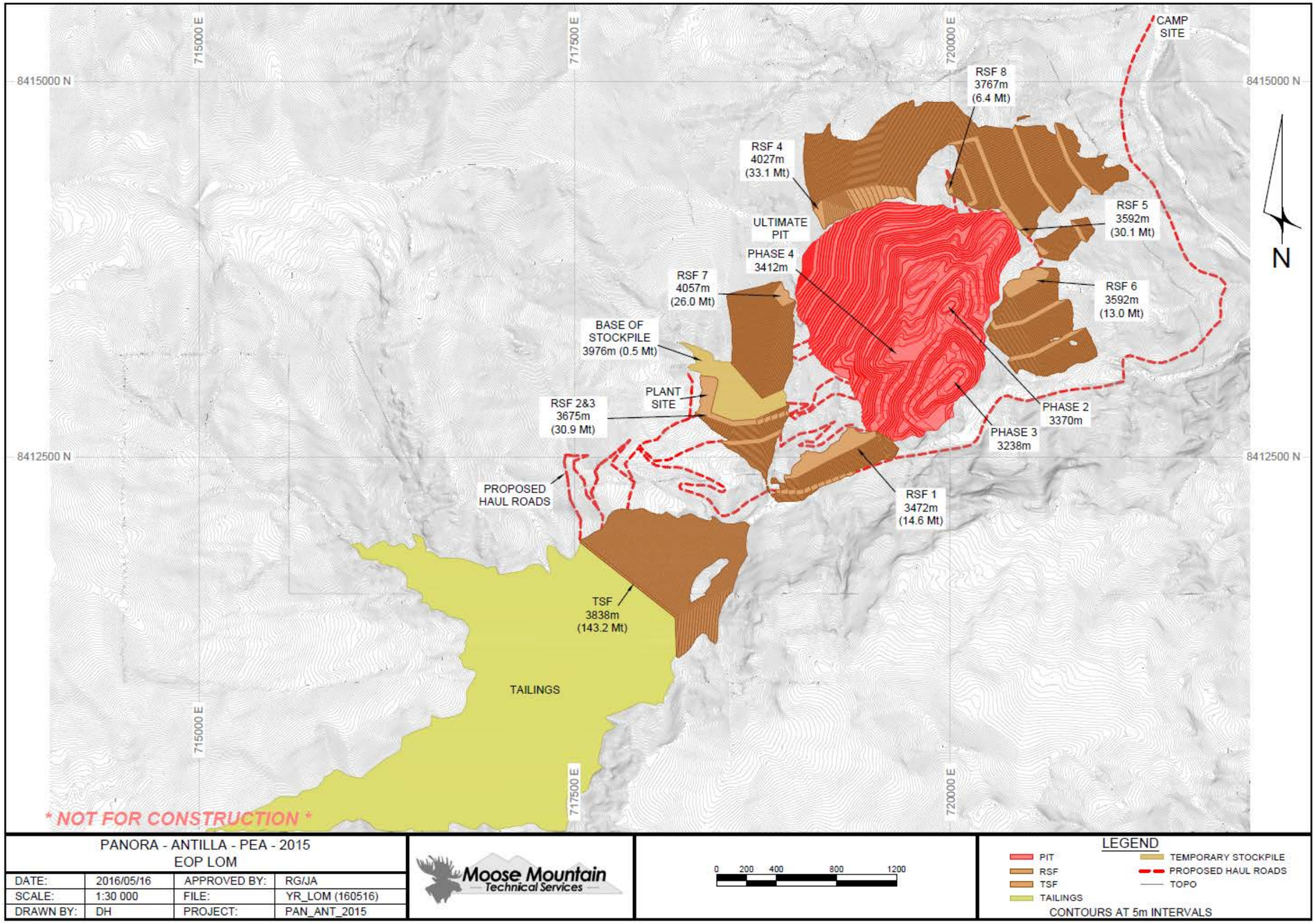


Figure 46: Life of Mine Detail

The end of the pre-production period, referred to as time zero, is signified by the start-up of the processing facilities. At time zero, there will be two active pit phases. Phase 1 is mined down to the 3,646 metre bench while Phase 2 is mined to the 4,036 metre bench. A large portion of mining operations during this time period involves narrow development benches which are blasted and pushed down slope by track dozers for shovel-loading and truck-hauling on wider operational benches. Approximately 9.5 million tonnes of mine waste rock from these phases is used to build the TSF starter embankment. Any remaining mine waste rock from these phases is placed in either rock storage facilities 3 or 4, depending on the source elevation. All mineralized material mined during pre-production is placed in the temporary stockpile adjacent to the plant site as the processing facilities have not begun operations.

By the end of Year 10 of the mine schedule mining of Phase 1 and 2 is completed down to pit bottom elevations of 3,304 metres and 3,370 metres respectively. Phase 3 and Phase 4 of the open pit are both active, mined down to an elevation of 3,508 metres and 3,718 metres respectively. Approximately 116 million tonnes of mine waste rock is used to construct the Year 10 tailings storage facility embankment. By Year 9, all rock storage facilities locations have been filled to capacity, therefore all remaining mine waste rock reports to the tailings storage facility embankment. By the end of Year 10 approximately 119 million tonnes of mineralized material is sent to the temporary stockpile, however there has also been approximately 32 million tonnes of stockpile reclaimed and sent to the processing plant, leaving a balance of approximately 87 million tonnes in the temporary stockpile.

By the end of life of mine operations, the ultimate pit has multiple pit bottoms established by the completion of Phases 2, 3, and 4, at 3,370 metres, 3,238 metres, and 3,412 metres, respectively. The Phase 1 pit bottom is mined out by subsequent phases. After Year 9 all mine waste rock is sent to the tailings storage facility embankment. At end of life of mine it contains approximately 143 million tonnes of mine waste rock. The temporary stockpile is reclaimed leaving behind only the base of the stockpile, equating to approximately 0.5 million tonnes.

15.8.5 Mine Operations

The mining operations proposed for the Antilla project are typical of open pit, truck-and-shovel mining methods for year round operations in a mountainous terrain.

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 accounts 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 24 hours per day, seven days a week, on a 12-hour shift rotation.

15.9 Mine Equipment

The Antilla mining fleet consists of a conventional diesel fleet with a peak material movement capacity of approximately 62,000 kilo tonnes per year, and a life-of-mine average material movement capacity of approximately 31,000 kilo tonnes per year.

The Antilla Mobile Mining Fleet is summarized in Table 43.

Table 43: Mobile Mining Fleet

Major Equipment	Function	Quantity				
		Y-2	Y-1	Y1	Y5	Y10
Hydraulic Shovel - 15m^3	Loading Ore & Waste	1	2	4	5	4
Haul Truck - 144t	Hauling Ore/Waste	4	7	17	22	14
Drill - Diesel Hydraulic - 270mm	Primary Drill	1	2	3	3	3
Drill - Diesel Hydraulic - 150mm	Secondary Drill, High Wall Drilling	0	0	1	1	1
Support Equipment						
Dozer - 306kW	Shovel Support - In Pit	3	3	5	6	6
Wheel Dozer - 372kW	Pit Clean Up, Shovel Support	1	1	1	2	2
Fuel / Lube Truck - 4000gal	Fuel Truck	1	1	1	2	2
Water Truck - 20 000gal	Haul Roads Water Truck	1	1	1	2	2
Grader - 221kW	Road Grading	1	1	2	2	2
FEL - 373kW	Multi-Tool, Tire Changes, Road Crush	1	1	1	1	1
Ancillary Equipment						
Excavator - 301kW	Utility Excavator	1	1	2	2	2
Mobile Screening Plant	Screening Plant	1	1	1	1	1
Jaw Crusher	Road Crush	1	1	1	1	1
Forklift - 10t	Forklift	1	1	1	1	1
Light Plant - 20kW	Light Plant	4	4	4	10	10
Crane - 40t	Crane, Truck Boxes	1	1	1	1	1
Crew Van - 15 Passenger	Crew Transport	2	3	3	3	3
Warehouse Truck - 1t	Warehouse Truck	1	1	1	1	1
Crew Cab Pickup	Crew Cabs, Supervisor trucks	4	6	8	8	8
Service Truck	Maintenance + Overhauls	1	1	2	2	2
Welding Truck	Welding Truck	1	1	1	1	1
Picker Truck	Picker Truck	0	1	1	1	1

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

Equipment productivity factors are estimated based on information provided by equipment vendors as well as historical experience and data from similar operations. Equipment productivity factors for the major equipment is detailed in Table 44.

Table 44: Equipment Productivity Factors

Major Equipment	Operating Efficiency	Availability %			Equipment Life
		0-35,000 hrs	35,000+ hrs	Utilization %	
Hydraulic Shovel - 15m^3	82.8%	86%	84%	97%	80,000
Haul Truck - 144t	82.8%	86%	84%	97%	80,000
Drill - Diesel Hydraulic - 270mm	83.0%	88%	86%	96%	70,000
Drill - Diesel Hydraulic - 150mm	83.0%	88%	86%	96%	70,000

15.9.1 Drilling and Blasting

Based on the production schedule, a requirement for production drilling hours is calculated based on hole size, and pattern, bench height, material density, and penetration rate of the drill. High wall and pre-shear drilling is included as an allowance for a 2,000-metre high wall length per year, and uses the secondary drill rig.

Drill productivity is estimated at 1.66 boreholes per hour based on a theoretical penetration rate of 28 metres per hour. The blasting powder factor is estimated as 0.3 kilograms/tonne and 0.28 kilograms/tonne for mill feed and mine waste rock respectively.

Drilling and blasting estimates are based on a 12-metre high operating bench.

15.9.2 Loading & Hauling

The primary loading units for the Antilla project are 15 cubic metres diesel hydraulic shovels matched with a 144-tonne haul truck. This truck and shovel pairing is used for both mill feed and mine waste rock.

The hydraulic shovels operate on 12-metre benches in a five pass loading arrangement with an 85% shovel bucket fill factor is assumed, resulting in an estimated shovel productivity of 1,646 tonnes/hour.

Haul trucks are limited by payload capacity rather than volume, due to the density of the material. Representative cycle times are used to calculate truck hours, with three source points distributed across each phase (top, middle, bottom) routed to each destination (mill, stockpile, tailing storage facility, or rock storage facility 1-8).

Figure 47 and Figure 48 show the shovel and truck fleets allocated over the life of the mine, versus the calculated requirements generated by the production schedule.

15.10 Open Pit Personnel

The mine plan assumes that Antilla will operate with four crews rotating to fill the mine roster of 12 hours per shift (two shifts per day). Owner manpower is the most intensive during the first eight years of production when total material movement is the highest. The manpower rises to a maximum of 255 (average is 237). During Years 9 through 18, the average manpower is 148 personnel and during Years 19-24 (stockpile reclaim) the average manpower drops to 75. Mining manpower includes hourly equipment operators, mine maintenance crews, technical staff and mine operations supervision.

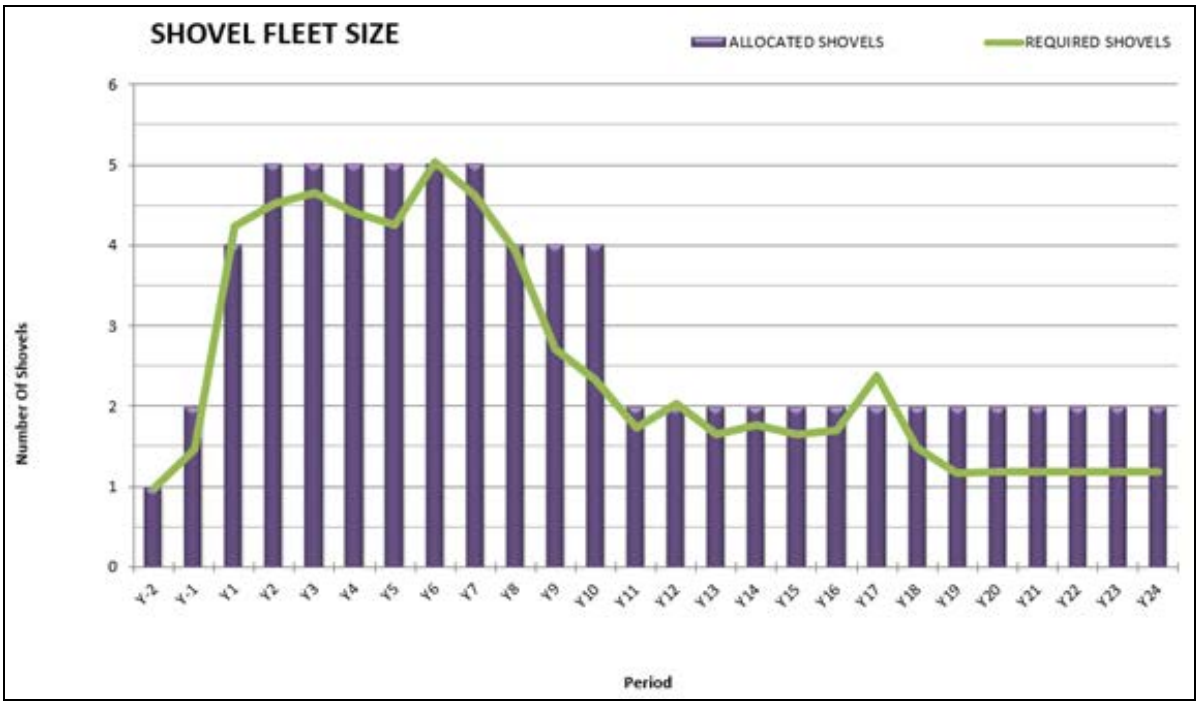


Figure 47: Shovel Fleet Size

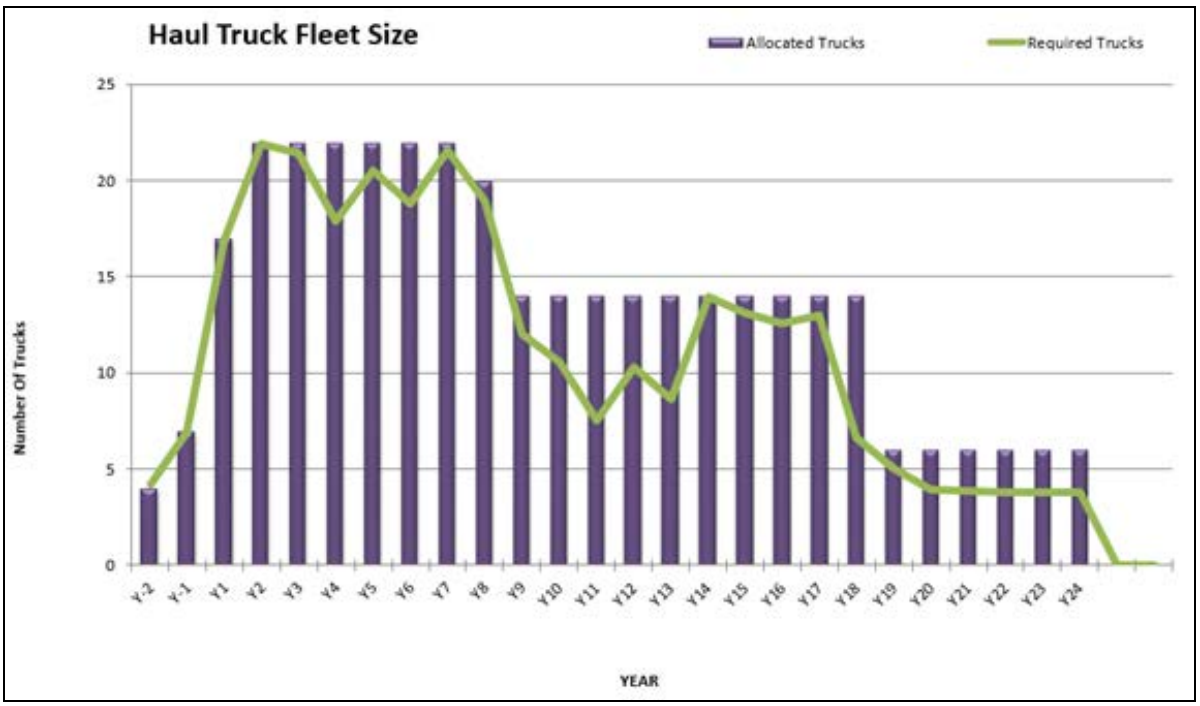


Figure 48: Haul Truck Fleet Size

16 Recovery Methods

The mine production plan requires the plant to process 40,000 tonnes/day (or 14.6 million tonnes/a) for a 24-year mine life. A very high proportion of Supergene mineralization will be processed in the first half of the plan with a transition after Year 11 to predominantly Primary Sulphide feed. Leach Cap material will be stockpiled until the end of the mine life for separate processing though the plant. Figure 49 shows a breakdown of the mineralization domains according to the conceptual mine production presented in Table 41 and Figure 41.

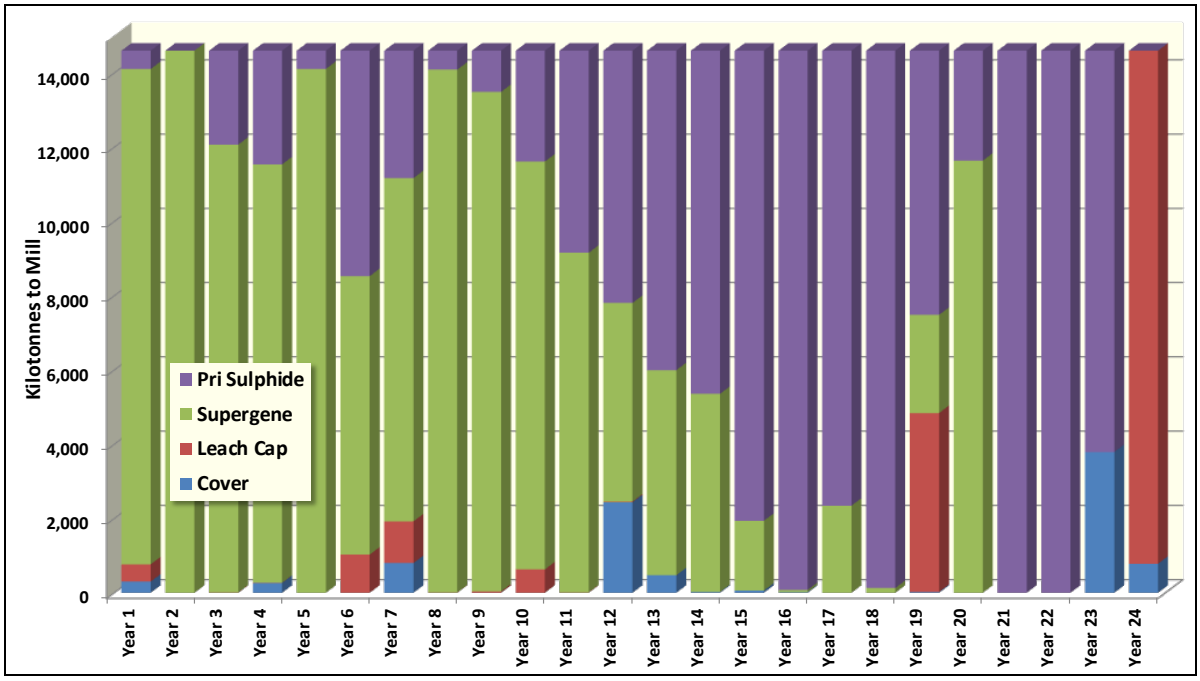


Figure 49: Mill Feed Blend of Mineralization Domains for Life of Mine

The process plant will generate a bulk, copper-molybdenum flotation concentrate which will then be separated into a saleable molybdenite concentrate of >32% molybdenum as well as a 20% to 30% copper concentrate with minor/no penalty elements.

At this stage, treatment of the molybdenite concentrate to reduce the copper content level is not included in the flowsheet, but may be required to achieve saleable impurity levels. This should be defined in the future with additional metallurgical testwork.

The effect of higher grade Supergene material in the first few years is evident in the trend of copper and molybdenum head grade values shown in Figure 3. After Year 5, copper head grades remain below 0.5% and drop to below 0.3% in Year 11 until the end of the mine life. The only composite Primary Sulphide sample tested was 0.3% copper, so metallurgical performance and copper concentrate quality are a concern when the head grade drops to below 0.2% copper for the last five years of the mine life.

It is recommended that additional samples be tested to confirm expected performance, in particular Primary Sulphide samples below 0.2% copper.

Figure 50 shows molybdenum head grade remains relatively stable while processing both Supergene and Primary Sulphide mineralization domains.

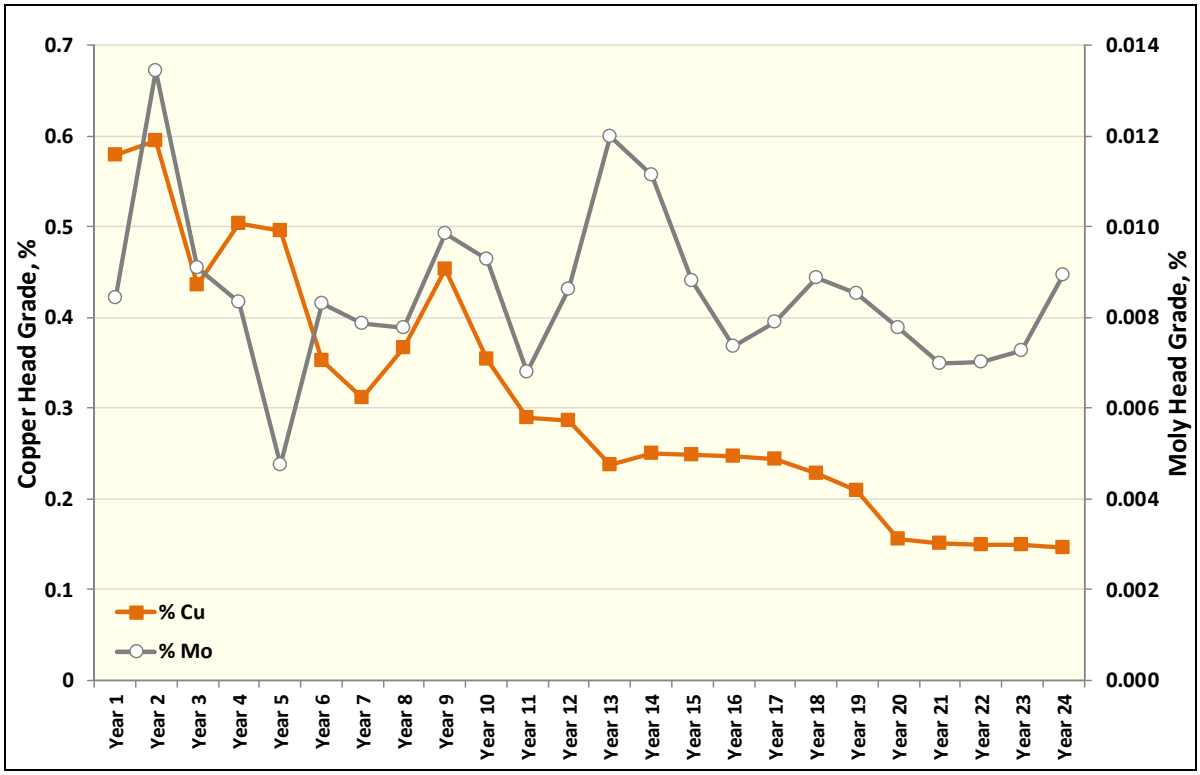


Figure 50: Mill Feed Grade of Life of Mine

16.1 Process Flowsheet

Based on the testwork results reported to date, the Antilla mineralization domains appear to be suitable for a flowsheet involving bulk concentrate flotation of copper and molybdenite, followed by molybdenite separation into saleable molybdenite and copper concentrates.

A possible process flowsheet is shown in Figure 51 and involves the following stages:

- Crushing/grinding to a P_{80} size of 100µm
- Bulk flotation of a copper-molybdenite concentrate
- Regrinding of the bulk concentrate prior to cleaning
- Conditioning with NaHS prior to copper-molybdenite separation
- Three or more stages of molybdenite cleaning
- Tailings from molybdenite separation circuit is the final copper concentrate
- Thickening of both concentrates
- Filtering of both concentrates
- Drying of molybdenite concentrate before bagging
- Tailings thickening prior to deposition in the tailings storage facility (TSF)

The availability of the primary crusher is expected to be 75% and the remainder of the plant is 92%.

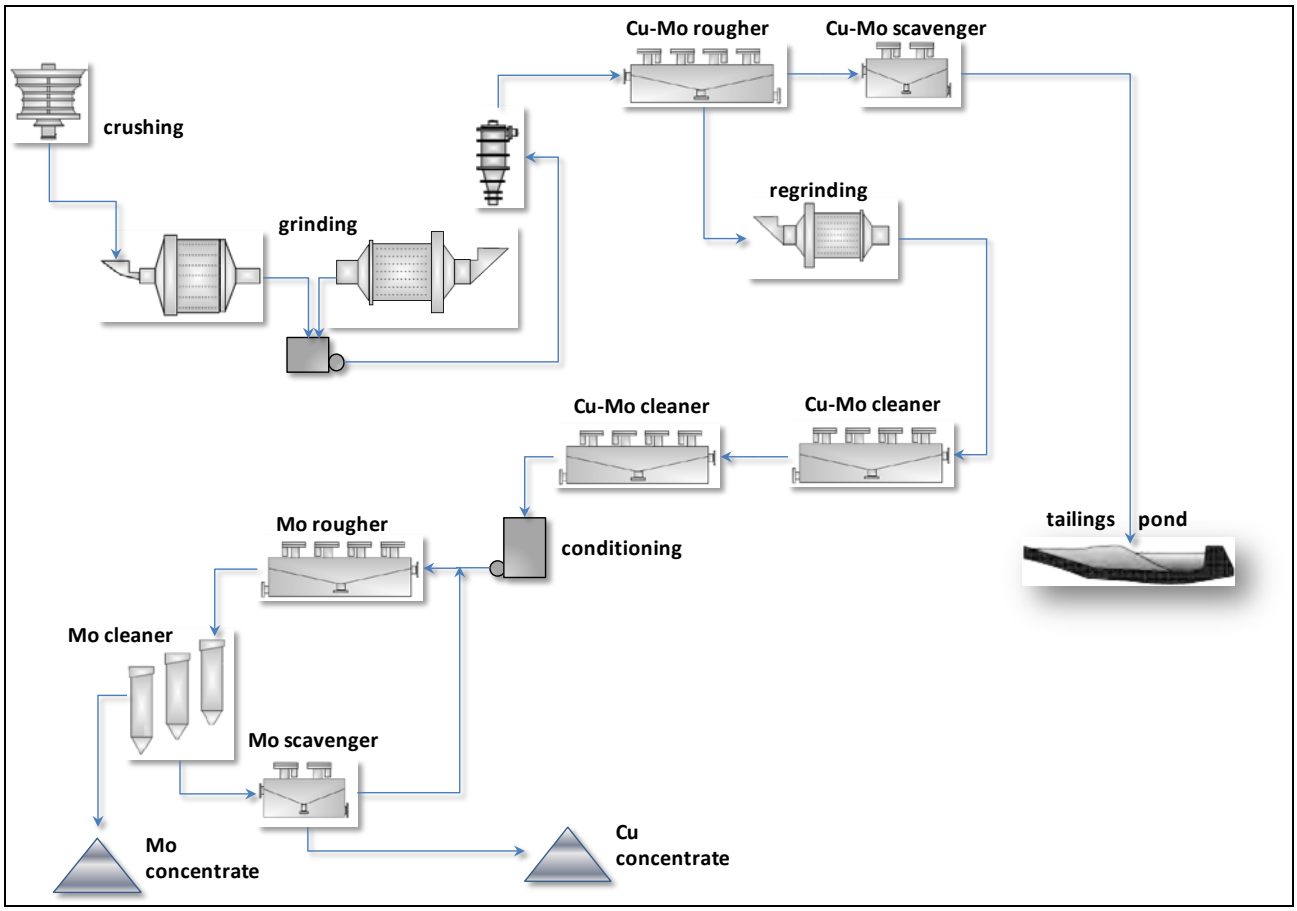


Figure 51: Schematic of Possible Process Flowsheet for Antilla Project

Due to local topography, it is expected that the copper concentrate will be pipelined to a remote filter plant located at the truck loadout site. Water from the filter plant may be returned to the plant or disposed of locally.

