BATERO GOLD CORP · COLOMBIA

Linares Americas Consulting SAC

Technical Report
NI 43-101

BATERO GOLD CORP.

LA CUMBRE GOLD PROJECT,

DEPARTMENT OF RISARALDA, COLOMBIA

NI 43-101 TECHNICAL REPORT ON UPDATED MINERAL RESOURCE ESTIMATE AND PRELIMINARY ECONOMICS ASSESSMENT

Prepared for:
Batero Gold Corp.

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IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for Batero Gold Corp. (Batero) by Linares Americas Consulting S.A.C. (LINAMEC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in LINAMEC's services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Batero subject to terms and conditions of its contract with LINAMEC. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

1.0 SUMMARY

1.1 INTRODUCTION

LINAMEC SAC was retained in October 2021 by Sociedad Minera Quinchia S.A.S. (Quinchia), a Colombian subsidiary of Batero Gold Corp., to review the latest information about core logging of the drilling programs and the QA/QC protocols, especially the data related to the primary sulfide zone, to update the mineral resource estimate of the La Cumbre Deposit including the primary sulfide zone in compliance with National Instrument 43-101, and to prepare a Preliminary Economic Assessment (PEA).

The new resource estimate incorporates the results of all drilling conducted between 2006 and 2017 in the area known as "La Cumbre", 143 drill holes and 41,338.50 metres. The main objective was to evaluate the metal content of the oxide, transition and primary zones, updating the total mineral resource estimate for the La Cumbre deposit. This mineral resource estimate now includes the resources of the primary sulfide zone.

1.2 PROPERTY DESCRIPTION, OWNERSHIP AND HISTORY

Part of the information in this section has been extracted and updated from Evans et al. (2013).

The Batero-Quinchia property comprises three concession contracts in a contiguous block totaling 1,407.43 ha, located within the Municipality of Quinchia, Department of Risaralda, Colombia. The property is located on the IGAC Planchas 186-IV-6 and 205-II-A topographic maps (1:25,000 scale), within a rectangular area extending for approximately 6.3 km in an east-west direction and 5.4 km in a north-south direction. The Universal Transverse Mercator (UTM) co-ordinates for the approximate center of this area are 422,500mE, 584,500mN (WGS-84, Zone 18N).

The licenses that comprise the Quinchia property were initially acquired by Juan David Uribe Hurtado and Silvia Stella Rios Martinez in 1998. The property was optioned to AngloGold Ashanti (AGA) in 2005. In November 2007, Caribbean Copper and Gold Corporation (Caribbean) entered into an agreement with AGA whereby it could earn a 100% interest in the property. Caribbean assumed AGA's responsibilities with respect to the underlying agreement. In 2009, Caribbean and AGA terminated the option agreement and the property reverted to the vendors. In 2010, Batero acquired the Quinchia property through its acquisition of all the issued and outstanding shares of Bahia, as described under Land Tenure.

Artisanal mining has taken place in the Batero-Quinchia property area from Pre-Colombian to modern times. Artisanal gold production in the area was greatest during the 1950s. Interest was renewed in the area in the late 1970s and culminated in the 1980s in the Miraflores area with the "Asociación de Mineros de Miraflores", a local artisanal mining cooperative.

During the 1990s, the Quinchia area drew the attention of various Canadian junior companies, some of which acquired ground in the general area. In 1997, a subsidiary of TVX Gold Inc., TVX Mineria de Colombia, completed a comprehensive review of the Quinchia property but did not follow up with ground work.

In 2000, INGEOMINAS undertook a series of technical studies in the area including geological mapping, geochemical and geophysical surveying, and prognostic (non-NI 43-101 compliant) resource estimations.

In May 2005, a subsidiary of AGA, Sociedad Kedahda S.A. (Kedahda), completed reconnaissance sampling in selected areas within the current property. During 2006, Kedahda completed geological mapping, soil sampling, channel sampling, and a 15-hole drilling program totalling 4,090.7 m on the Dos Quebradas, La Cumbre, and El Centro (Mandeval) targets. In April 2008, Kedahda completed a combined magnetometer and radiometric helicopter-borne survey over a large area including the current Quinchia property.

In May 2011, Batero acquired two historic gold mines within the Batero-Quinchia concessions in an arm's length cash transaction. La Cumbre Mine includes at least nine tunnels with lengths varying between 15 m and 250 m. The Mandeval Mine is located about 600 m northwest of La Cumbre sector and includes at least seven tunnels, the principal tunnel being approximately 160 m in length. Exploitation by the previous owners was halted in 2008 due to lack of a mining tenement and lack of a license for explosives.

In November 2007, Caribbean Copper and Gold Corporation (Caribbean) entered into an agreement with AGA whereby it could earn a 100% interest in the property. Caribbean assumed AGA's responsibilities with respect to the underlying agreement.

In July 2009, Caribbean Copper and Gold Corporation, AGA, and the registered holders of the concessions agreed to terminate the agreement. The Quinchia concessions reverted to the original vendors.

In October 2009, Angus Resources Inc. prepared a National Instrument 43-101 compliant technical report for the Quinchia gold porphyry project.

In July 2010, the company formerly known as Angus Resources Inc. changed its name to Batero Gold Corp., which was incorporated in 2008 and is headquartered in Toronto, Canada.

In 2010, Batero acquired ownership of Quinchía by acquiring all of the issued and outstanding shares of Bahía, a Panamanian company formed for the purpose of owning all of the issued and outstanding capital of Minera Quinchía, a Colombian company that owns all of the rights in the Quinchía property.

From January to December 2011, Batero completed a 62 holes diamond drilling program totaling 27,262.34 m on the La Cumbre target.

Roscoe Postle Associates Inc. (RPA), prepared an independent technical report on the Batero-Quinchia Project, Department of Risaralda, Colombia dated February 24, 2012.

From July to September 2012, Batero completed a 29 holes diamond drilling program totaling 4,252.70 m on the La Cumbre target.

Roscoe Postle Associates Inc. (RPA), prepared a preliminary economic assessment (PEA) and corresponding NI 43-101 technical report on the Batero-Quinchia Project, Department of Risaralda, Colombia dated December 16, 2013.

During 2014, Minera Quinchia, a Colombian subsidiary of Batero Gold, conducted an auger soil and rock sampling campaign to evaluate and characterize the oxidized saprolite.

In 2015, a systematic soil sampling program using augers was carried out on a 50m x 50m grid. A total of 205 samples were collected for gold and multi-element analysis with samples reaching depths of up to 5m.

From January 2016 to March 2017, Batero completed a 40 holes infill-drilling program totaling 4,574.16 m to evaluate the oxide zone (ZO) and the transition zone (TR) of the La Cumbre deposit.

In November 2018, LINAMEC prepared a Mineral Resource update which was filed with the Toronto Stock Exchange and disclosed on SEDAR.

In October 2022, SRK CONSULTING - PERU S.A. undertook engineering studies for the deposits of low-grade and sulfide stockpile located at Matecaña, as well as stockpile of oxide-transition material, leaching pad, processing plant, agglomeration plant and waste dump located in La Perla area.

1.3 GEOLOGY & MINERALIZATION

Information in this section has been extracted and updated from Evans et al. (2013).

The La Cumbre Project is located along the eastern margin of Colombia's physiographic Western Cordillera. The region is underlain by a highly complex basement known as the Romeral Terrane, which may be characterized as a tectonic mélange. The basement took form when Middle to Upper Mesozoic-aged volcanic and sedimentary oceanic rocks collided with, and were accreted to, the northern Andean paleo-continental margin, beginning in the Early Cretaceous. The resulting suture is known as the Romeral fault system and the mélange can be traced for over 1,000 km along the northern Andes.

The La Cumbre and surrounding area is underlain by four principal rock units. These include: 1) a basement complex consisting of mafic and ultramafic oceanic volcanic rocks and granitoid intrusive rocks belonging to the Romeral Terrane, 2) stratified clastic sedimentary rocks of the Amaga Formation, 3) basalt-andesite through felsic volcanic and pyroclastic rocks of the Combia Formation, and, 4) dioritic to monzonitic hypabyssal porphyritic intrusive rocks.

The Dos Quebradas, El Centro and La Cumbre porphyry gold deposits are associated with three Miocene intrusive centers in a north-south trend that have a strike extension of approximately two kilometers at elevations between 1,600 meters above sea level (MASL) and 1,050 MASL.

The Dos Quebradas, El Centro and La Cumbre porphyry gold deposits are copper-poor porphyry gold systems in which intermediate argillic alteration locally extensively overprints an early potassic assemblage and its associated quartz veinlet stockwork. Gold in these deposits occurs in altered dioritic intrusions and in the diorite-basalt contact zones. The highest gold and silver grades occur in the early diorite phases characterized by potassic (mainly biotite with subordinate K-feldspar) and potassic-calcic alteration that is characterized by addition of traces of actinolite and garnet to the potassic assemblage. Significant amounts of quartz ± sulfide veinlets and greater than 3% hydrothermal magnetite are common in these early phases.

The intrusions that host mineralization consist of several phases of diorite and later andesitic dike phases exhibiting characteristic alteration zoning, possibly because of telescoped porphyry and epithermal systems and progressive leaching of gold by overprinting argillic alteration.

La Cumbre is a discrete porphyry gold center in which the average gold content reaches economic tenor even though the quartz-veinlet intensity is relatively low. The La Cumbre porphyry gold center is truncated to the north and west by the structural corridor of La Amarilla.

This structure has a NW trend and separates the La Cumbre deposit from the El Centro and Dos Quebradas deposits. On the basis of the drilling to date, the gold grade attains a maximum in biotite-rich porphyry, and is clearly progressively leached as the intermediate argillic overprint intensifies (Sillitoe, 2006).

1.4 MINERAL RESOURCE ESTIMATE

The current mineral resource estimate (Batero Gold Corp. La Cumbre Gold Project, Department of Risaralda, Colombia NI 43-101 Technical Report Update on Mineral Resource Estimate, December 2021) is an update to the 2018 resource estimate prepared by Linares Americas Consulting (LINAMEC). It is accompanied by a technical report prepared by LINAMEC SAC and utilized ordinary kriging interpolation as the estimation methodology on a more extensive database than previously available.

The main objective was to publish a revised mineral resource estimate for the oxide, transitional and primary zones of the La Cumbre deposit. This updated resource estimate includes the forty diamond drillholes drilled in 2016 and 2017.

The mineral resources have been defined taking into account the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves. The mineral resource estimate has an effective date of December 31, 2021, that is the cut-off date for information used in the estimate. Mineral resources that are not reserves do not have demonstrated economic viability. Table 1-1 summarizes the mineral resource estimate for the La Cumbre deposit.

Table 1-1:
Total Mineral Resource Statement for All Mineral Zones - December 31, 2021:
Batero Gold Corp. – La Cumbre Gold Project

Total Resources All Mineral Zones							
Resource	**Volume**	**Density**	**Tonnage**	**Au g/t**	**Au oz**	**Ag g/t**	**Ag oz**
Measured	49,317,902	2.624	129,421,866	0.509	2,117,649	1.52	6,336,330
Indicated	2,411,421	2.606	6,283,667	0.383	77,476	0.45	91,432
Meas. + Ind.	**51,729,322**	**2.623**	**135,705,533**	**0.503**	**2,195,124**	**1.47**	**6,427,763**
Inferred	356,987	2.533	904,088	0.413	12,005	1.32	38,472

Notes to accompany La Cumbre Mineral Resource tables:

1. Mineral Resources have an effective date of December 31, 2021. The Qualified Person for the estimate is Mr. Walter La Torre, CP and MAusIMM.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
3. Mineral Resources are reported within a conceptual optimized pit that uses the following input parameters: Au price: US$1,750/troy oz and US$22.0/troy oz Ag, mining cost: US$1.95/t, process cost (including G&A): US$9.08/t processed, gold selling cost: US$47.00/troy oz and overall slope angle of 38°.
4. Gold recovery in the oxide and transitional zones was fixed at 85.5%. Gold recovery in the primary zone was fixed at 84.1%.
5. Mineral Resources (Oxide) are reported using a 0.218 Au g/t cut off grade.
6. Mineral Resources (Transitional) are reported using a 0.218 Au g/t cut off grade.
7. Mineral Resources (Primary) are reported using a 0.179 Au g/t cut off grade.
8. Totals may not sum due to rounding as required by reporting guidelines.

1.5 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical research has determined that heap leaching is the most suitable process for the beneficiation of ore from the oxide and transitional zones. The grain size of the oxide zone ore is very fine, with a high percentage of clay content (over 80% by weight, with a particle size of less than 6 mm); therefore, the permeability of this ore is not sufficient to leach it without prior agglomeration. Furthermore, due to the low permeability, it is estimated that the amount of cement required for the agglomeration process and to be able to stack the oxide to a height of 6 m is 23 kg/t.

From the fifth month of mining, the ore will be composed of oxides and transition zone ore, improving the conditions for the leaching process because the transition material has a coarser and more consistent grain size, improving permeability accordingly, and decreasing cement consumption to 14 kg/t for a ratio of 30% transition material and 70% oxide material. Column leach tests exceed 90% recovery for both oxide and mixed oxide/transition materials. The leaching process is designed for a maximum of 18 days of irrigation, obtaining a recovery in this period of 85.5%. Sodium cyanide and lime consumption is about 1.2 kg/t and 7 kg/t respectively.

The rich solution resulting from leaching has a copper content of up to 400 ppm. To reduce the copper content and improve the quality of the final gold doré, copper precipitation tests were performed, resulting in copper recoveries between 87% and 92.5%. Subsequently, the rich solution streams are destined to the Merrill-Crowe and ADR processes, which obtained recoveries above 99% in both cases.

Heap leaching tests of the primary zone material resulted in low recoveries. The most suitable beneficiation process for this type of mineral material is concentration and subsequent leaching of the concentrate. At the laboratory level, gravimetric and flotation recoveries of 35.6% and 49.4% were obtained, respectively, for a total of 85.0%.

1.6 MINING METHODS

The mining method defined for La Cumbre Project is an open pit method with a predominant bank height of 6 meters, 12-meter ramp width two way only and inter-ramp angles between 38° and 43°.

The project is segmented into two stages involving different capital investments in order to meet the infrastructure requirements of each stage of the project's LoM.

The first stage corresponds to the mining of the ore contained in the oxide and transition zones, which will be agglomerated and processed in leach pads at a throughput of 15 ktpd. At this stage of the project, the stripping ratio is an average of 0.28, allowing for the disposal of the waste dump and the stockpile in Matecaña area during the 5 years of this stage; that is, the transport of ore and waste will be done through the overland conveyor to maximize its use and take advantage of its regenerative capacity.

The second stage involves the mining of the primary sulfide zone that will be processed in a flotation and gravimetry plant whose ramp up will be in the 5th year and will last until the 14th year of the project. During this stage, starting in year 6, the stripping ratio is an average of 0.49 and the dumps will be available north of the pit, this will allow us to increase the throughput to 30 ktpd transported by the overland conveyor.

The final pit designs were based on an economic pit using the Lerchs and Grossman algorithm and involving the costs of extraction and processing, recovery, sales costs, general and administrative expenses provided by Minera Quinchia and reviewed by Linamec and the commodity prices estimated by Linamec for the project's LoM. Consequently, the pit and phase designs were made using Datamine Studio OP tools, and to guarantee the best extraction sequence, the Mine Plan Schedule Optimizer tool was used in different scenarios.

The mining and processing schedule presented in this document were selected over other scenarios due to their technical and economic feasibility. In this section, the mining schedule is of a “Potentially Economic Mineral Resource” nature, in addition to the fact that at all times the mineral present in the category of Inferred Resource is considered as waste.

The LoM production schedule is shown in Chapter 16, in Figures 16-14, 16-15, and in Tables 16-31 and 16-32.

1.7 RECOVERY METHODS

The proposed beneficiation process for the La Cumbre Project ore consists of two stages:

The first stage, designed to treat the ore from the oxide and transition zones at a rate of 15,000 DMT per day, has as its principal process the extraction of gold by leaching with cyanide solution in dynamic pads with estimated recovery of 85.5% in 18 days of spraying. For better efficiency in the leaching process, prior unit operations are required: crushing and agglomeration of fines at La Perla. After leaching, the leached ore will be deposited in a leached ore deposit. The rich solution is divided into two streams according to gold concentration: PLS and ILS. Both streams are treated separately in the SART-AVR process to precipitate the dissolved copper in the form of copper sulfides. Gold from the PLS solution stream is extracted by the Merrill-Crowe process, while gold from the ILS solution is extracted by activated carbon adsorption in an ADR plant. Finally, the gold precipitates from both the Merrill-Crowe and the ADR plant are smelted to form gold doré.

The second stage, designed to treat 30,000 DMT per day of ore from the primary zone, aims to concentrate the gold-bearing ore by gravimetric and flotation processes, with laboratory tests estimating a recovery of 84.9% between the two concentrates. Subsequently, the concentrates are leached by the CIL process, where a recovery of 95% is estimated, giving an overall recovery of 80.6% gold. The concentration stage requires the installation of the primary crushing station at La Cumbre, milling, gravimetry and flotation; the secondary crushing station at La Perla is maintained from the first processing stage, as are other facilities such as SART-AVR, Merrill-Crowe and ADR. The gravimetric and flotation concentrate is regrinded in a closed-circuit mill and hydrocyclones, the fine concentrate pulp is separated from the rich solution using a thickener. The gold in the rich solution is extracted by the Merrill-Crowe process, while the coarse pulp is leached in agitator tanks in the presence of activated carbon (CIL process). The gold adsorbed on the activated carbon is recovered in the ADR plant. Finally, the gold precipitates from both the Merrill-Crowe and the ADR plant are smelted to form gold doré.

1.8 PROJECT INFRASTRUCTURE

The La Cumbre Project will have two main entrances, from the north via Yarumal-Paramillo-La Cumbre or Dosquebradas-La Cumbre and the south via La Perla-Matecaña-Aguas Claras-La Cumbre.

In addition, the La Cumbre Project includes the opening of an open pit mine, a continuous conveyor system using regenerative belts, two processing plants, two TSF's and major facilities to support the operation. Most of the planned infrastructure is basically engineered.

The project contemplates the construction of three mine waste dumps with an estimated capacity of 71.6 million tonnes; two organic material storage facilities with a capacity of 4 million tonens; two temporary stockpiles with a capacity of 3.15 million tonens; one leached material storage facility with a capacity of 24.8 million tonnes; and one flotation tailings storage facility with a capacity of 82.5 million tonnes.

The material transport system will be through the construction of the conveyor belt that will take the material to the storage and processing points, and whose Conceptual Engineering was developed by Tarma Bulk Solids, while the Basic Engineering was developed by HLC Engineering and Construction with the support of the French company RBL REI.

The system will transport ore or waste from inside the pit to the transfer points at Matecaña and La Perla, with an approximate length of 2 kilometers and, due to its regeneration capacity due to the slope, it is estimated to contribute $0.09/t of transported material.

Batero plans to recruit local labor and transport workers from regional populated areas to the mine site by bus.

1.9 ENVIRONMENTAL STUDIES, PERMITTIND AND SOCIAL OR COMMUNITY IMPACT

Minera Quinchía, in the development of the exploration phase of the La Cumbre mining project, applied to the Colombian Environmental Authority (CARDER) for the renewal of the concession permits for domestic and industrial use and water discharge, in accordance with current regulations, having obtained the extension of these permits until 17 January 2022. Minera Quinchía is currently in the process of renewing these permits.

In order to advance to the Exploitation stage, the Environmental and Geographic Services-SAG consulting firm was hired to prepare the EIA, a study that will identify the characterization of the biotic, abiotic and socioeconomic components, and consequently the prior identification of the impacts associated with the development of the construction, development and closure stages of the La Cumbre Project, and proposing socio-environmental management plans to prevent, mitigate and/or compensate the effects caused in the Area of Influence. In addition, the EIA will propose the request to the environmental authority for permits for the demand, use, exploitation and/or impact of natural resources in the different stages and works of the project.

With respect to the socioeconomic component, the identification of interest groups and non-ethnic communities, the community information and participation program (PIPC) was implemented in three stages during the preparation of the EIA, with the objective of socializing and informing communities and interest groups in the Area of Influence about project activities, providing opportunities to identify impacts and formulate management measures with ongoing feedback and community participation and input.

Minera Quinchía is currently developing the process of Prior Consultation with the ethnic communities (Embera Chami and Embera Karamba), these processes are in the final stage, and are expected to culminate with the notarization before the National Directorate of Prior Consultation, DANCP, in September of 2022.

Consequently, once this process is completed, the EIA will be presented for the respective evaluation and process of granting the Environmental License by the National Authority of Environmental Licenses (ANLA).

Furthermore, Minera Quinchía focuses its social management in the Area of Influence of the La Cumbre Project, on issues pertaining to: Education, Social Infrastructure Development, Improvement of Productive Projects, Health Systems, Culture and Heritage Development formulating an analysis matrix that includes the implementation of projects, quantification of beneficiaries, actions and establishment of favorable impacts of the mentioned projects.

1.10 CAPITAL AND OPERATING COSTS

1.10.1 Capital Costs Estimates

The initial capital cost for Phase I totals US$169.5 million, including contingencies and is expected to complete construction in one year. Table 1-2 summarizes the initial capital cost expenditure for the initial Phase I, oxide processing plant.

Before the oxide ore is depleted, the Phase II sulfide plant processing expansion commences with the view to complete construction in two years with a distribution of 60% in the first year and 40% in year two as shown in Table 1-3.

Table 1-2:
Initial Capital Cost Summary – Phase I Oxides

Description	Total (US$k)
Conveyors	6,758
Overland	34,370
Conveyor transport	12,270
Matecaña deposit	7,803
Crushing / agglomeration circuit	9,364
Leaching circuit	19,622
Detox and neutralization treatment circuit	2,637
Conveyors feeding / Unloading of dynamic pad	18,019
Domestic and drinking water treatment circuit	1,356
Dynamic pad, leached deposit, left over material	36,878
Infrastructure and services	6,023
Contingency	14,388
Total initial CAPEX	**169,489**

Table 1-3:
Initial Capital Cost Summary – Phase II

Description	Total (US$k)
Crushing plant	3,855
Flotation plant 30 ktpd	132,777
Tailing deposit	33,966
Tailing pipeline 2.9 km	642
Waste deposit 60 Mt	36,200
DME	1,662
Land acquisition	2,745
Contingency	36,479
Total initial CAPEX	**248,325**

1.10.2 Operating Costs

The operating cost estimate is based on a conveyor plus contractor-operated truck and shovel mining operation, leaching processing facility, flotation processing facility and Tailings Storage Facility. Mine operating cost estimates are provided in Table 1-4. The PEA estimates that the C1 operating costs will average US$684/oz of Au.

Table 1-4:
Unit Operating Costs per Ounce

Item	LoM costs (US$/oz)
Mining costs	240.1
Processing costs	457.3
Site G&A	14.5
Treatment, refining, penalties	8.2
By-product credits	(35.9)
C1 cash cost	**684.2**
Royalties	56.9
Sustaining capital expenditures	29.8
AISC	**770.9**

*AISC = All-in sustaining cost

1.11 ECONOMIC ANALYSIS

The economic analysis was performed assuming the base case gold price of US$1,750/oz, and silver price of US$22/oz. These metal prices were based on consensus analyst estimates and recently published economic studies.

The economic analysis was performed assuming a 5% discount rate. The pre-tax NPV discounted at 5% is US$730 million; the IRR is 47.5%, and payback period is 1.9 years. On a post-tax basis, the NPV discounted at 5% is $481 million, the IRR is 32.1%, and the payback period is 2.5 years. A summary of project economics is shown in Table 1-5.

Table 1-5:
LoM Financial Valuation and Parameters

Item	Unit	Open Pit
Commodity Prices (Long term)		
Gold Price	US$/oz	$1,750
Silver Price	US$/oz	$22.00
LoM Mine Plan Summary		
Mine Life	Years	14.0
Minable resource	kt	106,594
Gold grade	g/t	0.56
Silver grade	g/t	1.57
Processing Rate	tpd	15,000-30,000
LoM Processing Recovery*		
Gold Recovery	%	85.5%
Silver Recovery	%	46.9%
LoM Revenue		
Net Revenue	US$M	$2,905.4
LoM Operating Cost		
Mining	$/t processed	3.66
Processing	$/t processed	6.98
Site G&A	$/t processed	0.22
Treatment, Refining, Freight	$/t processed	0.13
By-product credits	$/t processed	(0.55)
C1 Cash Operating Cost	US$/oz	684.22
AISC Cost	US$/oz	770.89

Item	Unit	Open Pit
Operating Costs	US$M	$1,171.5
Royalties	US$M	$92.5
LoM Cash Flow		
EBITDA	US$M	$1,641.4
Net Cash Flow		
Less: Cash taxes	US$M	($352.4)
Less: Change in working capital	US$M	$0.0
Less: Capital expenditures	US$M	($466.3)
Net Cash Flow	US$M	$822.7
Post-Tax NPV 5%	US$M	$480.6
Post-Tax IRR	US$M	32.1%
Payback (1st phase)	Years	2.5

* Only leachable materials (oxides+transitional zones)

1.12 QUALIFICATIONS AND ASSUMPTIONS

This report is considered by Linares Americas Consulting S.A.C. to meet the requirement of a Technical Report as defined in Canadian NI 43-101 guidelines for a preliminary economic assessment (PEA). There is no guarantee that the La Cumbre Project will be placed into production as this is contingent on successfully obtaining all the requisite consents, permits or approvals, regulatory or otherwise.

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. The quantity and grade of reported inferred mineral resources in this preliminary economic assessment are uncertain in nature and there has been insufficient exploration to define these inferred mineral resources as an indicated or measured mineral resource and it is uncertain if further exploration will allow conversion to the measured and indicated categories or that the measured and indicated mineral resources will be converted to proven or probable mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability; the estimate of mineral resources in this preliminary economic assessment may be materially affected by higher operating costs, lower metal prices, lower process recoveries, environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

2.0 INTRODUCTION

LINAMEC S.A.C. was retained in October 2021 by Sociedad Minera Quinchia S.A.S. (Quinchia), a Colombian subsidiary of Batero Gold Corp., to review the latest information about core logging of the drilling programs and the QA/QC protocols, especially the data related to the primary sulfide zone, to update the mineral resource estimate of the La Cumbre Deposit including the primary sulfide zone in compliance with National Instrument 43-101, and to prepare a Preliminary Economic Assessment (PEA).

Mr. Walter La Torre and Mr. Fernando Linares visited the property from December 9 to 12, 2021, and examined the La Cumbre deposit outlined within the Quinchia property. During the site visit, sufficient opportunity was available to examine several rock exposures, conduct a general overview of the property, mapping, geochemistry, density determination procedure and core logging and observe the condition of stored cores and reject samples, which are in fairly good condition.

Based on their experience, qualifications and review of the site and resulting data, the authors, Mr. La Torre and Mr. Linares are of the opinion that the exploration to date has been conducted in a professional manner and the quality of data and information produced from the efforts meets with acceptable industry standards. The work has been directed and supervised by qualified geologists. In preparing this report, the authors have followed proper methodology and procedures and exercised due care consistent with the intended level of accuracy using their professional judgment and reasonable care.

While actively involved in the preparation of the report, LINAMEC had no direct involvement or responsibility in the collection of the data and information or any role in the execution or direction of the work programs conducted for the project on the property or elsewhere. Much of the data has undergone thorough scrutiny by project staff as well as certain data verification procedures by LINAMEC.

2.1 EFFECTIVE DATE

The effective date of this Technical Report and Preliminary Economic Assessment is August 31, 2022.

2.2 ABBREVIATIONS, UNITS AND CURRENCIES

A list of abbreviations that may appear in this report is provided in Table 2-1. All currency amounts are stated in Colombian Pesos (COP) or US dollars (US$, USD). Quantities are stated in metric units, as per standard Canadian and international practice, including metric tonnes (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, percentage (%) for copper grades, and gram per tonne (g/t) for gold and silver grades. Wherever applicable, imperial units have been converted to the International System of Units (SI units) for consistency (Table 2-1).

Table 2-1:
Units of Measure, Abbreviations, Acronyms

Abbreviation/Symbol	Description
°C	degrees Celsius
3D	Three-dimensions
A	ampere
a	annum (year)
ABA	Acid Base Accounting
ADR	Adsorption-desorption-recovery

Abbreviation/Symbol	Description
Ai	Abrasion index
AISC	All-in sustaining cost
amsl	above mean sea level
AN	Ammonium nitrate
ARD	Acid rock drainage
As	Arsenic element
ASTM	American Society for Testing and Materials
Au	Gold
avg.	Average
AVR	Acidification, Volatilization and Reneutralization
B	billion
BD	Bulk density
BFA	Bench Face Angle
BHD	Blast Hole Drill
BQ	Drill core diameter of 36.5 mm
BRT	Bottle Roll Test
Bt	billion tonnes
BWi	Bond Work index
bya	billion years ago
C$	Canadian dollar
C&F	Cut and Fill
Ca	Calcium
CAF	Cut and Fill
CAPEX	Capital expenditures
cc	cubic centimeter
CEP	Construction Execution Plant
cfm	cubic feet per minute
Cía. (Spanish abbrevation)	Company
CIC	Carbon in Columns
CIM	Canadian Institute of Mining and Metallurgy
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
CMA	Compañía Minera Kolpa S.A.
CPP	Cumulative Probability Plot
CSV	Comma Separated Values file
Cu	Copper
CV	Coefficients of variation
d	day
d/a	days per year (annum)
d/wk	days per week
dB	decibel
dBa	decibel adjusted
DDH	Diamond drill holes
DGPS	Differential Global Positioning System
DME	Department of Minerals and Energy
dmt	dry metric ton
DSHA	Deterministic seismic hazard analysis
DSO	Deswik Stope Optimizer software
DWT	Dead weight tonnes

Abbreviation/Symbol	Description
E.I.R.L. (Spanish initials)	Individual Limited Liability Company
EA	environmental assessment
EAU	Economic Administrative Unit
EBITDA	Earnings Before Interest Taxes Depreciation and Amortization
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
ELC	Ecological Land Classification
EP	Engineering and procurement
EP	Engineering and procurement
ERD	Explosives Regulatory Division
EWP	Engineering work packages
FEED	Front End Engineering Design
FEL	Front-end loader
FS	Feasibility Study
ft	foot
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second
g	gram
G&A	General and Administration
g/cm^3	grams per cubic metre
g/L	grams per litre
g/t	grams per tonne
GA	General arrangements
GSI	Geological Strength Index
GSLIB	Geostatistical Software Library
GTZ	Glacial Terrain Zone
GW	gigawatt
h	hour
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m^2)
ha	hectare
HBZ	Health buffer zone
HCT	Humidity cell test
HCT	Humidity Cells Test
HG	High-grade
Hg	Mercury element
HGU	Hydrogeological Unit
HK	Hypabyssal kimberlite
HLEM	Horizontal loop electro-magnetic
HLF	Heap leach facility
HLF	Heap leach facilities
hp	horsepower
HPGR	High-pressure grinding rolls
HQ	Drill core diameter of 63.5 mm
Hz	hertz
ICP-MS	Inductively coupled plasma mass spectrometry
ID	identification, identifier

Abbreviation/Symbol	Description
IDW	Inverse distance weighted
IMC	Independent Mining Consultants
in	inch
in^2	square inch
in^3	cubic inch
IP	Induced polarization
IRA	Inter-Ramp Angle
IRA	Inter-Ramp Angle
IRR	Internal rate of return
JV	Joint venture
JV	Joint venture
K	hydraulic conductivity
k	kilo (thousand)
KBM	Kuzey Biga Madencilik
KCA	Kappes Cassiday & Associates
kg	kilogram
kg/h	kilograms per hour
kg/m^2	kilograms per square metre
kg/m^3	kilograms per cubic metre
KIM	Kimberlitic indicator mineral
km	kilometre
km/h	kilometres per hour
km^2	square kilometre
kPa	kilopascal
kt	kilotonne
ktpd	kilo tons per day
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
L	litre
L/min	litres per minute
LDD	Large-diameter drill
LG	Low grade
LGM	Last glacial maximum
LOM	Life of mine
LRL	Left - Right - Left rotation
m	metre
M	million
m/min	metres per minute
m/s	metres per second
m^2	square metre
M2M	Mine to Mill
m^3	cubic metre
m^3/h	cubic metres per hour
m^3/s	cubic metres per second
Ma	million years

Abbreviation/Symbol	Description
MAAT	mean annual air temperature
MAE	Mean annual evaporation
MAGT	mean annual ground temperature
mamsl	metres above mean sea level
MAP	mean annual precipitation
masl	metres above mean sea level
MASW	Multichannel Analysis of Surface Waves
Mb/s	megabytes per second
mbgs	metres below ground surface
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum
mbs	metres below surface
mbsl	metres below sea level
MEU	Ministry of the Environment and Urbanization
mg	milligram
mg/L	milligrams per litre
min	minute (time)
ML	Metal leaching
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMSIM	Metamorphosed Massive Sulphide Indicator Minerals
mo	month
MPa	megapascal
MPEI	Mechanical, piping, electrical, and instrumentation
MQ	Minera Quinchia
MRE	Mineral Resource Estimation
MRMR	Mineral Resources and Mineral Reserves
MSC	Mineral Services Canada Inc.
MST	Magnetic susceptibility trace
Mt	Million metric tonnes
Mton	Million metric tonnes
Mtpy	Million metric tonnes per year
mV/V	Millivolts per volt
MVA	megavolt-ampere
MW	megawatt
NA	Non-acidic
NAFZ	North Anatolian Fault Zone
NAG	Net acid generation
NAG	Net Acid Generation
NE	Northeast
NI 43-101	National Instrument 43-101
NN	Nearest-neighbor
NNP	Net Neutralization Potential
NPR	Neutralization Potential / Acidification Potential Ratio
NPV	Net Present Value
NQ	Drill core diameter of 47.6 mm
NSR	Net Smelter Return
NTU	Nephelometric Turbidity Unit
NWRD	North Waste Rock Dump

Abbreviation/Symbol	Description
°	degree
OC	Open Cast
Ohm·m	Ohm·meter (Ω·m)
OK	Ordinary Kriging
OP	Open pit
OPEX	Operational Expenditure
OSA	Overall Slope Angles
oz	Troy ounce
oz/t	Troy ounce per ton
P&ID	Piping & Instrumentation Diagrams
P. Geo.	Professional Geoscientist
PA	Potentially acidic
Pa	pascal
PACP	Potentially Acidic Contact Water Ponds
PAG	Potential Acid Generating
PDF	Probability density functions
PEA	Preliminary Economic Assessment
PEN	Peruvian Soles
PEP	Project Execution Plan
PFS	Preliminary Feasibility Study
PGA	Peak horizontal ground accelerations
PGE	Platinum group elements
PLA	Project Labour Agreement
PLC	Programmable logic controllers
PM	Project management
PMF	probable maximum flood
PP	Pre-production
ppb	parts per billion
ppm	parts per million
PSHA	Probabilistic seismic hazard analysis
psi	pounds per square inch
QA/QC	Quality assurance/quality control
QAQC	Quality assurance/quality control
QFP	Quartz-Feldspar Porphyry
QMP	Quality Management Plan
QP	Qualified Persons
RC	Reverse circulation
RC	reverse circulation
RF	Revenue Factor
RMA	Reduction-to-Major Axis method
RMR	Rock Mass Rating
RMR89	Bieniawski's 1989 update of the Geomechanical Classification
ROM	Run-of-Mine
rpm	revolutions per minute
RQD	Rock Quality Designation
RTP	Reduction to Pole
s	second (time)
S.G.	specific gravity
SART	Sulfidization, Acidification, Recycling and Thickening
Sb	Antimony element
Scfm	standard cubic feet per minute

Abbreviation/Symbol	Description
SE	Southeast
SEDEX	Sedimentary exhalative
SEM	Scanning electron microscope
SFD	Size frequency distribution
SG	specific gravity
SLS	Sub Level Stopping
SMP	Safety Management Plan
SMU	Selective Mining Unit, smallest mining unit
SPLP	Synthetic Precipitation Leaching Procedure
SSDS	Small scale direct shear
st/kg	stones per kilogram
st/t	stones per metric tonne
SW	Southwest
SWMP	Site water management plan
t	tonne (1,000 kg) (metric ton)
t/a	tonnes per year
t/d	tonnes per day
t/h	tonnes per hour
TC&RC	Treatment Charges and Refining Charges
TCR	Total core recovery
TDS	Total dissolved solids
TMF	Tailings management facility
tph	tonnes per hour
ts/hm^3	tonnes seconds per hour metre cubed
TSF	Tailings Storage Facility
UCS	Unconfined compressive strength
UEA (Spanish initials)	Economic Administrative Unit
UNI (Spanish initials)	National University of Engineering
Us	litres per second
US	United States
USCS	Unified Soil Classification System
US$	dollar (American)
USD	American dollar
USGS	United States Geological survey
UTM	Universal Transverse Mercator
V	volt
VEC	Valued ecosystem components
VMS	Volcanic massive sulphide
VSA	Vuggy silica alteration
VSEC	Valued socio-economic components
w/w	weight/weight
WAD	Weak acid dissociable
wk	week
wmt	wet metric ton
WRD	Waste rock dump
WRSF	Waste rock storage facility
WSF	Waste Storage Facility
WTP	Water Treatment Plant
XRD	X-ray diffraction
µm	micrometre, also called micron,

3.0 RELIANCE ON OTHER EXPERTS

LINAMEC prepared this technical report for Batero Gold Corp. (Batero). The quality of information, conclusions and estimates contained herein are based on industry standards for engineering and evaluation of a mineral project. In preparing this report, the authors hereof have followed methodology and procedures and exercised due care consistent with the intended level of accuracy using their professional judgment and reasonable care.

This report is intended to be used by Batero, subject to the terms and conditions of its contract with LINAMEC; however, LINAMEC disclaims any liability to any third party in respect of any reliance upon this document without LINAMEC's written consent.

Parts of this report, relating to the legal aspects of the ownership of the mineral claims, rights granted by the Government of Colombia and environmental and political issues, have been prepared or arranged by Batero. While the contents of those parts have been generally reviewed for reasonableness by the authors for inclusion in this report, the information and reports on, which they are based, have not been fully audited by the authors.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Batero-Quinchia property consists of three concession contracts in a contiguous block totaling 1,407.43 ha, located within the Municipality of Quinchia, Department of Risaralda, Colombia (Figure 4-1 and Figure 4-2). The property is located on the Instituto Geográfico Augustín Codazzi (IGAC) Plancha 186-IV-6 and 205-II-A topographic maps (1:25,000 scale). Table 4-1, lists the subject titles and the relevant tenure information. The subject titles were map-staked and therefore no boundary markers exist. As of the effective date of this report, none of the licenses have been surveyed. The subject titles are held under the name of Sociedad Minera Quinchia S.A.S. (Minera Quinchia).

The three parcels of land are located within a rectangular area extending for approximately 6.3 km in an east-west direction and 5.4 km in a north-south direction. The Universal Transverse Mercator (UTM) coordinates for the approximate center of this area are 422,500mE, 584,500mN (WGS-84, Zone 18N).

In accordance with Colombian law, the holder of the mining concession has a right to access the parcel of land covered by such concession and may perform exploration and exploitation work thereon, subject to indemnification for damages to the owners of such parcel of land that may arise from such access and the activities carried out by the holder of the mining concession.

Surface ownership is held privately by numerous individuals for agricultural use. Colombian law allows for exploration on private lands with notification of the surface landowners and reasonable compensation for surface disturbance caused by exploration activities. To date Minera Quinchia SAS - Batero has negotiated access and drill platform locations with individual landowners to compensate for any disturbance or loss of crop. As the company used a man portable drill rig there has been little disturbance.

Parts of this report, relating to the legal aspects of the ownership of the mineral claims, rights granted by the Government of Colombia and environmental and political issues, have been prepared or arranged by Minera Quinchia SAS - Batero. While the contents of those parts have been generally reviewed for reasonableness by the authors of this report, for inclusion into this report, the information and reports on which they are based has not been fully audited by the authors.

Table 4-1:
Batero-Quinchia Concession Areas

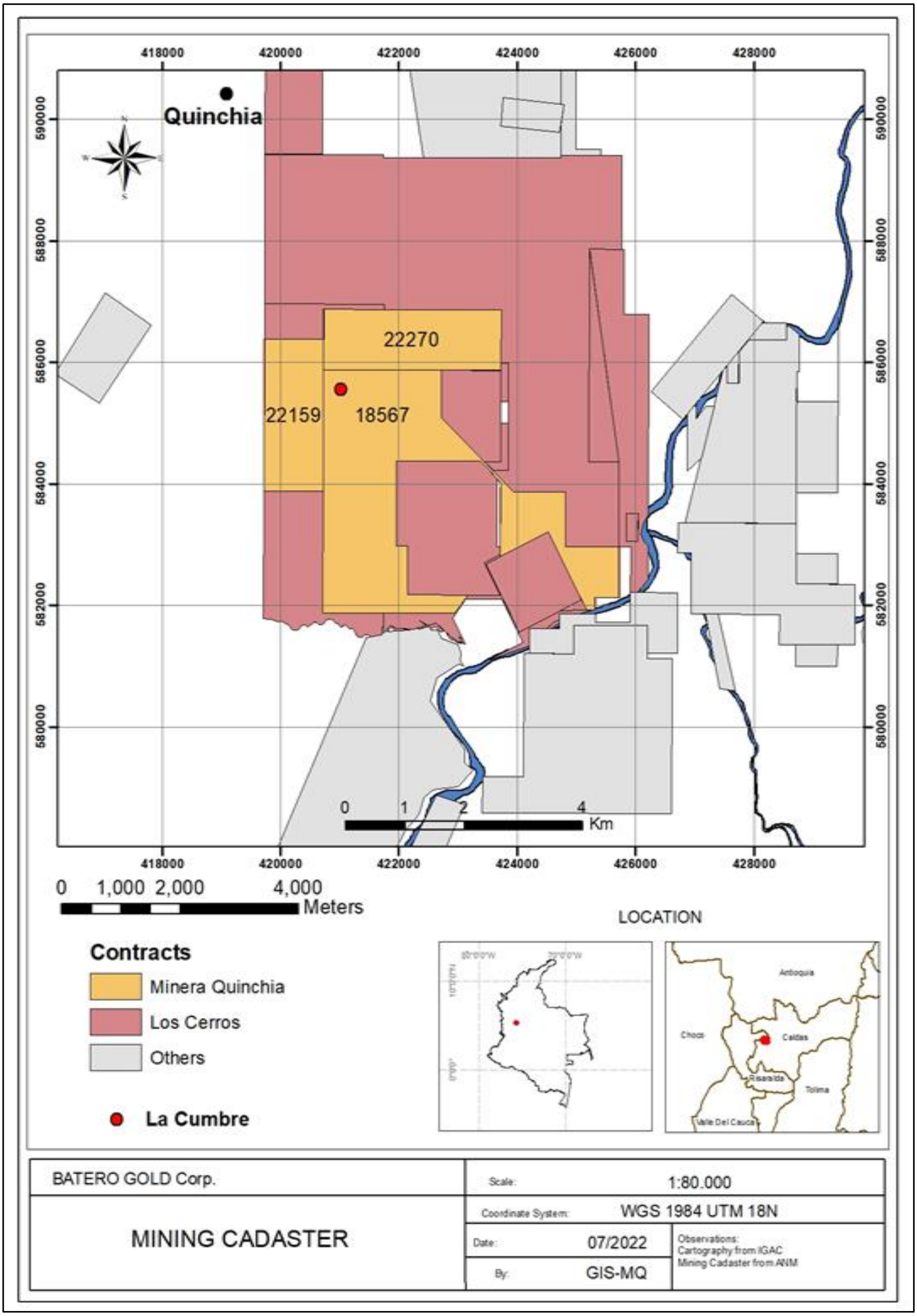
Contract Number	Contract/License Type	Original Registration Date	Minera Quinchia Registration Date	Area (ha)
18567	28 Year Concession Contract	15/04/1998	23/09/2013	859.30
22270	30 Year Concession Contract	10/07/2006	18/05/2009	298.35
22159	Exploration License*	26/07/2005	25/06/2009	250.60

* Concession Contract with a 28-year term

Certain types of exploration activity require a land use permit, issued by the Colombia Government, prior to conducting work on a mineral property. Mineral rights in Colombia are reserved for the federal government and governed by the Colombian Mining Code. The Colombian Mining Code has been changed and amended on several occasions. The oldest version relevant to the Batero-Quinchia Project is Decree 2685 of 1988 (the Previous Mining Code), which has been replaced and superseded in its entirety by Law 685 of 2001. Proposed Law 1382 of 2010 expired in May 2013, thus leaving Law 685 of 2001 the valid mining legislation. The mining law is administered by the Ministry of Mines and Energy which has relegated the administrative duties concerning concession issues to Agencia Nacional de Mineria (ANM; ministerial decree # 4134, November 3, 2011) and the institution formally known as INGEOMINAS will change its name to Servicio Geológico Colombiano (SGC; Ministerial Decree 4131, November 3, 2011) and be responsible for basic and applied geological investigations.