Due to limited comminution testwork, details on the crushing and grinding circuit are very preliminary. However, it appears likely that at the required plant capacity, primary crushing will be followed by a semi-autogenous grinding (SAG) mill and ball mill. This is based on the indication of ore hardness from the two BWi test results. A number of alternate crushing/grinding flowsheets exist which should be evaluated. It is likely that a pebble crusher to handle oversize in the SAG mill discharge will not be required.

Reagents to be used in the flowsheet include:

- Lime (as a pH regulator)
- Collectors (xanthate, 3418A)
- Depressants (sodium cyanide, sodium metabisulphite, sodium silicate)
- Frother (MIBC)
- Sulphidizing agent / redox control (sodium hydrosulphide)
- Flocculant

Optimized reagent additions and locations in the process flowsheet remain to be determined based on additional metallurgical testwork. Similarly, abrasion test results are needed to estimate the wear rate of grinding mill liners and media. (In the process operating cost estimate of Section 20, an abrasion index for moderate hardness material was assumed.)

16.2 Expected Metallurgical Performance

To date, only samples of Primary Sulphide and Supergene mineralization domains have been tested. Based on the Certimin 2013 testwork results on the two composite samples, the grade of the copper and molybdenite concentrate and recoveries shown in Table 45 are expected for the two main domains.

Table 45: Summary of Expected Concentrate Grade & Recovery

Concentrate	Copper		Molybdenite	
	Cu	Recovery	Mo	Recovery
Primary Sulphide	20%	85%	32%	65%
Supergene	30%	85%	40%	70%
Cover/Overburden*	25%	80%	32%	65%
Leach Cap*	25%	75%	32%	65%

* No testwork completed to date to support these estimates

Primary Sulphide reported 85% recovery to a 20% copper concentrate with 32% molybdenite concentrate at 65% recovery. Tests on supergene mineralization show 80% recovery to a higher 35% copper concentrate due to the greater percentage of cyanide-soluble, secondary copper minerals present in the sample. Molybdenum recovery was similar at 70% to a 40% grade concentrate.

To estimate the expected Supergene copper concentrate grade at a higher recovery, cleaner flotation kinetics testwork was relied upon (see Figure 52). The open circuit cleaner tests by Certimin show after 70% of the copper was recovered, the cleaner circuit concentrate grade drops dramatically with extended flotation times (with slower flotation rate copper minerals floating). The shape of the grade-recovery curve from the cleaner kinetics test was applied to the locked cycle test results to estimate the change in concentrate grade from 80% recovery (test reported result) to 85% recovery.

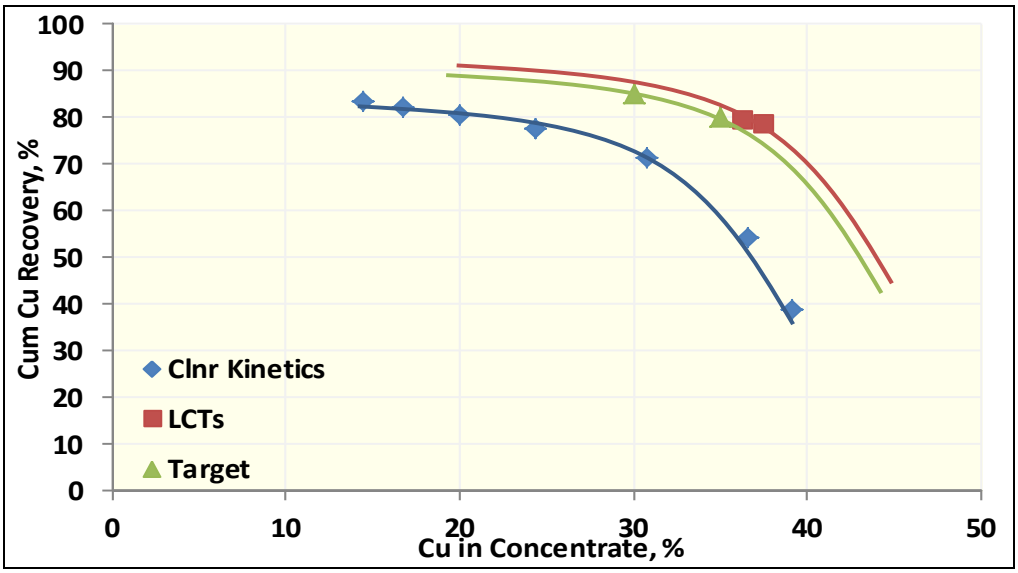


Figure 52: Extrapolation of Supergene Grade-Recovery Curve Based on Cleaner Kinetics

Applying the cleaner test grade-recovery curve suggests that the final concentrate should grade 30% copper at 85% recovery. However, even higher recoveries were expected to rapidly decrease the concentrate grade to below 20% copper. The application of the cleaner test results in this manner should be investigated in future metallurgical testwork programs. For the conceptual mine production plan discussed herein, expected Supergene recovery is set to 85% with a 30% copper grade concentrate being produced. Molybdenite performance was not modified.

For the Cover/Overburden and Leach Cap domains, the expected recoveries shown in Table 46 are based **solely** on the copper speciation data available and the overall copper grades. Based on the average copper speciation results included in the block model, the relative amounts of acid-soluble, cyanide-soluble and residual copper minerals are similar to that observed in Sample A from the Certimin 2013 testing (see Table 46).

It can be seen that for the Cover/Overburden, Leach Cap, and Primary Sulphide domains, the majority of copper mineralogy appears to chalcopyrite (residual copper), which is readily recovered by flotation. Additional mineralogical analysis such as QEMSCAN should be undertaken to confirm this.

The Supergene results show a much greater percentage of cyanide-soluble, secondary copper minerals, which result in a higher copper concentrate grade.

The recoveries and grades estimated in Table 46 reflect the apparent similarity in copper mineralogy and head grade to the two tested domains: Primary Sulphide and Supergene. It is recommended that samples of Cover and Leach Cap material be tested to confirm the estimates provided in Table 46.

It is not expected that the copper concentrate will incur any penalties based on the chemical assays available for the bulk copper-molybdenite concentrates (see Table 19). Smelter contracts will pay for 96.5% of the contained copper but likely not pay for any precious metals, due to their low content.

Molybdenite concentrate may require additional treatment to reduce the copper content and possibly zinc content as well. This should be determined in future testwork.

Table 46: Summary of Copper Speciation Results by Mineralization Domains

	Block Model Values				Met Sample A	Met Sample B
	Cover/OB	Leach Cap	Pri Sulphide	Supergene	Pri Sulphide	Supergene
Acid Soluble Cu	15%	21%	9%	19%	17%	25%
CN Soluble Cu	24%	26%	18%	61%	10%	57%
Residual Cu	61%	53%	73%	20%	72%	18%

17 Project Infrastructure

The property is accessible via 60-kilometre-long gravel road from the village of Santa Rosa on the paved Carretera Interoceánica Highway. The infrastructure to be developed for the project includes roads, water and power supply, mine waste containment and storage facilities, accommodation complex, support buildings, support facilities (i.e., fuel storage, warehousing etc.), and ore processing facilities. Infrastructure development at Antilla pose significant logistical and practical challenges which is compounded by the site remoteness, distance from major urban areas, high altitude, and rugged terrain.

The majority of the site buildings, including a permanent camp, will be located adjacent to the plant site area. The rock storage facilities will be located in the vicinity of the pit area. Mill feed will be stockpiled at a location adjacent to the process plant and reclaimed throughout and at the end of the mine life. The mill feed will be crushed and milled at the process plant in order to produce a final concentrate. The concentrate will be pumped to a remote filter plant / truck loadout site southeast of the pit area, where it will be loaded and truck hauled to the final destination. Tailings will be delivered via a pipeline to the tailings storage facility located in the Huancaspaco River basin southwest from the process plant.

The overall project site layout is presented in Figure 53.

17.1 Roads and Logistics

Two main routes are proposed for transportation of goods to and from the Antilla project site.

For the study discussed herein, it is assumed that the route from the port of Callao (Lima) will be utilized for delivery of supplies, equipment, and personnel during construction and through-out operations. It may also be possible to utilize the port of Marcona, which is located closer to the Antilla project site, for importation of some of the equipment and operating supplies.

Copper concentrate is assumed to be transported by truck from the Antilla site to the port of Marcona where it will be offloaded and shipped to the end user. The total route length is 533 kilometres and includes 60-kilometre-long gravel road from the Antilla site to the village of Santa Rosa and a paved Carretera Interoceánica Highway, as illustrated in Figure 54.

Molybdenum concentrate is assumed to be trucked 1,757 kilometres to a roasting plant in Chile, via a route that includes the 60-kilometre-long gravel road to the village of Santa Rosa and the paved Carretera Interoceánica Highway and Panamericana Sur highways.

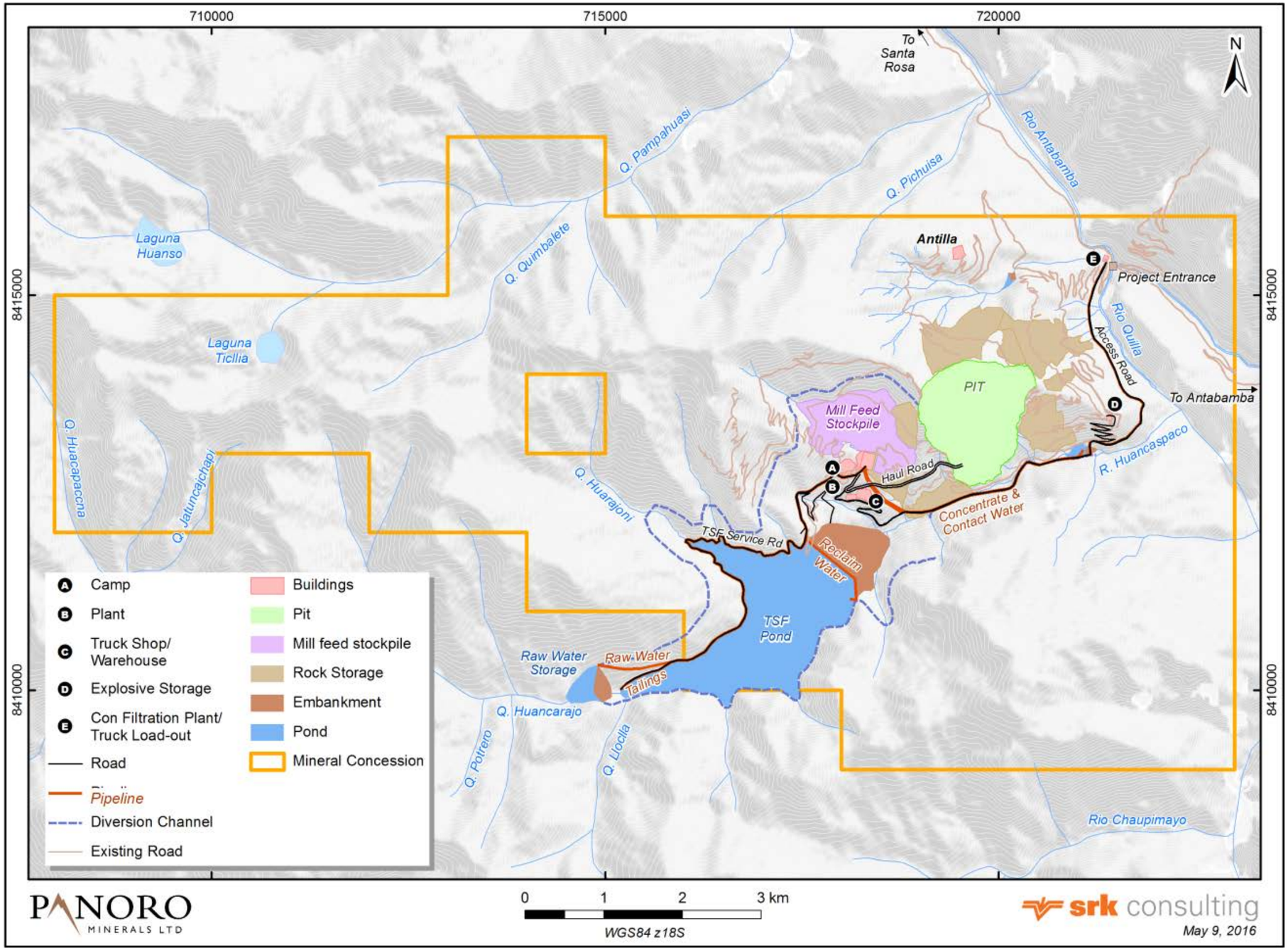


Figure 53: Overall Site Layout

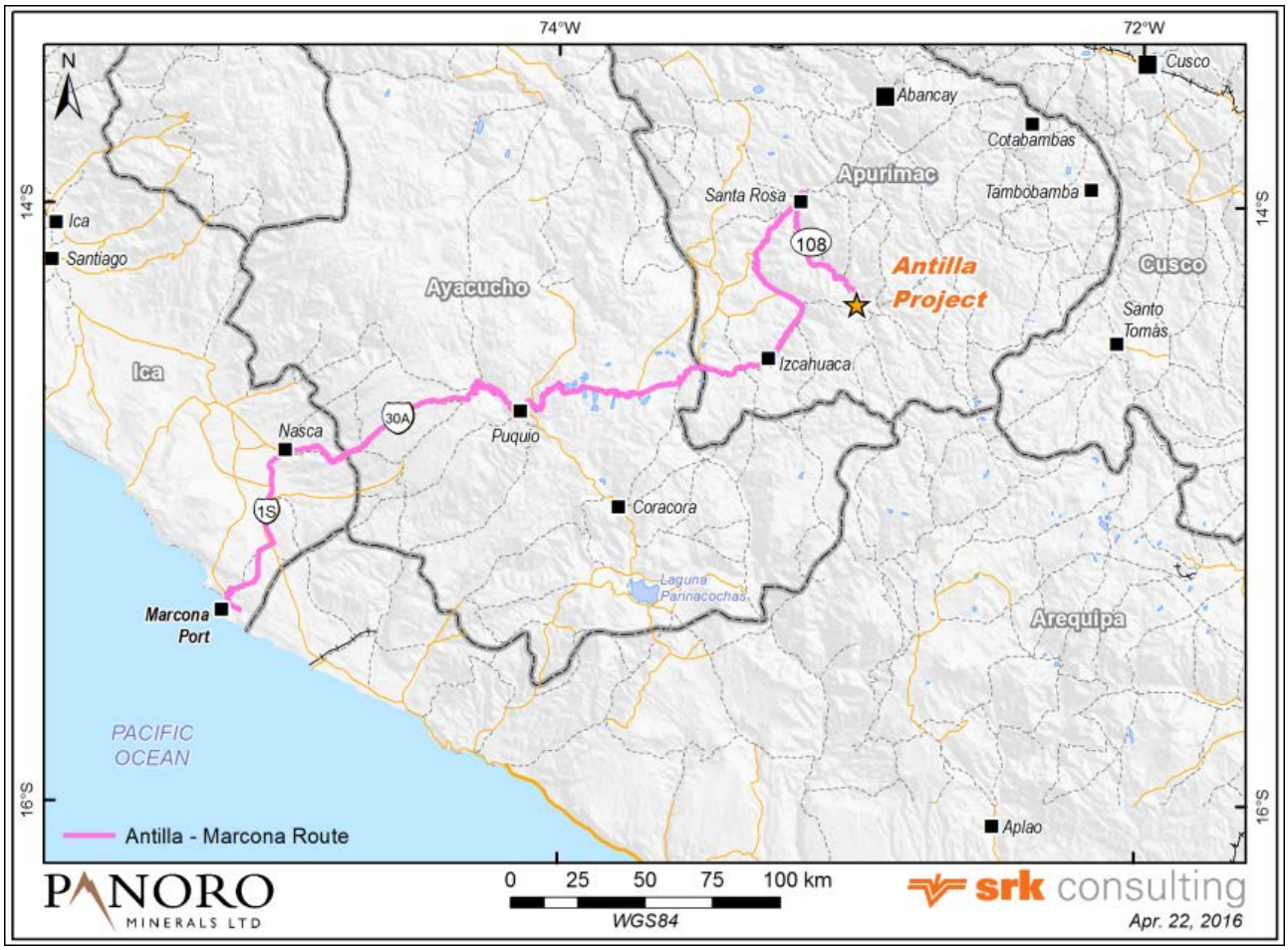


Figure 54: Copper Concentrate Transport Route

17.1.1 Site Access Road

From Highway 26 at Santa Rosa, the Antilla project site is accessed by 60-kilometre-long gravel and dirt road, approximately 4 to 5 metres wide. Although a public road, it is not maintained as well as paved roads. Road upgrades are necessary to allow a safe handling of traffic in and out of the Antilla site and these include improvements to drainage (road ditch and culverts at stream crossings), and switchback radius enhancements near the village of Pocohuanca.

17.1.2 On-Site Roads

The proposed mine access road is 8 kilometres long and it will be constructed to connect the mine, process plant, residential camp, office, maintenance and repair facility, and warehouse with the site access road. The road will be 8 metres wide with the maximum grade of 12%. It would be suitable for transportation of mining equipment, fuel trucks, mobilization of construction equipment, and ongoing operational requirements. The mine access road will partially accommodate the concentrate pipeline and process water pipeline. Road construction includes two bridges for crossing the Huancaspaco River to avoid potential rockfall hazards downhill from the Antilla open pit. The river itself will serve as a buffer zone to prevent falling rock from reaching the roadway. On-site roads are depicted on Figure 53.

The other on-site roads include:

- Mine haul roads connecting the pit with the process plant and the tailings embankment during its construction. Mine haul roads are presented in Section 15.0.
- The TSF service road, which will be 6 metres wide to accommodate tailings and process water pipelines and to provide access to the tailings pond and fresh water storage reservoir.
- Access to explosive storage area, via a 4-metre wide gravel road.

17.2 Power Supply and Distribution

The Antilla mine operation is estimated to require a power supply of approximately 65 MW. Power will be supplied from the Peruvian national grid system (power transmission line L.T. 220kV Machu Picchu - Abancay - Cotaruse). A 50-kilometres-long transmission line will connect the Cotaruse substation to the main Antilla substation, which is to be built near the project's process buildings. The proposed alignment for the 220 kV line (Figure 55) and cost estimate was provided by PANAPEX S.A. (2014).

From the Antilla substation, electrical power will be distributed to the plant buildings and primary crusher, residential camp and ancillary buildings, various pumping stations, and concentrate filtration plant, at either 25 kV or 13.8 kV depending on the power demand.

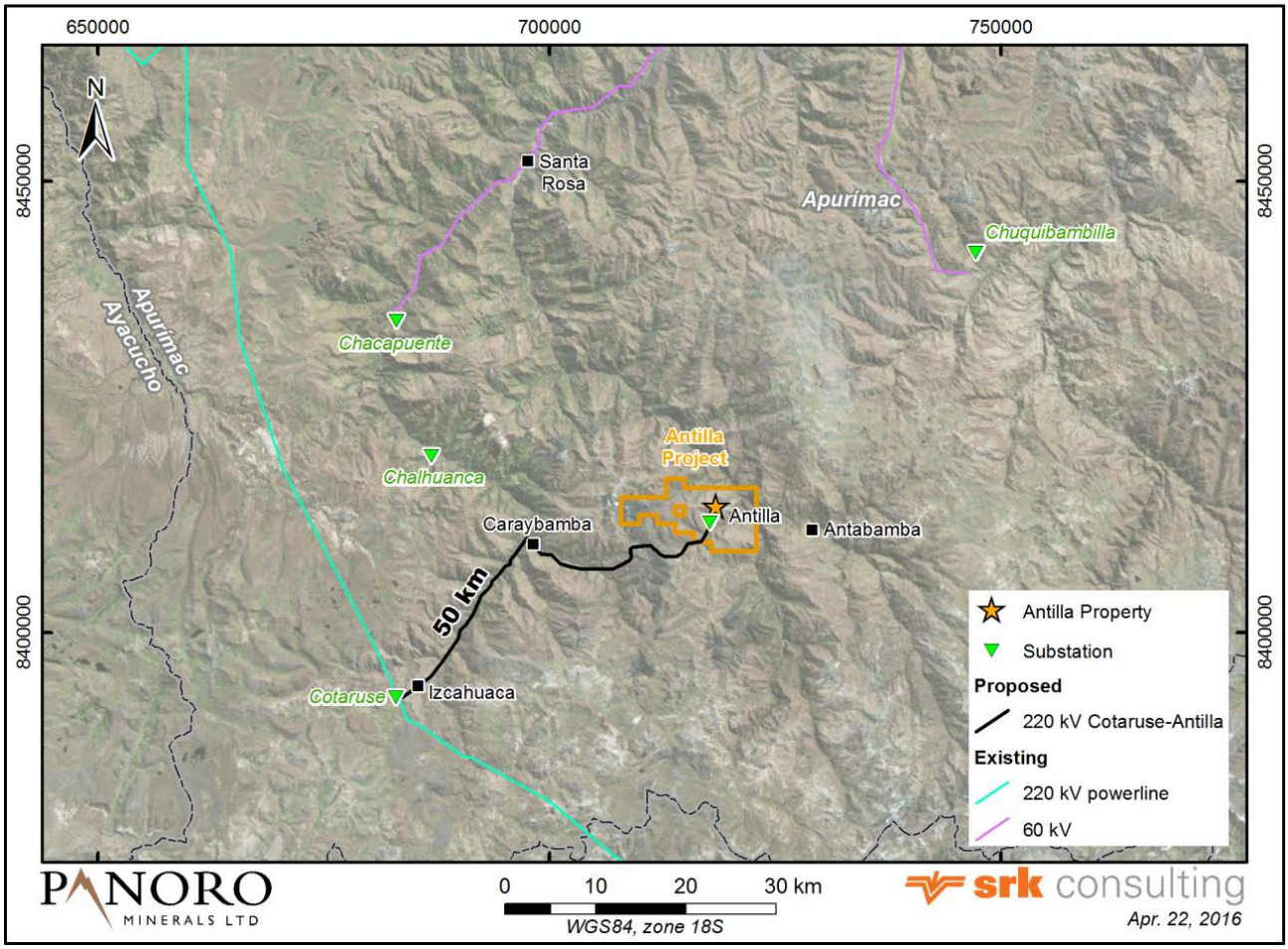


Figure 55: 220 kV Overhead Power Transmission Line Route

17.3 Water Supply and Management

Surface water will be used to supply all of the water needs for the Antilla project. This water can be obtained through the collection of runoff water during the wet season.

Tailings water is planned to be reclaimed from the tailings storage facility pond utilizing a floating pump and a pipeline to deliver the water to the process water tank. Tailings reclaim water is the source of most of the process plant water requirements. The process water make-up requirements will be sourced from the contact water collection pond and the fresh water reservoir.

The surface water management plan for the Antilla project will preserve to the maximum extent possible the non-contact status of surface runoff. Both contact water and non-contact water will be used in the process plant operation and will be transferred throughout the project via conveyance pipelines.

Contact water consist of runoff from construction-related areas, such as the pit, RSF, and tailings embankment, will be collected immediately downstream of the tailings storage facility in the Huancaspaco River valley collection pond and used as process plant makeup water. A preliminary site water balance indicates that contact water can meet most of plant makeup water demand, and excess water discharges are only expected at the end of the mine life. No allowance for a water treatment plant has been considered in this study.

Non-contact or fresh water consists of storm water runoff from the upper catchment areas within the project site, i.e., north of the tailings storage facility, plant site area, and rock storage facility, will be directed via a diversion channel to a fresh water reservoir located upstream of the tailings storage facility in the Huancaspaco River valley. The fresh water reservoir has a designed capacity of 4.2 million cubic metres and will be charged during the wet season. From this reservoir, water will be distributed to the processing plant, workshop, offices, camp, and the kitchen (via potable water filtration system).

The fresh water diversion system includes a south bypass channel from the fresh water reservoir to allow Huancaspaco River flows in excess of water supply demands to be routed around the south side of the TSF and released to the natural drainage downstream of the contact water collection pond. This south diversion ditch will also capture and divert runoff from the catchment area to the south of the tailings storage facility.

The contact water from the drainage area east of the proposed pit and below the northern rock storage facility will be collected in a settling pond and sediments removed before being discharged to the Antabamba River.

The sanitary sewer collection network and treatment facilities include pipelines, treatment tank, and pump stations and should be designed in accordance with local and regional government health regulations.

An allowance has been made for a potable water system including high density polyethylene piping to the process plant and ancillary buildings. It is anticipated that a pre-engineered packaged water treatment system will be utilized.

17.4 Tailings Storage Facility

The tailings storage facility siting and design was prepared by ATC Williams (ATC Williams 2015, 2016a, 2016b) and was reviewed by SRK.

17.4.1 Preliminary Design Assumptions

Approximately 350.4 million tonnes of mill feed (40,000 tonnes/day) are expected from the open pit during the project's 26-year life of mine development plan. The primary facts and assumptions used in the development of the conceptual tailings management system design were:

- The amount of tailings produced will be approximately equal to the amount of mill feed processed (this is conservative as the final product is a concentrate).
- Tailings physical characterization has not been carried out. Based on experience with similar tailings, an in place settled dry density of 1.20 tonnes per cubic metre (t/m^3) was used in the analysis. This converts to a required design volume of approximately 292 million cubic metres (Mm^3) of tailings.
- Beaching characteristics of hydraulically placed tailings was not considered, and storage calculations were based on struck level (i.e., horizontal) tailings.
- The project is located in a seismically active area.
- There has not been any geotechnical or hydrogeological characterization studies carried out in any of the areas considered for the construction of the tailings storage facility. Design concepts presented here are based on experiential judgement from projects in similar regions.
- There has not been any local or regional borrow characterization studies carried out for the project to determine candidate construction materials for embankment and/or containment structures.
- Mine waste rock is assumed to be the primary source of construction material. No geotechnical characterization has been carried out on this material. However, experiential judgement suggests a compacted in situ dry density of 1.87 t/m^3 for design purposes.
- No geochemical characterization studies have been carried out on the tailings, and only four preliminary acid base accounting tests have been conducted on waste rock to evaluate acid rock drainage (ARD) potential. This test data suggest the waste rock has no neutralization potential and two of the four samples tested is potentially acid generating (PAG). The fact that the samples tested have no neutralization potential also suggests that material with low sulphur content (such as > 0.02%S) may produce ARD.
- Regional climate data suggest that the site precipitation exceeds evaporation.
- Recycle water from the tailings storage facility will be used to supplement the mill water demand. Plant make-up water will be required mainly due to interstitial water retained in the settled tailings.
- Preliminary indications are that the contact water volumes are less than the plant make-up water requirements. It is assumed in this study that contact water can be utilized as plant make-up water rather than discharged to the environment. No allowance for water treatment has been considered at this time.
- At any given time, the tailings storage facility will have freeboard allowance of 1 metre. This value is not based on a hydro technical evaluation, but an arbitrary assumption.
- Work was completed using a 25-metre contour interval topographical mapping provided by Panoro.

17.4.2 Alternatives

A number of different tailings management strategies were considered including conventional low solids content slurry tailings, thickened tailings, paste tailings, and filtered tailings. Filtered and paste tailings were deemed to be economically prohibitive. Thickened tailings were not specifically excluded, but designing a containment facility allowing for conventional low solids content slurry was deemed more conservative given the limited state of information at this time.

The different containment systems for the slurry tailings evaluated included cyclone dam construction, quarry rock fill dam construction, and waste rock dam construction. For both the quarry and waste rock dams, environmental containment options included a conventional low permeability core and an upstream geomembrane liner. The high seismicity of the project region led to a decision to not consider a cyclone dam at this time. The abundance of mine waste rock, coupled with limitations of suitable waste rock dump locations led to the decision to use waste rock as the primary dam construction material. The steep terrain results in inefficient containment dam structures, and therefore the use of a low permeability core was discounted, as there is unlikely to be sufficient local borrow material available to construct it. Environmental containment for the dam would therefore be provided through an upstream geomembrane liner.

Different tailings management facilities were considered, ranging from sites immediately downstream of the open pit to areas as far away as 20 kilometres from the project site, in adjacent watersheds. The steep and variable terrain effectively only lends itself to the construction of valley fill containment structures, and generally the storage efficiency is low. The significant topographical relief requires considerable pumping heads, especially when crossing watershed boundaries. The preferred tailings management facility location was therefore a site immediately upstream of the proposed plant site and due west of the open pit.

Staged construction of the containment structure was considered given the extended mine life. Upstream raises were not considered due to the high seismicity of the area and Peruvian regulatory constraints associated with this method of construction. Centerline raises were not considered as the use of an upstream geomembrane liner makes this challenging. Staged construction will therefore be done using the downstream raise method.

Two different environmental containment strategies for the tailings storage facility were considered. The first assumed that the tailings storage facility basin comprises predominantly low permeability materials and that there is limited fractured bedrock present. The environmental containment for this scenario comprises construction of a keyway into competent bedrock, thereby sealing the upstream containment dam liner to bedrock. The alternate strategy entails complete lining of the tailings storage facility basin with a geomembrane liner, under the assumption that the foundation materials are highly permeable, and that deep groundwater seepage would result in unacceptable downstream environmental impacts. The design assumed the former case, i.e., no requirement for lining of the entire tailings storage facility basin.

17.4.3 Preliminary Design

Concept

The tailings storage facility containment dam is a compacted rock fill dam constructed predominantly from direct hauled mine waste rock, supplemented as needed by manufactured borrow materials. Environmental containment is provided using an upstream geomembrane liner placed onto a porous concrete curb overlying filter and transition zones, tied in to competent bedrock. The achievable upstream and downstream slopes when utilizing the porous concrete curb

and geomembrane technology are assumed to be 1.7H:1V and 1.75H:1V, respectively, based on engineering judgement. This containment dam technology has been under similar conditions at other projects.

The final crest elevation of the containment dam will be 3,834 metres, which translates to a maximum centreline height of approximately 290 metres. The crest length will be approximately 1,130 metres and the crest width adopted is 10 metres. The total volume of material contained in the dam structure (above the existing surface) is about 76 Mm^3, and the containment volume for tailings behind this dam is about 295 Mm^3. This translates to a storage efficiency (i.e., volume of tailings/volume of dam) of about 3.9.

Foundation Preparation

The containment dam will be constructed in the Huancaspaco River valley. To allow construction, a temporary coffer dam is required to cut off the flow and allow water to be diverted through active pumping. The coffer dam will be an earthen structure with an upstream geomembrane liner keyed into the foundation. At this point, this coffer dam has not been sized, and the pumping requirements have not been evaluated.

With the dam foundation drained, the entire dam footprint will be excavated to expose suitable foundation materials. Without knowledge of the actual foundation conditions, it was assumed that an average of 2 metres of topsoil and unsuitable material would be required to be stripped. This material would be disposed of in a stockpile(s) for later use in site reclamation at an undefined location assumed about 2 kilometres from the site.

A keyway for the upstream liner to tie into bedrock will be excavated by removing any fractured rock. The keyway was assumed to be approximately 10 metres deep (below the foundation level) and the cross-sectional area of the keyway excavation was assumed to be 150 square metres (m^2). Material excavated from the keyway would be reused as dam construction fill material. No allowance has been made for grouting of the foundation. It is assumed that the base of the keyway would be competent in-tact rock which, once tied in with the geomembrane liner, would yield a sufficiently tight seal against seepage from the tailings storage facility.

No dedicated shear key has been allowed for in the design at this time. The size of the structure and profile of the foundation are assumed to be sufficient to address the potential for sliding along the foundation. This would have to be confirmed through proper engineering analysis as the project advances.

Rock Fill Placement

The bulk of the containment dam will be constructed using run-of-mine waste rock, directly hauled from the open pit. This material will be placed by mine haul trucks, in lifts of approximately 1 metre thick and compacted using vibratory methods. Given the height of the dam, and the seismicity of the area, considerable compaction is required to ensure long-term stability of the structure. Over the life of mine, there are periods when the mine waste rock production cannot match the dam construction demands, and any such shortfall will be supplemented through the development of local quarry rock, assumed to be available within 5 kilometres from the dam site, or by advancing the dam construction schedule to make use of periods when excess mine waste rock is available. The latter option, i.e., advancing dam construction to utilize mine waste rock when available, has been assumed in the Antilla project economic analysis. Adjustments to the mine schedule, may also offer opportunities for minimizing the need of quarry rock.

Although the limited information pertaining to waste rock suggests that ARD may be a concern, no environmental containment underneath the dam has been provided for at this time.

Geosynthetic Liner Installation

The upstream face of the containment dam, extending all the way to the base of the keyway trench is lined with a 2-millimetre thick high density polyethylene geomembrane liner placed over deformable concrete curbs. The curbs are constructed over two zones of filters and transition materials respectively 3 metres and 5 metres thick. These filter and transition materials will be manufactured on site through crushing and screening using either waste rock or local quarry rock.

Dam Drainage

To ensure that any leakage through the liner, or groundwater pressure from beneath the foundation, does not result in buildup of excess pore pressure in the containment dam compromising its stability, a basal finger drain is included in the design. 200- and 300-millimetre diameter perforated high density polyethylene pipes are placed inside manufactured filter material (i.e., gravel from crushing and screening or a local borrow source), which in turn is wrapped in 270 g/m^2 non-woven geotextile.

Dam Raises

The TSF containment dam will be raised in six stages, including the starter dam. Details of the downstream staged construction are summarized in Table 47. However, it is important to note that in actual fact dam construction will be continuous as the rock fill shell, which forms the bulk of the structure, is constructed from mine waste rock that is hauled directly to the dam. Filter material, transition material, deformable concrete curb construction, and liner placement will be campaigned as per the schedule in Table 47.

Table 47: Summary of Staged Construction Tailings Storage Facility Containment Dam

Stage	Storage Period (Year)	Construction Complete (Year)	Crest Elevation (masl)	Embankment Volume (Mm^3)	Contained Tailings (Mt)
E10-1 (Starter)	0 – 1	-1	3,642	3.64	14.7
E10-2	2 – 3	2	3,686	8.84	44.0
E10-3	4 – 5	4	3,712	13.35	73.1
E10-4	6 – 10	9	3,756	26.17	146.1
E10-5	11 – 18	17	3,805	51.65	263.5
E10-6.1	19 – 26	25	3,834	75.96	353.5*

* The small difference between the actual contained volume of tailings and the required design volume is due to the precision of the design completed to date and is not considered material.

Deposition Plan

Tailings will be pumped from the plant site at a solids content of about 30% (by weight) using conventional centrifugal pumps. Tailings deposition will be down-valley, i.e., the deposition points will be placed upstream of the containment dam such that the permanent supernatant pond will be located immediately upstream of the dam. At this time a detailed deposition plan has not been developed as all volumetric calculation assume a struck (i.e., horizontal) tailings elevation.

Water Management

A tailings water balance has not been developed. The tailings storage facility will have a permanent supernatant pond that will allow for maximum recycling of water to the processing plant. Since the area has a positive climatic water balance, i.e., mean annual precipitation exceeds mean annual evaporation, there will be excess water available that is assumed to provide part of the necessary mill make-up water demands. It has also been assumed that there would be no need for discharge of excess water to the environment and therefore there is no allowance for water treatment. The tailings

storage facility does not have an operational spillway. Recycle water will be returned to the plant site using a decant pump and pipeline.

To minimize contact water, upstream runoff will be diverted around the tailings storage facility using a series of engineered contour channels.

Closure and Reclamation

The tailings is assumed to be acid generating and therefore a low infiltration cover is required. At closure, the tailings 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. All the soil materials 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 waste rock piles. 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 tailings storage facility to attenuate and direct water to a permanent overflow closure spillway on the containment dam.

17.5 Rock Storage Facilities

The rock storage facilities are discussed in Section 15.8.2.

17.6 Mill Feed Stockpile

The mine plan includes provision for a mill feed stockpile as discussed in Section 15.8.3.

17.7 Site Buildings and Facilities

Project buildings and facilities consist of mine, process and tailings facilities, and ancillary buildings.

17.7.1 Mine Facilities

The mine service facilities include trucks maintenance and repair shop, fuel station and explosive magazine.

The truck shop will house offices, tool cribs, machine shop, welding repair area, electrical shop, three large equipment repair bays, two light vehicle repair bays, light vehicle parts storage, compressor room, lubricants storage room, emergency vehicle parking, first aid room, tire repair area, and exterior wash bay.

Fuel storage consists of a horizontal cylindrical carbon steel tank with a 75,000-litre capacity located within a bermed containment area lined with an impermeable membrane. The fuel storage area will be clearly marked and will have a perimeter fence and fire extinguishers.

Explosives and blasting agents required for mining will be stored at permitted locations to the southeast of the pit, at 1.0 kilometre safe distance from personnel and operating activities. All explosive related structures will be located within an appropriately barricaded and fenced area in accordance with the applicable standards. The actual magazines will be of the pre-fabricated container type including a powder magazine and ammonium nitrate storage silo.

17.7.2 Process Facilities

The process facilities described in Section 16 will be complemented with the assay laboratory, concentrate handling system, and concentrate filtration plant and load out building.

The assay laboratory will handle samples from the mine and process plant and include sample receiving, sample drying, sample preparation, metallurgical lab, wet lab, fire assay, electrical and mechanical rooms, a computer room, men's and women's restrooms and locker facilities, a lunch room, a loading dock, and various offices.

The copper concentrate will be pumped via a 6.5-kilometre-long pipeline to the filtration plant. Copper concentrate will be loaded into storage bins prior to loading into trucks. The load-out facility will have a covered area and loading docks to load highway transport trucks.

Copper concentrate transportation will be carried out using 40-tonne capacity B-train trucks. Before the trucks depart truck loadout facility they will be weighed at the truck scale. All trucks will be systematically registered so that the delivered concentrate weighed at the storage port reconciles with that which left the project site.

Copper concentrates are envisioned to be stored in the port of Marcona, while awaiting vessels for ongoing shipment to smelters in Asia. Molybdenite concentrate, is assumed to be packaged in 1-tonne bulk bags, loaded into trucks and transported to a roasting plant located in Mejillones, Chile.

17.7.3 Tailings Disposal Facilities

Tailings facilities are complemented with the tailings and reclaim water pipelines and pumps.

17.7.4 Ancillary Buildings

The ancillary buildings include administration offices, residential camp, warehouse, water and wastewater management, communications centre, and security guard house. Structures will be pre-fabricated, flat-packed and sent to a site ready for quick erection. In most cases, the buildings will be double storied to limit their footprint.

The administration buildings will be a pre-engineered, steel-framed structure and will provide offices for administrative and technical staff, including management, training, accounting, safety, and security.

A residential camp is sized to accommodate approximately 350 employees and will provide designated accommodations for technical, management, and labour personnel. The camp will include dining and food preparation facilities, laundry, recreational and medical facilities. The camp will be located within walking distance of the mine administration buildings. It is envisioned the permanent camp will be built early in the construction period so that it can be used initially for construction and made available to operations and commissioning staff as they are assigned to the site. It is assumed in this study that construction camp will be located near the Antilla village.

The warehouse building will be located near the process plant facilities with the storage area that will contain critical spares for the process and mining operations. Offices in the building and workshop areas are included.

The communications centre includes a satellite system to provide a telephone/fax and network servers for email, internet, and data services. The entertainment systems at the camp will consist of multiple satellite TV channels. The mine site telephone system will link all essential areas of the site, and through the satellite system, to outside of the project site. The mine radio system will include one base station and a number of repeater stations for local communications and connection with the mine dispatcher.

The main mine guard house will be located at the entrance to the mine site on the access road, near truck load-out facility. The main security office will be located in the administration building.

18 Market Studies and Contracts

18.1 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 smelter charges, including treatment, refining, penalty details, payment timing, metal accountability, and other contract terms, as well as transportation, port, and shipping costs.

18.2 Principal Assumptions

The Antilla project will produce a copper and a molybdenite concentrate.

For this marketing assessment, assumptions are based on metallurgical data with respect to the characteristics of the copper and molybdenum concentrates, 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$4.50 per pound, with an average of US$3.13 per pound. Figure 56 shows the trend in copper prices since 2011.

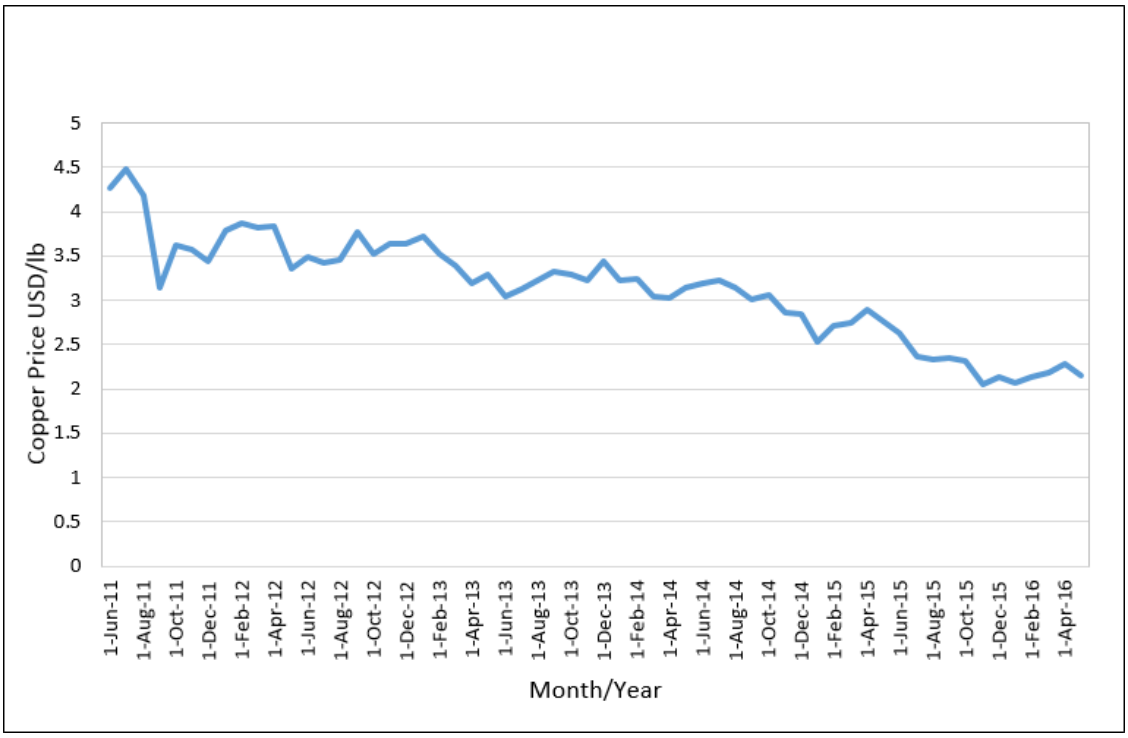


Figure 56: Five Year Copper Monthly Price (Source: COMEX, 2016)

A copper price of US$3.00 per pound was selected for the duration of the project life and represents 96% of the five-year trailing price for copper as of the date of this report. This price is supported by a long term (2020+) copper price forecast by a large banking institution. It is assumed that the copper concentrate produced at the Antilla mine at 8% to 10% moisture content would be marketed to smelters in Asia.

The molybdenite concentrate at 5% moisture content is assumed to be sold to roasting plant in Chile. The concentrate price is based on the published price for molybdenum oxide, less a discount incurred to roast the sulphide concentrate to oxide. This study has been carried out using a price of molybdenum of US$12.00 per pound. This is supported by a long term (2020+) molybdenum price forecast by a large banking institution.

Commercial terms assumed for sale of two concentrates are summarized in Table 48 and Table 49. Price participation and possible transport losses were excluded in this study.

Table 48: Commercial Terms for Sale of Copper Concentrate

Parameter	Unit	Copper	Remarks
Realization Charges			
Smelter Copper Deduction	units	1	
Payable Copper in Concentrate	%	95.0 - 96.7	Four metallurgical types
Concentrate Selling Costs			
Concentrate Transport to Port	US$/wmt con	40.0	Equivalent to $0.075/t km
Port Charges	US$/wmt con	10.0	
Insurance & Marketing	%	0.075	% of concentrate value
Ocean Freight	US$/wmt con	50.0	
Smelting Charges	US$/dmt con	75.0	
Refining Charges	US$/lb	0.075	

Table 49: Commercial Terms for Sale of Molybdenum Concentrate

Parameter	Unit	Molybdenum	Remarks
Realization Charges			
Roaster Molybdenum Deduction	units	0	
Payable Molybdenum in Concentrate	%	100	
Concentrate Selling Costs			
Concentrate Transport	US$/wmt con	271	Equivalent to $0.15/t km
Insurance & Marketing	%	0.075	% of concentrate value
Roasting Charges	US$/lb	1.62	13.5% of molybdenum price

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 concentrate freight, treatment and refining charges.

19 Environmental Studies, Permitting, and Social or Community Impact

19.1 Environmental Setting

Limited environmental baseline data has been collected for the site. Baseline studies have focussed 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. 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 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 concentrations.

19.2 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.

No formal consultation with any of the local communities has been completed, 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.

19.3 Regulatory Requirements

Prior to the initiation of mine development and processing, the Peruvian Environmental Regulations, administered by the Development of Mining and Metallurgical Activities, require the proponent to conduct a comprehensive environmental impact assessment. The environmental impact assessment must be approved by the General Environmental Directorate of the Ministry of Energy and Mines 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, 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 concentrate 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 50. The cost of conducting the environmental impact assessment and obtaining all of the necessary permits and approvals is estimated to be approximately US$1 million. This excludes development of the project description and execution of baseline studies.

Table 50: List of Regulatory Permits and Approvals Required

Authority	Mining Project	Transport of Concentrates	Transmission Line	CIRA	Closure Plan	Water 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			

19.4 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.

19.5 Mine Waste Geochemistry

Detailed geochemical characterization studies have not been carried out on mine waste materials for the Antilla project. Four waste rock samples were submitted 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 51 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 51: Samples Submitted for Acid Base Accounting Testing

Sample ID	Mineralogical Domain	Description
COV-ANT-1A	Surface Cover	Colluvium with quartzite clasts and some claystone clasts.
EL-ANT-2A	Leach Cap	Quartzite (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	Quartzite (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 quartzite (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 52 and the full report is included in Appendix C.

Table 52: 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 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.

19.6 Mine Closure

19.6.1 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.

19.6.2 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 and chemical stability of the site such that there would be no long-term effects associated with the mining activities after operations cease.

19.6.3 Tailings Storage Facility Closure

The tailings are assumed to be acid generating; therefore, a low infiltration cover is required. At closure, the tailings 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 tailings storage facility to attenuate and direct water to a permanent overflow closure spillway on the containment dam.

19.6.4 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 tailings 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.

19.6.5 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 are 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. 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.

19.6.6 Processing and Crushing Plant Closure

At closure, all hazardous waste material from the processing plant and ancillary facilities will be gathered and disposed of at an off-site licensed hazardous waste disposal facility. The processing 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 processing 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.

19.6.7 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.

19.6.8 Concentrate and Tailings Pipeline Closure

All site pipelines will be placed on the surface with containment berms or open trenches as necessary. For the most part, pipeline routes are coincident with site roads. Therefore, the closure of berms and/or open trenches will coincide with road closure activities.

Pipelines not containing hazardous material will be dismantled and relocated to the on-site non-hazardous landfill site or transported off site for disposal in a licensed hazardous water landfill site if necessary.

19.6.9 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.

19.6.10 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.

19.6.11 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 tailings 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 tailings cover may have to occur many years after closure to allow for the surface to be trafficable.

19.6.12 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$91.7 million, as presented in Table 53.

Table 53: Closure Cost Estimate

Component	Cost (US$)
Tailings	36,298,032
Rock Storage Facilities	5,645,935
Open Pit Mine	88,167
Processing Plant	11,334,697
Roads	554,531
Concentrate Pipeline	10,313
Tailings Pipeline	10,197
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	**$56,445,032**
Indirect Costs (25% of Direct Costs)	$14,111,258
Subtotal	**$70,556,290**
Contingency (30% of Direct Costs)	$21,166,887
Total	**$91,723,177**

20 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.

20.1 Capital Cost Estimate

20.1.1 Capital Cost Summary

The capital cost of the proposed project has been estimated based on the scope defined in previous sections of this report. The following parties have contributed to the preparation of the capital cost estimate:

- Mining operations (Moose Mountain)
- Mineral processing (SRK)
- Tailings storage facilities (ATC Williams/SRK)
- Site infrastructure and services (SRK)
- Owner's costs (SRK)

The capital cost for the proposed project consists of direct and indirect costs, owner's costs and contingency. Direct project costs are developed for the four major components: mining operations, mineral processing, tailings storage facilities, and site infrastructure and services.

The pre-production initial capital costs have been estimated at US$603 million and sustaining capital costs at US$324 million. Both estimates include contingency. The accuracy of this estimate is considered to be within -20% to +50%.

The capital cost estimate is expressed in US dollars for the fourth quarter of 2015. A summary of the project capital costs is shown in Table 54.

Table 54: Capital Cost Estimate Summary

Description	Initial CAPEX (US$M)	Sustaining CAPEX (US$M)	Total CAPEX (US$M)
Mine Equipment	51	111	162
Mine Development	55	-	55
Processing	236	-	236
Tailings Storage Facility	25	145	169
Site Infrastructure and Services	110	9	119
Owner's Costs	28	-	28
Subtotal, before Contingency	**$506**	**$264**	**$770**
Contingency	97	60	157
Total Capital Cost	**$603**	**$324**	**$927**

Capital cost for the Antilla project was developed from the reference projects, budget quotations for major equipment, SRK/Moose Mountain/ATC Williams in house databases, and industry cost reference guides. Items not included in the capital cost estimate are as follows:

- Sunk costs
- Exploration costs
- Further, more detailed technical studies, including EIS, and prefeasibility and feasibility studies
- Permitting cost
- Escalation beyond Q4 2015
- Value added tax
- Working capital (included in economic model)
- Reclamation and closure costs (presented in Section 19.6.12)
- Interest and financing cost
- Foreign currency exchange rate fluctuations

Costs associated with political upheaval, government policy changes, labour disputes, permitting delays, weather delays or any other *force majeure* occurrences are also excluded.

Temporary construction facilities and services, construction equipment, freight, insurance, and engineering/procurement/construction management services are included in the estimate as indirect costs using factors applied to the sum of direct installed costs.

Contingency costs have been estimated using various percentages by major category and applied to the direct and indirect capital cost of a particular category. The initial capital contingency allowance of US$97 million is 16% of total initial capital costs. The US$60 million sustaining capital contingency allowance is 19% of total sustaining capital costs. The overall contingency allowance represents 17% of total capital.

20.1.2 Mining Capital Cost Estimate

Moose Mountain has provided costs specific to mining equipment and the operation of that mining equipment. Mining capital and operating costs are prepared as a scoping level estimate, or an American Association of Cost Engineers Class 5 estimate. A Class 5 estimate is defined as having an expected accuracy range of -20% to +50% and is generally based on factored or scaled costs. This is considered sufficient for a preliminary economic assessment.

The capital cost estimate for major mining equipment is based on quotes provided by vendors to Panoro. Moose Mountain reviewed these quotes and they were deemed sufficient based on benchmarks from similar operations at this level of study. Quotes were assumed to be before freight, but including assembly. Moose Mountain has applied a freight factor to each equipment piece, as well as separating the assembly cost (using a factor if assembly was not specifically provided by the vendor). Mine Equipment capital costs include estimates for freight and assembly.

Mine Development capital costs include all mining operating costs during the pre-production period prior to mill start-up. These include drilling, blasting, loading, hauling and GME costs related to pre-stripping material as well as costs for stockpile pad preparation and the construction of external haul roads required for start-up.

The mining fleet summary is shown in Table 55.