Figure 4-1:
Project Location Map

Figure 4-2:
Concessions Properties Map



In Colombia, mineral concession agreements consist of three phases, namely the exploration, construction, and exploitation phases, and are governed by Law 685 of 2001. Under the mining code, the exploration phase is for a three-year period, which can be extended for up to four additional two-year periods for a maximum of eleven years. During the exploration phase, annual surface payments, Canon Superficiario (Canon), are payable to the Colombian government on the basis of one minimum daily salary per hectare. The current canon rate is COP$26,848 per hectare (approximately US$9.15/ha) for holdings of less than 2,000 ha. The surface payment is calculated as one minimum daily wage per contracted hectare per year for the first five years of the exploration phase. During years six and seven of the exploration phase, the payment increases to 1.25 minimum daily wages per contracted hectare per year, and in years eight to eleven it increases to 1.5 minimum daily wages per contracted hectare per year. Upon completion of the exploration phase of a concession, the construction phase is for a period of three years, and may be extended for a period of one year, after which it enters its exploitation phase, in which canon fees are no longer payable but are replaced by a production royalty payable to the Colombian government.

License 18567 is a Concession Contract formally registered on September 23, 2013. The Exploration License, which was first issued under Decree 2655 of 1988. License 18567 was granted to TVX Minera Ltda. in 1994 and subsequently assigned to Juan David Uribe Hurtado (Hurtado) in 1998. On January 26, 2000, a request was filed for the term of the license to be suspended due to force majeure. The suspension was approved on April 13, 2000, and the license was renewed in 2004. On May 18, 2009, INGEOMINAS authorized the assignment of 100% of the rights and obligations in License 18567 in favor of Minera Quinchia. In November 2011, the INGEOMINAS office in Ibaque in its official minutes assigned the concession the status of Concession Contract to License 18567 for a 28-year term. On January 10, 2012, Batero requested that the Concession Contract be formally documented in the Colombian Mineral Registry.

License 22159 is an Exploration License, which was first applied for in 1998 but was issued under Decree 2655 of 1988 on July 26, 2005. In 2002, the Embera - Chami native community located within the area of License 22159 was given an opportunity to exercise its preferential right to the area by INGEOMINAS' predecessor agency, the Empresa Nacional Minera Ltda. This right was not exercised, and the license was granted to Silvia Estela Rios Martinez (Martinez) and Hurtado in 2005. On June 25, 2009, INGEOMINAS authorized the assignment of 100% of the rights and obligations in License 22159 in favor of Minera Quinchia. In November 2011, the INGEOMINAS office in Ibaque in its official minutes assigned the concession the status of Concession Contract to License 22159 for a 28-year term. On January 10, 2012, Batero requested that the Concession Contract be formally documented in the Colombian Mineral Registry.

License 22270 is a Concession Contract issued under Mining Law 685 of 2001. It was first applied for by Martinez and Hurtado on July 9, 1998. The concession contract was executed on October 24, 2005. It was registered in the National Mining Registry on July 10, 2006. On May 18, 2009, INGEOMINAS authorized the assignment of 100% of the rights and obligations in License 22270 in favor of Minera Quinchia. The term of the contract is for 30 years from the date of its registration in the National Mining Registry with the right to renew for a further 30-year period.

In 2015, an application for the integration of the three mining titles was filed with the National Mining Agency (ANM), the application was approved according to the PARMZ No. 605 order issued on October 2, 2015. Currently, the expectation is that the Agency proceed with the preparation, subscription and registration of the unified contract.

In 2017, the concession for the use of surface water was renewed for five more years, under resolution No. 0072 of January 19, 2017 issued by the Regional Autonomous Corporation of Risaralda (CARDER), which had been granted through the Resolution No. 0730 of March 7, 2011.

The current and future operations of Batero-Quinchia, including exploration, development and commencement of production activities on the property require such a permit. Other permits governed by laws and regulations pertaining to development, mining, production, taxes, labor standards, occupational health, waste disposal, toxic substances, land use, environmental protection, mine safety and other matters, may be required as the project progresses.

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

5.1 ACCESSIBILITY

Information in this section has been extracted from Evans et al. (2013).

The Batero-Quinchia property is located approximately 95 km by road from Pereira, the capital city of the Department of Risaralda, which has a population of approximately 800,000 in its metropolitan area. Pereira is serviced by multiple flights daily from Bogota and Medellin as well as international flights from Panama City, Panama. The drive to the town of Quinchia from Pereira takes approximately two hours on paved roads. The property is located about nine kilometers south-southeast from the town of Quinchia. From Quinchia a gravel road provides direct access to the La Cumbre project site and a separate gravel road provides direct access to the Dos Quebradas area. Driving time from the town of Quinchia to the property is about 30 minutes.

5.2 CLIMATE

The Project area lies within the warm temperate wet forest zone according to the Holdridge Life Zone climatic classification system and in the tropical monsoonal zone of the Koppen climate classification chart. Climatic zones vary with elevation and are defined as hot (greater than 24°C) below 1,000 m in the Cauca River valley; temperate (18°C to 24°C) between 1,000 m to 2,000 m; and cold above 2,000 m (12°C to 18°C).

Many station-years of daily/monthly climate data from 16 regional climate stations are available (1962 to present). In addition, climate data from up to four on-site climate stations is available from March 2011 to present. Based on the regional climate stations, rainfall ranges from 900 mm to 3,000 mm per year and typically averages greater than 1,000 mm per year. Without significant changes in temperature, the seasons are defined by variations in precipitation with two rainy seasons occurring from March to May and from September to December.

Table 5-1 illustrates climatic data for the La Cumbre weather station (altitude 1,915 m), the climate data are available since 2012.

Table 5-1:
La Cumbre Weather Station Data

Parameter	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Max. Temp. (°C)	28.2	25.9	26.5	27.1	27.2	24.2	26.2	26.5	27.2	24.9
Min. Temp. (°C)	13.2	13.4	13.3	13.7	13.8	13.7	12.9	13.8	13.2	13.6
Mean Temp. (°C)	18.3	18.2	18.4	19	18.9	18	17.8	18.81	18.62	18.14
Precipitation (mm)	138.8	58.3	26	64	118.9	145.3	132	122.6	106.1	164.9
ET -(mm)	83.4	75	76	75	78	72	45	99.8	95.9	94.3
Sunshine (h/month	170	162	161	173	165	156	152	158	158.6	143.3
Days/Precip- > 0,1 (mm)	14	12	8	11	17	19	17	18	17	21

*ET = Evapotranspiration

Topography has an influence on the amount of precipitation. Due to the site's higher elevation, rainfall totals are likely higher than those measured by the local climate station.

The ecological zones defined by the Holdridge Life Zone system are zoned by elevation. The ecological zones that pertain to the Project area include:

Premontane (sub-tropical) wet forest transitional to tropical and dry forest; defined as temperatures greater than 24°C, annual rainfall of 1,500 mm to 2,800 mm and elevation of 700 m to 1,000 m.

Premontane (sub-tropical) wet forest defined as temperatures of 18°C to 24°C, rainfall of 2,000 mm to 4,000 mm and elevation of 1,000 m to 1,900 m.

Lower montane (warm temperate) wet forest defined as temperatures of 12°C to 18°C, rainfall of 2,000 mm to 4,000 mm and elevation of 1,900 m to 2,900 m.

Exploration activities are possible year-round.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The La Cumbre Project is located in the Municipality of Quinchia, also known as "Valle de Los Cerros", Department of Risaralda; having a territorial extension of 149.8 km². According to censuses conducted by the National Administrative Department of Statistics (DANE), in 2005, there was a population register of 31,996 inhabitants; while in 2016 the population is estimated at 33,816 inhabitants (51.2% men and 48.8% women). The distribution by age groups allows identifying that about 30% of the population is between 0 and 14 years of age, while 32% between 30 and 59 years of age; having that 76% of the population is located in rural areas, while 24% of the population is located in urban areas. The illiteracy index is 11.4%, with a net secondary education coverage of 39.5%.

Limited resources are available in the town of Quinchia including emergency medical services, temporary accommodations, and fuel. Quinchia has a daily bus service to Pereira. A greater range of services are available in Pereira. Any mining development on the Project would have access to the national electrical grid. Three 230 kV high tension power lines run along the Cauca River Valley. A 132 kV sub-station at Marmato supplies power to the surrounding area. Locally, there is a 33 kV substation at Irra, approximately 5 km to the southeast, and there is a 13.2 kV substation at Quinchia, approximately 6 km to the northwest. An abandoned railway line runs along the east side of the Cauca River. Abundant water is available locally for process purposes.

5.4 PHYSIOGRAPHY

The property is located within the Cauca-Patia or Inter-Andean physiographic province, which lies on the eastern slope of the Western Cordillera of the northern Andean mountains, on the west side of the Cauca River. As such, it is located in moderately steep-to-steep, mountainous, and relatively rugged terrain at elevations ranging from 800 masl to 2,800 masl. Elevation within the Batero-Quinchia property varies from 1,300 masl to 2,000 masl.

The Project is situated mostly within the temperate zone between 800 masl and 2,000 masl where the climate is warm (18°C to 24°C) and humid. The natural vegetation is scarce there are some relicts of gallery forest, predominantly the tall and low stubble. Much of the original forest cover has been cleared for agriculture and grazing, particularly at lower elevations. Land is used primarily for growing coffee and sugar cane and lesser areas for cattle grazing. Subsistence farming crops include plantain, beans, bananas, and manioc.

6.0 HISTORY

Artisanal mining has taken place in the Batero-Quinchia Project area from Pre-Colombian to modern times. Recent historical accounts indicate that artisanal gold production in the area was greatest during the 1950s (Rodriguez et al., 2000).

Interest was renewed in the area in the late 1970s and culminated in the 1980s in the Miraflores area with the "Asociación de Mineros de Miraflores", a local artisanal mining cooperative.

During the 1990s, the Quinchia area drew the attention of various Canadian junior companies, some of whom acquired ground in the general area.

In 1997, a subsidiary of TVX Gold Inc., Mineria TVX de Colombia, completed a comprehensive review of the Quinchia property but did not follow up with ground work.

The licenses which comprise the Quinchia property were acquired by Juan David Uribe Hurtado and Silvia Stella Rios Martinez (the Vendors) in 1998. The property was optioned by the Vendors to AngloGold Ashanti (AGA) in 2005.

In 2000, INGEOMINAS undertook a series of technical studies in the area including geological mapping, geochemical and geophysical surveying and prognostic (non-NI 43-101 compliant) resource estimations (Baldys and Anderson, 2010).

From 2005 to 2008, a subsidiary of AngloGold Ashanti (AGA), Sociedad Kedahda S.A. (Kedahda) carried out exploration programs consisting of geological mapping and sampling, soil sampling, diamond drilling, preliminary metallurgical testing, and airborne geophysics.

In June 2006, AGA completed bottle roll testing at SGS Lakefield on one sample (Met-15) from a drill hole located within the Dos Quebradas South area. Met-15 was a composite of 10 one-metre samples taken from 162.0 m to 172.0 m in drill hole DD005. The assayed head grade of the composite was 1.53 Au g/t and the calculated head grade was 1.68 Au g/t. The results showed 78.6% gold dissolution after 24 hours and indicated a continued recovery trend after that time.

From January to November 2006, Kedahda completed an 18-hole diamond drilling program totalling 4,701.67 m on the Dos Quebradas, La Cumbre, and El Centro (then Mandeval) targets.

From April to November 2006, Kedahda completed geological mapping at a scale of 1:10,000 along the eastern part of the current property and soil sampling and detailed geological mapping at a scale of 1:2,500 at Dos Quebradas South and La Cumbre. Channel sampling in saprolitic diorite at Dos Quebradas returned values of greater than 3.0 Au g/t across 50 m. B-horizon soil sampling was completed on a 100 m by 25 m grid at Dos Quebradas and on a 200 m by 50 m grid at La Cumbre. Follow up diamond drilling of soil anomalies on both properties intersected porphyry style Au + Cu mineralization.

In November 2007, Caribbean Copper and Gold Corporation (Caribbean) entered into an agreement with AGA whereby it could earn a 100% interest in the property. Caribbean assumed AGA's responsibilities with respect to the underlying agreement.

In July 2009, Caribbean Copper and Gold Corporation, AGA, and the registered holders of the concessions agreed to terminate the agreement. The Quinchia concessions reverted to the original vendors.

In October 2009, Angus Resources Inc., prepared a National Instrument 43-101 compliant Technical Report for the Quinchia gold project.

In July 2010, the company formerly known as Angus Resources Inc., changed its name to Batero Gold Corporation, and was incorporated in 2008. The headquartered is in Toronto, Canada.

In 2010, Batero acquired the Quinchia property through its acquisition of all the issued and outstanding shares of Bahia, a Panamanian company incorporated to hold all of the issued and outstanding capital in Minera Quinchia, a Colombian company which holds all the rights in the Quinchia property.

From January to December 2011, Batero completed a 62-hole diamond drilling program totaling 27,262.34 m on the La Cumbre target.

Roscoe Postle Associates Inc. (RPA), prepared an independent Technical Report, dated February 24, 2012, on the Batero-Quinchia Project, Department of Risaralda, Colombia. The purpose of this report was to document the technical information available on the project and to support the initial mineral resource estimates for the La Cumbre, El Centro, and Dos Quebradas Zones.

From July to September 2012, Batero completed a 29-hole diamond drilling program totaling 4,252.70 m on the La Cumbre target.

Roscoe Postle Associates Inc. (RPA), prepared a Preliminary Economic Assessment (PEA) and NI 43-101 Technical Report on the Batero-Quinchia Project (the Project), Department of Risaralda, Colombia dated December 16, 2013.

During 2014, Minera Quinchia, a Colombian subsidiary of Batero Gold, conducted an auger soil and rock sampling campaign to evaluate and characterize the oxidized saprolite. During the exploration campaign, 17 rock samples and 28 auger soil samples were taken and analyzed for gold and silver. Additionally, geological mapping of outcrop in the area was made to characterize the oxide zone of the La Cumbre deposit.

In 2015, a systematic geochemical auger soil sampling program was carried out on a 50m x 50m grid, 205 samples were collected for gold and multi-element analysis. The samples reached depths of up to 5m. Additionally, 13 rock samples and 13 saprolite concentrates were collected.

From January 2016 to March 2017, Batero complete a 40-hole infill-drilling program totaling 4,574.16 m to evaluate the oxide zone and transition zone of the La Cumbre deposit. A total of 2,222 samples were taken for gold and multi-element geochemical analysis.

In November 2018, LINAMEC prepared a Mineral Resource update which was filed with the Toronto Stock Exchange and disclosed on SEDAR.

In May 2021, TARMA BULK - PERU together with Minera Quinchia selected the overland conveyor with energy regeneration alternative as a result of a study for mineral transport between project components. This study forms part of the larger ongoing pre-feasibility study.

In May 2021, ANDDES - PERU carried out the geotechnical characterization and a permeability study of oxide and transition material, which allowed Minera Quinchia to establish the best binder doses for a 5m high leaching column.

In September 2021, HLC Ingeniería y Construcción, with the collaboration of the Universidad Nacional de Ingeniería, completed and delivered the Basic Engineering Studies for the Development of Overland Conveyor.

In October 2021, SRK CONSULTING - PERU S.A. completed and delivered the engineering studies for the deposits of low-grade and sulfide stockpile located at Matecaña, as well as stockpile of oxide-transition material, leaching pad, processing plant, agglomeration plant and waste dump located in La Perla area.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The following descriptions of the regional and project geology are modified from Baldys and Anderson (2009).

7.1 REGIONAL GEOLOGY

The northern Andean cordillera in Colombia has been uplifted by the subduction of the Nazca oceanic plate beneath the Guiana Shield along with interaction with the Caribbean plate to the north. The region has been tectonically active from the Mesozoic through to the present. Subduction has created magmatic island arcs that have since accreted to the continental margin in generally north-south oriented belts. These magmatic arc terrains host most of the precious metal mineralization in Colombia.

The Quinchia property is located along the eastern margin of Colombia's physiographic Western Cordillera. According to Cediel and Caceres (2000) and Cediel et al (2003), the region is underlain by a highly complex basement known as the Romeral Terrane, which may be characterized as a tectonic mélange. The basement took form when Middle to Upper Mesozoic-aged volcanic and sedimentary oceanic rocks collided with, and were accreted to, the northern Andean paleo-continental margin, beginning in the Early Cretaceous. The resulting suture is known as the Romeral fault system and the mélange can be traced for over 1,000 km along the northern Andes. The original Romeral fault system is generally north-striking and dextral transcurrent in nature whilst the Romeral mélange contains mega-scale blocks and fragments of the oceanic allochthon and crustal slivers of autochthonous Paleozoic metamorphic rocks which formed the paleo-continental margin. The structure of the Romeral system has been modified by various post-Romeral tectonic events.

Following accretion, the Romeral Terrane and mélange were conformably overlain in the Late Oligocene to Early Miocene by autochthonous siliciclastic sedimentary sequences of the Amaga Formation, including basal conglomerates, quartz sandstones, siltstones, shales and coal. In the Middle to Late Miocene, both the Romeral mélange and the Amaga Formation were overlain by mafic to intermediate volcanic flows and pyroclastics of the Combia Formation, associated with at least one Middle to Late Miocene volcanic arc emplaced into the Romeral Terrane basement during this time-period. Also associated with late arc formation was the syntectonic emplacement of a series of intrusive rocks, including polyphase hypabyssal stocks, dikes and sills of dioritic, granodioritic and monzonitic composition.

Figure 7-1:
Lithotectonic and Morphostructural Map of the La Cumbre Project Area

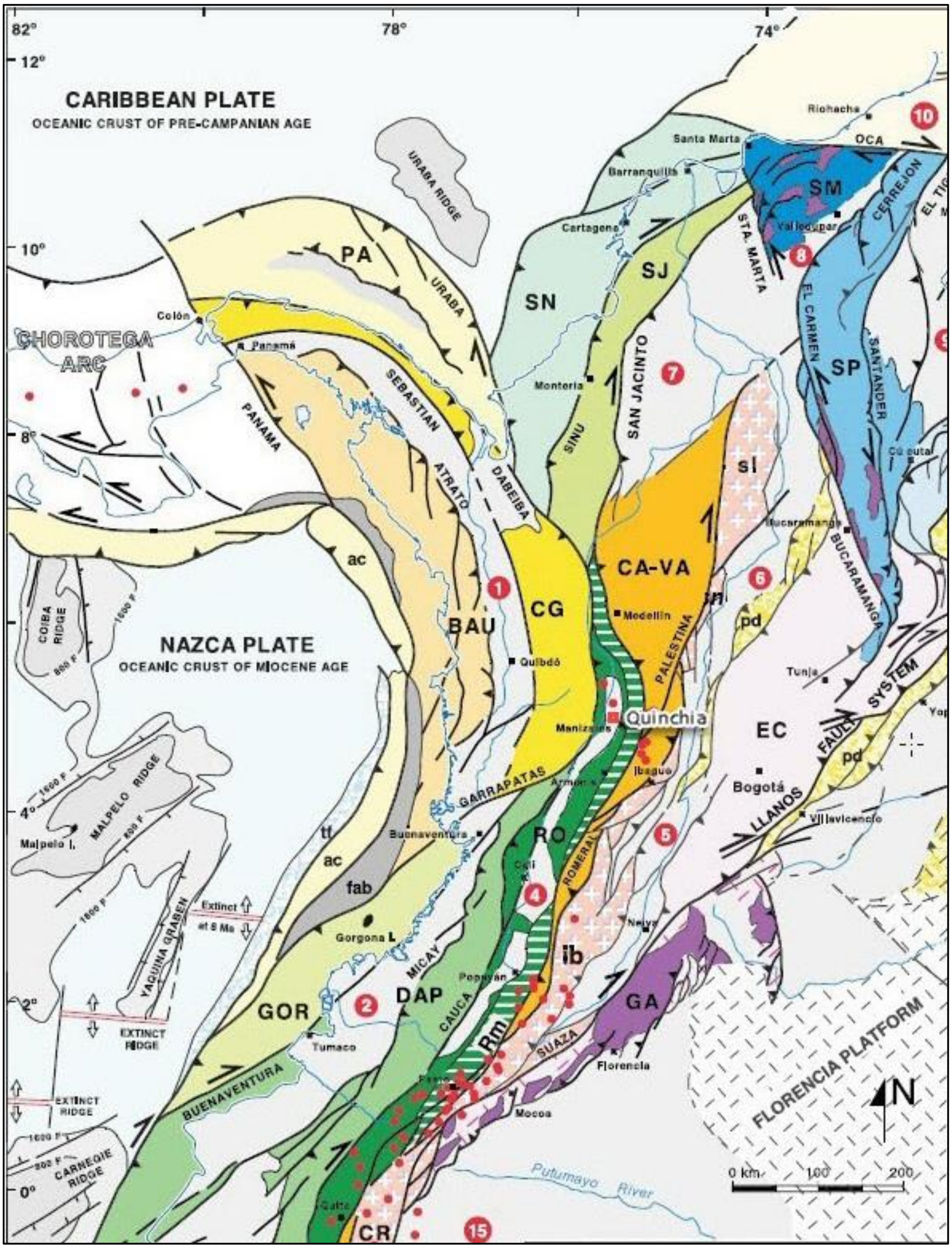


Figure 7-1. Lithotectonic and morphostructural map of northwestern South America. GS = Guiana Shield; GA = Garzon massif; SP = Santander massif-Serrania de Perija; ME = Sierra de Merida; SM = Sierra Nevada de Santa Marta; EC = Eastern Cordillera; CO = Carora basin; CR = Cordillera Real; CA-VA = Cajamarca-Valdivia terrane; sl = San Lucas block; ib = Ibague block; **RO = Romeral terrane**; DAP = Dagua-Pinon terrane; GOR = Gorgona terrane; CG = Canas Gordas terrane; BAU = Baudo terrane; PA = Panama terrane; SJ = San Jacinto terrane; SN = Sinu terrane; GU-FA = Guajira-Falcon terrane; CAM = Caribbean Mountain terrane; Rm = Romeral melange; fab = fore arc basin; ac = accretionary prism; tf = trench fill; pd = piedmonte; 1 = Atrato (Choco) basin; 2 = Tumaco basin; 3 = Manabi basin; 4 = Cauca-Patia basin; 5 = Upper Magdalena basin; 6 = Middle Magdalena basin; 7 = Lower Magdalena basin; 8 = Cesar-Rancheria basin; 9 = Maracaibo basin; 10 = Guajira basin; 11 = Falcon basin; 12 = Guarico basin; 13 = Barinas basin; 14 = Llanos basin; 15 = Putumayo-Napo basin. Source: Cediel et al, 2003

Following the accretionary events, the region was compressionally deformed in the Early to Middle Miocene and again in the Middle to Late Miocene, in both cases by further accretionary events taking place to the west along the active Pacific margin. The structural architecture of the Romeral fault and mélange system is essentially that of a more than 10 km wide series of north-south striking, vertically dipping dextral transcurrent faults. Virtually all lithologic contacts within the Romeral basement are structural, characterized by abundant shearing, mylonitization and the formation of clay-rich fault gouge. Structural reactivation during the Miocene resulted in orthogonal compression accompanied by mostly west- directed (back) thrusting and high-angle reverse fault development in the basement rocks. The Amaga Formation was deformed into generally open, upright folds with tilting and near isoclinals folding being associated with generally localized, west-verging thrusting. The Combia Formation records tilting and open folding and both the Amaga and Combia Formations exhibit moderate to strong diapiric doming were affected by the emplacement of the Mid to Late Miocene intrusive suite. North south, northeast, northwest and east-west conjugate shearing and dilational fracturing affects all of the above geologic units. Some of these elements can be observed as structural lineaments traversing the region.

7.2 PROJECT GEOLOGY

The surface geology of the Quinchía project is composed of several lithological units that form part of the eastern flank of the Western Cordillera. These include rocks of Cretaceous age corresponding to the Barroso Formation (basalts and diabases), hypabyssal bodies correlated with the dacitic and andesitic Irra porphyry, "intrusive" contact breccias related to the contacts between the different intrusive phases, and effusive volcanic rocks correlated with the Combia Formation (volcanic tuffs and agglomerates), see Figure 7-2.

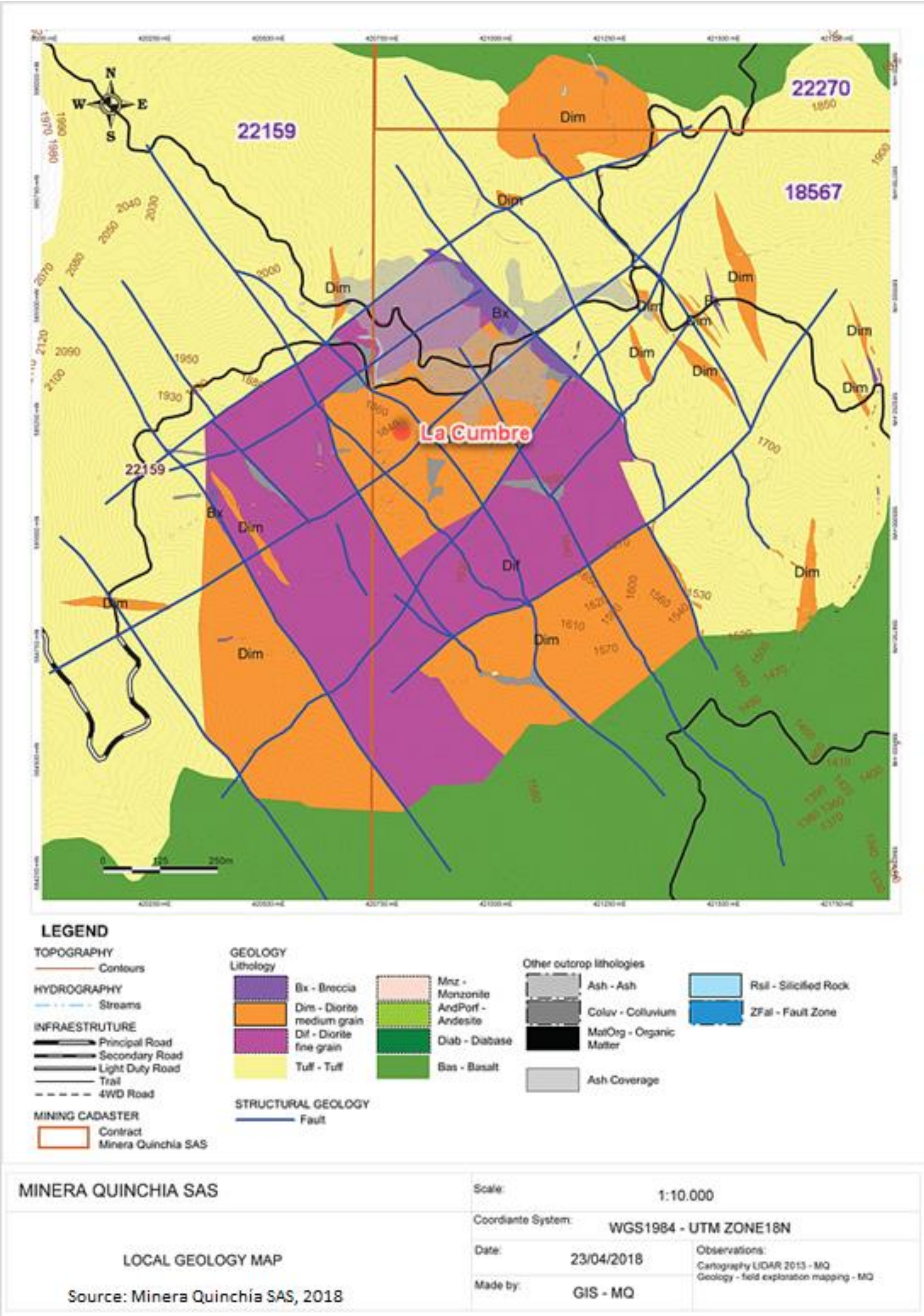
7.2.1 Barroso Formation

Composed of basalt and massive diabase, the color varies from greenish gray to dark gray, fine grained and aphanitic texture, sometimes with epidote veins and amygdales filled by zeolites. The Barroso Formation extends from the East of Viterbo to the Cauca Almaguer fault, this formation is separated from the Western Strip by a NS-NNE fault, which controls the eastern contact of the Anserma gabbro (Estrada & Viana, 1998). In the study area, basalt predominates over diabase. The color of these rocks is green with an aphanitic texture, most have cavities filled with quartz and zeolites. The mineralization consists of magnetite (7-10%), pyrite (2%) and chalcopyrite in trace amounts that occurs as veinlets and disseminated grains. The veinlets sometimes form a stockwork composed of quartz (qtz) veins mainly “A” type, magnetite (mt) veinlets “M” type and regularly “B” type veinlets (qtz + py).

7.2.2 Dos Quebradas - La Cumbre Porphyritic Stock

This intrusive body is named the dacitic - andesitic porphyry of Irra by Estrada & Viana (1998). Bedoya et al (1999) report the presence of a series of hypabyssal bodies in the Piedras Creek towards the western part of La Cumbre Hill. In this work, a series of hypabyssal bodies is mentioned, in intrusive contact with the basaltic rocks of the Barroso Formation. The rocks have a porphyritic texture and a medium to fine grain size and vary from green to light gray in color, depending on the type of hydrothermal alteration they have suffered.

Figure 7-2:
La Cumbre - Project Geology



The stock is a mixture of bodies with sub-tabular geometries that are cross cut according to the genetic order of formation; the early phases are intruded by younger phases. "*Stocks are generally composite, with early porphyries being intruded by intermineral and late-mineral phases, a mechanism that causes episodic inflation of the stocks. Progressively younger* *porphyry phases are commonly intruded into axial portions of stocks, giving rise to nested geometries, see Figure 8-2, (Sillitoe, 2000)*".

The porphyritic texture indicates that the magmas intruded and crystallized near the surface and due to their shallow nature are called "epizonal intrusives"; the rocks can be equigranular in texture with moderately coarse grain size (Maksaev, 2004). It is common to find flow structures subparallel to the lines of contact with basalts defined by the alignment of amphibole crystals and/or plagioclase and xenoliths assimilated by the intrusion and "A" type veinlets truncated by the intrusion.

The porphyritic stock is classified macroscopically as a diorite with a porphyritic texture, consisting of plagioclase phenocrysts altered to clays (illite and kaolin), minor quartz (<5%), and locally hornblende and augite. It is also common to find primary biotite as subhedral crystals embedded in a microcrystalline felsic matrix consisting of quartz, feldspar, plagioclase, biotite and/or magnetite.

Both geological mapping and diamond core logging have established that diorite is the most abundant lithology in the La Cumbre deposit, where at least two phases have been identified: a fine-grained diorite and a medium-grained green to gray diorite.

7.2.3 Contact Breccias

The contact breccias form a narrow band that belong to an oligomictic meso-breccia. The fragments are post-mineral intrusives with sub-rounded and subangular shapes replaced by kaolin and calcite. The matrix is composed of green colored rock dust due to the presence of illite and calcite is observed as cement. The mineralization is 3% pyrite, and traces of magnetite. No veinlets of any type are observed. One of the problems for the identification of this type of breccia is the strong superimposition of pervasive hydrothermal alteration, which destroys the original texture. Microscopically, the rock shows a total replacement of the plagioclase phenocrysts by calcite, sericite and illite, with a matrix replaced by sericite, illite, smectite and fine granular quartz.

7.3 LOCAL STRUCTURAL GEOLOGY

Information in this section has been extract from Harabi (2015, SRK internal report).

The Quinchia property is underlain by Mesozoic aged rocks that form an accretionary complex known as the Romeral Terrane assembled during the Early Cretaceous. The north-northeast-trending Romeral fault system marks the suture and can be traced for over 1,000 kilometres along the northern Andes. Oligocene to Late Miocene sedimentary and volcanic rocks conformably overlie the Romeral Terrane rocks. The Quinchia porphyry deposit is hosted in a Miocene arc-related series of intrusive rocks, including polyphase hypabyssal dioritic, granodioritic and monzonitic stocks, dikes and sills.

On a regional scale, the main structural trend in the Quinchia district is the generally north northeast-striking, subvertical basement architecture of the Romeral fault system. Previous mapping has identified a series of fault sets with northwest, north to north northeast, northeast, and east trends. The northwest-trending Amarilla structural corridor that bisects the deposit areas is a particularly prominent example.

Normal movement faults are major features in the map area, with the best expressed kinematics defined on the Granates fault, where well-developed slickenlines and steps define the normal movement on the fault. The similar orientation of the La Cumbre fault suggests it may be a related feature. The normal movement along moderately southwest to south southwest-dipping normal faults is associated with epithermal mineralization at the nearby Miraflores and at Mina Guayacanes deposits.

A set of north to north, north east striking faults with predominantly dextral movement and little alteration crosscut altered southeast-striking faults along the Miraflores trend. A set of west northwest- to west-striking faults also cut across the traces of most other fault sets and were observed to have sinistral strike-slip slickenlines and steps, or offset magnetic markers and/or mineralization at Dos Quebradas. These fault sets are interpreted to post-date mineralization. The relative late timing of these fault sets and the movement sense on them are consistent with a renewed northeast-southwest compression.

The Amarilla fault has a complex multi-movement history. Evidence for both sinistral separation of units and dextral strike slip kinematic indicators are observed. This apparent contradiction can be explained by either normal to oblique sinistral normal early movement on the fault followed by a dextral strike slip reactivation later in its history.

7.4 HYDROTHERMAL ALTERATION

The main hydrothermal alteration present in the area includes:

7.4.1 Potassium Alteration

It is divided into three alteration subtypes that are:

7.4.2 Potassium Alteration Rich in Potassium Feldspar

It is characterized by dominant potassium feldspar, biotite-chlorite as subordinate minerals, magnetite and chalcopyrite in thin scattered grains and gypsum as alteration of anhydrite in veinlets. It mainly affects the early phases of the Dos Quebradas intrusion to a depth of 160 to 200 meters.

7.4.3 Potassium Alteration Rich in Secondary Biotite

The main characteristic of this type of alteration is the development of new biotite crystals with a fine grain size. Guilbert and Park (1986) call it the biotite zone or biotite-chlorite zone, generated mainly in intermediate rocks (andesite-diorite), in which the primary biotite is altered to a more magnesian variety, together with rutile formed by the release of titanium during the chemical reaction (Beane and Titley, 1981). The secondary biotite is located at 240 to 275 m depth and is the main feature of the La Cumbre intrusive center. The biotite is fine grained, appears disseminated and semi-pervasive and is associated with magnetite and chalcopyrite giving a brown coloration to the rock.

7.4.4 Calcium Potassium Alteration

This alteration is developed mainly in the basalts of the Barroso Formation. The mineralogical association is actinolite, biotite, and chlorite as small crystals of subhedral to euhedral forms, or actinolite, anhydrite, garnet in veinlets. The color of the rock varies between green-brown and brown. In hand specimen, the alteration is mainly restricted to the reaction halos of type A and type M veinlets but also occurs as a patch of fine biotite in the basalt. At the microscopic level the alteration is pervasive and has a random texture typical of contact metamorphism. It is found in the basalts of the El Centro and Dos Quebradas areas close to the contacts of the intrusive body. Calcium-potassium alteration does not occur in the La Cumbre intrusive center.

In LINAMEC's opinion, it is considered relevant to determine an association between the distinct types of potassic alteration with their secondary magnetite contents potentially associated with Au and Ag contents.

7.4.5 Propylitic Alteration

The propylitic alteration comprises a wide alteration halo that affects both the wall rock (basalts of the Barroso Formation), the volcanic rocks of the Combia Formation and the late intermineral phases present in the porphyritic stock. Chlorite replaces hornblende and biotite and is seen microscopically as pseudomorphs or fine lamellae following the crystal exfoliation planes and replacing the matrix. The epidote occurs disseminated in crystalline aggregates with subhedral forms and short prismatic habits. Anhedral calcite is observed filling cavities in the matrix, replacing plagioclase crystals and in small fractures. Sericite associated with calcite appears as fine lamellae on plagioclase, sericite is developed by the potassium released by the chloritization of biotite (Beane and Titley, 1986).

7.4.6 Intermediate Argillic Alteration

Intermediate argillic alteration (IAA) is characterized by the formation of clay minerals and is a product of intense H + metasomatism (acid leaching) at temperatures between 100 and 300 °C. The occurrence of kaolinite group minerals and the partially destroyed texture of the rock define this alteration. Typical minerals are kaolinite, dickite, montmorillonite, illite, chlorite, and small amounts of sericite (Sillitoe, 2000). In the La Cumbre sector IAA was observed mainly in the El Centro area where, with the help of drill logs, a NW strip parallel to Mandeval Creek has been mapped. Other sectors where the IAA occurs are La Lengüita and Dos Quebradas.

7.4.7 Supergene Argillic Alteration

In different outcrops and in all drillholes carried out in the La Cumbre area, the supergene argillic alteration (SAA) was found. This alteration forms an oxidation zone or soil profile with argillic supergene alteration that varies from 30 to 70 meters in average thickness. This leached zone, within the mineralized zone, has a 1.0 Au g/t content. The alteration indicates that tuffs, when altered below the crystallization temperature of volcanic glass, are altered to palagonite (orange yellow amorphous mineraloid) and later to clay minerals such as smectite and then to zeolites (McPhie et al., 1993). In some sectors evidences of these minerals were found as a product of alteration of volcanic glass.

7.5 MINERALIZATION

The following is modified from Baldys and Anderson (2009).

The Dos Quebradas, El Centrol and La Cumbre porphyry gold deposits are associated with three Miocene intrusive centers in a north-south trend that have a strike extension of over two kilometers at elevations between 1,600 MASL and 1,050 MASL. These intrusive centers are composed of dikes and stocks separated into three groups: i) early inter-mineral, ii) late inter-mineral, and, iii) post mineral. These dioritic phases were emplaced into intermediate to felsic volcanic rocks of the Miocene Combia Formation and Cretaceous basalts of the Barroso Formation.

The three porphyry gold deposits on the Batero-Quinchia property are described as follows:

1. **Dos Quebradas**: the mineralized area occurs in the incised Dos Quebradas valley and covers an area of approximately 700 m by 700 m trending north of Batero's Concession Contract 22270.

2. **El Centro (previously Mandeval)**: a lower grade, possibly deeper deposit covering an area of approximately 800 m by 500 m.
3. **La Cumbre**: the deposit covers an area of approximately 1,000 m by 600 m.

These deposits display typical features described by Sillitoe (2000). The three deposits are copper-poor porphyry gold systems in which intermediate argillic alteration locally overprints an early potassic assemblage and its associated quartz veinlet stockwork (Jahoda, 2007).

According to Jahoda (2007), gold in these targets occurs in altered dioritic intrusions and in the diorite-basalt and diorite-volcanoclastic contact zones. The highest gold and copper grades encountered at Quinchia to date occur in the early diorite phases characterized by potassic (mainly biotite with subordinate K-feldspar) and potassic-calcic alteration that is characterized by the addition of traces of actinolite and garnet to the potassic assemblage. Significant amounts of quartz ± sulfide veinlets and more than 3% hydrothermal magnetite is common in these early phases.

Gold values in the early diorite are highest where hydrothermal biotite and fine-grained chalcopyrite reach maximums. Gold grades are lower in the inter-mineral phases; they still have potassic alteration with a lower density of veinlets compared with the early intrusive phases. Sulfide contents in early intra-mineral phases are normally lower than 1% but up to 3% and include pyrite, and locally trace amounts of chalcopyrite, bornite, and molybdenite.

Late inter-mineral intrusive phases present moderate to strong intermediate argillic alteration with an average sulfide content of 3% to 5% composed mainly of pyrite with traces of molybdenite and chalcopyrite. The late inter-mineral phases are devoid of potassic alteration and quartz veins. Post mineral dikes exhibit argillic alteration (kaolinite) with subordinate chlorite and epidote.

Gold and copper grades in basaltic wall rock follow potassic biotite and potassic calcic (biotite-actinolite) alteration. A-veinlet densities reach up to 50 veinlets per meter. Most artisanal mining activity in the Quinchia area follows centimetric fault gouge along faults in tuffaceous volcanic rocks with strong intermediate argillic alteration. Gold occurs with the fault gouge that contains fine-grained pyrite.

The intrusions that host mineralization consist of several phases of diorite and later andesitic dikes exhibiting characteristic alteration zoning, possibly because of telescoped porphyry and epithermal systems.

Mineralization at La Cumbre was discovered through follow up drilling of surface geochemical anomalies by Kedahda in 2006. The mineralization was best tested in the central part of La Cumbre where four holes were drilled along an east-west section. Hole DD008 intersected 210 m of 0.80 Au g/t and 0.15% Cu from surface to a vertical depth of 150 m. Drilling to the south of this section confirmed the mineralization at surface which continued as a low-grade zone to 520 m depth in hole DD018.

Five holes were drilled by Kedahda in 2006 in the area of Dos Quebradas (Batero concessions). This resulted in the discovery of significant gold mineralization within the diorite (holes DD004 and DD005) and across the contact into the surrounding basalts where hole DD006 intersected 216 m averaging 0.746 Au g/t and 0.11% Cu. Kedahda also drilled two holes on the adjoining Dos Quebradas North property owned by Seafield Resources Ltd.

7.5.1 Characterization of the Oxide Zone

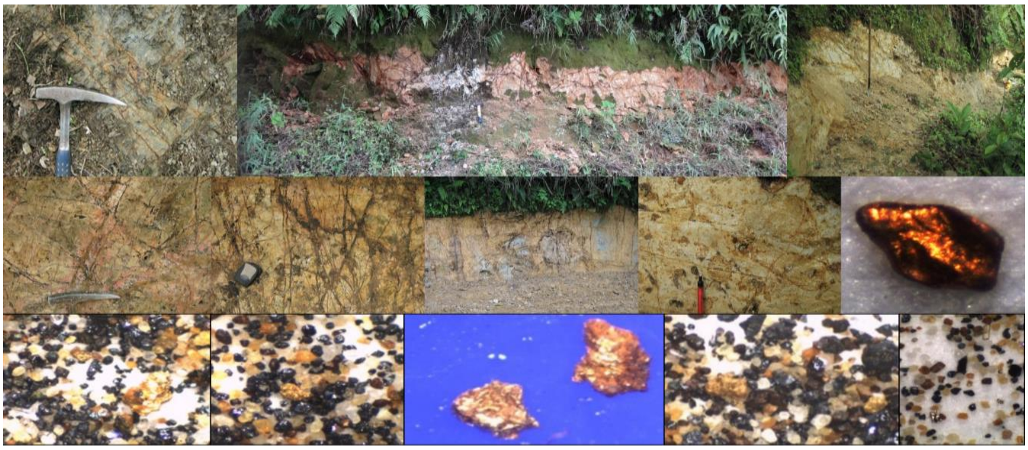
The leaching processes generated on the rock, the action of late mineralizing fluids and later supergene processes, have allowed the development of an oxidation zone within the saprolitized rock. This oxidation zone directly affects the mineralization contained in the porphyry system, and the surrounding wall rock mineralization. These late and supergene processes allow the oxidation of the sulfides, releasing the metals in a natural way by dissolution generated by meteoric water. The dissolution releases the gold, copper and silver contained within the sulfide matrix, and generates an enrichment within the superficial layer of the rock (see Figure 7-3 and Figure 7-4).

During the fourth quarter of 2014, the Company developed an exploration program for the oxide zone located within the La Cumbre deposit. The program determined that the mineralized body has a greater extension than that defined in the initial exploration stages and contains a potential resource exploitable in the short term. Additionally, the exploration defined new potential areas, which must be examined in more detail.

Figure 7-3:
Mineralized Rock in the Oxide Zone

(Source Minera Quinchía SAS).

Figure 7-4:
Oxide Zone, Mineralized Rock and Free Gold Panned from the Oxide Zone



(Source Minera Quinchía SAS).