Table 55: Mining Equipment CAPEX

Major Equipment	Function	Unit Cost (US$'000)	Quantity Y-2	Y-1	Y1	Y5	Y10
Hydraulic Shovel - 15m^3	Loading Ore & Waste	4,727	1	1	4	3	3
Haul Truck - 144t *	Hauling Ore/Waste	2,198	4	6	17	13	12
Drill - Diesel Hydraulic - 270mm	Primary Drill	2,441	1	1	3	2	2
Drill - Diesel Hydraulic - 150mm	Secondary Drill, High Wall Drilling	1,040	0	1	1	1	1
Support Equipment							
Dozer - 306kW	Shovel Support - In Pit	1,156	4	4	8	8	8
Wheel Dozer - 372kW	Pit Clean Up, Shovel Support	1,173	1	1	1	1	1
Fuel / Lube Truck - 4000gal	Fuel Truck	1,221	1	1	1	1	1
Water Truck - 20 000gal	Haul Roads Water Truck	2,224	1	1	1	2	2
Grader - 221kW	Road Grading	946	1	1	2	2	2
FEL - 373kW	Multi-Tool, Tire Changes, Road Crush	1,030	1	1	1	1	1
Ancillary Equipment							
Excavator - 301kW	Utility Excavator	987	1	1	2	2	2
Mobile Screening Plant	Screening Plant	275	1	1	1	1	1
Jaw Crusher	Road Crush	497	1	1	1	1	1
Forklift - 10t	Forklift	192	1	1	1	1	1
Light Plant - 20kW	Light Plant	25	4	4	4	10	10
Crane - 40t	Crane, Truck Boxes	877	1	1	1	1	1
Crew Van - 15 Passenger	Crew Transport	45	2	3	3	3	3
Warehouse Truck - 1t	Warehouse Truck	45	1	1	1	1	1
Crew Cab Pickup	Crew Cabs, Supervisor Trucks	38	4	6	8	8	8
Service Truck	Maintenance + Overhauls	125	1	1	2	2	2
Welding Truck	Welding Truck	125	1	1	1	1	1
Picker Truck	Picker Truck	300	0	1	1	1	1

The mining capital cost is summarized in Table 56. A 20% contingency factor is applicable on Mine Equipment and Mine Development capital costs.

Table 56: Mining Capital Cost Summary

Description	Initial CAPEX (US$M)	Sustaining CAPEX (US$M)	Total CAPEX (US$M)
Mine Development	55.3		55.3
Mine Equipment	46.9	107.3	154.1
Other Capital	2.8		2.8
Freight	1.3	3.6	4.9
Total Capital Cost	**106.4**	**110.9**	**$217.3**

20.1.3 Processing Capital Cost Estimate

For the process capital cost estimate, a plant design capacity of 40,000 tonnes/day will be constructed on site to produce separate copper and molybdenite concentrates.

Based on copper and copper-molybdenum projects in Peru at similar plant capacities, SRK estimated the process CAPEX to be US$283 million to an accuracy level of -20% to +50% accuracy, suitable for a preliminary economic assessment. The proposed plant includes the copper concentrate filter plant and 20% contingency, but exclude the concentrate pipeline and tailings storage facility. Working capital is also excluded. A breakdown of the process capital cost into different categories is shown in Figure 57.

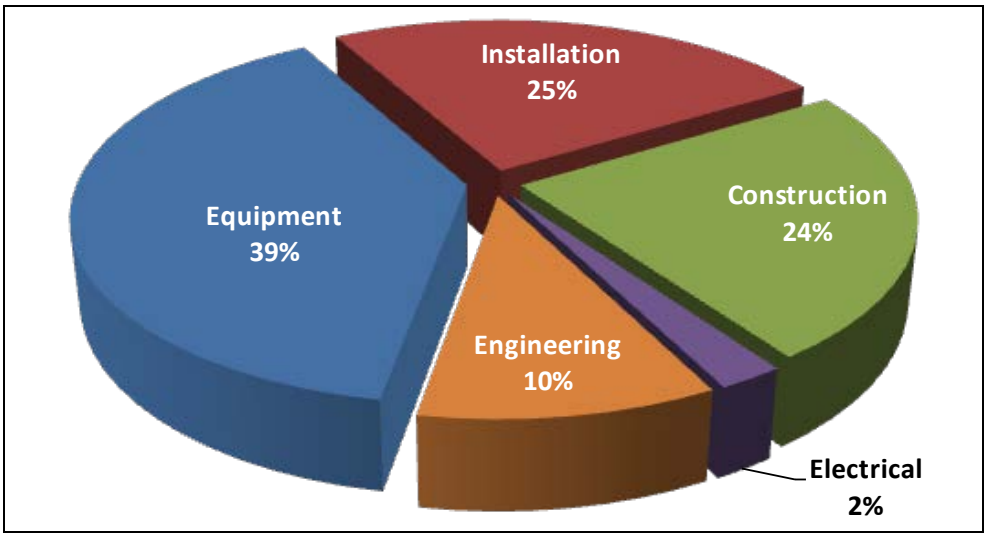


Figure 57: Breakdown of Process Plant Capital Cost

20.1.4 Tailings Storage Facility Capital Cost Estimate

Tailings storage facility capital costs are summarized in Table 57. It is assumed that pit waste rock will be utilized for the containment dam, and the rockfill excavation and hauling cost is included in the Antilla mining costs. The capital cost includes rockfill placement and compaction only. To minimize stockpiling and re-handle, the rockfill placement and compaction schedule is linked to the pit waste rock mining schedule.

In the pre-production period, the Stage 1 starter dam will be completed, and the Stage 2 raise will be started (i.e., the foundation will be completed and part of the rockfill requirement placed and compacted). The remainder of the Stage 2 raise and subsequent Stage 3 to 6 raises are scheduled to be completed during mine operation.

Table 57: TSF Capital Costs

Item	units	Quantity	Unit Cost	Initial Capital* (US$M)	Sustaining Capital* (US$M)	Total Capital, (US$M)
Mobilization, construction access	LS	1	2.0	2.00		2.00
Excavation - Topsoil, etc. removal	m^3	2,153,500	3.97	1.71	6.84	8.55
Excavation - Keyway trench ripped	m^3	233,800	4.43	0.50	0.54	1.04
Rock fill placement and compaction	m^3	75,338,500	0.83	4.15	58.38	62.53
Zone 1 fill (deformable concrete)	m^3	92,100	99.70	1.78	7.40	9.18
Zone 2A and 2B select fill	m^3	2,920,100	11.60	6.54	27.33	33.87
Dam lining (HDPE SST 2.0 mm GM)	m^2	378,100	7.10	0.52	2.16	2.68
Base drain soil removal	m^3	28,400	12.51	0.19	0.17	0.36
Base drain - Nonwoven geotextile	m^3	70,100	1.76	0.06	0.07	0.12
Base drain - HDPE perforated pipe	m	11,000	19.37	0.11	0.11	0.21
Base drain - Bedding sand/gravel	m^3	20,000	17.91	0.19	0.17	0.36
Base drain - Structural fill cover	m^3	8,400	14.71	0.06	0.06	0.12
Total direct costs				**$17.8**	**$103.2**	**$121.0**
Indirect costs (i.e., 40% direct costs)				7.1	41.3	48.4
TSF Dam Direct + Indirect Cost				**$24.9**	**$144.5**	**$169.4**
Contingency (25%)				6.2	36.1	42.4
Total TSF Cost, incl. contingency				**$31.2**	**$180.6**	**$211.8**

* Initial CapEx: Stage 1 and part of Stage 2. Sustaining CapEx: Completion of Stage 2, and Stages 3 to 6

20.1.5 Project Infrastructure and Services Capital Cost Estimate

The on-site infrastructure is described in Section 17 of the report. A summary of the infrastructure facilities costs for the proposed project is provided in Table 58.

No allowance has been made for temporary infrastructure facilities and services such construction camp and catering.

Table 58: Project Infrastructure Capital Costs Summary

Description	Initial CAPEX (US$M)	Sustaining CAPEX (US$M)	Total CAPEX (US$M)
Mine camp / housing facilities/ medical centre	8.0	-	8.0
Administrative / mine offices	1.0	-	1.0
Laboratory	1.0	-	1.0
Warehouse	2.5	-	2.5
Truck maintenance shop	8.0	-	8.0
Explosive storage magazine	0.5	-	0.5
Fuel station	0.5	-	0.5
Power transmission lines	14.5	-	14.5
Power substations	13.5	-	13.5
Communication systems	2.0	-	2.0
Access road to camp (8km) with bridge	4.4	-	4.4
Tailings pond/water reservoir service road (7km) + slurry line routes	1.6	1.5	3.1
Antilla - Santa Rosa gravel road upgrade, 60km + switchback section	7.0	-	7.0
TSF slurry pipeline	1.4	1.6	3.0
TSF slurry pumps	-	1.1	1.1
TSF reclaim water pipeline and barge	1.6	-	1.6
TSF reclaim water pump	0.1	-	0.1
Concentrate pipelines	0.3	-	0.3
Potable water supply system	0.2	-	0.2
Process water supply	0.1	-	0.1
Contact water pipeline to plant	0.5	-	0.5
Contact water pump	0.2	-	0.2
Water diversion channels	9.6	2.4	12.0
Water management structures (storage and settling ponds)	1.4	-	1.4
Site preparation (civil work)	5.0	-	5.0
Subtotal Infrastructure Direct Costs	**$84.9**	**$6.6**	**$91.5**
Indirect costs (30% of Direct costs)	25.5	2.0	27.4
Subtotal Direct + Indirect costs	110.3	8.6	118.9
Contingency (20%)	22.1	1.7	23.8
Total Infrastructure Costs (including Indirects and Contingency)	**$132.4**	**$10.4**	**$142.7**

20.1.6 Owner's Costs

The owner's cost are estimated at US$28 million to cover the costs associated with recruitment, training, transportation, accommodation, catering of the workforce, as well as security costs, project insurance costs, project permitting costs, external consultants expenses, and local community project costs. Owner's water and power supply costs, employee housing allowance are included in project infrastructure costs. Owner's cost includes an allowance of US$5 million for project site land acquisition, assuming a project area of 2,500 hectares.

Costs normally attributed to G&A in pre-production Years -2 and -1 have been included in the owner's cost and capitalized.

20.2 Operating Costs Estimate

Operating costs estimates presented in this study include on-site and off-site costs. On-site operating costs have been estimated for the operating areas of mining, processing, and administration. Off-site operating costs covers all activities associated with concentrate transport, insurance, smelting, refining and roasting charges.

Both on-site operating costs and off-site concentrate costs were benchmarked to the similar open pit projects in Peru, with the comparable mining and processing rates and considering the owner's fleet is engaged. All capital and operating costs are in US dollars, unless otherwise specified.

Total operating cost estimates are summarized in Table 59. On-site operating costs are estimated to average US$9.10/tonne milled and total US$3,188 million over the mine life. Off-site operating costs are estimated to total US$900 million over the mine life.

Table 59: Operating Costs by Major Functions

Operating Cost Component	Unit Cost (US$/t milled)	Total OPEX (US$M)	Percent of Total OPEX
Mining	3.57	1,250	31%
Processing	4.60	1,612	39%
Tailings pumping	0.18	64	2%
G&A	0.75	263	6%
Subtotal On-Site Costs	**$9.10**	**$3,188**	**78%**
Off-Site Costs		900	22%
Total Operating Costs		**$4,088**	**100%**

The operating cost estimates were based on energy prices of US$0.061/kWh for electricity and US$0.80/litre for diesel fuel.

The operating costs are considered to have accuracy of -20% to +50%, based on the assumptions in this section of the report.

20.2.1 Mining Cost

Moose Mountain has provided costs specific to mining equipment and the operation of that mining equipment. Mining Capital and Operating Costs are prepared as a scoping level estimate, or an American Association of Cost Engineers Class 5 estimate. A Class 5 estimate is defined as having an expected accuracy range of -20% to +50% and is generally based on factored or scaled costs. This is considered sufficient for a preliminary economic assessment.

The mine operating costs are estimated by Moose Mountain using a cost model built from first principles. Mine operating costs are based on owner-operated haul trucks as well as production quantities, consumable costs from budgetary quotations obtained by Panoro, and benchmarking.

The mine operating cost estimate includes the costs of drilling, blasting, loading, hauling, support and ancillary equipment operations and maintenance, and mine operations supervision required to produce the run-of-mine material to feed the process plant.

Mine operating costs during mill production (Years 1-24) total US$1,250 million, or an average of US$1.62/tonne moved, as shown in Table 60. Tonnes moved include material moved from the pit to the mill, pit to stockpile and stockpile to mill throughout Years 1-24.

Table 60: Operating Cost Breakdown

Consumable	Unit	Unit Price
Drilling	$US/t moved	$0.11
Blasting	$US/t moved	$0.24
Loading	$US/t moved	$0.29
Hauling	$US/t moved	$0.74
Pit Maintenance	$US/t moved	$0.13
Geotechnical	$US/t moved	$0.01
Unallocated Labour	$US/t moved	$0.01
GME	$US/t moved	0.08
Total Mining Cost	**$US/t moved**	**$1.62**

Figure 58 shows the breakdown of mine operating costs in a pie chart.

Table 61 shows the costs of major consumables utilized in the estimation of the mine operating costs. Mine staff wages and salaries are based on benchmarked wages and salaries from similar mine operations in Peru that Panoro is familiar with.

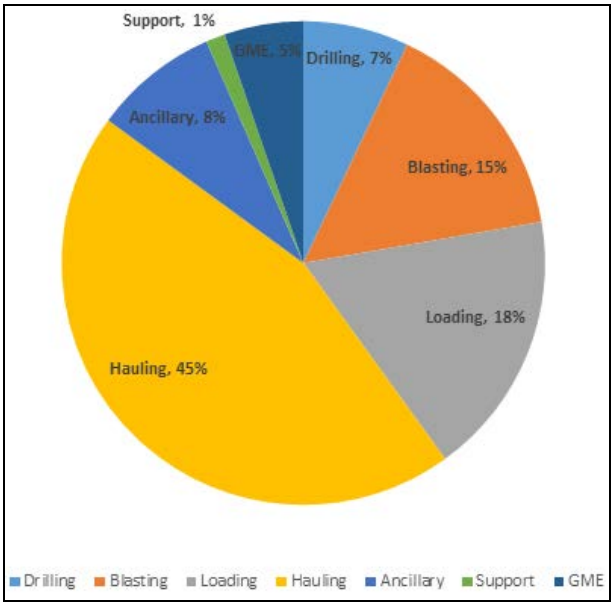


Figure 58: Breakdown of Mine Operating Costs

Table 61: Costs of Major Consumables

Consumable	Unit	Unit Price
Diesel	$US/L	$0.80
Bulk Explosives	$US/kg	$0.65
Tires	$US/tire	$33,000

20.2.2 Processing Cost

The process operating cost was estimated based on similar copper and copper-molybdenum projects in Peru and scaled to account for the lower labour and power costs of the region.

At 40,000 tonnes/day, SRK's estimates of process operating costs are US$4.60/tonne and should be considered -20 to +50% accurate, which is suitable for a preliminary economic assessment. A breakdown of the process operating costs is shown in Figure 59. The cost for grinding media/liner wear was estimated based on expected ore hardness as no abrasion testwork has been completed to date. Power costs were assumed to be US$0.061/kWh.

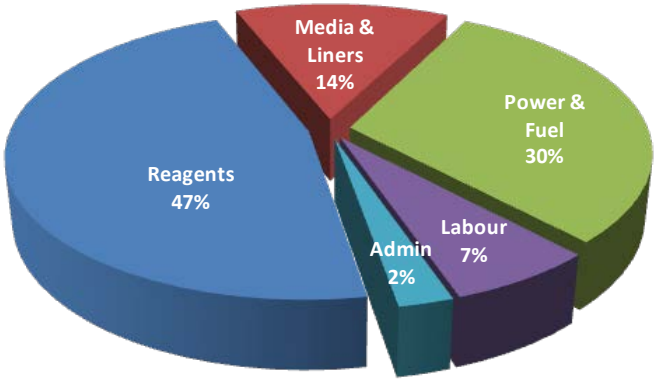


Figure 59: Breakdown of Process Plant Operating Cost

20.2.3 Tailings Pumping

The tailings storage facility is located upstream from the processing plant and tailings deposition is at elevations higher than the plant site throughout most of the 24 year processing period, which will necessitate tailings pumping throughout the mine life. Tailings pumping costs, principally for electric power, are estimated to increase from US$0.01/tonne milled in Year 1 to US$0.22/tonne milled in Year 24, and average US$0.18/tonne milled over the mine life.

20.2.4 General and Administrative Cost

General and administrative (G&A) costs estimate are based on benchmarking to similar open pit projects in Peru and other Latin American countries. The Antilla life-of-mine G&A cost is estimated to average US$0.75/tonne of mill feed or US$11 million/year. Costs attributed to G&A in the pre-production period Year -2 and Year -1 have been included in the owner's cost and capitalized.

G&A costs include the salaries for an estimated 60 administrative and support staff, camp catering and meals for the entire workforce, and employee transportation costs, assuming a majority of the workforce resides in Abancay and Cusco. Other G&A costs includes office supplies, insurance, legal services, environmental monitoring, business travel, communications, information technology, community relations, recruitment, safety and training supplies, security and first aid, building and support vehicle maintenance.

Worker profit sharing and supplementary worker retirement fund contributions are excluded from G&A estimates. These items are related to mine income and are included separately in the economic analysis cash flow model. Corporate head office costs are excluded from the G&A estimate.

20.2.5 Off-Site Costs

Off-site costs are summarized in Table 62 based on the commercial terms presented in Section 18.

The standard shipment lot size for sea borne copper concentrate cargos is 10,000 wet metric tonnes. For Peruvian and Chilean producers, the majority of shipments and contracts are based on a CIF Main Asian Port destination.

Table 62: Off-Site Cost Summary

	Units	Unit Cost	Total Cost (US$M)
Copper Concentrate			
Transport to Port	US$/wmt	40.00	156
Port Charges	US$/wmt	10.00	39
Insurance & Marketing	% con value	0.075	4
Ocean Freight	$/wmt	50.00	195
Smelting Charges	US$/wmt	75.00	270
Refining Charges	US$/lb	0.08	146
Subtotal, Copper Concentrate			**$810**
Molybdenum Concentrate			
Transport	US$/wmt	271	18
Insurance & Marketing	% con value	0.075	0.3
Roasting Charges	US$/lb	1,268	72
Subtotal, Molybdenum Concentrate			**90**
Total Off Site Costs			**$900**

21 Economic Analysis

This section summarizes the economic analysis completed for the Antilla project preliminary economic assessment. A preliminary economic assessment is a conceptual study of the potential viability of mineral resources. A preliminary economic assessment should not be considered to be a pre-feasibility or feasibility study, and the economics and technical viability of the Antilla project have not been demonstrated at this time. The preliminary economic assessment is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. Furthermore, there is no certainty that the conclusions or results as reported in the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves have not demonstrated economic viability.

21.1 Economic Model Assumptions

21.1.1 Methodology

The Antilla project has been evaluated on a discounted cash flow basis assuming 100% equity project financing. The cash flow analysis has been prepared in constant fourth quarter 2015 US dollars. No inflation or escalation of revenue or costs has been incorporated.

Key economic indicators assessed include net cash flow (NCF), payback period, present value of the net cash flow (PVNCF) at various discount rates, and internal rate of return (IRR).

Discounting of revenues and costs for present value of the net cash flow purposes is to the start of a two-year development and construction period. Project costs that will be incurred prior to this point (for example, costs for further exploration drilling, field and laboratory investigations, environmental studies, and more detailed technical studies) are not included in the economic analysis.

Payback period is defined as the time after process plant start-up that is required to recover the initial expenditures incurred developing the Antilla project. At this point in time the project's cumulative undiscounted net cash flow is zero.

21.1.2 Metal Prices

The Antilla project is considered to be far from commercial production since additional exploration and more detailed technical studies including declaration of mineral reserves are needed prior to a production decision, and once operating, the processing period is projected to be lengthy (24 years). For these reasons long-term metal price forecasts are considered appropriate for evaluating the project's potential viability.

Evaluation of the Antilla project is based on long-term metal prices of US$3.00/pound for copper and US$12/pound for molybdenum. These metal prices are supported by long-term forecasts by banking institutions, which are projecting an improvement to the current depressed copper and molybdenum prices by 2019+.

21.1.3 Production and Mill Feed

The proposed mining schedule and plant feed schedule is presented in Table 41 in Section 15.8 of this report.

The mining schedule is based on a two-year development and construction period followed by approximately 19 years of pit operation. During the pit operation, higher grade mineral resources encountered in the pit are preferentially fed to the processing plant and the remaining mineral resources encountered are stockpiled. The plant feed schedule is based on processing 40,000 tonnes per calendar day (14.6 Mt/a) for 24 years. Plant feed after Year 19 is principally reclaim of relatively low grade mineral resources stockpiled in prior years.

The plant feed comprises four metallurgical types. Process recovery and concentrate grade by metallurgical type is presented in Table 36 of Section 16. Process recovery over the mine life is estimated to average 84.5% for copper and 67.4% for molybdenum. Smelter payable copper is estimated based on a one-unit deduction from the forecast copper concentrate grade, and averages 96.1% of copper contained in the concentrate. Payable molybdenum is 100% of molybdenum contained in the concentrate since negligible roaster molybdenum losses are anticipated.

Life of mine mining and processing quantities, recovered metal, and payable metal after smelter deductions are summarized in Table 63. Annual payable copper and molybdenum are estimated to average 81 million pounds and 1.9 million pounds, respectively, over the 24-year processing period.

Table 63: Mine Production Summary

			Units	LOM
Mine Production				
Indicated Mineral Resources			Mt	291.1
Inferred Mineral Resources			Mt	59.8
Waste Rock			Mt	297.2
Total Mined			Mt	648.1
Re-handle - Stockpile to Plant			Mt	153.1
Total Moved			Mt	801.2
Plant Feed				
Process Years			years	24
Plant Feed - Cover			Mt	9.0
Plant Feed - Leach Cap			Mt	22.0
Plant Feed - Supergene			Mt	164.7
Plant Feed - Primary Sulphides			Mt	154.8
Plant feed - Total	40	kt/d	Mt	350.4
Copper Grade			%	0.310
Recovered Copper	84.5%	Avg recovery	kt	918
Copper Concentrate (dry)	25.5%	Avg con grade	kt	3,594
Payable Copper	96.1%	Avg pay factor	Mlb	1,944
Molybdenum Grade			%	0.009
Recovered Molybdenum	67.4%	Avg recovery	kt	20.2
Molybdenum Concentrate (dry)	35.5%	Avg con grade	kt	56.8
Payable Molybdenum	100%	Avg pay factor	Mlb	44.5

21.1.4 Capital and Operating Costs

Capital and operating cost estimates are presented in Section 20 of this report and are summarized in Table 64. Initial capital over a two-year construction period is estimated at US$603 million. Sustaining capital, principally for mining equipment and tailings storage facility expansion, is estimated at US$324 million.

Table 64: Initial and Sustaining Capital Costs

Area	Initial Capital (US$M)	Sustaining Capital (US$M)	Total Capital (US$M)
Mine Equipment	51	111	162
Mine Development	55		55
Process Plant	236		236
Tailings Storage Facility	25	145	169
Infrastructure	110	9	119
Owner	28		28
Subtotal, Before Contingency	**$506**	**$264**	**$770**
Contingency	97	60	157
Total Capital Cost	**$603**	**$324**	**$927**

As presented in Table 59 in Section 20 life of mine on-site operating costs for mining, processing, and G&A average YS$9.10/tonne processed and total US$3,188 million over the mine life. Off-site operating costs consist of copper and molybdenum concentrate transportation, smelting, and refining and total US$900 million over the mine life as presented in Table 62.

21.1.5 Working Capital

It is estimated that approximately six weeks of mill production will be contained within concentrate inventory on site or in transit, which will delay receipt of copper and molybdenum revenue and contribute to project working capital requirements. Working capital is also required to maintain an operating supplies inventory on site. Accounts payable, estimated at one month on site operating cost, partially offsets these working capital requirements. Working capital requirements in Year 1 are estimated at US$43 million, which is recovered over the remaining project life.

21.1.6 Mine Closure and Salvage Value

The mine closure cost as presented in Section 20 is estimated at US$92 million. This includes approximately US$10 million related to the open pit mine (principally for waste rock dump closure) which is assumed will be incurred in Years 20 and 21 when pit mining ceases. For the purposes of economic evaluation the remaining closure cost is assumed to be incurred in Year 25, after the processing plant ceases operation.

The Peru mine closure law requires mine owners to provide a guaranty for the estimated costs associated with their mine closure plans. The amount of the guaranty, to be provided in annual installments, may be in the form of insurance, cash collateral, a trust agreement or other forms permitted under Peruvian law. Mine closure financial assurance requirements are not addressed in this PEA but should be assessed in subsequent more detailed project studies.

No allowances for salvage value of equipment and facilities are included in the project economic evaluation.

21.1.7 Unit Cash Cost and Production Cost Estimates

Mining industry commonly report non-GAAP measures including C1 cash costs and C2 production costs as defined by Brook Hunt, a Woods Mackenzie company. C1 cash costs are defined by Brook Hunt as the costs of mining, milling and concentrating, on site administration and general expense, property and production royalties not related to revenues or profits, metal concentrate treatment charges, and freight and marketing costs less the net value of by-product credits. C2 production costs are defined as the sum of C1 cash costs and depreciation, depletion and amortization. Estimated Antilla project C1 cash costs and C2 production costs in terms of US dollars per pound of payable copper are summarized in Table 65 below.

Table 65: C1 and C2 Unit Cost Estimates

	LOM Total (US$M)	Unit Cost* $/lb Cu payable
Cash Cost C1		
Mining	1,250	0.64
Process	1,675	0.86
G&A	263	0.14
Treatment Charges	342	0.18
Refining Charges	146	0.08
Freight & Marketing	413	0.21
Total Operating Costs	**4,088**	**2.10**
Molybdenum By-product Credit	(534)	(0.27)
Total C1 Cash Costs	**3,554**	**1.83**
Production Cost C2		
Depreciation & Amortization	1,011	0.52
Total C2 Production Costs	**4,566**	**2.35**

* Based on life of mine total 1,944 million pounds of payable copper

21.1.8 Peru Taxation

Peru taxes and royalties included in the economic model are described below. SRK 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.

Tax Stability Agreement

Mine developers often enter into a Tax Stability Agreement (TSA) with the Peru government. Advantages include stabilization of tax regime for the duration of the agreement, and in the case of 15-year stability contracts, accelerated annual depreciation of mining assets. The disadvantage is a 2% premium on the corporate income tax rate. A 15-year duration TSA was found to be slightly beneficial to the Antilla project's economics and is included in the economic model.

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.

Sunk exploration and other eligible project costs are carried forward and amortized over the mine life. Panoro estimates that its eligible sunk project costs to date total US$7.2 million. Amortization estimates also include mine closure expenses in the year incurred.

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.

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 totalling 0.29% of sale revenue were assumed for the Antilla project's economic analysis.

Retirement Fund Contributions

Mining companies contribute 0.5% of annual income before taxes to a supplementary fund for the retirement of mining, metallurgical, and steel workers.

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.

Corporate Income Tax

The Peru corporate tax rate is scheduled to be reduced to 26% by 2019. The Antilla project is far from commercial production and a statutory corporate tax rate of 26% has been assumed for the project. As described above, a 15-year TSA is included in the economic analysis, and during the 15-year TSA term a 2% premium on corporate income tax is applicable. After production year 13, i.e., 15 years from the start of construction, the corporate tax rate reverts to the statutory 26% rate.

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

- Depreciation and amortization
- Mining royalty
- Special mining tax
- Regulatory fees
- Retirement fund contributions
- Worker profit sharing

21.2 Economic Results

The results of the analysis show the Antilla project mineral resources to be potentially viable, warranting further study. Base case economic results are summarized in Table 66. At a base case copper price of US$3.00/pound and molybdenum price of US$12/pound, the potential pre-tax present value of the net cash flow at the start of the projected two-year construction period using a 7.5% discount rate ($PVNCF_{7.5\%}$) is estimated at US$491 million, and potential project post-tax $PVNCF_{7.5\%}$ is estimated at US$225 million. Potential internal rates of return (IRR) are respectively 22.2% pre-tax and 15.1% post-tax. The payback period is estimated to be slightly over four years.

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

Table 66: Base Case Economic Results Summary

			Units	Total
Plant Feed			Mt	350
Payable Copper			Mlbs	1,944
Payable Molybdenum			Mlbs	44.5
Revenue				
Gross Revenue - Copper	3.00	US$/lb	US$M	5,831
Gross Revenue - Molybdenum	12.00	US$/lb	US$M	534
Total Gross Revenue			**US$M**	**6,365**
Off Site Operating Costs				
Copper Concentrate Transport & Insurance			US$M	-394
Copper Concentrate, Smelting & Refining Charges			US$M	-415
Molybdenum Concentrate Transport & Insurance			US$M	-18
Molybdenum Concentrate Roasting Charges			US$M	-72
Total Off-Site Costs			**US$M**	**-900**
Net Smelter Return (NSR)	15.60	US$/t milled	US$M	5,465
On-Site Operating Costs				
Mining	3.57	US$/t milled	US$M	-1,250
Processing & Tailings Pumping	4.78	US$/t milled	US$M	-1,675
General & Administrative	0.75	US$/t milled	US$M	-263
Total On-Site Operating Costs	**9.10**	**US$/t milled**	**US$M**	**-3,188**
Operating Margin			US$M	2,277
Initial Capital Costs			US$M	-603
Sustaining Capital Costs			US$M	-324
Reclamation and Closure			US$M	-92
Project Net Cash Flow, Pre-tax			**US$M**	**1,258**
Payback Period, Pre-tax			Years	3.3
PVNCF 5%, Pre-tax			US$M	676
PVNCF 7.5%, Pre-tax			**US$M**	**491**
PVNCF 10%, Pre-tax			US$M	350
IRR, Pre-tax			**%**	**22.2%**
Mining Royalty			US$M	-66
Special Mining Tax			US$M	-31
Retirement Fund & Regulatory Contributions			US$M	-25
Worker Profit Sharing			US$M	-98
Income Tax			US$M	-312
Total Statutory Charges			**US$M**	**-533**
Project Net Cash Flow, Post-tax			**US$M**	**725**
Payback Period, Post-tax			Years	4.1
PVNCF 5%, Post-tax			US$M	348
PVNCF 7.5%, Post-tax			**US$M**	**225**
PVNCF 10%, Post-tax			US$M	131
IRR, Post-tax			**%**	**15.1%**

Annual cash flow estimates are presented in Table 67 and Figure 60.

Table 67: Base Case Annual Cash Flow

			Units	Total	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14	Y 15	Y 16	Y 17	Y 18	Y 19	Y 20	Y 21	Y 22	Y 23	Y 24	Y 25
Mine Production					**Development**		**> Operation**																								**Closure**
ROM Mined			Mt	351	1.5	4.2	15.1	30.7	26.1	18.8	38.8	17.5	13.2	21.7	23.6	22.1	12.8	16.0	13.6	17.3	13.6	9.1	24.0	7.4	3.7	0.1	0.1	-	-	-	-
Waste Rock Mined			Mt	297	10.6	13.8	34.9	25.7	31.1	33.5	13.8	33.6	34.5	23.9	9.8	4.5	2.6	5.2	0.8	0.6	3.2	6.2	5.3	2.3	1.2	0.1	0.0	-	-	-	-
Total Mined			Mt	648	12.1	18.0	50.0	56.4	57.2	52.3	52.6	51.1	47.7	45.6	33.4	26.7	15.5	21.2	14.4	17.8	16.7	15.3	29.3	9.7	4.9	0.2	0.1	-	-	-	-
ROM Re-handle to Plant			Mt	153	-	-	3.0	-	0.7	2.7	-	10.9	9.3	2.9	-	2.1	5.9	3.8	5.8	3.7	3.7	5.8	0.4	8.8	10.9	14.5	14.5	14.6	14.6	14.6	-
Total Moved			Mt	801	12.1	18.0	52.9	56.4	57.9	55.0	52.6	62.0	57.0	48.6	33.4	28.7	21.3	25.0	20.2	21.5	20.4	21.1	29.7	18.4	15.8	14.7	14.6	14.6	14.6	14.6	-
Plant Feed - Cover			Mt	9.0	-	-	0.3	-	0.0	0.3	-	-	0.8	-	-	-	-	2.4	0.5	0.0	0.1	0.0	-	-	0.0	-	-	-	3.8	0.8	-
Plant Feed - Leach Cap			Mt	22.0	-	-	0.5	-	0.0	0.0	-	1.0	1.1	0.0	0.0	0.6	0.0	0.0	0.0	-	-	-	-	-	4.8	-	-	-	-	13.8	-
Plant Feed - Supergene			Mt	164.7	-	-	13.3	14.6	12.1	11.3	14.1	7.5	9.2	14.1	13.5	11.0	9.1	5.3	5.5	5.3	1.9	0.1	2.3	0.1	2.6	11.6	-	-	-	-	-
Plant Feed - Primary Sulphides			Mt	154.8	-	-	0.5	-	2.5	3.1	0.5	6.1	3.4	0.5	1.1	3.0	5.4	6.8	8.6	9.2	12.7	14.5	12.3	14.5	7.1	3.0	14.6	14.6	10.8	-	-
Plant Feed - Total	**40.0**	**kt/d**	**Mt**	**350.4**	**-**	**-**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**14.6**	**-**
Copper Grade			%	0.310	-	-	0.579	0.594	0.435	0.504	0.496	0.352	0.312	0.367	0.453	0.354	0.289	0.286	0.238	0.250	0.248	0.246	0.244	0.228	0.209	0.156	0.150	0.150	0.149	0.147	-
Recovered Copper	84.5%	avg recovery	kt	918	-	-	72	74	54	62	62	43	38	46	56	44	36	35	29	31	31	31	30	28	25	19	19	19	18	16	-
Copper Concentrate (dry)	25.5%	avg con grade	kt	3,594	-	-	243	246	196	228	208	174	146	154	193	162	141	146	125	134	146	153	141	141	109	71	93	93	86	65	-
Payable Copper	96.1%	avg pay factor	Mlb	1,944	-	-	153	157	115	133	131	91	81	97	120	93	76	74	62	65	65	64	64	59	52	41	39	39	38	34	-
Mo Grade			%	0.009	-	-	0.008	0.013	0.009	0.008	0.005	0.008	0.008	0.008	0.010	0.009	0.007	0.009	0.012	0.011	0.009	0.007	0.008	0.009	0.009	0.008	0.007	0.007	0.007	0.009	-
Recovered Mo	67.4%	avg recovery	kt	20.2	-	-	0.9	1.4	0.9	0.8	0.5	0.8	0.8	0.8	1.0	0.9	0.7	0.8	1.2	1.1	0.8	0.7	0.8	0.8	0.8	0.8	0.7	0.7	0.7	0.8	-
Mo Concentrate (dry)	35.5%	avg con grade	kt	56.8	-	-	2.2	3.4	2.4	2.2	1.2	2.3	2.2	2.0	2.5	2.5	1.9	2.4	3.3	3.1	2.6	2.2	2.3	2.6	2.5	2.0	2.1	2.1	2.2	2.7	-
Payable Mo	100%	avg pay factor	Mlb	44	-	-	1.9	3.0	2.0	1.8	1.1	1.8	1.7	1.7	2.2	2.1	1.5	1.9	2.6	2.4	1.9	1.5	1.7	1.9	1.8	1.7	1.5	1.5	1.5	1.9	-
Revenue																															
Gross Revenue - Copper at $3.00/lb			US$M	5,831	-	-	458	471	344	398	394	274	243	291	359	279	228	223	186	196	194	192	191	178	156	123	117	117	115	102	-
Gross Revenue - Mo at $12/lb			US$M	534	-	-	23	36	24	22	13	22	20	21	27	25	18	22	31	29	22	18	20	22	22	21	18	18	18	22	-
Total Gross Revenue			US$M	6,365	-	-	480	508	369	420	406	296	264	312	385	304	246	246	218	225	216	211	211	200	178	144	135	135	133	125	-
Off-Site Operating Costs																															
Copper Concentrate Transport & Insurance			US$M	-394	-	-	-27	-27	-22	-25	-23	-19	-16	-17	-21	-18	-15	-16	-14	-15	-16	-17	-15	-15	-12	-8	-10	-10	-9	-7	-
Copper Concentrate, Smelting & Refining Charges			US$M	-415	-	-	-30	-30	-23	-27	-25	-20	-17	-19	-23	-19	-16	-17	-14	-15	-16	-16	-15	-15	-12	-8	-10	-10	-9	-7	-
Mo Concentrate Transport & Insurance			US$M	-18	-	-	-0.7	-1.1	-0.8	-0.7	-0.4	-0.8	-0.7	-0.6	-0.8	-0.8	-0.6	-0.8	-1.1	-1.0	-0.8	-0.7	-0.7	-0.9	-0.8	-0.7	-0.7	-0.7	-0.7	-0.9	-
Mo Concentrate Roasting charges			US$M	-72	-	-	-3.1	-4.9	-3.3	-3.0	-1.7	-2.9	-2.8	-2.8	-3.6	-3.3	-2.4	-3.0	-4.2	-3.9	-3.0	-2.5	-2.7	-3.0	-2.9	-2.8	-2.4	-2.4	-2.5	-3.0	-
Total Off-Site Costs			US$M	-900	-	-	-60	-63	-49	-56	-50	-43	-37	-39	-49	-41	-35	-36	-33	-35	-36	-36	-34	-34	-28	-20	-23	-23	-22	-18	-
Net Smelter Return (NSR)			US$M	5,465	-	-	420	444	320	364	356	253	227	273	336	263	211	210	184	190	181	175	177	166	150	124	112	111	111	107	-
Unit NSR			$/t milled	15.60	-	-	28.78	30.44	21.90	24.94	24.37	17.35	15.55	18.68	23.04	17.98	14.45	14.35	12.64	13.04	12.37	11.95	12.11	11.37	10.29	8.53	7.64	7.63	7.59	7.30	-
On-Site Operating Costs																															
Mining			US$M	-1,250	-	-	-67	-81	-88	-86	-86	-85	-89	-79	-60	-56	-44	-46	-39	-48	-50	-44	-53	-34	-28	-18	-19	-18	-16	-16	-
Processing & Tailings Pumping			US$M	-1,675	-	-	-67	-68	-68	-69	-69	-69	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-70	-
General and Administrative			US$M	-263	-	-	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-11	-
Total On-Site Operating Costs			US$M	-3,188	-	-	-145	-160	-167	-165	-166	-166	-169	-159	-141	-137	-125	-128	-120	-130	-131	-125	-134	-115	-110	-100	-100	-100	-97	-97	-
Unit On-Site Operating Costs			$/t milled	9.10	-	-	9.97	10.95	11.41	11.32	11.39	11.35	11.61	10.92	9.65	9.38	8.57	8.75	8.21	8.87	8.97	8.59	9.20	7.86	7.50	6.82	6.88	6.83	6.67	6.66	-
Operating Margin			US$M	2,277	-	-	275	284	153	199	189	88	58	113	196	126	86	82	65	61	50	49	42	51	41	25	11	12	13	9	-
Initial Capital Costs			US$M	-603	-231	-372	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital Costs			US$M	-324	-	-	-67	-34	-25	-17	-9	-11	-23	-21	-14	-7	-8	-9	-10	-10	-14	-8	-16	-6	-4	-4	-3	-2	-2	0	-
Reclamation and Closure			US$M	-92	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-5	-5	-	-	-	-82
Change in Working Capital			US$M	0	-	-	-43	-2	16	-6	1	13	4	-7	-10	9	5	0	2	0	1	0	0	0	2	2	2	0	0	1	8
Project Net Cash Flow, pre-tax			**US$M**	**1,258**	**-231**	**-372**	**164**	**249**	**144**	**176**	**182**	**89**	**38**	**86**	**172**	**128**	**83**	**73**	**57**	**51**	**37**	**42**	**27**	**45**	**38**	**18**	**5**	**9**	**11**	**10**	**-74**
Cumulative NCF, pre-tax			US$M		-231	-603	-438	-189	-45	131	313	402	441	527	699	826	909	982	1,039	1,090	1,127	1,169	1,195	1,241	1,279	1,297	1,302	1,312	1,323	1,332	1,258
Payback Period, pre-tax			Years	3.3																											
PVNCF 7.5%, pre-tax			US$M	**491**																											
IRR, pre-tax			%	**22.2%**																											
Depreciation and Amortization			US$M	1,011	0	0	178	111	117	121	124	28	22	20	19	19	19	17	14	11	12	13	13	14	12	16	12	6	5	4	85
Mining Royalty			US$M	-66	-	-	-5	-5	-4	-4	-4	-3	-3	-3	-6	-3	-2	-2	-2	-2	-2	-2	-2	-2	-2	-1	-1	-1	-1	-1	-
Special Mining Tax			US$M	-31	-	-	-2	-5	0	-2	-1	-1	-1	-2	-5	-3	-1	-1	-1	-1	-1	-1	-1	-1	-1	0	-	0	0	0	-
Retirement Fund & Regulatory Contributions			US$M	-25	-	-	-2	-2	-1	-2	-2	-1	-1	-1	-2	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	0	0	0	0	0	-
Worker Profit Sharing			US$M	-98	-	-	-7	-13	-2	-6	-5	-4	-3	-7	-13	-8	-5	-5	-4	-4	-3	-3	-2	-3	-2	-1	-	0	-1	0	-
Income Tax			US$M	-312	-	-	-23	-42	-8	-18	-15	-14	-8	-22	-42	-26	-16	-15	-12	-11	-8	-8	-6	-8	-6	-2	-	0	-2	-1	-
Total Statutory Charges			US$M	-533	0	0	-38	-67	-16	-31	-27	-24	-15	-36	-68	-41	-26	-25	-20	-19	-15	-14	-12	-14	-11	-4	-2	-2	-4	-3	0
Project Net Cash Flow, post-tax			**US$M**	**725**	**-231**	**-372**	**126**	**182**	**129**	**145**	**155**	**66**	**23**	**50**	**104**	**87**	**57**	**48**	**37**	**32**	**22**	**28**	**15**	**31**	**27**	**14**	**4**	**7**	**7**	**7**	**-74**
Cumulative NCF, post-tax			US$M		-231	-603	-477	-294	-166	-21	134	200	223	273	377	464	521	569	606	638	660	688	703	734	761	775	778	785	792	799	725
Payback Period, post-tax			Years	4.1																											
PVNCF 7.5%, post-tax			US$M	**225**																											
IRR, post-tax			%	**15.1%**																											

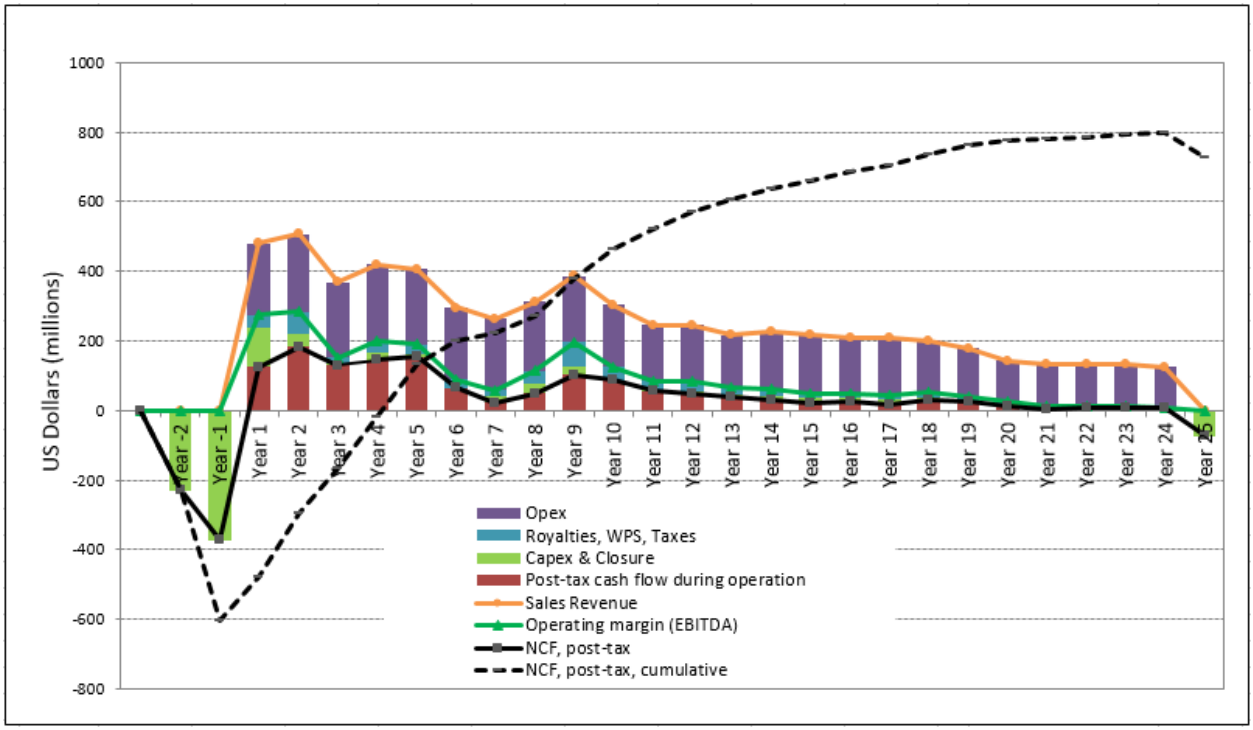


Figure 60: Undiscounted Annual Net Cash Flow

21.3 Sensitivity Analysis

The proposed project's $PVNCF_{7.5\%}$ and IRR sensitivity to changes in copper price is shown in Table 68. The base case copper price of US$3.00/pound provides a post-tax $PVNCF_{7.5\%}$ of $225 million and IRR of 15.1%. If the copper price rises US$0.50/pound to US$3.50/pound, the IRR would rise 7.5% to 22.6%. Conversely, a 0.50/pound reduction in the copper price to US$2.50/pound results in an 11.3% reduction in IRR to 3.7%. For all cases the molybdenum price is held constant at US$12/pound.

Table 68: Project Economics Sensitivity to Copper Price

Copper Price*	US$/lb	2.50	2.75	**3.00***	3.25	3.50
Project NCF Pre-tax**						
Project NCF	US$M	287	773	**1258**	1744	2230
PVNCF 5%	US$M	102	389	**676**	964	1251
PVNCF 7.5%	US$M	30	261	**491**	721	951
PVNCF 10%	US$M	-27	161	**350**	538	726
IRR	%	8.8%	16.2%	**22.2%**	27.7%	32.8%
Payback	years	5.5	4.0	**3.3**	2.7	2.2
Project NCF Post-tax						
Project NCF	US$M	79	406	**725**	1033	1339
PVNCF 5%	US$M	-26	163	**348**	529	708
PVNCF 7.5%	US$M	-73	78	**225**	369	512
PVNCF 10%	US$M	-112	11	**131**	248	365
IRR	%	3.7%	10.5%	**15.1%**	19.0%	22.6%
Payback	years	8.4	4.8	**4.1**	3.6	3.2

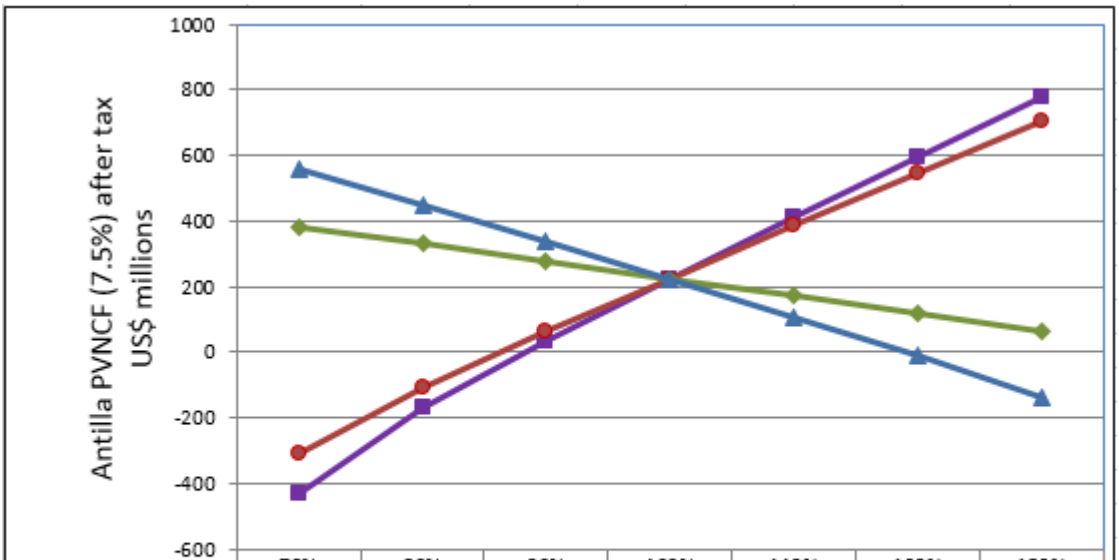
* Base case copper price US$3.00/pound. Molybdenum price held constant at US$12/pound for all cases.

** Pre-tax excludes Peru statutory charges, i.e. profit sharing, regulatory fees, mining royalty, special mining tax, income tax.

The sensitivity of the proposed project $PVNCF_{7.5\%}$ and IRR to +/- 30% changes in the key input parameters of metal prices, metal grades, capital costs, and operating costs are shown in Figure 61 and Figure 62. The sensitivities results due to a particular parameter change assume the remaining parameters remain unaffected.

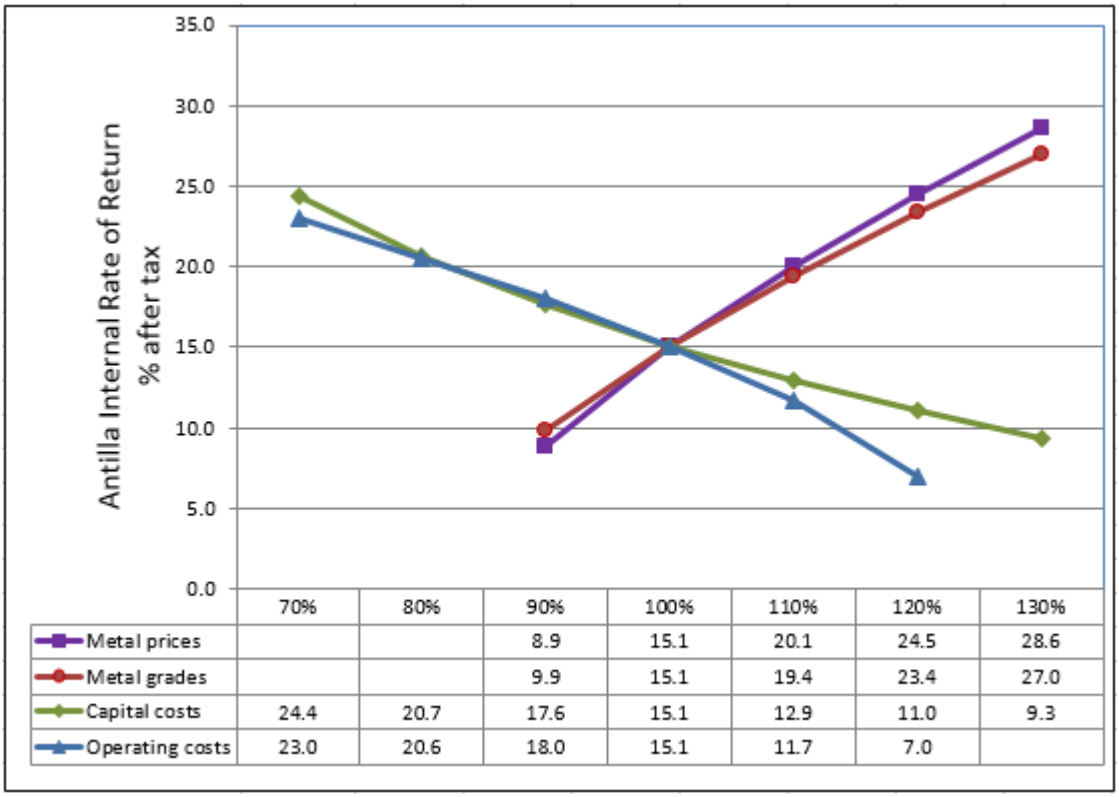
A review of Figure 61 and Figure 62 shows that, like most mining projects, the $PVNCF_{7.5\%}$ and IRR are most sensitive to metal prices (copper and molybdenum), which directly impact revenue. The project is slightly less sensitive to metal grades, since the revenue impact of a change to head grade is partially offset by related changes to concentrate tonnage and off-site operating costs.

The project $PVNCF_{7.5\%}$ and IRR is more sensitive to changes in operating costs than to capital costs. This is attributed to the fact that although capital costs are weighted heavily at the front-end of the project, total operating costs are about four times the total capital costs.