8.0 DEPOSIT TYPES

The porphyry intrusions genetically related to all gold-rich porphyry deposits belong exclusively to the I-type, magnetite series suites (e.g., Ishihara, 1981). The abundance of hydrothermal magnetite in gold-rich porphyry deposits may be taken to suggest that their host intrusions are highly oxidized, sulfur-poor representatives of the magnetite series (Sillitoe, 1979). Bulk-tonnage gold deposits are also hosted by and related genetically to more reduced, either magnetite or ilmenite series intrusions, but these are of sheeted-vein rather than truly porphyry type (Thompson et al., 1999; Thompson and Newberry, 2000). Similarly, several “porphyry" copper-gold deposits related to ilmenite series intrusions (Rowins, 2000) are not considered to be porphyry type in the strict sense (Sillitoe, 2010), see Figure 8-1 and Figure 8-2.

La Cumbre at Quinchia is a discrete porphyry gold center in which the drilling campaigns have revealed an average gold content reaching economic tenor even though the quartz-veinlet intensity is relatively low (Sillitoe, 2006).

According to Sillitoe (2006), the Quinchia district is located within Colombia’s late Miocene, Middle Cauca porphyry belt, which is a well-defined volcano-plutonic arc with confirmed potential to host gold (copper)-rich porphyry systems.

The Dos Quebradas, El Centro and La Cumbre porphyry gold targets are associated with three Miocene intrusive centers in a N-S trend that have a strike extension of approximately 3km at elevations between 1600 m and 1950 m. These intrusive centers are composed of dikes and stocks separated into three groups: early inter-mineral, late inter-mineral and post-mineral dioritic phases emplaced into the intermediate to felsic volcanic rocks of the Miocene Combia Formation.

“Gold-rich porphyry deposits worldwide conform well to a generalized descriptive model. This model incorporates six main facies of hydrothermal alteration and mineralization, which are zoned upward and outward with respect to composite porphyry stocks of cylindrical form atop much larger parent plutons. This intrusive environment and its overlying advanced argillic lithocap span roughly 4 km vertically, an interval over which profound changes in the style and mineralogy of gold and associated copper mineralization are observed. The model predicts a number of geologic attributes to be expected in association with superior gold-rich porphyry deposits. Most features of the descriptive model are adequately explained by a genetic model that has developed progressively over the last century. This model is dominated by the consequences of the release and focused ascent of metalliferous fluid resulting from crystallization of the parent pluton. Within the porphyry system, gold- and copper bearing brine and acidic volatiles interact in a complex manner with the stock, its wall rocks, arid ambient meteoric and connate fluids. Although several processes involved in the evolution of gold-rich porphyry deposits remain to be fully clarified, the fundamental issues have been resolved to the satisfaction of most investigators. Exploration for gold-rich porphyry deposits worldwide involves geologic, geochemical, and geophysical work but generally employs the descriptive model in an unsophisticated manner and the genetic model hardly at all. Discovery of gold-rich porphyry deposits during the last 30 years has resulted mainly from basic geologic observations and conventional geochemical surveys and has often resulted from programs designed to explore for other mineral deposit types.” (Abstract, Sillitoe 2000, p. 315)

As with other authors (Sillitoe, 2006), LINAMEC agrees that in the case of the La Cumbre Project, the mining potential is primarily related to the potassic zone where there are high contents of hydrothermal secondary magnetite probably associated with Au and Ag contents.

Figure 8-1:
Conceptual Porphyry Cu-Au Deposit Model (after Sillitoe, 1995)

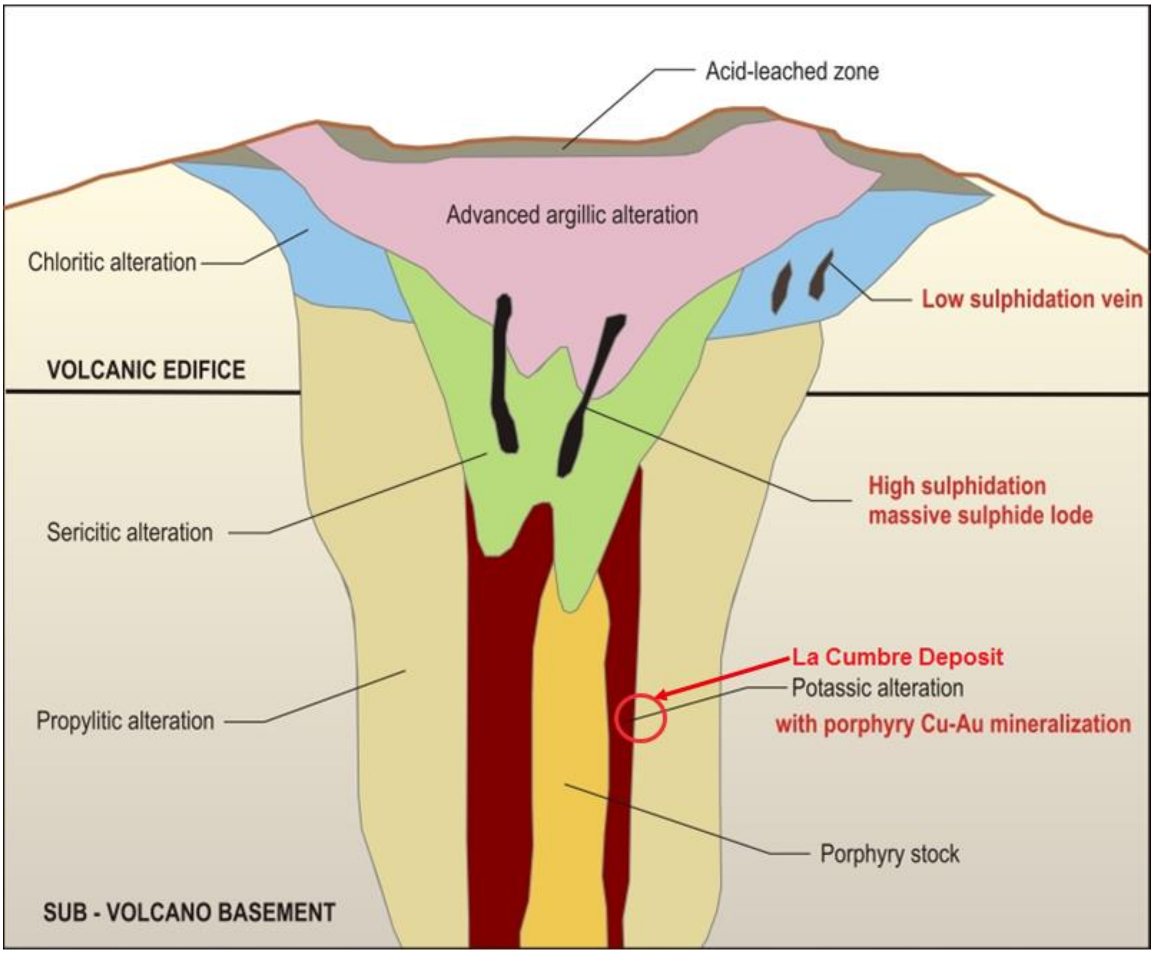
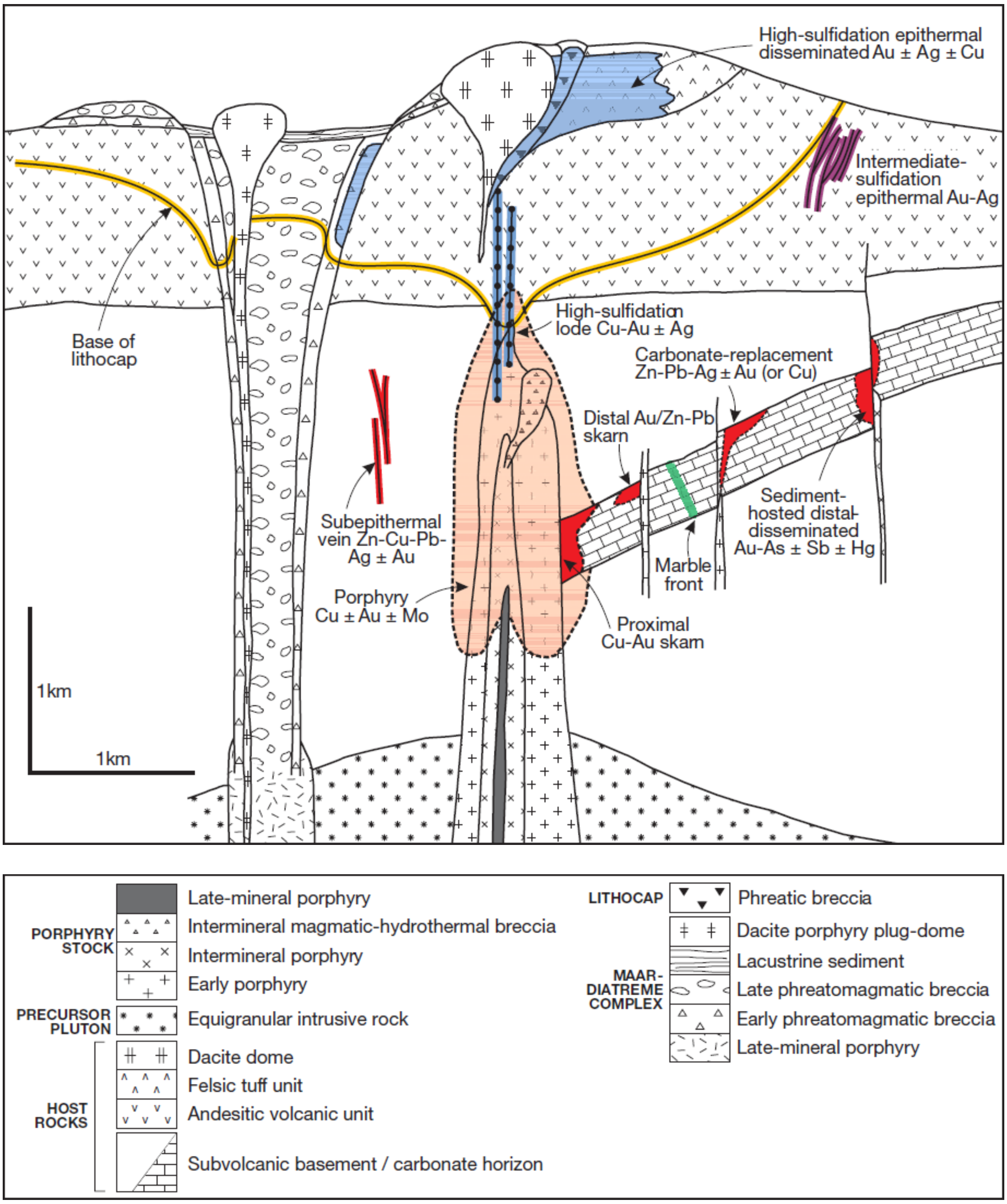


Figure 8-2:
Descriptive Model of Typical Gold-Rich Porphyry System (after Sillitoe, 2009)



Anatomy of a telescoped porphyry Cu system showing spatial interrelationships of a centrally located porphyry Cu ± Au ± Mo deposit in a multiphase porphyry stock and its immediate host rocks; peripheral proximal and distal skarn, carbonate-replacement (chimney-manto), and sediment-hosted (distal-disseminated) deposits in a carbonate unit and subep-ithermal veins in noncarbonate rocks; and overlying high- and intermediate-sulfidation epithermal deposits in and alongside the lithocap environment. The legend explains the temporal sequence of rock types, with the porphyry stock predating maardiatreme emplacement, which in turn overlaps lithocap development and phreatic brecciation. Only uncommonly do individual systems contain several of the deposit types illustrated, as discussed in the text. Notwithstanding the assertion that cartoons of this sort add little to the understanding of porphyry Cu genesis (Seedorff and Einaudi, 2004), they embody the relationships observed in the field and, hence, aid the explorationist. Modified from Silli-toe (1995b, 1999b, 2000).

9.0 EXPLORATION

Information in this section has been extracted and updated from Evans et al. (2013).

Work undertaken on the Batero-Quinchia properties up to 2013 was focused over and adjacent to the La Cumbre - Dos Quebradas mineralized area and included outcrop mapping, rock channel and chip sampling, soil sampling, test pitting, ground based magnetic, induced polarization and radiometric surveys and drilling (see Chapter 10.0 Drilling).

Mapping of outcrops is limited to creek, trail, and road exposures and is completed by tape and compass methods supported by hand held GPS and a network of topographic control points. This mapping has provided insight into the potential structural controls on the mineralization in the areas surrounding the porphyry mineralization, as well as lithological and alteration distributions. The mapping has also discovered and documented the presence of approximately 30 artisanal mine tunnels that were working along narrow fault zones and veins bearing epithermal style mineralization.

Rock channel and chip sampling (two-meter continuous samples of outcrop) collected 3,550 samples (maximum result of 10 g/t Au, mean result 0.240 g/t Au) principally in road cuts and stream exposures. Where possible, a rock sawn channel sample was collected, in other cases a hammer and chisel sample were collected (average sample weight was 1.5 kg to 2.0 kg). Of the collected samples, 3,470 are from the La Cumbre - Dos Quebradas mineralized corridor, with the other samples taken from within the remainder of the concession areas. The results of the rock sampling were used to assist in targeting the 2011 drill program. In the case of the porphyry mineralization, the samples approximately delimit the extents of the mineralized centers and in the case of the areas between the porphyry centers establish areas of structurally controlled mineralization.

Auger soil sampling, completed in stages, was conducted along north-south lines spaced 100 m apart with a 20 m station spacing. The 2011 to 2012 survey covers the main La Cumbre - Dos Quebradas corridor and was extended to the west on a 50 m by 50 m grid and to the east within the concession boundaries on 100 m line spacing with 50 m sample spacing along the lines.

In 2013, the remainder of the concession area was covered by reconnaissance sampling on a 100 m by 100 m grid. Sample depths range from five meters to seven meters with the C-horizon (saprolite) being sampled. The total area of the survey covers approximately 1,300 ha and includes 3,640 samples (maximum analytical result 6.5 g/t Au with a mean of 0.13 g/t Au). The soil grids highlight a strong mineralized trend in both Au (Figure 9-1) and Cu, especially over the La Cumbre - El Centro - Dos Quebradas areas highlighting the three principal mineralized centers in the northwest portion of the Batero-Quinchia property. As a result of the soil survey, seven additional areas of interest have been outlined throughout the remainder of the concession area. The interpolated (ID2) copper distribution map clearly shows the break in the mineralization trend at the Amarilla structural corridor between the La Cumbre - El Centro - Dos Quebradas deposits. The Amarilla corridor is host to fault and vein controlled epithermal style mineralization previously exploited by artisanal miners (samples from the area assayed up to 36.6 g/t Au).

Test pits, shallow excavations of up to two meters in depth and one to 1.5 m, were sampled in vertical profile and when the base encountered bedrock an additional sample was collected. From the test pits 560 samples were collected (maximum analytical result of 3.29 g/t Au with a mean of 0.251 g/t Au). This sampling confirmed the near surface mineralization at the La

Cumbre deposit area, and identified additional areas to the east named the Antena target, and to the southwest named Cumbre Sur.

In 2010, ground based geophysical surveying of 53.05-line kilometers comprising induced polarization (IP), magnetics, and natural radiation spectrometry (NRS) was completed over the La Cumbre-Dos Quebradas area by ARCE Geofisicos of Peru. Survey specifics include:

1. 24 profiles averaging approximately 2,000 m running north - south.
2. NRS reading every 10 m along lines with 30 second sample durations (52.99 km in 24 profiles).
3. Magnetometer readings every 10 m with two Scintrex ENVI proton precession units, one used as a base station (53 km in 24 profiles).
4. IP readings every 50 m along lines using pole-pole (2 array) electrode configuration with a plotting point at the mid distance between the moving electrodes C1 and P1, seven successive "a" spacings of 50 m, 100 m, 150 m, 200 m, 250 m, 300 m and 350 m were used with self-potential, chargeability and resistivity (53.05 km in 24 profiles).
   - Transmitter: IRIS VIP4000; 220V-60Hz generator
   - Energizing field: direct current (time-domain) with 2 in. pulse duration
   - Maximum available tension: 3,000 V; applied current up to 3 A true intensity
   - Receiver: IRIS ELREC PRO with 1 uV resolution, 100M ohm, 20 partial chargeability windows

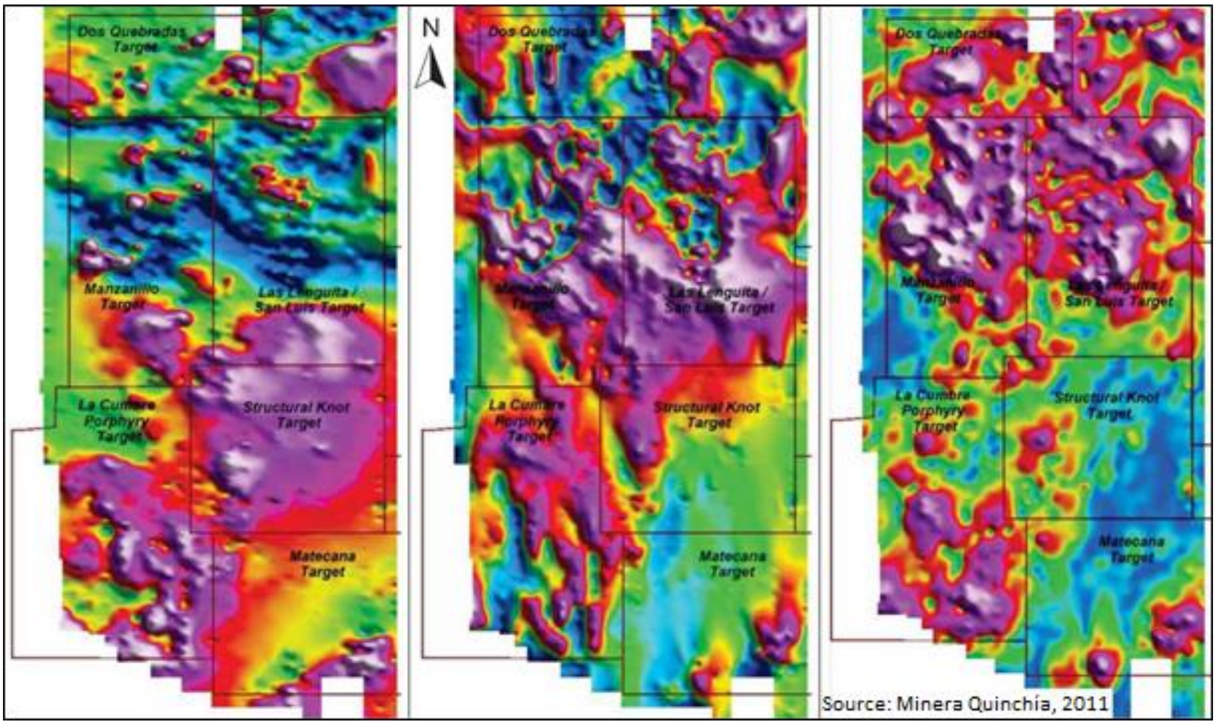
Targets generated by Batero in 2013 via data compilation, new mapping, and sampling included two areas with potential for gold mineralization within an oxide zone, both adjacent to the La Cumbre deposit (Antena and Cumbre Sur), six targets for vein or fault controlled mineralization with sample results ranging from 1.0 Au g/t to 36.6 Au g/t (Kobey, Matecaña, Triunfo, La Perla, Llanadas, Esmeralda), and a single target for a gold-copper porphyry style deposit (Esmeralda SW), see Table 9-1 for exploration targets.

Table 9-1:
La Cumbre – Exploration Targets

Target Name	Mineralization Style	Soil Anomaly Components	Geophysical Anomaly type
Antenna	Oxide Au	Au	
Cumbre Sur	Oxide Au	Au, Cu	
Kobey	High grade vein	Au, Ag, As, Sb, Hg	Linear structure
Matecana	High grade vein	Au, Cu, Ag, As, Sb, Pb, Zn	Linear & intrusion structures
Triunfo	High grade vein	Au, Cu, Ag, As, Sb, Hg, Pb, Zn	Linear structure
La Perla	High grade vein	Au, Cu, Zn	Linear structure
Llanadas	High grade vein	Au, Cu, Ag, Hg	Linear structure
Esmeralda	High grade vein	Cu, Ag, As, Sb, Hg, Pb, Zn	Linear structure
Esmeralda SW	Porphyry	Au, Cu, Ag, Sb, Mo	Intrusion structure

The results of the IP, magnetics and NRS surveys were supplied to consultant TEP Ltda. (TEP) for interpolation, modelling, interpretation and target recommendation (Hernandez- Pardo, 2011). TEP's processing was later supplied to 3D Geoscience Inc. of Canada to reprocess and complete 3D modelling and target recommendations (Ballantyne, 2011). This data was provided as the 2011 drill program was initiated. Figure 9-1 illustrates the results of the total magnetic intensity, reduced to the pole anomalies and analytical signal. In LINAMEC's opinion, the geophysical method of magnetometry yielded the best results in this type of deposit and may be used in future geological prospecting activities including diamond drilling.