	70%	80%	90%	100%	110%	120%	130%
Metal prices	-432	-170	34	225	412	596	780
Metal grades	-311	-110	60	225	386	546	705
Capital costs	382	330	278	225	172	119	66
Operating costs	558	449	338	225	108	-10	-139

Figure 61: Antilla PVNCF7.5% Sensitivity to Key Input Parameters



	70%	80%	90%	100%	110%	120%	130%
Metal prices			8.9	15.1	20.1	24.5	28.6
Metal grades			9.9	15.1	19.4	23.4	27.0
Capital costs	24.4	20.7	17.6	15.1	12.9	11.0	9.3
Operating costs	23.0	20.6	18.0	15.1	11.7	7.0	

Figure 62: Antilla IRR Sensitivity to Key Input Parameters

22 Adjacent Properties

There are no adjacent properties that are relevant to the subject of this technical report.

23 Other Relevant Data and Information

There are no other relevant data available about the Antilla project.

24 Interpretation and Conclusions

24.1 General

This technical report provides a summary of the results and findings from each major area of investigation including exploration, geological modelling, mineral resource and plant feed estimations, mine design, metallurgy and process design, infrastructure, environmental management, capital and operating costs, and economic analysis. The level of investigation for each of these areas is considered to be consistent with that normally expected with preliminary economic assessment for resource development projects.

The results of the preliminary economic assessment indicate the proposed Antilla project is potentially economically viable. The favorable economic potential warrants further technical studies to examine the potential viability of the proposed project at a higher level of confidence and the preparation of a preliminary feasibility study.

Exploration activities to date have been completed under the appropriate Peruvian permits. Most of the infrastructure proposed for the conceptual project is within the mineral concession boundaries. However, the fresh water storage reservoir and a portion of the tailings storage facility are located outside the current mineral concession limits.

Panoro holds no surface rights in the area, and any future mining activities will require agreements to be negotiated with both local communities and individual surface rights holders. An allowance for the land acquisition is included in the owner's cost estimate. The inability to purchase or lease the lands on which the proposed Antilla project would be developed poses a significant risk.

24.2 Geology and Mineral Resources

24.2.1 Conclusions

Tetra Tech has prepared a revised mineral resource estimate for the Antilla copper-molybdenum deposit using a block geostatistical block modelling approach constrained by sulphide mineralization wireframes. The block model was populated with copper and molybdenum values using ordinary kriging, informed from capped core composites. The mineral resources have been estimated in conformity with the generally accepted CIM *Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines*. The Mineral Resources Statement is reported in accordance with 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 resource will be converted into mineral reserve. The effective date of the Mineral Resource Statement is October 19, 2015. At a cut-off grade 0.175% copper equivalent (CuEq), the Antilla project is estimated to contain 291.8 million tonnes at 0.34% copper, 0.009% molybdenum, and 0.36% CuEq in the Indicated category and 90.4 million tonnes at 0.26% copper, 0.007% molybdenum, and 0.28% CuEq in the Inferred category. The mineral resources are constrained within a conceptual pit shell.

24.2.2 Risks

There are no significant additional geological risks at the Antilla deposit outside of those risks reflected in the classification of mineral resources.

24.2.3 Opportunities

The sulphide mineralization is interpreted to be open along strike. There is an opportunity to expand the mineral resources laterally by further exploration drilling. There are also a number of additional targets identified from geochemical exploration that warrant additional exploration, including the Chabuca and North Block targets.

24.3 Mining

24.3.1 Conclusions

The mineral resources are well defined inside the selected conceptual pit limit. The proposed mill feed in the conceptual open pit mine schedule comprises 83% of material classified as Indicated.

The geotechnical recommendations for the pit wall slopes have been developed based on geotechnical logging of core, rock quality evaluation, and compressive strength testing on a limited number of core samples. The geotechnical information is considered adequate for defining conceptual pit slope bench-berm configurations for a preliminary economic assessment.

24.3.2 Risks

The preliminary final pit high wall is over 900 metres in vertical height. If future geotechnical investigations result in shallower overall pit slope angles, then the stripping ratio will increase and have a negative impact on the project mining costs.

24.3.3 Opportunities

Potential opportunity exists to optimize overall pit slopes by incorporating controlled blasting to steepen bench face angles.

24.4 Mineral Processing & Metallurgy

24.4.1 Conclusions

The metallurgy of the Antilla sulphide mineralization has been investigated in two testwork programs: a single composite sample in 2011 at Inspectorate in Vancouver and two composite samples in 2013 at Certimin in Lima. Provided the two samples are representative of the sulphide mineralization as a whole, the expected metallurgical performance assumed in this study should be appropriate. Additional testwork is required to confirm these assumptions.

The 2013 testwork program tested two main mineralization domains: Primary Sulphide and Supergene and show that a bulk copper-molybdenum concentrate could be recovered by flotation to a saleable grade with good recovery. There does not appear to be any deleterious elements present at penalty levels. Precious metal content is likely below payable levels.

There is a potential for acid generation from the flotation tailings due to the sulphide and, in particular, pyrite content. The molybdenite concentrate separated from the bulk concentrate may contain elevated levels of copper (and possibly zinc) which would require additional treatment such as ferric chloride leaching to reduce to non-penalty levels.

The metallurgical testwork results indicate a relatively soft material amenable to conventional crushing, grinding and flotation of a bulk concentrate, followed by molybdenum separation. The limited testwork to date does not suggest any issues with concentrate quality, recovery, thickening, or filtering. Costs associated with such a process plant should be quite predictable and comparable to existing operations in this region of Peru.

24.4.2 Risks

Limited metallurgical testwork was completed on samples from the sulphide mineralization. A number of risks have been identified, including the following:

- Inclusion of the Cover/Overburden and Leach Cap domains without any metallurgical testwork poses a possible risk to copper recovery in the final year of production.
- The mine production plan indicates Primary Sulphide material below the copper grade tested in the single composite sample. This could result in lower bulk concentrate recoveries and therefore molybdenum concentrate production as well.
- The possibility of greater amounts of acid-soluble copper minerals in all domains which would negatively impact copper recovery.
- The Supergene recovery below that estimated herein due to a very low concentrate grade being produced with extended cleaner flotation time.
- Costs associated with the acid-generating tailings.
- The requirement for additional treatment of the molybdenum concentrate to reduce its copper grade to non-penalty levels (and possibly zinc).

24.4.3 Opportunities

In contrast, there are a number of opportunities:

- Mill feed hardness could remain soft and therefore reduce crushing and grinding circuit power requirements and lower steel media and liner wear rates.
- Higher copper recovery could be demonstrated through additional metallurgical testwork.

24.5 Project Infrastructure

24.5.1 Conclusions

Planning of logistics will play a key role in supporting construction and operation of the proposed project. Site infrastructure for the proposed project includes access roads, camp facilities, offices, shops, warehouses, open pit, tailings and rock storage facilities, raw-water storage facility, water diversion structures, explosive storage, and settling ponds.

The project’s annual power consumption is estimated at 65 MW and it will be supplied by connecting to the national Peru electricity grid via a 50-kilometre transmission line from the Cotaruse substation. Site roads would be constructed with local borrow material and mine waste rock.

Tailings would be pumped to the tailings storage facility via a pipeline. Water will be reclaimed from the tailings storage facility and returned to the plant via a dedicated reclaim barge and pipeline. Surface runoff will be diverted away from the mining, tailings, and rock storage areas to minimize contact water volumes. Surplus contact water will be collected and directed to two contact water settling ponds. Water collected in the main contact water settling pond and in a fresh water reservoir will provide the mine water supply.

The capacity of the proposed tailings storage facility is designed to contain the tailings solids generated by the milling process over the mine life. The upstream lined containment dam will be constructed in stages, using predominantly waste rock material.

24.5.2 Risks

Project Infrastructure

- The proposed project assumes that the concentrate can be shipped from the existing Marcona port facilities without requiring additional capital costs. There is a risk that the Marcona port facilities will be inadequate, requiring either additional project capital expenditures or additional concentrate transportation costs to an alternate port.
- The project site is accessible from Carretera Interoceánica Highway at Santa Rosa via a 60-kilometre-long gravel road. The road is publicly maintained and may require more upgrades than the proposed switchback radius enhancements and drainage improvements. Additional upgrades may include road widening, road strength increase at some sections (compaction of a rock base formation) to accommodate heavy loads, and the bridge improvement at the Antabamba River crossing near the village of Saraica. Road upgrades would need to be completed prior to the project's construction phase.
- Water for the proposed project is considered to be obtained directly from creeks and rivers within the project area. If needed, water could also be pumped from the Antabamba River or from two small lagoons in the vicinity. Although water storage facilities are planned, prolonged dry seasons could make water unavailable for the project at times during construction or operations.
- No provision has been included for treatment of the contact water coming from the pit, process plant, tailings and rock storage facilities should discharge to the environment prove necessary. Future studies or permitting requirements may indicate a need for water treatment, thus increasing the cost and potentially impacting negatively the proposed project schedule.

Tailings

The following risks should be evaluated as the proposed project advances:

- Following completion of additional tailings and waste rock geochemical testing, as well as hydrogeological testing of the proposed tailings storage facility basin and containment dam foundation, and in accordance with Peruvian regulations, it is possible that additional seepage reduction and/or interception, collection and treatment works may be required. Should this be necessary, it would result in increases in the capital and sustaining capital cost of the project.
- Foundation characterization investigations are needed for the containment dam and the basin. These studies should include comprehensive geotechnical and hydrogeological characterization. Identification of deep overburden soils, or significant fractured rock would require cost comprehensive changes to the preliminary design as presented. This includes a need for additional excavation and possibly additional seepage control mitigation measures such as grouting with a commensurate increase in costs.

- Completion of a detailed tailings and site wide water balance may result in a need for increased water storage capacity for the basin which would require a larger dam, with subsequent direct impact to the capital costs.
- Unavailability of nearby suitable borrow materials to supplement the dam construction demands, and the closure requirements may result in increase in capital and sustaining capital costs.
- Detailed geotechnical analysis of the containment dam, especially under seismic loading conditions may demonstrate a need for a shear key, as well as possible flatter slopes and/or a buttress. Such modifications to the dam would result in increased capital and sustaining capital costs.
- Geotechnical and hydrogeological technical analysis of the coffer dam design may result in increased cost for this component of the project.

24.5.3 Opportunities

Opportunities that should be explored in future stages of development include:

- Use of tailings thickeners to increase tailings density with a commensurate reduction of required storage space. This would result in a reduced dam height with opportunities for cost reductions.
- Following comprehensive tailings physical characterization, it may be possible to demonstrate increased settled dry densities which would improve storage efficiency and allow for reduction of the dam size.
- Tailings physical characterization coupled with additional geotechnical analysis may allow for consideration of alternate tailings management strategies such as reconsidering a centerline raise method or considering construction of the containment dam from cyclone tailings. Such changes can result in cost savings over the life of mine.
- Through development of a detailed tailings deposition plan, taking into consideration the beaching characteristics of the tailings, it may be possible to reduce the overall height of the containment dam.
- Further geochemical testing of the tailings may eliminate the need to construct a low permeability cover over the tailings at closure, which would subsequently provide a considerable cost savings at closure.
- The next project phase should examine the potential for adding the concentrate filtration plant and truck load out facility in the plant site general area in order to potentially reduce the costs for concentrate pipeline handling.

24.6 Environmental and Permitting

24.6.1 Conclusions

Limited environmental baseline data has been collected for the site focussing on environmental permitting in support of exploration activities. Additional baseline work will have to be initiated together with public consultation in support of commencing with and environmental impact assessment for the proposed project.

24.6.2 Risks

Environmental, social, regulatory and closure risks that should be considered include:

- Completion of the baseline studies may require two or more years of data collection which would have an impact on both cost and schedule.
- Approval for the construction of the proposed tailings storage facility in the Huancaspaco River valley will require explicit regulatory approval as the current regulations impose limitations on this type of facility. Should regulatory approval for this site be denied, it would have a material impact on the project's economics.
- To date there has been some formal public consultation, limited to exploration permiting objectives, with interested and affected communities. Additional work will need to be conducted as part of the development of the project.
- It is possible that a low infiltration cover, or perpetual capture and treatment of leachate water from the rock storage facility and downstream tailings storage facility containment dam slope may be required as a result of the waste rock geochemistry. This will result in a significant increase in the overall closure costs.
- Water quality in the pit lake may not be of suitable quality to discharge to the environment as a result of the pit wall geochemistry. This may require covering of pit walls and/or perpetual water treatment. These measures would significantly increase the closure costs.
- Placement of a closure cover on the tailings storage facility surface could be problematic due to the surface not being trafficable for a long period post-closure. This may require a long care and maintenance period or the possible inclusion of a trafficability layer to the cover design, consisting of waste rock which would significantly increase the closure costs.

24.6.3 Opportunities

From the environmental, social, regulatory, and closure perspective the following opportunities should be explored:

- Further geochemical testing of the tailings may change the need to construct a low permeability cover over the tailings at closure, which would subsequently provide considerable cost savings at closure.
- As engineering on the proposed project advances, there may be increased opportunities for progressive reclamation which would reduce the post-closure bond requirements.

24.7 Market Studies

No market studies specific to Antilla concentrates have been performed and no concentrate sale contracts are in place. The study is based on long term copper and molybdenum prices of US$3 per pound and US$12 per pound, respectively.

It is assumed that copper concentrate will be shipped to Asia smelters. Assumed copper concentrate terms include a copper payable deduction of one unit which, at the forecast copper concentrate grades ranging from 20% to 30% copper, average 96.1% payable copper. Off-site costs for copper concentrate consist of transportation and smelting/refining charges. Transportation costs, including trucking to the port, port charges, ocean freight and insurance are estimated to total approximately US$101/wet metric tonnes copper concentrate. Smelting and refining charges of US$75/dry metric tonnes concentrate and US$0.075 per pound copper payable are assumed.

Assumed molybdenum concentrate off-site costs include roasting at US$1,268/dry metric tonne concentrate, and transportation and insurance costs of US$276/wet metric tonne concentrate, nominally based on trucking to a molybdenum roaster in Chile.

24.8 Capital Costs

Initial capital costs for the proposed project are estimated at US$603 million over a two year construction period. Sustaining capital costs, principally for mining equipment and tailings storage facility expansion, are estimated at US$324 million over a 24 year operational phase, for total life-of-mine project capital cost of US$927 million. Mine closure costs are estimated at US$92 million.

24.9 Operating Costs

Estimated on-site operating costs for the proposed project average US$9.10/tonne plant feed and total of US$3,188 million over the 24 year mine life. Estimated off-site operating costs for concentrate transportation, smelting, refining and roasting total US$900 million over the mine life.

24.10 Economic Analysis

24.10.1 Conclusions

The proposed Antilla project has been evaluated on a discounted cash flow basis in constant fourth quarter 2015 US dollars assuming all equity project financing. Project costs that will be incurred prior to the start of a two year construction period (i.e., for additional exploration and more detailed project studies) are excluded from the evaluation.

The economic analysis shows the Antilla project mineral resources to be potentially viable. Antilla C1 cash costs and C2 production costs are estimated at US$1.83/pound payable copper and US$2.35/pound payable copper, respectively. At a base case copper price of US$3.00/pound and molybdenum price of US$12/pound, the potential post-tax present value of the net cash flow at the start of the projected two-year construction period using a 7.5% discount rate ($PVNCF_{7.5\%}$) is estimated at US$225 million. The potential post-tax internal rate of return (IRR) is estimated at 15.1%. The project payback period is estimated to be slightly over four years.

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

A preliminary economic assessment should not be considered to be a preliminary feasibility or feasibility study, and the economics and potential technical viability of the proposed Antilla project have not been demonstrated at this time. The preliminary economic assessment is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. Furthermore, there is no certainty that the conclusions or results as reported in this preliminary economic assessment will be realized.

24.10.2 Risks

There is a risk of lower metal prices since current copper and molybdenum spot prices are less than the base case long term metal prices assumed in this study. The potential viability of the proposed project is very sensitive to metal prices, particularly the price of copper, which is the source of 92% of base case revenue. A US$0.50/pound reduction in the copper price to US$2.50/pound reduces the

post-tax IRR by 11.3%. A US$0.50/pound increase in the copper price to US$3.50/pound increases the post-tax IRR by 7.5%.

There is a risk to the potential viability of the proposed project, should head grades, operating costs, or capital costs differ from the estimates in this conceptual study. The proposed project's potential $PVNCF_{7.5\%}$ and IRR are shown to be quite sensitive to these input parameters.

Mine closure annual financial assurances in the form of insurance, cash collateral, a trust agreement or other forms permitted under Peruvian law are not addressed in the study discussed herein. If financial assurances are in the form of cash collateral this would advance closure related costs and reduce the proposed project's potential $PVNCF_{7.5\%}$ and IRR.

24.10.3 Opportunities

The annual cash flow model shows that the last four years of the proposed operation, i.e. Years 21 to 24, exhibit very marginal, albeit positive, net cash flow contributions. During this period low grade mineralization stockpiled in prior years is processed. Possible options that may improve overall project economic results:

- Exclude the low grade material from plant feed, which would reduce sustaining capital costs for the tailings storage facility planned in prior years to develop tailings capacity for the low grade mineralization.
- Increase the plant capacity and process the low grade material in earlier years, which would increase plant capital cost but would eliminate the significant fixed costs associated with mining re-handle, processing and G&A after Year 20.

Exploration success could increase the mineral resource base and potentially improve project economics by extending the mine life or justifying an increase in the plant throughput rate.

Joint operation with Panoro's other project in the region, Cotabambas, could provide synergies that would reduce the G&A costs attributed to the Antilla project.

25 Recommendations

25.1 Geology and Mineral Resources

Tetra Tech recommends additional drilling be completed to investigate further mineralization on the property currently not in the mineral inventory. Additional drilling should increase the confidence in both the continuity and extent of the copper and molybdenum mineralization. The recommended drilling includes infill drilling within the centre of the deposit where borehole spacing is greater than 100 metres, and areas along the edge of the know deposit where borehole data are relatively scarce.

Tetra Tech recommends a minimum of 24 boreholes at a minimum depth of 200 metres per borehole, or a minimum of 4,800 metres. The proposed minimum budget for such a drill program is approximately US$1.4 million.

Tetra Tech considers that additional exploration targets exist on the property; Tetra Tech recommends Panoro continue exploration work with a focus on the Chabuca and North Block targets.

25.2 Mining Methods

The following activities are recommended to progress the project forward:

25.2.1 Mining Recommendations

To advance the project Moose Mountain recommends additional mining related work as follows:

- Update and optimize the ultimate economic pit limits using only Measured and Indicated mineral resources from the block model using the latest infill drilling.
- Design pits, dumps and roads at a level to support a preliminary feasibility study.
- Trade-off study to optimize mining equipment size based on various ultimate pit limits and mill throughputs.
- Examine alternate pit phase designs to allow backfilling strategy to be used.
- Optimize the production schedule through examining various stockpiling scenarios and cut-off grade strategies along with stockpile and rock storage facility locations.
- Examine the potential of a contractor mining fleet to reduce initial capital costs.

The total costs estimated are between US$200,000 and US$300,000.

25.2.2 Geotechnical, Geochemistry, Hydrology, and Hydrogeology

- Conduct geotechnical site investigation program that includes five additional geotechnical core boreholes and provide samples for laboratory testing to further evaluate rock mass conditions and structural orientations at the next level of study.
- Full geochemical characterization of all the materials will be required in assessing the scale of potential acid rock drainage.
- Further hydrogeological and hydrological site characterization, taking seepage and runoff management requirements into account.

- Detailed water balance model that shows sufficient sources of water available for the project.

The total costs for the proposed rock geotechnical, hydrological, hydrogeological and geochemical investigations are estimated at approximately US$1,650,000.

25.3 Mineral Processing & Metallurgy

Metallurgical testwork to date have tested the two main mineralization domains and show the sulphide mineralization in these domains is amenable to conventional processing by grinding and flotation to achieve saleable copper and molybdenite concentrates. No testwork to date has been conducted on the minor Cover and Leach Cap zones. Future metallurgical testwork is recommended.

Additional metallurgical samples should be collected for testing from all four mineralization domains, representing different locations in the deposit:

- The samples should cover a wider range of copper grade and mineralogy to quantify the effect on copper flotation performance. In particular, samples of 0.2% copper grade or lower.
- Samples of Cover/Overburden and Leach Cap should be tested.
- Variability testing program should be completed, collecting a greater number of localized samples rather than broad-range, composite samples.
- Testwork should include comminution, flotation and dewatering. If significant variability within a mineralization domain is indicated, then a geometallurgical testwork program is recommended.

The testwork program should include:

- Range of comminution tests to estimate crusher, SAG mill, ball mill and regrind power requirements. Similarly, media and liner wear rates need to be estimated.
- Copper mineralogy should be investigated using an automated analysis such as QEMSCAN to quantify liberation levels and association with pyrite and molybdenite.
- Complete full minor element analysis of final concentrates.
- Determine the grind-recovery and grade-recovery relationships for copper flotation.
- Determine the molybdenum grade-recovery relationship with additional cleaning stages.
- Investigate ferric chloride leaching of final molybdenite concentrate.
- Conduct ABA testwork on the flotation tailings for acid-generating potential.
- Perform settling/filtration testwork for thickener/filter equipment sizing.

Depending on the number of samples tested, it is expected that this testwork program would cost between US$250,000 and US$500,000 and be completed in 6 to 12 months. Additional drilling will be required to collect the metallurgical samples (2 to 5 core boreholes at a cost of approximately US$300,000.

For molybdenite concentrate testing, processing of a bulk sample of representative feed (or of each mineralization domain) through a pilot plant is recommended. Due to the small amounts of molybdenite concentrate generated from small-scale batch testing, operation of a pilot plant will provide much larger amounts of concentrate for downstream testing. Similarly, samples of copper concentrate and tailings can be tested for dewatering equipment sizing.

25.4 Project Infrastructure

Completion of additional infrastructure-related studies, including:

- Complete process and infrastructure design and associated costs to a preliminary feasibility study level.
- Complete an updated power supply study.
- Complete hydrogeological studies including the site water balance, contact water and fresh water strategy, confirmation of water treatment requirements.
- Complete logistics and transportation study that defines the best transportation route and upgrades required to the access road.
- Complete a logistics and marketing study. Ideally Marcona Port would serve as the logistics hub for the project. Panoro does not have an agreement with the port authorities to establish a customs port of entry, therefore such agreement and confirmation of available port facilities and costs associated with concentrate handling at the port, will be required to meet the needs of the project.

For the tailings management system design, the following work is recommended as the project advances:

- Complete a trade-off study on alternate tailings deposition strategies to consider opportunities such as cycloning, centerline raise, or alternate sites.
- Confirm the viability of construction of the tailings storage facility within the Huancaspaco River valley from a regulatory perspective.
- Conduct geotechnical and geohydrological characterization studies along the dam foundation as well as the storage basin foundation to confirm the design concept.
- Carry out detailed hydrology studies to establish the coffer dam and diversion structure design criteria, as well as the final closure spillway.
- Complete geochemical and physical characterization studies on the tailings material to confirm closure and operational water discharge limitations as well as beaching and density characteristics to confirm containment dam size.

25.5 Environmental and Permitting

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.

25.6 Economic Analysis

It is recommended that Panoro proceed with further technical studies leading toward the preparation of an Antilla preliminary feasibility study. The results of the preliminary economic assessment

discussed herein indicate the proposed Antilla project is potentially viable, and the favorable economic potential warrants further, more detailed, technical studies.

For economic modelling, further project studies should include more detailed analysis of:

- The form of mine closure annual financial assurance requirements, and associated taxation implications;
- Categorization of project capital expenditures into appropriate Peru asset classes with associated depreciation rates;
- Tax rates and depreciation rates with and without a TSA.
- VAT payments and recovery.

25.7 Recommendations and Estimated Budget

Table 69 shows a summary of recommendations per discipline and the total costs.

SRK is unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform the exploration work recommended for the Antilla project.

Table 69: Summary of Cost for Recommended Future Work

Task	Estimated Cost (US$'000)
Geology / Mineral Resource	1,400
Mining	300
Metallurgy	800
Infrastructure	1,950
Geotechnical, Hydrological, Hydrogeological, Geochemical Studies	1,650
Environmental Impact Studies	2,900
Marketing Study	100
Preliminary Feasibility Study	2,000
Total Cost for Recommended Future Work	**$11,100**

26 References

ATC Williams. 2015. Antilla: summary of Conceptual TSF Design Approaches – TSF E9. Memorandum to Minera Panoro (Peru) S.A.C. Reference No. 115131.01.002. July 22, 2015.

ATC Williams. 2016a. Antilla Project, Preliminary optimization and evaluation of opportunities for TSF (valley fill) – Draft I. Technical note to Minera Panoro (Peru) S.A.C. Reference 115131.01-C001. January 26, 2016.

ATC Williams. 2016b. Antilla_PEA_Financial_Model_40 Ktpd-TSF_E10 staged-DraftC.xlsx. Excel Spreadsheet prepared for Minera Panoro (Peru) S.A.C.

Certimin, 2013. Informe Metalúrgico Final (Rev-01) (COT SM 0026 03 13/CERTIMIN) O/T Certimin No. JUN4007.R13. Pruebas Metalúrgicas para Mineral de Cu-Mo. 8 August, 2013. p. 408.

DGAAM-MEM, 1995, Guía Ambiental para el Manejo de Drenaje Acido de Minas (Environental Guideline for Acid Mine Drainage Prediction, Ministry of Energy and Mines of Peru), p. 23-24.

Dunn, K., 2008. Petrographic Description of 19 Drill Core Samples and 2 Hand Samples, Antilla Project, Peru. Unpublished report dated 3 November, 2009. p. 99.

Fox, J., 2006. Ore Processing for Antilla and Contabamba (sic) Projects, Peru. Unpublished report concerning process options for the Antilla property. 1 September, 2006. p. 7.

Kirkham and Sinclair, 1975. Porphyry copper, gold, molybdenum, tungsten, tin, silver, in Eckstrand, O.R., Sinclair, W.D., and Thorpe, R.I., eds., Geology of Canadian Mineral Deposit Types: Geological Survey of Canada, Geology of Canada, no. 8, p. 421-446.

KPMG, 2013. Peru - Country Mining Guide. KPMG International. Available from https://www.kpmg.com/Ca/en/industry/Mining/Documents/Peru.pdf. Accessed May 2016. p. 9.

KPMG, 2016. Peru - Country Mining Guide. KPMG International. Available from https://www.kpmg.com/Ca/en/industry/Mining/Documents/KPMG-Mining-Country-Guides-Peru.pdf. Accessed May 2016. p. 11-13.

Lee, C. Nowak, M. and Wober, H. H., 2007. Independent Technical Report on the Mineral Exploration Properties of the Cordillera de las Minas S.A. Andayualas-Yauri Belt, Cusco Region, Peru. Unpublished NI 43-101 compliant technical report for the Antilla, Cotabambas and other regional exploration properties. p. 125.