Figure 9-1:
Magnetic Survey Results – Quinchia Project

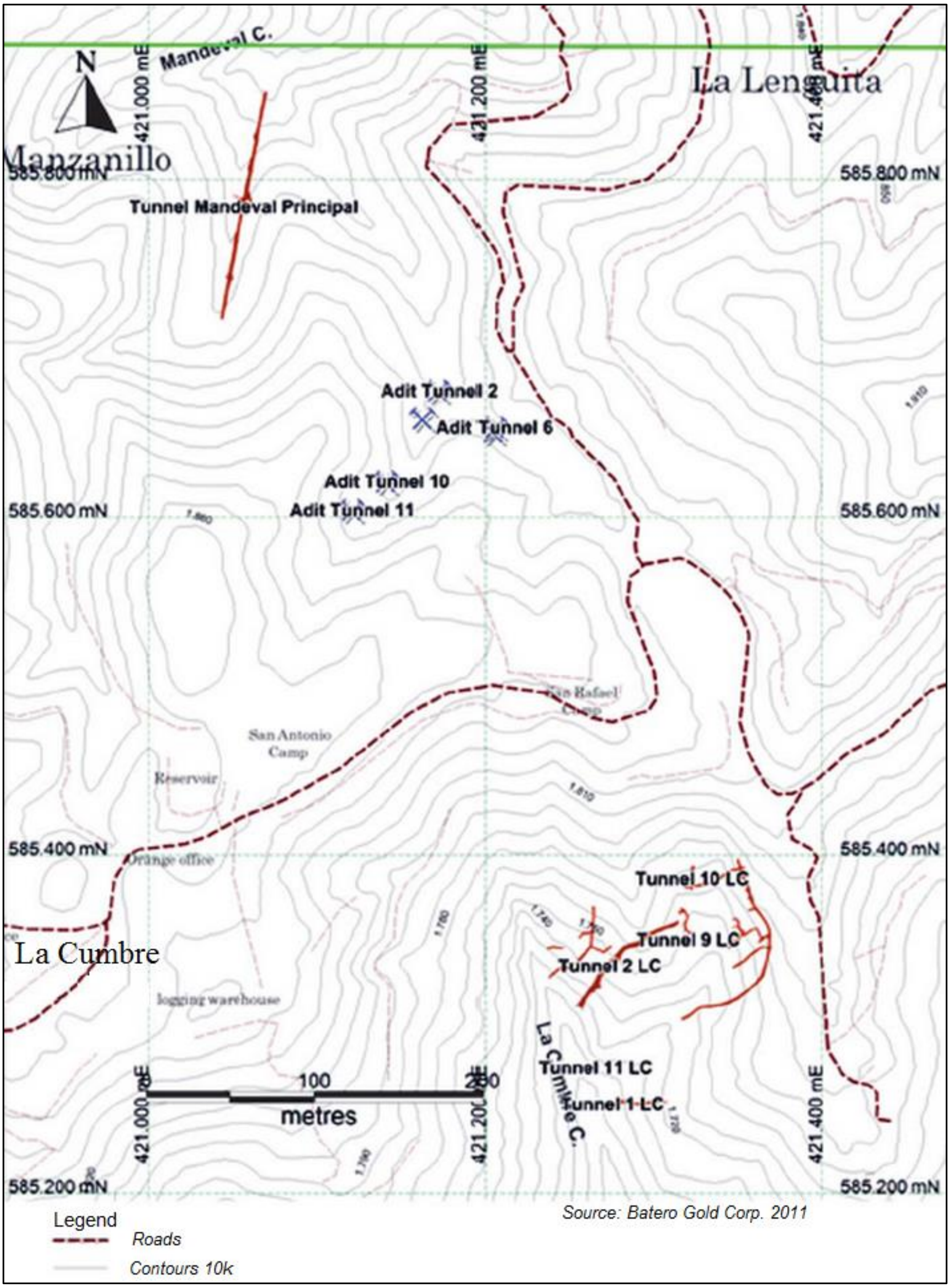


In May 2011, Batero announced it had acquired a 100% interest in two historic, artisanal gold mines (La Cumbre and Mandeval) at the Batero-Quinchia Project. The locations of the La Cumbre and Mandeval adits are shown on Figure 9-2. Subsequently, Batero rehabilitated, mapped and sampled the old workings. A total of 670 m of old workings have been rehabilitated to date and 1,299 samples were collected and analyzed at ALS Chemex. These samples yielded a maximum result of 16.65 g/t Au and a mean result of 0.32 g/t Au. Samples were taken using the same procedure as surface rock samples, i.e., continuous cut channel or hammer and chisel samples, with two-meter intervals. Samples were collected along both sides of the tunnel approximately one meter from the floor and a sample of the tunnel back, as an arc between the lateral samples continuous across the back, approximately every 20 m.

Mineralization in the tunnels shows an epithermal style of mineralization and also occurs as mineralized fault breccia. During the 2013 soil-sampling program, an additional 15 historical adits and two operational adits were encountered and sampled. None of these adits has been rehabilitated. Further exploration and drilling are required to fully evaluate this structurally controlled epithermal gold trend.

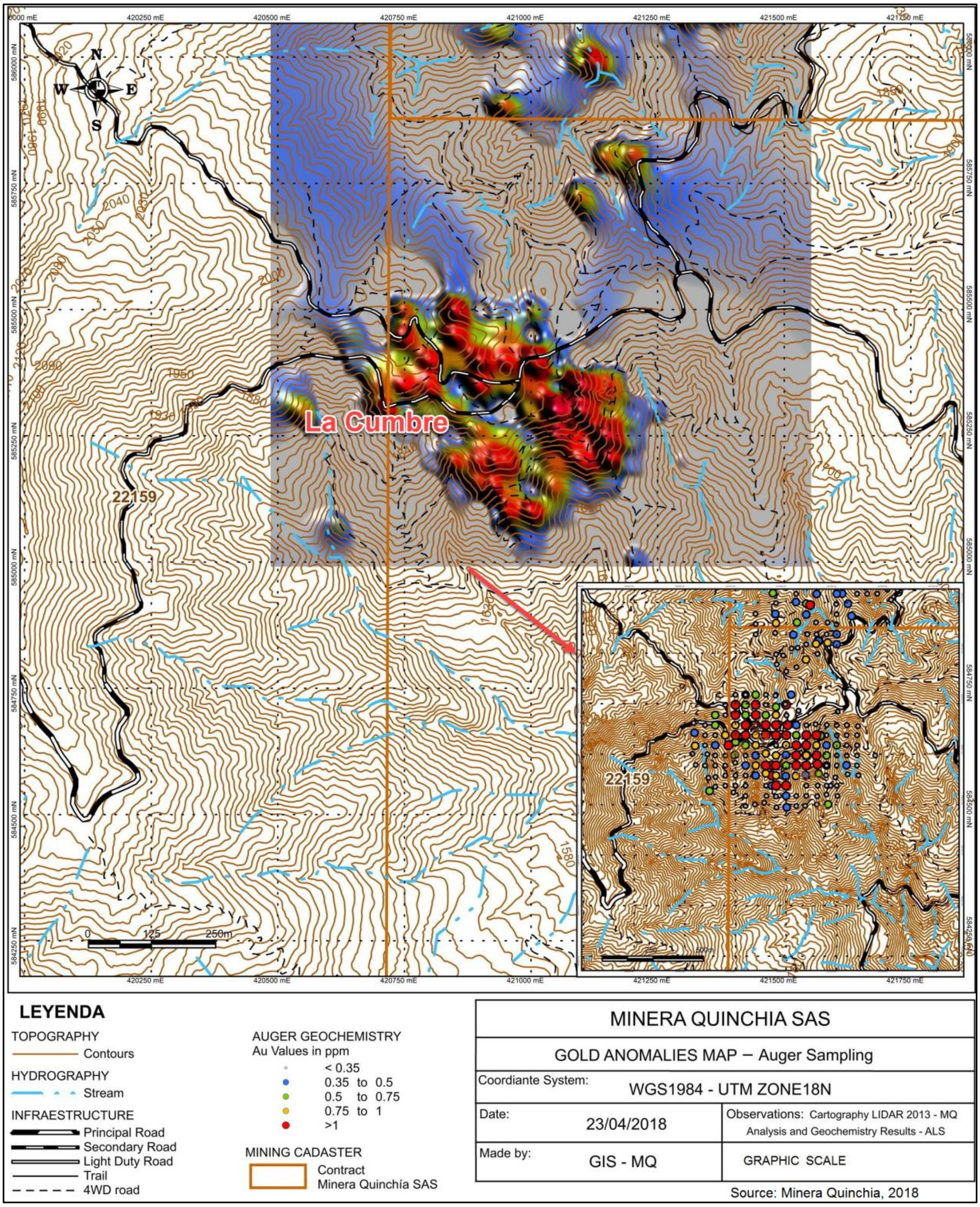
During 2014, a rock sampling and auger soil sampling campaign was developed to evaluate and characterize the oxidized saprolite. During these exploration campaigns, 17 rock samples (maximum analytical result 1.465 g/t Au with a mean of 0.552 g/t Au) and 28 soil samples were taken (maximum analytical result 2.30 g/t Au with a mean of 0.657 g/t Au). Additionally, geological mapping of the exposed oxide zone was undertaken to characterize the zone at the La Cumbre deposit.

Figure 9-2:
Adits (Tunnels) Location – Quinchia Project



Subsequently, in 2015, a systematic geochemical auger sampling was carried out on a 50m x 50m grid, 205 samples were collected for gold and multi-element analysis (maximum gold content of 2.64 g/t Au with a mean of 0.522 g/t Au), reaching depths of up to 5m. In addition, 13 rock samples (maximum gold content of 1.255 g/t Au with a mean of 0.520 g/t Au) and 13 tray concentrates were taken (Figure 9-3).

Figure 9-3:
Auger Sampling Map at the La Cumbre Deposit with Anomalies



In 2017, 6 rock samples were taken for gold and multielement geochemical analysis (maximum result 1.575 g/t Au, mean of results 0.843 g/t Au). All rock and soil samples were analyzed at ALS Chemex Laboratories (ALS Chemex Peru).

9.1 EXPLORATION POTENTIAL

Epithermal mineralization occurs within and adjacent to the La Cumbre deposit, as well as structurally controlled mineralization throughout the El Centro and Dos Quebradas deposit areas. Higher grade intersections including two meters grading 43.4 Au g/t in hole LC003, 31.3 m grading 2.85 Au g/t in hole QAP034, 10 m grading 1.4 Au g/t in hole SB007, and 53.6 m grading 1.31 Au g/t in hole LC001, occur in the porphyry mineralization.

In addition, there are high-grade results from samples collected from rehabilitated tunnel exposures. Highlights of samples from mineralized fault-vein structures in the La Cumbre Mines tunnels are summarized in Table 9-2, Table 9-3 and Table 9-4.

Free gold has been observed in samples from tunnels at both La Cumbre Mines and Mandeval Mines. To date, a total of 710 linear channel, 452 back samples, and 207 select vein samples representing approximately 280 m of tunnels of the Mandeval Mines and 445 m of tunnels in the La Cumbre Mines have been collected from eight tunnels at Mandeval and 12 tunnels at La Cumbre Mines. Tunnel lengths vary from five meters to 150 m.

Table 9-2:
La Cumbre Underground Mines Vein Grab Sample Highlights

Location	Length (m)	Au (g/t)	Ag (g/t)
Tunnel TULC6	Grab	16.65	5.3
Tunnel TULC6	Grab	7.26	4.5
Tunnel TULCPP	Grab	7.76	51.5
Tunnel TULCPP	Grab	6.92	34.9
Tunnel TULC2	Grab	5.45	69.8
Tunnel TULCPP	Grab	5.40	47.8
Tunnel TULCPP	Grab	4.12	25.0
Tunnel TULCPP	Grab	2.69	3.9
Tunnel TULCPP	Grab	2.97	2.9

Table 9-3:
La Cumbre Underground Mines Two-Meter Vein Sample Highlights

Location	Length sampled (m)	Au g/t	g/t Ag
Tunnel TULC8	2	13.2	11.7
Tunnel TULC3	2	3.53	8.3
Tunnel TULC8	2	3.13	20.6

Table 9-4:
La Cumbre Underground Mines Wallrocks Sample Highlights

Location	Length sampled (m)	Au g/t	g/t Ag
Tunnel TULC2	4	3.62	4.4
Tunnel TULCPP	8	1.23	1.3
Tunnel TULCPP	4	1.19	1.2
Tunnel TULCPP	6	1.10	2.3
Tunnel TULC2	10	0.69	2.9
Tunnel TULC2	81	1.19	15.0

10.0 DRILLING

In 2016 - 2017, Batero completed 40 diamond drill holes totaling 4,574.16 m in the area known as “La Cumbre”. This infill drilling campaign aimed to increase the resources in the oxide and transitional zones of La Cumbre Deposit. This technical report includes the results of this drilling campaign (see Table 10-1 for drill hole collar information and Figure 10-1 for the location of the drill holes).

The 2016-2017 drilling campaign, started on January 06, 2016 and concluded on March 16, 2017. All the drill holes were vertical and were drilled to various depths (see Table 10-1 for more details).

The collar coordinates, for all 2016-2017 drillholes, were surveyed using a Topcon GTS 226 Total Station by “Corporación Lonja Nacional de Propiedad Raíz y Consultorías Catastrales de Pereira - Risaralda”. Only the hole collars were surveyed after the rig had moved. Since all boreholes were vertical, downhole measurements were considered unnecessary.

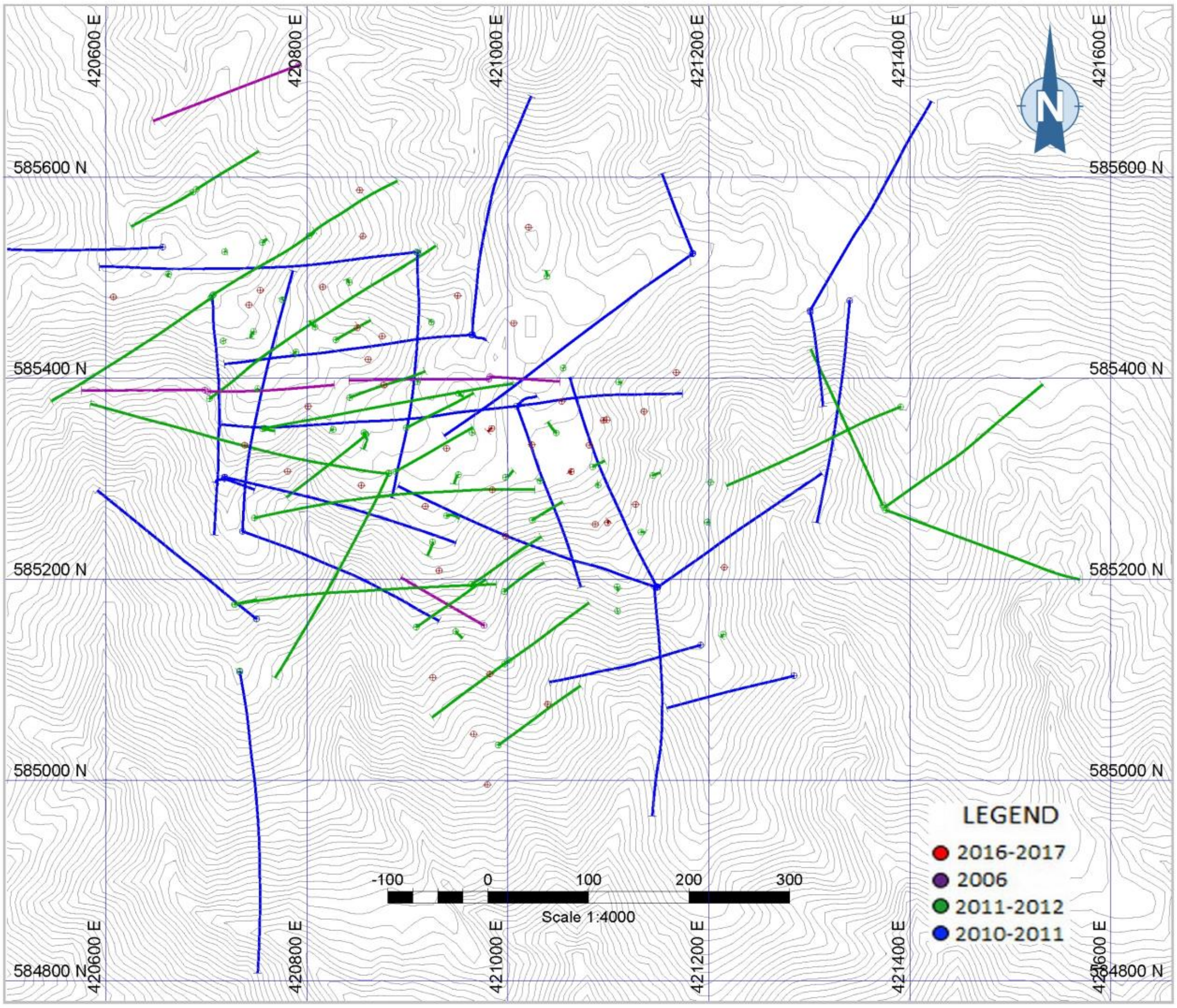
Table 10-1:
2016-2017 Drilling Program – La Cumbre Deposit

Hole-ID	East UTM*	North UTM*	Elevation	Azimuth	Dip	Length (m)
DDH-ZO-001	420780.75	585307.19	1865.69	360	-90	73.20
DDH-ZO-002	420854.15	585293.29	1870.99	360	-90	100.00
DDH-ZO-003	420801.08	585371.84	1902.15	360	-90	91.00
DDH-ZO-004	420874.88	585441.72	1907.49	360	-90	80.25
DDH-ZO-005	420876.73	585393.28	1892.38	360	-90	101.55
DDH-ZO-006	420949.91	585482.27	1868.17	360	-90	28.20
DDH-ZO-007	420855.73	585541.31	1904.28	360	-90	72.20
DDH-ZO-008	421005.70	585454.59	1868.90	360	-90	100.00
DDH-ZO-009	420738.06	585333.30	1897.82	360	-90	100.00
DDH-ZO-010	420607.54	585480.86	1964.99	360	-90	50.15
DDH-ZO-011	421098.46	585358.53	1846.98	360	-90	111.40
DDH-ZO-012	421023.74	585333.82	1859.12	360	-90	97.30
DDH-ZO-013	420852.60	585587.14	1893.82	360	-90	88.10
DDH-ZO-014	420815.71	585490.86	1901.46	360	-90	100.00
DDH-ZO-015	420753.71	585487.55	1941.90	360	-90	87.80
DDH-ZO-016	421020.55	585550.21	1888.27	360	-90	50.00
DDH-ZO-017	420917.81	585272.30	1869.15	360	-90	100.00
DDH-ZO-018	420931.55	585208.47	1839.25	360	-90	100.00
DDH-ZO-019	421086.88	585254.36	1822.76	360	-90	100.00
DDH-ZO-020	420925.40	585102.05	1766.61	360	-90	100.00
DDH-ZO-021	420982.08	585105.34	1766.45	360	-90	100.26
DDH-ZO-022	420965.97	585045.71	1725.99	360	-90	96.60
DDH-ZO-023	420979.68	584995.35	1701.30	360	-90	100.00
DDH-ZO-024	421039.54	585075.38	1738.05	360	-90	44.80
DDH-ZO-025	420997.92	585242.41	1825.06	360	-90	80.00
DDH-ZO-026	421053.53	585376.91	1863.72	360	-90	95.70
DDH-ZO-027	421080.89	585333.26	1854.88	360	-90	110.00
DDH-ZO-028	421135.39	585366.80	1813.15	360	-90	100.00
DDH-ZO-029	421167.46	585405.62	1810.77	360	-90	50.00
DDH-ZO-030	421127.01	585274.34	1809.81	360	-90	106.80
DDH-ZO-031	421215.24	585211.64	1733.60	360	-90	100.30
DDH-ZO-032	420984.45	585288.82	1861.92	360	-90	94.40

Hole-ID	East UTM*	North UTM*	Elevation	Azimuth	Dip	Length (m)
DDH-ZO-033	420938.98	585330.03	1876.96	360	-90	100.00
DDH-ZO-034	420860.91	585418.59	1911.91	360	-90	100.05
DDH-ZO-035	420742.38	585473.01	1940.98	360	-90	100.00
DDH-ZO-047	421095.00	585358.00	1847.69	360	-90	264.10
DDH-ZO-048	421063.00	585307.00	1855.33	360	-90	300.00
DDH-ZO-049	420984.00	585350.00	1870.75	360	-90	300.00
DDH-ZO-050	421099.00	585256.00	1818.74	360	-90	300.00
DDH-ZO-051	420850.00	585450.00	1911.71	360	-90	300.00

*UTM System: WGS 84 Zone 18N

Figure 10-1:
La Cumbre Deposit - Location Map of Drillholes



A Minera Quinchia technician was present at the drill platform at all times. The technician cleaned the core, carefully reconstructed the core and calculated the core recovery (CR), core loss (CL) and rock quality designation (RQD) at the drill site. In 2016-2017, the technician also completed first pass geotechnical logging. A Minera Quinchia geologist is present at the drill for the beginning and termination of each hole. Core recovery is generally very good in the fresh rock averaging over 95%, and it averages over 80% in the saprolite. Triple tube core barrels were used for all drill holes to improve core recovery.

The core is brought to a central core handling facility at La Cumbre on a daily basis by Minera Quinchia personnel. At the La Cumbre core handling facility, the core is photographed with a digital camera and logged by Minera Quinchia geologists

The collars of DDH-ZO-026, DDH-ZO-027 and DDH-ZO-032 holes were located by a LINAMEC geologist using a hand-held GPS during the site visit, as part of the verification process (see Figure 10-2, Figure 10-3 and Figure 10-4). Differences of the collar coordinates between the hand-held GPS and the Total Station are considered acceptable.

Figure 10-2:
Collar Monument of DDH-ZO-026 at La Cumbre Area

**Figure 10-3:
Collar Monument of DDH-ZO-027 at La Cumbre Area**

**Figure 10-4:
Collar Monument of DDH-ZO-032 at La Cumbre Area**

Table 10-2, shows a selection of intersections through the main resource zones to illustrate typical grades and widths of the La Cumbre Deposit.

Table 10-2:
La Cumbre Significant Drill Intervals – 2016-2017 Drilling Campaign

Hole-ID	From (m)	To (m)	Intersection Length (m)	Au g/t	Ag g/t	Zone
DDH-ZO-004	6.00	8.00	2.00	2.480	0.88	ZOX
DDH-ZO-011	4.50	22.00	17.50	2.079	0.73	ZOX
DDH-ZO-011	48.00	60.00	12.00	1.572	2.67	ZOX
DDH-ZO-011	88.00	98.00	10.00	2.308	4.40	ZTR
DDH-ZO-012	22.00	24.00	2.00	2.490	5.64	ZOX
DDH-ZO-015	24.00	32.00	8.00	2.182	1.56	ZOX
DDH-ZO-026	50.00	56.00	6.00	1.945	1.81	ZTR
DDH-ZO-027	14.00	24.00	10.00	2.450	1.85	ZOX
DDH-ZO-028	34.00	44.00	10.00	2.066	1.22	ZOX
DDH-ZO-030	24.00	26.00	2.00	2.970	6.45	ZOX
DDH-ZO-030	46.00	60.00	14.00	2.417	2.59	ZOX
DDH-ZO-030	72.00	74.00	2.00	4.980	5.07	ZOX
DDH-ZO-047	8.00	16.00	8.00	2.039	7.03	ZOX
DDH-ZO-049	80.00	84.00	4.00	2.185	3.50	ZTR
DDH-ZO-051	26.00	28.00	2.00	2.680	2.68	ZOX
DDH-ZO-051	38.00	48.00	10.00	2.259	2.26	ZOX
LC025	216.00	218.00	2.00	11.000	0.41	ZSP
QAP035	484.40	485.07	0.67	8.990	10.25	ZSP
QAP008	461.10	463.10	2.00	7.290	16.35	ZSP
QAP030	47.90	48.80	0.90	5.590	75.70	ZSP
LC007	282.70	284.00	1.30	5.070	2.90	ZSP
DDH-ZO-033	72.00	74.00	2.00	4.770	1.82	ZSP
LC063	43.10	45.42	2.32	4.750	1.81	ZSP
LC056	75.50	76.50	1.00	4.470	35.60	ZSP
QAP006	141.70	143.70	2.00	4.390	72.60	ZSP
QAP011	415.00	417.00	2.00	4.270	4.45	ZSP
LC001	326.69	328.00	1.31	4.170	1.93	ZSP
QAP041	409.00	411.00	2.00	4.120	1.96	ZSP
LC001	305.00	307.00	2.00	4.040	3.69	ZSP
QAP046	408.00	410.00	2.00	4.000	2.49	ZSP
DDH-ZO-048	100.00	104.00	4.00	3.790	3.68	ZSP
LC014	693.00	693.60	0.60	3.690	4.32	ZSP
LC011	341.00	341.63	0.63	3.650	8.32	ZSP
LC006	279.95	281.28	1.33	3.540	1.06	ZSP
QAP022	466.00	467.00	1.00	3.410	13.25	ZSP
LC003	108.85	110.00	1.15	3.290	1.88	ZSP
LC014	495.00	495.60	0.60	3.290	6.38	ZSP
LC007	165.77	166.51	0.74	3.270	3.27	ZSP
LC001	75.70	76.70	1.00	3.190	4.37	ZSP
LC004	145.00	146.11	1.11	3.090	4.65	ZSP

Abbreviations: ZOX = oxide zone; ZTR = transition zone; ZSP = primary zone. Lengths are down hole intersections and are not considered true widths as insufficient information is available to determine true width.

During the period March 25, 2006 to March 16, 2017, Batero completed 143 diamond drill holes totaling 41,338.50 m within the La Cumbre area. The purpose of the drilling was primarily to delineate resources by testing for extensions of the known mineralization at the La Cumbre target. All exploration drill programs (several phases) performed since 2006 to 2017 are listed below in Table 10-3.

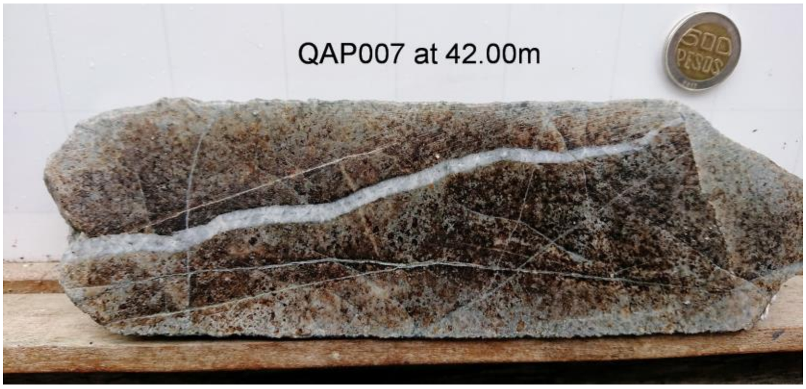
Table 10-3:
La Cumbre – Drilling Campaigns

Period	Drill holes	Total Length (m)
2006	5	1,399.55
2010-2011	69	31,112.09
2012	29	4,252.70
2016-2017	40	4,574.16
Total		41,338.50

Core logging, systematic sampling of core cut by a diamond saw, storage of cores and bagging of samples for sending to international labs was carried out by Minera Quinchia geologists following NI 43-101 protocols (QA/QC, etc.) as described in Chapter 12. Figure 10-5 shows core warehouse and logging facility and Figure 10-6 shows "B" type quartz vein surrounded by potassic alteration as logged by Minera Quinchia geologists. Finally, several systematic geologically interpreted cross-sections were developed for each drill hole in order to outline a model of the mineralized porphyry systems with mineral zones (potential grade-shell).

Figure 10-5:
Core Warehouse and Logging Facility at La Cumbre Site

Figure 10-6:
Drillhole QAP007, Quartz "B" Type Veinlet Hosted in a Porphyry Diorite with Strong Potassic Alteration (fine biotite)



11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 DRILL CORE SAMPLE PREPARATION

A Minera Quinchia technician is present at the drill rig at all times when the drill is operational. The technician monitors the drilling, verifies that the metreage markers inserted by the drillers are correct, washes and reconstructs the core, calculates the core recovery (CR) and logs the rock quality designation (RQD). The technician also ensures that the core boxes are properly secured for transportation to the core handling facility at La Cumbre.

Drill core is placed sequentially in wooden core boxes at the drill rig site. The core boxes are carried by Minera Quinchia personnel on a daily basis and transported by truck to the La Cumbre core logging and sampling facility. The core facility is located on a fenced property with 24-hour security present. At the La Cumbre core facility, the core is photographed with a digital camera and is quick logged by Minera Quinchia geologists.

The drill cores from past campaigns are stored in a core shed located near the town of Quinchia at a site known as Chorroseco. This facility has metallic fences and is considered to have adequate security (see Figure 11-1).

Figure 11-1:
Core Shed at the Chorroseco Site with Metal Fences and Security for Old Core Storage

The following summary describes the sampling methodologies employed during the La Cumbre 2016 – 2017 campaign.

- Samples are blocked on two-meter intervals and shorter intervals may be selected based on geological contacts, structures, changes in texture, alteration and mineralization. The length of samples ranged from 0.60 – 5.9 m, with 87.8% measuring 2.0 m in length (see Figure 14-8 for sample length histogram).
- A sample number, defined by the geologist logging the drillhole, was assigned to the designated sample interval, and the range (from and to) of the sample along with corresponding remarks (logging as described above) was captured and entered in the drill log. The range and sample number were also marked on the core box.
- Once the logging and sampling intervals are completed by the geologist, the drill core is delivered to the geo-technician and a photographic record of all core boxes belonging to each drill hole is captured before the cutting/sampling of the drill core. The photographic record is performed again for all split/sampled drill core.
- Drill core is cut in half (symmetrically) by diamond saw and after cutting the core boxes go to the sampling area where the sample (half of the split core) is packed in a clear

heavy-duty plastic sample bag, weighed and coded for delivery. The other half of the cut core is retained in its core box and stored at the on-site company warehouse.

- Several samples are packed in larger rice bags (average of 7 samples per bag) and a delivery report is made and submitted to the laboratory along with the samples maintaining a chain of custody until delivery. Minera Quinchia personnel typically ship samples and drive samples to delivery points once per week or when the drill hole is completed.

This more disciplined approach to sampling will result in:

- A better understanding of the mineralizing controls and distribution of gold.
- Better enable the interpretation of mineralized zones and therefore provide a better interpretation of the deposit(s) as a whole.
- Possible enhancement of gold grade over better defined mineralized zones and lenses.
- Engineering design and modelling will have more precise grade data and better-defined mineralized zones to incorporate into modelling and mine design.

11.2 BULK DENSITY DETERMINATIONS

To verify the results of density determinations made by Minera Quinchia, LINAMEC and Minera Quinchia selected 27 samples. The samples were analyzed by ALS Lima. The results indicate the analytical precision and accuracy of the Minera Quinchia density measurements are not comparable with the results of ALS laboratory. Results are shown as X-Y dispersion graphs using the Reduction-to-Major Axis (RMA) method (Sinclair, 1999), which offers a non-biased adjustment on both series of results. This mathematical procedure treats both series as independents.

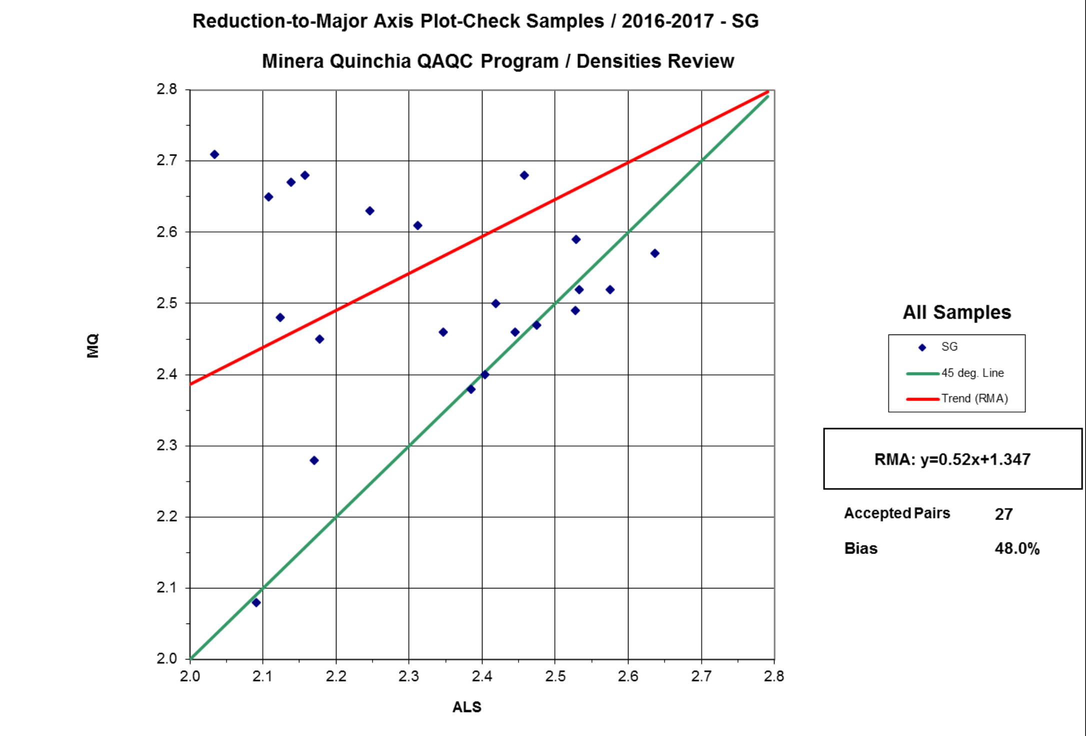
LINAMEC's conclusion, based on the results obtained in the verification process of density determinations is that density measures obtained by Minera Quinchia are not comparable with the results obtained by ALS Laboratory; the bias of Minera Quinchia samples is greater than 10% (bias = 48%). RMA statistics for the Minera Quinchia is presented in Table 11-1 and plotted in Figure 11-2.

Table 11-1:
Accuracy of MQ Relative to ALS for Density Determinations

Minera Quinchia QAQC Program - RMA Parameters - All Samples								
Element	R^2	N (total)	Pairs	m	Error (m)	b	Error (b)	Bias
SG	0.0055	27	27	0.520	0.100	1.347	0.043	48.0%

To establish bulk densities for the La Cumbre Project updated mineral resource estimate, 59 samples were collected for bulk density determinations to be used in the tonnage calculation of both oxide and transitional zones. The SG value of 2.520 was used to calculate the tonnage in both mineral zones.

Figure 11-2:
RMA Plot Check Samples for Bulk Density in La Cumbre (MQ vs. ALS)



The samples were submitted to ALS Laboratory in Lima, Peru for Specific Gravity (SG), Bulk Density (BD) determinations, SG is determined by weighing a sample in air and in water, and it is reported as a ratio between the density of the sample and the density of water. The BD is the density of a material in weight per unit volume, and it is determined by the weight of the sample and the volume of water the sample displaces. Calculations for BD were corrected for water temperature and the density of the wax coating. The SG of 20 samples was determined with this ALS method (OA-GRA08a).

For the determination of the SG for the saprolitic samples, the pycnometer method was used. A total of 39 samples were submitted to ALS Laboratory in Lima, Peru. SG was determined by weighing 3.0 g of soil sample into an empty pycnometer, and then the pycnometer is filled with a solvent (either methanol or acetone) and weighed again (ALS OA-GRA08b method). From the weight of the sample and the weight of the solvent displaced by the sample, the specific gravity is calculated according to the following equation:

$$SG = \frac{\text{Weight of sample (g)}}{\text{Weight of solvent displaced (g)}} \text{ x } SG\ of\ Solvent$$

The current update of the mineral resource estimate for the Quinchía project now includes the primary sulfide zone, to report the estimated tonnes from this zone, Minera Quinchía and LINAMEC have selected 56 diamond drill core halves samples, which were submitted, with their respective chain of custody documents, to the Bureau Veritas (BV) laboratory located in the city of Medellín, Colombia. The water displacement method (Archimedes method) of wax-coated samples (procedure code SPG03) was used to determine the bulk density of these samples.

The average value obtained for the bulk density for the primary zone was 2.644 g/cm^3, after discarding the maximum and minimum values of the results obtained, see Table 11-2.

Table 11-2:
Bulk Density Values by Method Used for Mineral Resource Estimate

Method	Mineral Zone	No. Samples	SG g/cm^3
Pycnometer	OVB	39	2.590
Paraffin	OXD & ZTR	20	2.390
Waxed Core	ZPR	56	2.644
Total/weighted average		**115**	**2.582**

LINAMEC encourages an increase in the number of density determination tests by type of mineralization and even a differentiation by lithological type where possible.

11.3 QUALITY ASSURANCE AND QUALITY CONTROL

A QA/QC program was performed for the 2016-2017 drilling campaign. The details of the Batero QA/QC program in general will be discussed later in Chapter 12.3 of this report.

All of the La Cumbre samples in the mineral resource database have been submitted with standard reference materials to control assay accuracy and, depending on the program, have included twin samples, coarse crush duplicates and pulp duplicates to control sampling, sub-sampling and analytical precision.

An independent check-assaying program has also been used to demonstrate the reproducibility of the assays carried out in the primary laboratory and to help establish assaying accuracy.

11.4 DATABASES

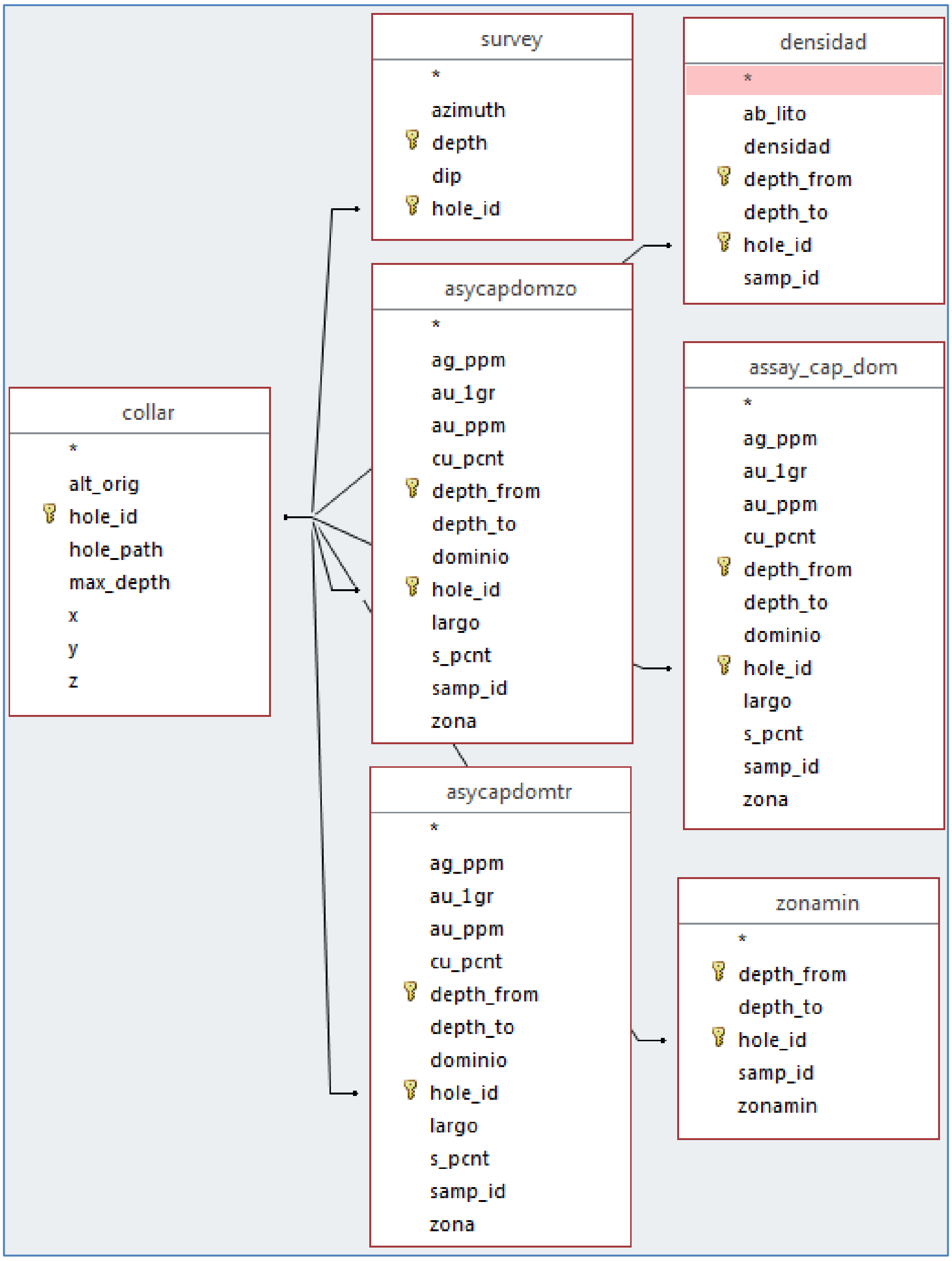
La Cumbre drilling data is currently stored in an Access database. The drill data (collar, assay, survey, and logging) is manually uploaded to the database and the data verification of data input is conducted visually. The assay certificates are stored in their original formats (*.CSV, *.XLS, *.PDF) and geological logs are recorded on paper by hand, and manually entered in the Access form (see Figure 11-3 for the structure of the Access Database Tables).

The database, for the present mineral resource update, consists of 143 DDH's, with 41,338.5 m drilled and 22,974 assay records. The oxide, transitional and primary zones total 18,983 assays records, with a total of 14,119 composites in all mineral zones. See Table 11-3 for detailed information.

Table 11-3:
Databases Used in the La Cumbre Updated MRE 2021

Zone	Meters	Assays	Composites
Overburden	232.00	100	130
Oxide	4,800.12	2,400	2,400
Transitional	2,734.32	1,491	1,290
Primary	33,572.06	18,983	10,299
Total	**41,338.50**	**22,974**	**14,119**

Figure 11-3:
La Cumbre – MS-Access Database Tables



11.5 SAMPLE SECURITY

Each day the drill core samples are transported from the drilling platform to the core shed in wooden boxes properly marked with the drill hole and box number. Each box is carefully sealed with lids and nails and placed in a backpack for transportation on foot until the 4x4 truck can pick up the core for transportation to the core shed which is guarded by Minera Quinchia personnel. The samples are accompanied by the respective custody documents, duly completed and signed. Once the core trays are laid out on tables, the nails and lids are removed in preparation for logging and core photographs.

The core samples are measured (marked-up), logged, and labelled following the internal procedures that have been endorsed by outside consultants. These samples are then cut and packed into size 8" double plastic bags, which were previously marked with stickers showing a sample number assigned by the geologist. Before the batches are sent to the laboratory, geologists and technicians prepare a batch checklist to track the movement of the material. This checklist gives the number of samples, batches, and quality control (QC) samples, with the type, and number. At this stage, the checklist must be signed by the geologists, security guard, and the driver of the vehicle. When this process is completed, the batches are sent to the ALS preparation laboratory in Medellin. Here, the warehouse foreman receives the batches from the driver, and must check against the batch checklist, and sign to verify the contents of the batches. The foreman is the individual responsible to hand deliver each of the samples to the ALS Medellin laboratory.

Upon delivery, the ALS shift supervisor verifies that all samples as specified in the laboratory request sheet are the same as delivered, then signs for their receipt. These samples are logged into the internal ALS system called "Webtrieve" (used globally by ALS clients) and assigned a work order number known as the internal way lot. Every time a sample goes through this process, it is followed by the system indicating its stage.

The samples go through the initial preparation process (crushing, splitting, and pulverizing) at ALS Colombia and the pulp is sent to ALS Peru (as defined below) in Lima. This pulp is packed in a paper bag and coated with plastic, then sent in heavy gauge cardboard boxes with ALS tape and coded security straps, which identifies those boxes if any that have been opened during transit between ALS Medellin and ALS Peru by customs. The leftover pulp and coarse rejects are sent to the Batero warehouse in Itagui within 45 days of the date of issue of the certificate.

LINAMEC has reviewed the entire sample chain of custody at La Cumbre, from the drilling of the samples to the receiving of final analytical results. LINAMEC is of the opinion that the in-house Batero custody control systems in place are of industry standard and are adequate and appropriate for use in mineral resource and reserve estimation.

11.6 ANALYTICAL LABORATORIES

The samples of the 2016-2017 drilling campaign in the La Cumbre Area, were submitted to the ALS Minerals Medellin Colombia, for mechanical preparation and then shipped for analysis to the ALS certified assay laboratory in El Callao, Peru (ALS Peru).

Batero receives the analytical data from ALS Chemex laboratory electronically as CSV or XLS files and the final certificates as PDF files. The migration of the assay data to Century is completely automated.

LINAMEC checked 100% of the gold, silver, copper, and sulfur assays in the Batero drill hole database against 2016-2017 laboratory certificates and no issues were found.

11.7 SAMPLE PREPARATION & ANALYSIS

11.7.1 Sample Preparation

Sample preparation of the 2016-2017 samples was conducted at the ALS Minerals preparation laboratory in Colombia ("ALS Colombia") located at Bodegas San Bartolome Bodega 3, Carrera 48B No 99 Sur-59, La Estrella, Medellin. ALS Colombia is independent from Batero.