Lowell, J.D., and Guilbert, J.M., 1970. Lateral and vertical alteration-mineralization zoning in porphyry ore deposits: Economic Geology, v. 65, p. 373-408.

MEND Report 1.20.1, 2009, Prediction Manual from Drainage Chemistry from Sulphidic Geologic Materials, p. 14-8 to 14-14.

Pecho, V, 1981. Geología de los cuadrángulos de Chalhuanca, Antabamba y Santo.

Perelló J., Carlotto V. Zarate A.,Ramos P., Posso H., Neyra C., Caballero A., Fuster N., and Muhr R., 2003, Porphyry Style Alteration and Mineralization of the Middle Eocene to Early Oligocene Andahuaylas – Yauri Belt, Cuzco region, Peru; Econ.Geology, vol.98, 2003, p. 1575-1605.

Sector Energia y Minas, Instituto Geologico, Minero y Metalurgico (INGEMMET), 2004. Mapa geologico 50,000 sheet 29-p and 29-q

Sinclair, W.D., 2007. Porphyry deposits, in Goodfellow, W.D., ed., Mineral Deposits of Canada: A Synthesis of Major Deposit-Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods: Geological Association of Canada, Mineral Deposits Division, Special Publication No. 5, p. 223-243.

Southern Copper, 2012. Crecimiento Informe Annual 2012. p. 9.

Tejada, F.L., Mamani, J.V., and Espoinoza, B., I., 2013. Descubrimiento y Geología del Pórfido Cu – Mo Trapiche, Apurímac – Perú. Abstract presented at the Perumin Convention, Arequipa.

Tetra Tech, 2013. Technical Report and Resource Estimate of the Antilla Copper-Molybdenum Project, Peru, Document No. 1397600100-REP-R0001-05, 16 December 2013. p. 130.

Vale, 2002. Press release dated July 19, 2002. p. 1.

Wright, C., 2009. Restated, Amended Technical Report for the Antilla Property, Apurimac, Peru. Project No. 160972. 23 August 2009. 100 pages.

APPENDIX A

Mineral Tenure Information
And Legal Title Opinion

ABOGADOS | ATTORNEYS AT LAW ROSSELLÓ

March 10th, 2016

Messers
SRK Consulting
Present. -

Attention: Andric Goran

Dear Sirs,

We have been asked to issue a legal opinion about the status of the 12 mining concessions of the Antilla Project (Antilla Concessions), owned by PANORO APURIMAC S.A., and set out in Annex I hereto.

For the purpose of this letter, we have reviewed originals or copies of the mining concessions, entries of them in the Register of Mining Rights, information available on the INGEMMET (authority that approves the concessions and oversees the compliance with obligations to maintain the concessions currently) as well as various information provided by PANORO or obtained directly by us, detailed in the Annex.

The legal opinion included in this letter is based on the laws of Peru, in force to date, and assumes:

i. the truth and validity of all signatures, stamps and securities, the compliance of all originals, copies, photocopies and certified copies of such documents,

ii. that there is no agreement in existance to transfer any of the mining concessions.

Based on these assumptions, we are of the opinion that, as of the date of this letter:

1. The 12 Antilla Project Mining Concessions are in force and registered in the name of PANORO APURIMAC S.A. as its owner.

   The Antilla mining concessions have been requested and processed thus complying with the standards of the General Mining Law, having obtained of the competent authority the title of mining concession, which is in force and free of burden or lien.

Av. Camino Real 348, Torre El Pilar, Piso 12, San Isidro, Lima 27, Perú T (511) 230 0000 F (511) 222 2555 | rossello@rossellolaw.com I www.rossellolaw.com

ABOGADOS | ATTORNEYS AT LAW ROSSELLÓ

2. The company has paid the concessions fees (annual fees and penalty) in respect of all of the Antilla Concessions until the year 2014, which has met the requirements to grant the concessions until June 2016. Until June 2016 The Company must pay the concessions annual fees of one year (2015) to keep alive concessions until June 2017, subject to mining concessions are approved indefinitely. In the case of Macla 2003 concession, The Company has already payed the year 2015 annual fee.

We assume no obligation to update or supplement this opinion to reflect facts or circumstances that may hereinafter be our knowledge, or any changes in laws, rules, regulations or judicial or arbitral decisions that may occur hereinafter.

Yours faithfully,

Humberto Martínez Aponte

ROSSELLÓ
ABOGADOS | ATTORNEYS AT LAW

MINING PROPERTIES FROM ANTILLA PROJECT - PANORO APURIMAC S.A. (PASA)

N°	DENOMINATION OF THE CONCESSION	OWNER	CODE	ZONE	PAGE	HECTARS	DATE COMPLAINT	Certification date	N° Entry	Departmental resolution	REVISION ESTUDIO ROSELLO AT 09th of March 2016	
											Ownership OK?	Validity ok?
	PROYECTO ANTILLA											
1	ALUNO CINCO 2002	Panoro Apurimac	010170402	18	29-Q	100.00	2/09/2002	31/01/2003	11027526	172-2003-INACC	PASA	Jun-16
2	ALUNO CUATRO 2002	Panoro Apurimac	010170302	18	29-Q	800.00	2/09/2002	31/01/2003	11027524	144 y 9142003-INACC	PASA	Jun-16
3	ALUNO QUINCE 2002	Panoro Apurimac	010200202	18	29-Q	900.00	15/10/2002	21/03/2003	11074624	708-2003-INACC	PASA	Jun-16
4	VALERIA QUINCE 2003	Panoro Apurimac	010043803	18	29-P	1,000.00	3/03/2003	27/01/2004	11074940	150-2004-INACC	PASA	Jun-16
5	VALERIA DIECISEIS 2003	Panoro Apurimac	010043903	18	29-P	900.00	3/03/2003	9/09/2003	11076530	2540-2003-INACC	PASA	Jun-16
6	VALERIA TREINTAIDOS	Panoro Apurimac	010329903	18	29-Q	800.00	13/10/2003	4/03/2004	11074941	649-2004-INACC	PASA	Jun-16
7	ANTILLANA 2003	Panoro Apurimac	010344303	18	29-P	1,000.00	30/10/2003	26/02/2004	11087019	588-2004-INACC	PASA	Jun-16
8	ANTILLANA UNO 2003	Panoro Apurimac	010344203	18	29-Q	800.00	30/10/2003	16/02/2004	11074938	364-2004-INACC	PASA	Jun-16
9	VALERIA SESENTAIUNO 2004	Panoro Apurimac	010166404	18	29-P	400.00	1/06/2004	9/08/2004	11074939	2802-2004-INACC	PASA	Jun-16
10	MACLA 2003	Panoro Apurimac	010002003	18	29-P	300.00	6/11/2003	17/06/2003	11030017	1488-2003INAAC	PASA	Jun-17
11	DON MARTIN 1	Panoro Apurimac	010313306	18	29-Q	300.00	18/07/2006	17/11/2006	11067064	4851-2006-INAAC	PASA	Jun-16
12	ANTILLA UNO	Panoro Apurimac	010059709	18	29-Q	200.00	25/02/2009	16/07/2009	11092677	2170-2009-INGEMET	PASA	Jun-16
						7,500.00						

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Annex 2

- Records of mining concessions revised at INGEMMET (grants approving authority).
- Electronic entries of each concession, entered in the Register of Mining Rights - SUNARP.
- Mining Record 2016 (named Padron Minero) posted by INGEMMET (authority controlling the validity of concessions)
- Historical record of payments (annual fees and penalty) in the INGEMMET.
- Payment voucher of annual fees and penalty delivered by PANORO

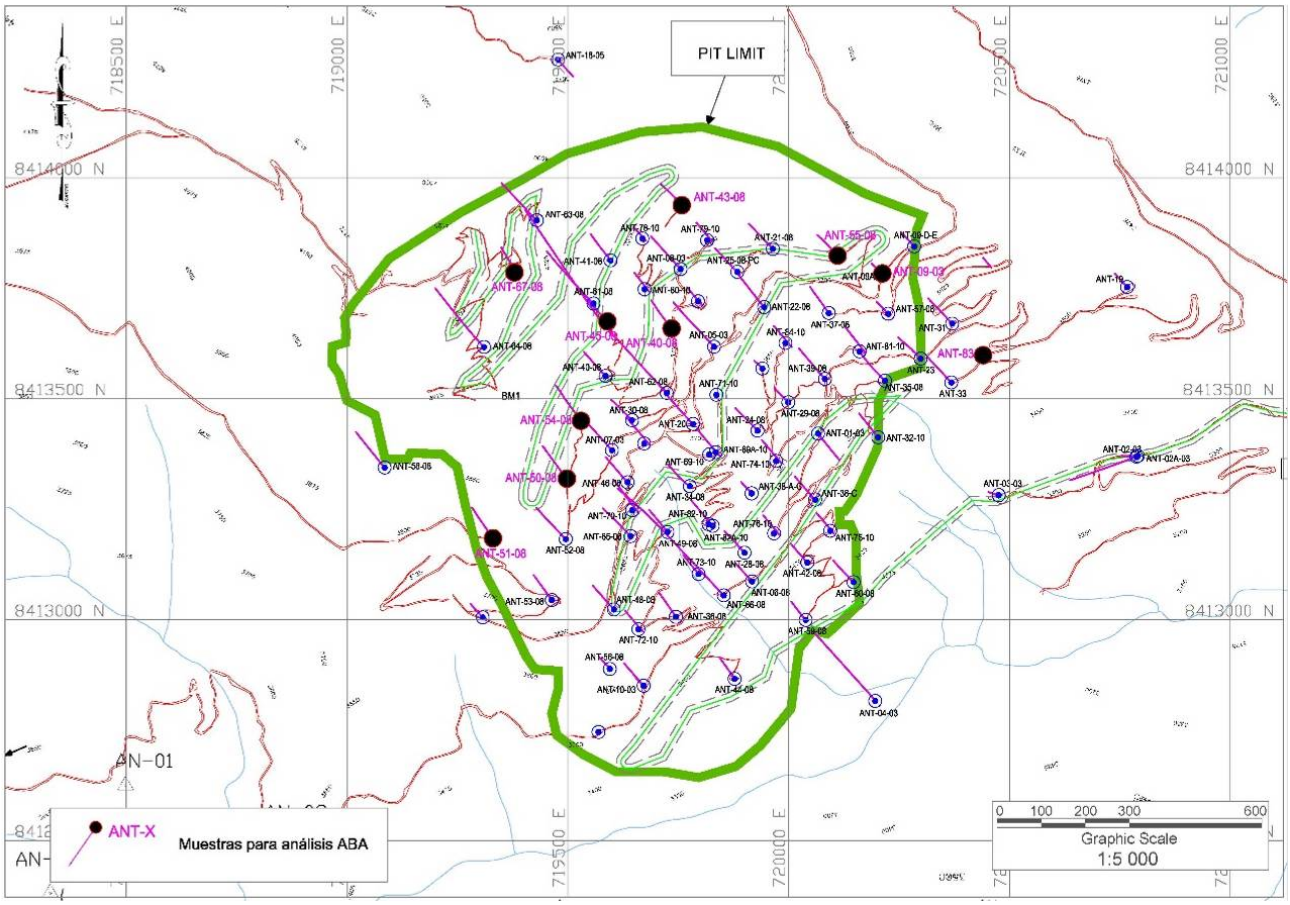
March 10th, 2016

Humberto Martínez Aponte

APPENDIX B

Testwork for Geochemical Characterization
Sample Locations
Sample Descriptions
Acid Rock Drainage Prediction

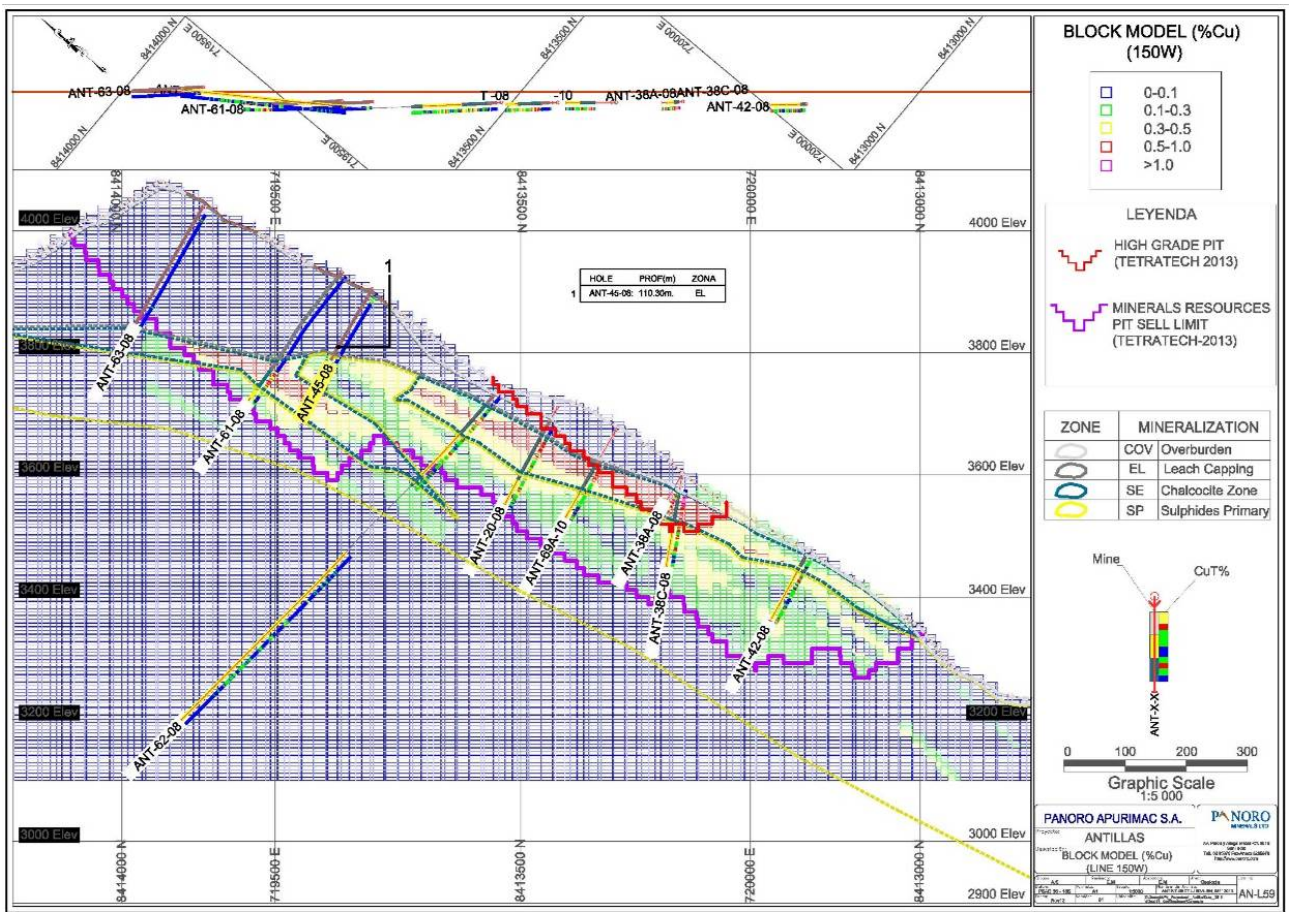
Sample Location



Map of the Location of the Boreholes Selected for Collecting Samples (From Panoro)

Most Representative Sections of the Locations of the Boreholes Selected for Collecting the Samples. Sections provided by Panoro

Section 150W



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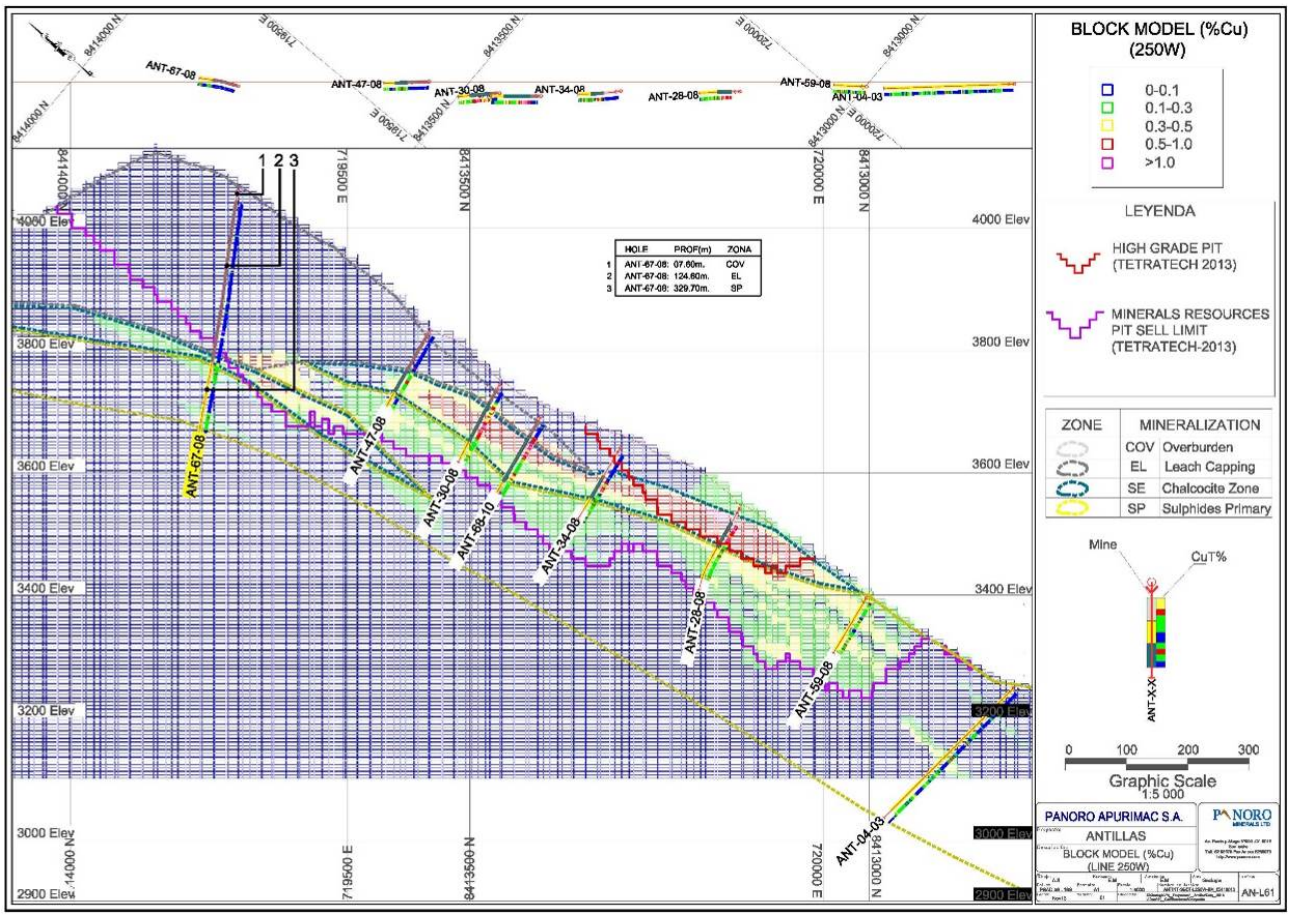

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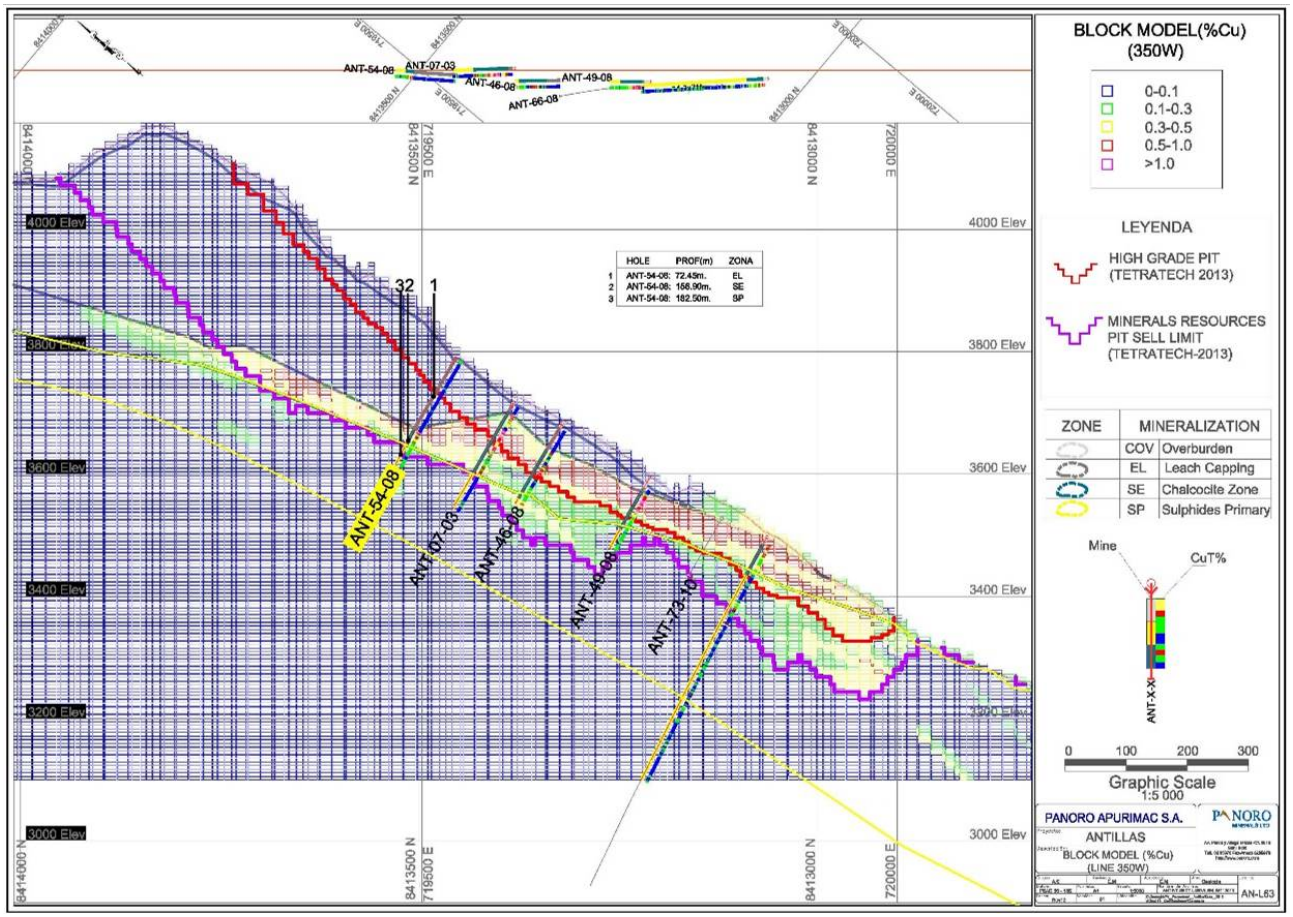

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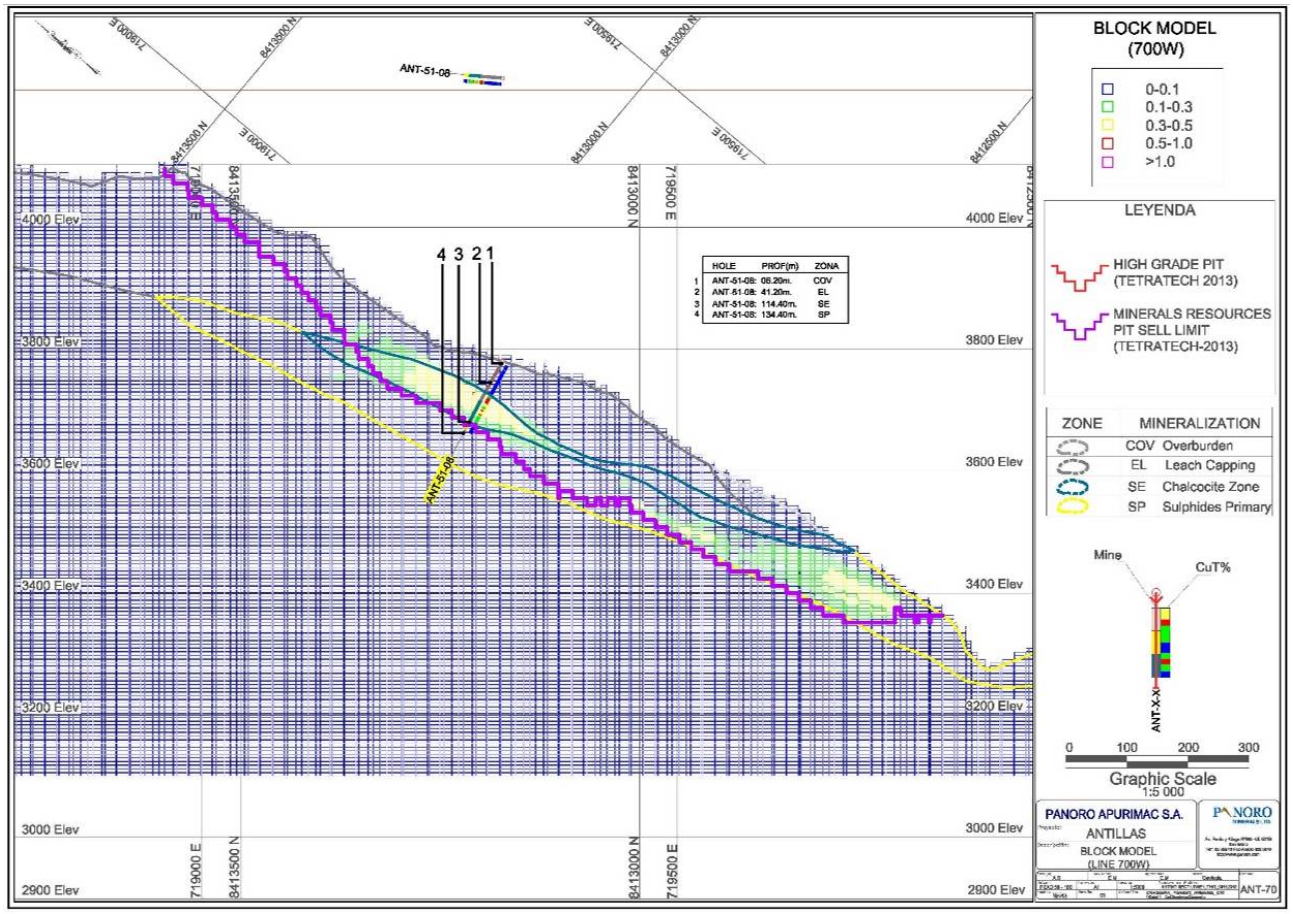
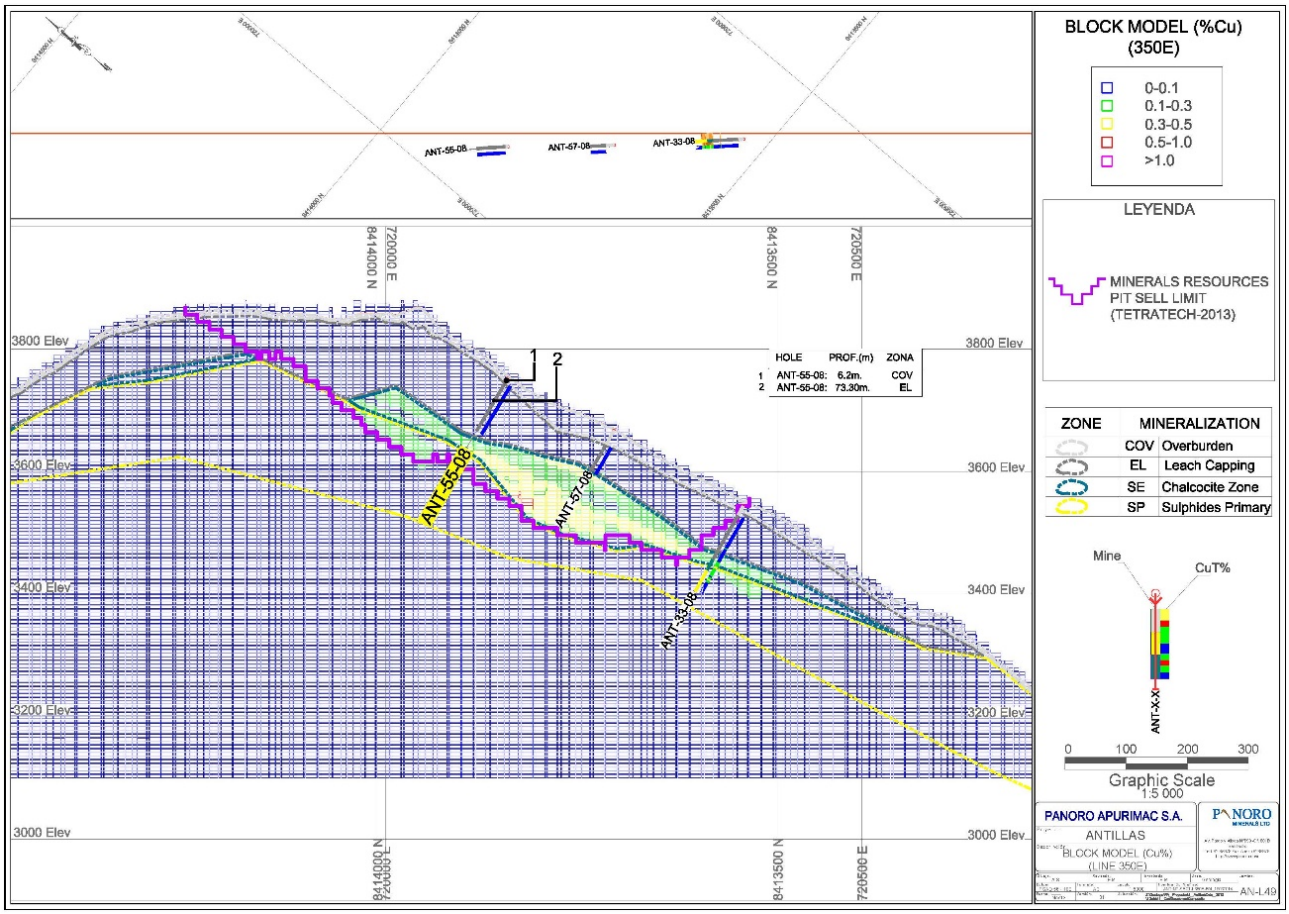


Section 350E



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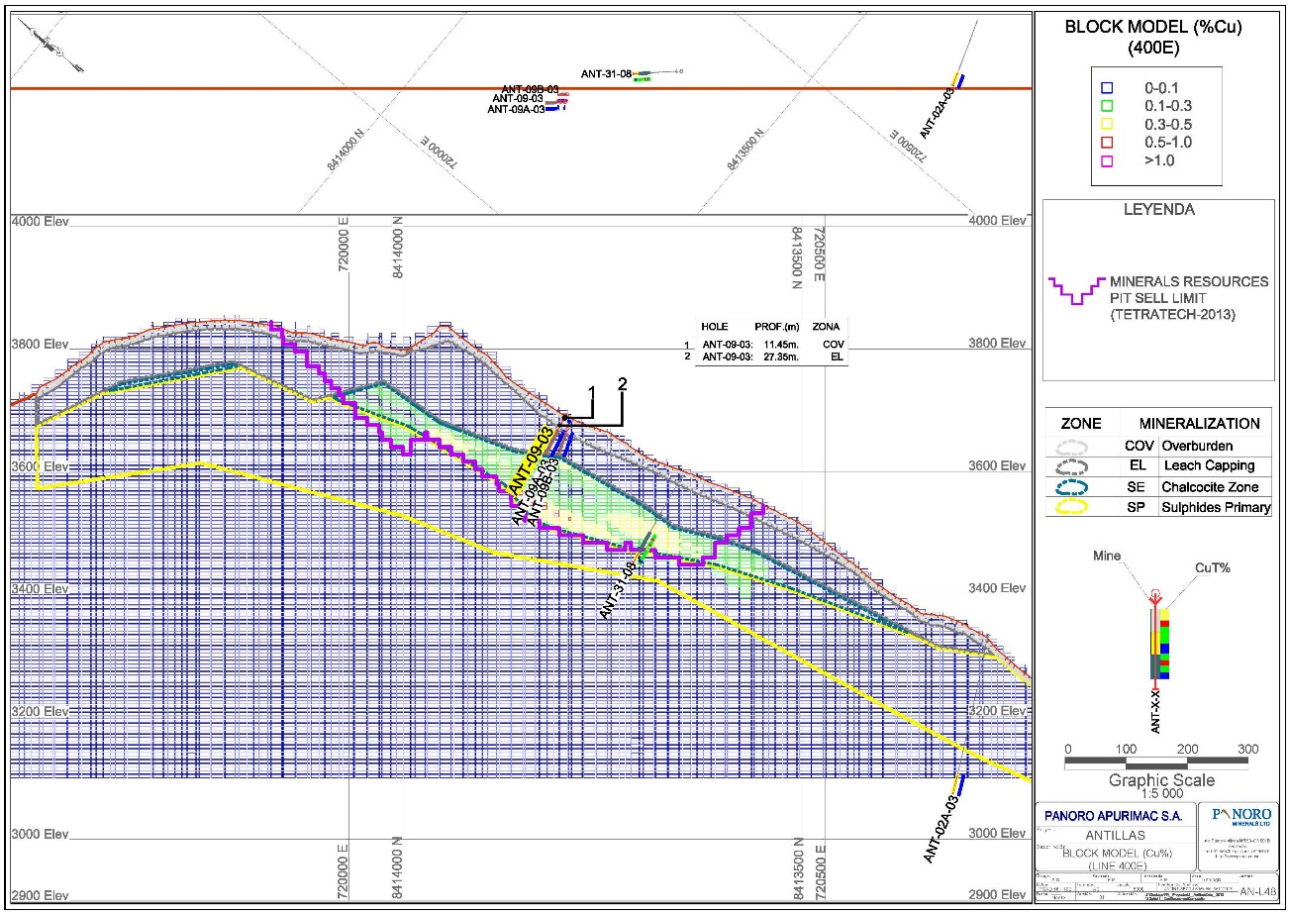

Sample Identification

ID	Mineralogical Domain	Description
COV-ANT-1A	Overburden	Colluvium with quartzite clasts and some claystone clasts
EL-ANT-2A	Leach cap	Quartzites (whitish grey, beige), with some 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 zone	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 zone	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).

Note: Samples were obtained and prepared by Panoro

Sample Description

Zone	Hole	From (m)	To (m)	Depth (m)	Weight (Approx.)	Lithology	Mineralization
COV	ANT-09-03	0	26.5	11.45	400g	Overburden. Quartzite clasts.	
	ANT-40-08	0	6	4.60	375g	Quaternary Coverage.	
	ANT-43-08	0	22	9.20	400g	Overburden zone with some quartzite clasts subrounded to subangular.	
	ANT-50-08	0	4.65	3.70	370g	Overburden. Quartzite clasts.	
	ANT-51-08	0	4.4	3.80	370g	Overburden zone with quartzite clasts subangular. Limonite in clasts.	
	ANT-55-08	0	8.2	6.20	350g	Colluvium.	
	ANT-67-08	0	17.5	7.60	375g	Overburden.	
	ANT-83-10	0	18.6	13.60	390g	Overburden.	
EL	ANT-09-03	26.5	29.15	27.35	380g	Whitish gray QUARTZITE.	Presence of FeOx (Jar, Goe and lesser Hem).
	ANT-40-08	6	11	10.60	340g	QUARTZITE, with some sandstone horizons. Stock Works weakly to moderately fractured.	Presence of strong and moderate Hem goethite, trace Py.
	ANT-45-08	0	134.7	110.30	340g	QUARTZITES, weakly sericitizised, moderately fractured, and presence of breccia.	Presence of FeOx (Jar, Goe and Hem).
	ANT-50-08	4.65	148	62.60	330g	QUARTZITE, weakly sericitizised, with layers of sandstone and presence of principal porphyry at the beginning of the section.	Goethite in fractures and traces of Hem Quartzite. In the main porphyry presence of Goethite in fractures are observed.
	ANT-51-08	4.4	76	41.20	380g	Late PORPHYRY, with layers of quartzites and sandstones, weakly sericitizised.	The PFT presents FeOx in fract., Cc disseminated in the matrix and in fract. and FeOx in fract. in quartzite and sandstone.
	ANT-54-08	0	135.8	72.45	360g	QUARTZITE, weakly sericitizised, with presence of Late Porphyry at the top.	Presence of goethite and jarosite on fractures, little Hematite.
	ANT-55-08	8.2	113	73.70	340g	QUARTZITE, slightly fractured, with layers of sandstone.	Quartzites with moderate leaching of FeOx (Hem, Goe); sandstones have horizons of Py.

Sample Description 2

Zone	Hole	From (m)	To (m)	Depth (m)	Weight (Approx.)	Lithology	Mineralization
EL	ANT-67-08	17.5	284.2	124.60	330g	QUARTZITE, whitish, moderately fractured, with some, medium to fine grained, sandstone horizons.	Presence of FeOx (hem, goe, Jar) in fractures; Py in fractures and patches of Cc and trace disseminated Cpy.
	ANT-83-10	18.6	57.8	35.20	350g	QUARTZITE beige, with some levels of fine-grained sandstones.	Sporadic traces of Py.
SE	ANT-40-08	11	62.8	45.50	500g	QUARTZITE s with levels of fine-grained sandstones.	Sporadic Feox (Hem, Goe); presence of Cc, Mo and Py.
	ANT-43-08	22	74	38.60	500g	QUARTZITE, whitish gray, weakly sericitizised. Presence of faults.	Some FeOx, more jar and less Hem, presence of Cc, Cpy, Mo and Py.
	ANT-50-08	148	185	149.30	520g	QUARTZITE with layers of sandstone and “principal” porphyry.	The quartzites and sandstones have disseminated Py and Cpy and fract, sporadic Mo; the PFP has disseminated Py.
	ANT-51-08	76	117	114.40	560g	Intercalation of QUARTZITE and SANDSTONES.	Cpy, Py and Cc, disseminated and in fract.
	ANT-54-08	135.8	166	156.90	550g	Intercalation of QUARTZITE and SANDSTONES.	Presence of Cc and Cpy, disseminated and in fract. Mo traces in quartzite, presence of disseminated Cpy, Py and Cc in sandstones.
	ANT-55-08	113	131.3	116.70	500g	QUARTZITE, moderately fractured, with yellowish brown sandstone horizons.	Some horizons of siltstone, some Py and Cc.

Sample Description 3

Zone	Hole	From (m)	To (m)	Depth (m)	Weight (Approx.)	Lithology	Mineralization
SP	ANT-40-08	62.8	160	84.40	600g	Intercalation of QUARTZITES and yellowish brown SANDSTONES and, slightly argilizised, SILTSTONES.	Some Feox (jar, goe), presence of Cc Cpy, Mo and Py.
	ANT-43-08	74	140	98.60	600g	Intercalation of white gray, weakly sericitized, QUARTZITES, with brown grey, strongly argilizised, SANDSTONES.	Sporadic presence of sulfides as Cc, Cpy, Py, Mo; Bn trace.
	ANT-51-08	117	180.8	134.40	650g	"Principal" PORPHYRY with sandstones and quartzites horizons.	Py and Cpy disseminated and in fract. Patches of Cc, Py in veinlets in SW; Mo traces.
	ANT-54-08	166	200.5	182.50	620g	Cuarcitas con niveles de areniscas e intercalaciones de PFP QUARTZITES and SANDSTONES with interbedded levels of PFP.	Py Cpy disseminated and in fractures, as well as patches of Mo and Cc.
	ANT-67-08	284.2	397.5	329.70	615g	Intercalation of silty SANDSTONES and moderately silicificated QUARTZITES.	Presence of Py, Cpy and Mo disseminated and fracturesa and in traces.

Borehole Collar Location

Borehole ID	UTM Easting	UTM Northing	Elevation (masl)	Depth (m)
ANT-09-03	720212.65	8413783.39	3691.50	29.15
ANT-40-08	719735.62	8413658.79	3763.64	160.00
ANT-43-08	719758.46	8413938.18	3880.71	140.00
ANT-45-08	719589.37	8413675.26	3909.42	134.70
ANT-50-08	719498.51	8413318.64	3787.58	185.00
ANT-51-08	719331.25	8413184.13	3789.28	180.80
ANT-54-08	719530.58	8413449.67	3792.03	200.50
ANT-55-08	720111.95	8413823.12	3752.00	131.30
ANT-67-08	719379.60	8413785.66	4069.87	401.10
ANT-83-10	720440.80	8413598.36	3556.87	57.80

Boreholes Orientation

Borehole ID	Depth (m)	Azimuth (°)	Dip (°)
ANT-09-03	29.15	318	-60
ANT-40-08	8.00	322	-60
ANT-40-08	50.00	322	-59
ANT-40-08	100.00	325	-60
ANT-40-08	160.00	329	-61
ANT-43-08	27.00	317	-60
ANT-43-08	50.00	317	-58
ANT-43-08	100.00	313	-62
ANT-43-08	140.00	313	-62
ANT-45-08	9.00	317	-60
ANT-45-08	50.00	317	-59
ANT-45-08	100.00	321	-60
ANT-45-08	122.00	319	-61
ANT-45-08	134.70	319	-61
ANT-50-08	6.00	323	-60
ANT-50-08	63.00	323	-58
ANT-50-08	124.00	324	-61
ANT-50-08	185.00	325	-61
ANT-51-08	50.00	325	-60
ANT-51-08	100.00	325	-62
ANT-51-08	150.00	327	-62
ANT-51-08	180.80	326	-63
ANT-54-08	50.00	324	-60
ANT-54-08	100.00	324	-59
ANT-54-08	150.00	328	-62
ANT-54-08	200.00	322	-62
ANT-54-08	200.50	326	-63
ANT-55-08	9.00	317	-60
ANT-55-08	50.00	317	-59
ANT-55-08	100.00	314	-59
ANT-55-08	131.00	313	-60
ANT-55-08	131.30	313	-60
ANT-67-08	100.00	336	-80
ANT-67-08	150.00	336	-83
ANT-67-08	200.00	331	-81
ANT-67-08	250.00	329	-80
ANT-67-08	350.00	328	-79
ANT-67-08	401.10	333	-81
ANT-83-10	57.80	320	-60

APPENDIX C

Laboratory Results of Acid Base Accounting Testing Modified Sobek Method

Analysis of the ABA Results

Modified Sobek Acid-Base Account Analysis – Methodology

Parameter	Reference	Description
Total Sulphur - outsourced	ASTM E1915-97	Modified Sobek
Inorganic carbon - outsourced	ASTM E1915-97	Modified Sobek
Fizz grade	MEND Project 1.16.3	Modified Sobek
Paste pH	MEND Project 1.16.3	Modified Sobek
Maximum acid potential (MAP)	MEND Project 1.16.3	Modified Sobek
Net Neutralization Potent (NNP)	MEND Project 1.16.3	Modified Sobek
Neutralization Potent (NP)	MEND Project 1.16.3	Modified Sobek
NP/MPA	MEND Project 1.16.3	Modified Sobek
CO_3 Leachable Sulphur - Gravimetric	MEND Project 1.16.3	Modified Sobek
HCl Leachable Sulphur - Gravimetric	MEND Project 1.16.3	Modified Sobek
Sulphur	MEND Project 1.16.3	Modified Sobek

Modified Sobek Acid-Base Account Analysis – Lab Results

N° ALS - Corplab	144675/2014-1.0	144676/2014-1.0	144677/2014-1.0	144678/2014-1.0
Sampling date	15/05/2014	15/05/2014	15/05/2014	15/05/2014
Sampling time	00:00:00	00:00:00	00:00:00	00:00:00
Type of simple	Rock	Rock	Rock	Rock
Identification	COV-ANT-1A	EL-ANT-2A	SE-ANT-3A	SP-ANT-4A

Parameter	Unit	Detection Limit	144676/2014-1.0	144677/2014-1.0	144678/2014-1.0	144676/2014-1.0
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		---	None	None	None	None
Paste pH		---	6,96	7,78	6,61	6,06
Maximum Acid Potential	$\frac{t\ CaCO_3}{1000t}$	0,5	0,6	< 0,5	10,3	15,3
Net neutralization potential	$\frac{t\ CaCO_3}{1000\ t}$	---	-2,6	-2,0	-14,3	-18,3
Neutralization potential	$\frac{t\ CaCO_3}{1000\ t}$	---	-2	-2	-4	-3
NP/MAP		---	-3,33	Not applicable	-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

Interpretation of Results

Modified Sobek Acid-Base Account Analysis – Interpretation

ID	Sample Description	Results	Conclusion
All samples		Negative neutralization potential, probably due to the presence of jarosite that produces acidity upon dissolution. Low % inorganic carbon, i.e. none or very low % of carbonates are present.	All samples are characterized by a negative neutralization potential
COV-ANT-1A	Overburden Cover: Colluvium with quartzite clasts and some claystone clasts	Low sulphur content: 0,02% Low MAP: 0,6 near neutral paste pH	Unlikely to produce acid rock drainage, based on low sulphur content (1) (2)
EL-ANT-2A	Leach Capping: Quartzites (whitish grey, beige), with some 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 chalcopyrite (disseminated and in fractures)	Low sulphur content: 0,01% Low MAP: < 0,5 Paste pH slightly alkaline	Unlikely to produce acid rock drainage , based on low sulphur content (1) (2)
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).	Sulphur content: 0,33% MAP: 10,3 NP/MAP: 0,39 paste pH near neutral	Likely, though uncertain, to produce acid rock drainage, based on sulphur content en NP/MAP. (1) (2)
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).	Sulphur content: 0,49% MAP: 15,3 NP/MAP: 0,20 paste pH near neutral	Likely, though uncertain, to produce acid rock drainage, based on sulphur content en NP/MAP. (1) (2)

1. If sulphur content would be higher the potential to generate acid drainage will be also as there is no positive neutralization potential.
2. The ferric-hydroxide-sulphate-jarosite is relatively soluble pH>4.5 (slow) and will undergo hydrolysis in moist environments to release the stored acidity (MEND report, pg. 5-35).

Modified Sobek Acid-Base Account Analysis – Interpretation Criteria

Criteria	Description
Sulphur content < 0,02%	Unlikely to produce acid rock drainage
NNP < - 20 t $CaCO_3$/1000t	Likely to produce acid rock drainage
- 20 t $CaCO_3$/1000t < NNP < 20 t $CaCO_3$/1000t	Uncertain to produce acid rock drainage
20 t $CaCO_3$/1000t < NNP	Unlikely to produce acid rock drainage
1 < NP/MAP [1]	Unlikely to produce acid rock drainage
NP/MAP < 1 [1]	Likely to produce acid rock drainage
pH paste	Gives an idea of pH of natural runoff water at present time

1. Not for tailings

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled **Preliminary Economic Assessment Technical Report for the Antilla Copper-Molybdenum Project, Peru**, dated June 16, 2016.

I, Goran Andric, residing at Etobicoke, Ontario do hereby certify that:

1) I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 1300 - 151 Yonge Street, Toronto, Ontario, Canada;
2) I am a graduate of the University of Belgrade; I obtained a Bachelor of Science degree in Mining Engineering in 1988. I have practiced my profession since 1989 including 14 years in operations and 8 years as a consultant and have experience working in a several copper projects in Canada and Peru;
3) I am a professional engineer registered with the Professional Engineers of Ontario, license number 100103151;
4) I have personally inspected the subject project from April 20 to 25, 2014;
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 co-author of this report and responsible for Sections 1, 2, 15.2, 17.1, 17.2, 17.3, 17.7, 18, 20.1.5, 20.2.5, 24.1, 24.3, 24.5, 24.7, 25.2.2, 25.4, 25.7 and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
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 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.

["Signed and Sealed"]

Toronto, Ontario, Canada
June 16, 2016

Goran Andric, PEng
Principal Consultant (Mining Engineering)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled **Preliminary Economic Assessment Technical Report for the Antilla Copper-Molybdenum Project, Peru**, dated June 16, 2016.

I, Adrian Dance, residing in West Vancouver, BC, do hereby certify that:

1) I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 2200-1066 West Hastings Street, Vancouver, BC, Canada;
2) I am a graduate of the University of British Columbia in 1987 where I obtained a Bachelor of Applied Science and a graduate of the University of Queensland in 1992 where I obtained a Doctorate. I have practiced my profession continuously since 1992 including 15 years as a consultant and have experience working in a number of iron and titanium operations around the world;
3) I am a professional engineer registered with the Association of Professional Engineers & Geoscientists of British Columbia, licence number 37151;
4) I have not personally visited the project area but relied on a site visit conducted by Goran Andric, PEng, a co-author of this technical report;
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 co-author of this report and responsible for Sections 12, 16, 20.1.3, 20.2.2, 24.4, 25.3, and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
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 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.

["Signed and Sealed"]

Vancouver, BC, Canada
June 16, 2016

Adrian Dance, PEng
Principal Consultant (Metallurgy)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled **Preliminary Economic Assessment Technical Report for the Antilla Copper-Molybdenum Project, Peru**, dated June 16, 2016.

I, Maritz Rykaart, residing in Surrey, British Columbia, Canada do hereby certify that:

1) I am a Principal Consultant with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 2200 – 1066 West Hastings Street, Vancouver, British Columbia, Canada;
2) I graduate from the Rand Afrikaans University in 1991 and 1993 and the University of Saskatchewan in 2001. I obtained a BEng, MEng, and a PhD in Civil Engineering. I have practiced my profession continuously since 1993. My entire career has entailed mine waste management design, construction, and closure;
3) I am a professional engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia (#28531). I am member of the Canadian Geotechnical Society and the Canadian Dam Association;
4) I personally inspected the subject project between May 5 and 9, 2014;
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, I am 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 Section 17.4 and Section 19 and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
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 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.

["Signed and Sealed"]

Vancouver, BC, Canada
June 16, 2016

Maritz Rykaart, PhD, PEng
Principal Consultant (Geotechnical Engineering)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled **Preliminary Economic Assessment Technical Report for the Antilla Copper-Molybdenum Project, Peru**, dated June 16, 2016.

I, Brian Connolly, PEng, do hereby certify that:

1) I am a Principal Mining Engineer with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 1300 - 151 Yonge Street, Toronto, Ontario, Canada;
2) I graduated with a Bachelor of Applied Science degree in Mineral Engineering from the University of British Columbia in 1973 and have practiced my profession continuously since graduation. My work has involved mine engineering and technical services management at operating mines for 18 years and consulting on open pit projects since 1995. My industry and consulting experience has frequently involved mining project economic analysis and financial modelling;
3) I am registered as a professional engineer with Professional Engineers Ontario (Registration # 90545203);
4) I have not personally visited the project site but relied on a site visit conducted by Goran Andric, PEng, a co-author of this technical report;
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;
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 a co-author of this report and responsible for Sections 2, 20.4.2, 21, 24.10, 25.6, and the portions of the executive summary that pertain to these sections of the technical report;
8) I have had no prior involvement with the subject property;
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 or securities of Panoro Minerals Ltd., and
11) 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.

[“Signed and Sealed”]

Toronto, Ontario, Canada
June 16, 2016

Brian H. Connolly, PEng
Principal Consultant (Mining Engineering)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled **Preliminary Economic Assessment Technical Report for the Antilla Copper-Molybdenum Project, Peru**, dated June 16, 2016.

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 Avenida Pardo y Aliaga #699, Oficina 601 B, San Isidro, Lima, Peru;
2) I am a graduate from the Universidad Nacional de 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 24 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 November 2 to 5, 2015;
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 3, 4, 5, 6, 7, 8, 9, 10, 11, 22, and 23 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.

[“Signed and Sealed”]

Lima, Peru
June 16, 2016

Luis Vela,CMC
VP Exploration (Panoro Minerals Ltd.)

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled **Preliminary Economic Assessment Technical Report for the Antilla Copper-Molybdenum Project, Peru** dated June 16 2016.

I, Paul Joseph Daigle, PGeo, do hereby certify that:

1) I am a Senior Resource Geologist contracted to Tetra Tech WEI Inc. (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; I obtained a Bachelor of Science degree in Geology in 1989. 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, Peru;
3) I am a professional geologist and a member of the Association of Professional Geoscientists of Ontario, membership number 1592;
4) I personally inspected the subject project between June 3 and 7, 2013;
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 co-author of this report and responsible for section 13.0 and section 11.2 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 December 23, 2013;
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 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.

["Signed and Sealed"]

Toronto, Ontario, Canada
June 16, 2016

Paul Daigle, PGeo
Senior Geologist

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled: **Preliminary Economic Assessment Technical Report for the Antilla Copper-Molybdenum Project, Peru**, dated June 16, 2016.

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 12 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:
    - 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. Preparation of budget level mine plans and cost inputs, oversaw operation of personal designs and acting in supervisory-role positions as needed.
    - 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 PGeo;
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) and the Association of Professional Engineers and Geoscientists of Alberat (#74969);
4) I have not personally visited the project;
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 a co-author of this report and responsible for the mining aspect of the study. This includes the mining components of the Executive Summary and Sections 17, 20, 24, and 25 as well as the entirety of Section 15 and I accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
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 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.

[“Signed and Sealed”]

Penticton, BC, Canada
June 16, 2016

Jesse J. Aarsen, PEng
Senior Associate (Mining Engineering)