Sample preparation is the most critical step in the entire laboratory operation. The purpose of preparation is to produce a homogeneous sub-sample that is fully representative of the material submitted to the laboratory. The sample is logged in the tracking system, weighed, dried and finely crushed to better than 70% passing a 2 mm (Tyler 9 mesh, US Std. No. 10) screen. A split of up to 1,000 g is taken and pulverized to better than 85% passing a 75-micron (Tyler 200 mesh) screen. This method is appropriate for rock chip or drill samples.

11.7.2 Sample Analysis

After preparation at ALS Colombia, the samples are then shipped for analysis to the ALS certified assay laboratory in Lima, Peru (ALS Peru).

At ALS Peru, gold analyses were performed utilizing the Au-AA24 (50g sample) method with Atomic Absorption completion. If the gold content exceeded 10 g/t Au the sample was then subjected to Au-GRA22 method analyzing a 50g split of sample by fire assay and completion with a gravimetric finish.

In addition, assaying for 48 elements (ME-MS61) ICP-MS analysis was performed on each sample.

11.7.3 Laboratory Independence and Certification

ALS Peru is independent from Batero and has the following accreditation: ISO 9001:2008 certification by IQNET, The International Certification Network, for chemical analysis of geological samples and products of its industrial processing chemical analysis of environmental samples from the mining and energy industries.

ISO/IEC 17025:2005 Accreditation by the Standards Council of Canada as a Testing Laboratory.

12.0 DATA VERIFICATION

The database audit covers only the data collected by Batero Gold (Batero) during the 2016-2017 drilling campaign performed in the La Cumbre area for the oxide and transitional zones. This constitutes the new data used to update the estimates of the mineral resource for the La Cumbre deposit.

LINAMEC has audited the data coming from:

- Collar coordinates.
- Downhole survey (dip and strike).
- Surface geological mapping.
- Geological logs, and
- Assay reports.

In the audit process, geological logs were scanned and compared with the data in Access files. LINAMEC was provided with assay reports from ALS laboratory, collar survey reports, downhole survey reports and field drilling reports for its audit.

Two projects were created one in Leapfrog and the other in GEMS® for modelling and resource estimation. The database for drill holes consists of 143 DDH's, 41,338.50 m and 4,075 assay records. See Table 12-1 for distribution of data in the Oxide Zone and Transitional Zone.

The databases were exported for audit purposes directly from the projects created in GEMS®, that were used for the updated resource estimation and reporting of the La Cumbre Project in November 2021.

Table 12-1:
Audited Data Base – Exported from GEMS

Mineral Zone	Meters	Assays	Composites
Oxides	4,800.12	2,400	2,400
Transitional	2,734.32	1,491	1,290
Primary	33,572.06	18,983	10,299
Total	**7,5306.50**	**4,075**	**3,690**

12.1 QUALITY ASSURANCE/QUALITY CONTROL (QA/QC)

Quality Assurance (QA) concerns the establishment of measurement systems and procedures to provide adequate confidence that quality is adhered to. Quality Control (QC) is one aspect of QA and refers to the use of control checks of the measurements to ensure the systems are working as planned. The QC terms commonly used to discuss geochemical data are:

Bias: the amount by which the analysis varies from the correct result.
Precision: the ability to consistently reproduce a measurement in similar conditions.
Accuracy: the closeness of those measurements to the “true” or accepted value.
Contamination: the transference of material from one sample to another.

LINAMEC has carried out an evaluation of the QA/QC samples for the 2016-2017 drilling campaign applying our own templates. The present evaluation is for gold and silver.

12.2 STANDARD SAMPLES REVIEW

Standards (or certified reference materials, CRM) are samples prepared by certified labs under special conditions, used to estimate the assay accuracy of the control batch. Two CRM's supplied by CDN Resource Laboratories Ltd. (CDN) were used during the 2016-2017 drilling campaign in the area of La Cumbre Deposit. The accepted Best Values (BV) or certified values and their corresponding Confidence Intervals (CI) are presented in Table 12-2.

Table 12-2:
List of Certified Reference Materials

Code	Assay No.	ALS	SGS	BV Au (g/t)	BV Ag (g/t)	CI
CDN-CM-12	45	17	28	0.686	3.55	0.006
CDN-CM-17	37	11	26	1.370	14.35	0.010

For evaluating the standard samples, control charts were constructed for each Au and Ag standard. The values reported for the inserted standard samples were plotted in a time (or pseudo-time) sequence. Lines corresponding to *BV, 1.05*BV+CI, 0.95*BV-CI* and *AV±2*SD* were also plotted (*BV*, *CI*: Best Value and Confidence Interval at the 95% confidence level, respectively, calculated as a result of round-robin tests; *AV*, *SD*: average (mean) value and standard deviation, respectively, calculated from the actual assay values of the inserted standards).

In principle, the standard value had to lie within the *AV±2*SD* boundaries to be accepted. Otherwise, the value was qualified as an outlier. The analytical bias was calculated as:

$$\text{Bias (\%)} = (AV_{eo} / BV) - 1$$

where, AV_{eo} represents the average recalculated after the exclusion of the outliers. The bias values are assessed according to the following ranges: good, between -5% and +5%; reasonable, with care, from -5% to -10% or from +5 to +10%; unacceptable, below -10% or above 10%.

During 2016-2017 drilling campaign, 128 standard analyses, representing 3.14% of the total samples included in submission batches, were submitted. The analyses comprised 65 samples of CDN-CM-12 (1.60%) and 65 samples of CDN-CM-17 (1.55%). Almost all gold assays were within the AV±2*SD range or very close to those limits, with exception of one isolated outlier for the CDN-CM-12 standard and three outliers for the CDN-CM-17 standard. Most of the individual bias values were acceptable. The overall biases were below ± 10% (-0.3% for CDN-CM-12 and -0.9% for CDN-CM-17), see Figure 12-1 and Figure 12-2 for Au Control Chart Standards.

The performance of these standards for silver are presented in Figure 12-3 and Figure 12-4. Three samples of CDN-CM-12 and four of CDN-CM-17 standard assays were above the AV±2*SD range. Five individual bias values were ± 10% for CDN-CM-12 standard and 13 individual bias values were upper ± 10% for CDN-CM-17 standard. The overall biases for silver were below ± 10% (4.9% for CDN-CM-12 and 4.6% for CDN-CM-17). It is also noted that the standard deviations for silver grades are very wide and do not provide a tight constraint on the precision of the analyses.

Figure 12-1:
Control Chart Standard for CDN-CM-12 – Au (g/t)

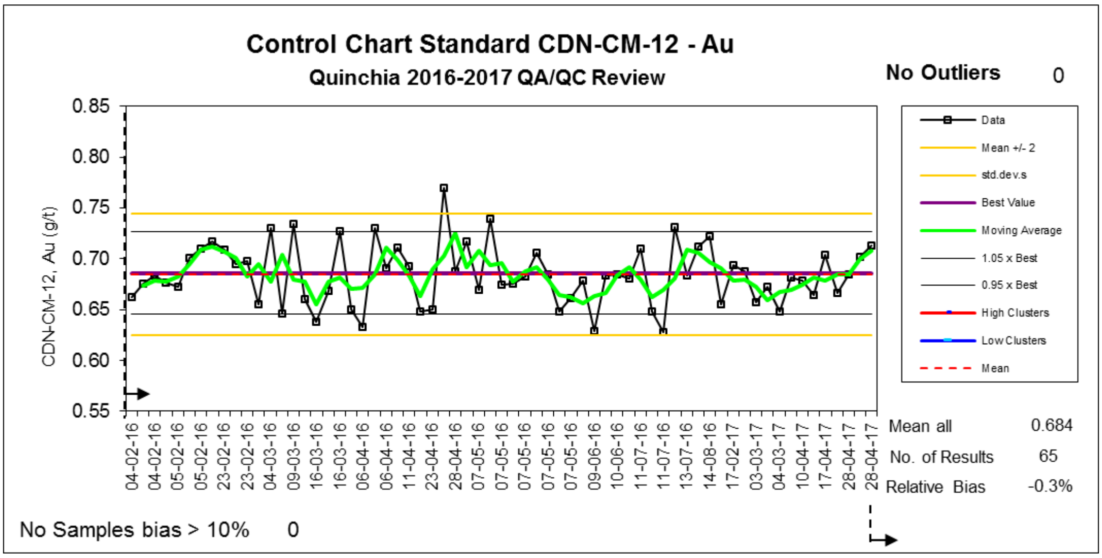


Figure 12-2:
Control Chart Standard for CDN-CM-17 – Au (g/t)

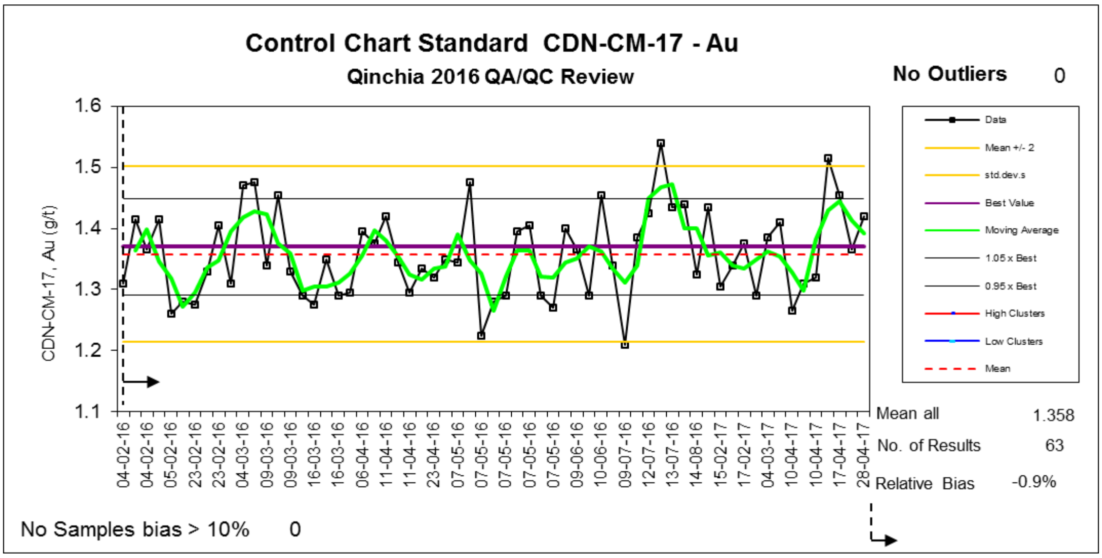


Figure 12-3:
Control Chart Standard for CDN-CM-17 – Ag (g/t)

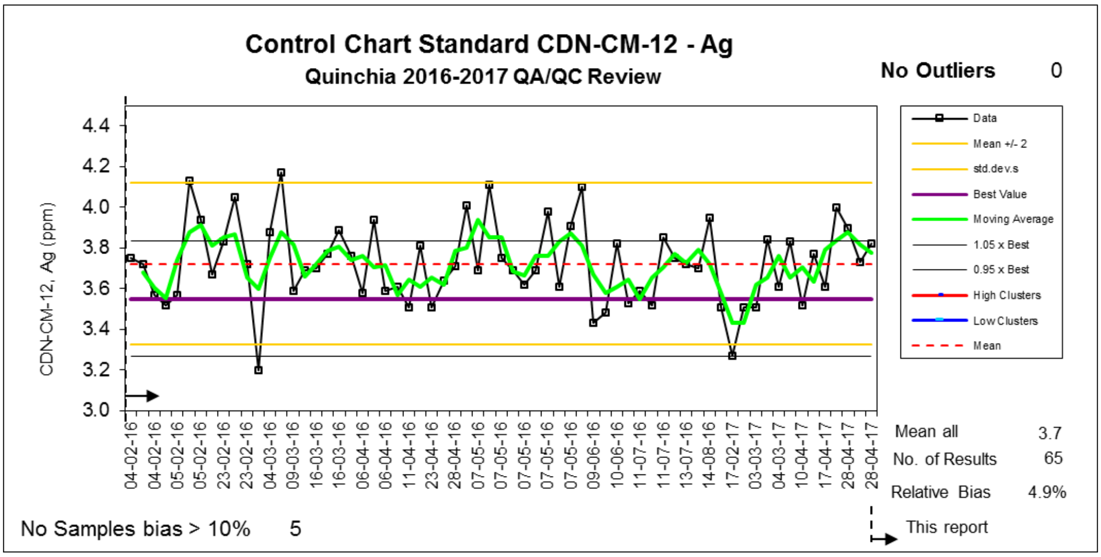
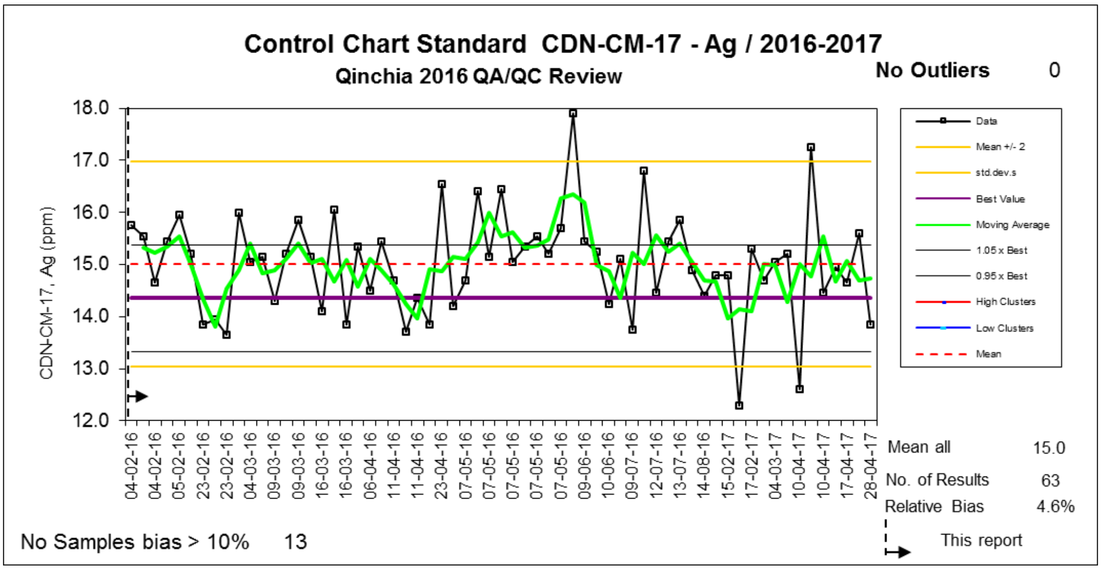


Figure 12-4:
Control Chart Standard for CDN-CM-17 – Ag (g/t)



12.3 CONCLUSION

The review of CRM's charts concludes that the relative bias for gold and silver is reasonable and that the accuracy and precision of assays from ALS laboratory is considered to be good. The analytical results are suitable for inclusion in mineral resource estimation.

12.4 INDEPENDENT DATA VERIFICATION

To verify the results of gold and silver grades for the La Cumbre oxide and transitional mineral zones, LINAMEC and Minera Quinchia selected 59 random pulps. The samples were analyzed by both SGS Medellin, Colombia and ALS Lima, Peru. The results indicate that the analytical precision and accuracy of the SGS laboratory are comparable to ALS.

To verify the results of gold and silver grades for the La Cumbre primary mineral zones, LINAMEC and Minera Quinchia selected 1,283 random pulps. The samples were analyzed by both SGS Medellin, Colombia and SGS Lima, Peru. The results indicate that the analytical precision and accuracy of the SGS Medellin laboratory are comparable to SGS Lima.

Results, for all mineral zones, are shown as X-Y dispersion graphs using the Reduction-to-Major Axis method (Sinclair, 1999), which offers a non-biased adjustment on both series of results. This mathematical procedure treats both series as independents.

In total, 59 samples were sent for external control to ALS Lima, which acted as secondary laboratory. The samples were assayed for Au, Ag and Cu. (representing 1.5% of the samples included in regular submission batches)

For processing the check assays for oxide and transitional zones, the few values below the detection limits were replaced by half the detection limits. The RMA plots indicate a good fit for Au, Ag and Cu between SGS Medellin and ALS Lima, reflected in the high values of the coefficient of determination R2 for both Au (0.997) and Ag (0.965) after the exclusion of 01 outliers (1.7%) for Au and 02 outliers (3.4%) for Ag, and good relative biases, 3.9% and 4.8%, respectively, see Figure 12-5. LINAMEC concludes that the accuracy of ALS Lima for Au and Ag, as compared to SGS Medellin, is good.

In total, 1,283 samples were sent for external control to SGS Medellin, which acted as secondary laboratory. The samples were assayed for Au, Ag and Cu. (representing 6.8% of the samples included in regular submission batches)

For processing the check assays for primary zone, the few values below the detection limits were replaced by half the detection limits. The RMA plots indicate an acceptable fit for Au, and good fit for Ag and Cu between SGS Medellin and SGS Lima, reflected in the high values of the coefficient of determination R2 for both Au (0.989) and Ag (0.971) after the exclusion of 36 outliers (2.8%) for Au and 19 outliers (1.5%) for Ag, and acceptable relative biases, -9.9% and good relative biases 2.1%, respectively; Figure 12-6. LINAMEC concludes that the accuracy of SGS Lima for Au and Ag, as compared to SGS Medellin, is acceptable for gold and good for silver.

LINAMEC's conclusion, based on the results obtained in the verification of gold grades from the La Cumbre drilling program, is that assays are acceptable to be used in the mineral resource update, with the bias of RMA for La Cumbre samples less than 10%. RMA statistics for the La Cumbre deposit is presented in Table 12-3 for oxide and transitional zones and Table 12-4 for primary zone. The RMA graphs are plotted in Figure 12-5 for the oxide and transitional zones and Figure 12-6 for primary zone.

Table 12-3:
Accuracy of ALS Relative to SGS for Gold on the Basis of Check Assays

Quinchia QAQC Drilling Program - RMA Parameters - All Samples								
Element	R2	N (total)	Pairs	m	Error (m)	b	Error (b)	Bias
Ag (g/t)	0.913	59	59	0.863	0.033	0.124	0.147	13.7%
Au (g/t)	0.997	59	59	0.958	0.007	0.015	0.010	4.2%
Quinchia QAQC Drilling Program - RMA Parameters - Outliers Excluded								
Element	R2	Accepted	Outliers	m	Error (m)	b	Error (b)	Bias
Ag (g/t)	0.965	57	2	0.952	0.023	-0.094	0.064	4.8%
Au (g/t)	0.997	58	1	0.961	0.007	0.013	0.010	3.9%

Table 12-4:
Accuracy of SGS Medellin Relative to SGS Lima for Gold on the Basis of Check Assays

Quinchia QAQC Drilling Program - RMA Parameters - All Samples								
Element	R2	N (total)	Pairs	m	Error (m)	b	Error (b)	Bias
Ag (g/t)	0.882	1283	1283	0.935	0.009	-0.099	0.029	6.5%
Au (g/t)	0.909	1283	1283	0.941	0.008	36.816	9.375	5.9%
Quinchia QAQC Drilling Program - RMA Parameters - Outliers Excluded								
Element	R2	Accepted	Outliers	m	Error (m)	b	Error (b)	Bias
Ag (g/t)	0.971	1264	19	0.979	0.005	-0.145	0.013	2.1%
Au (g/t)	0.989	1247	36	1.099	0.003	-1.585	1.857	-9.9%

12.5 COMMENTS ON CHAPTER 12

LINAMEC considers that the current drilling and sampling procedures undertaken by Minera Quinchia are adequate for use in the mineral resource estimation of the La Cumbre deposit. No major deficiencies or problems were found in the verification and audit procedure.

Figure 12-5:
RMA Plot Check Samples for Au in La Cumbre Deposit (Oxide and Transitional Zones)

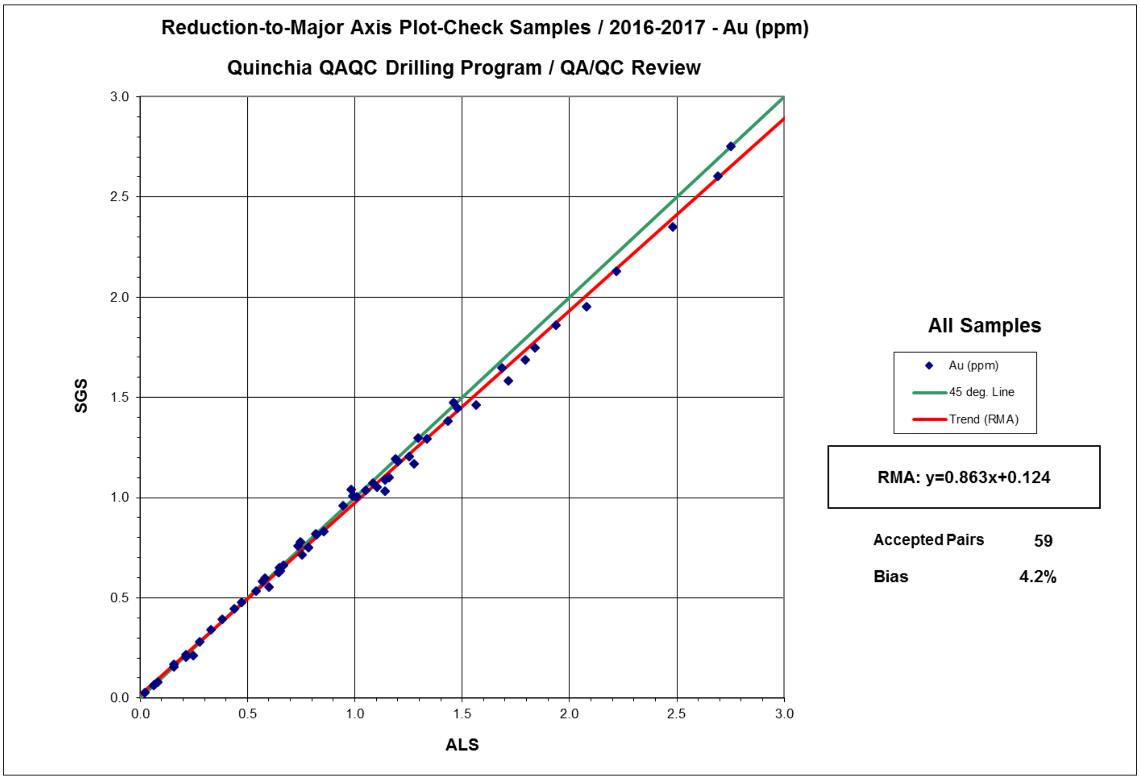
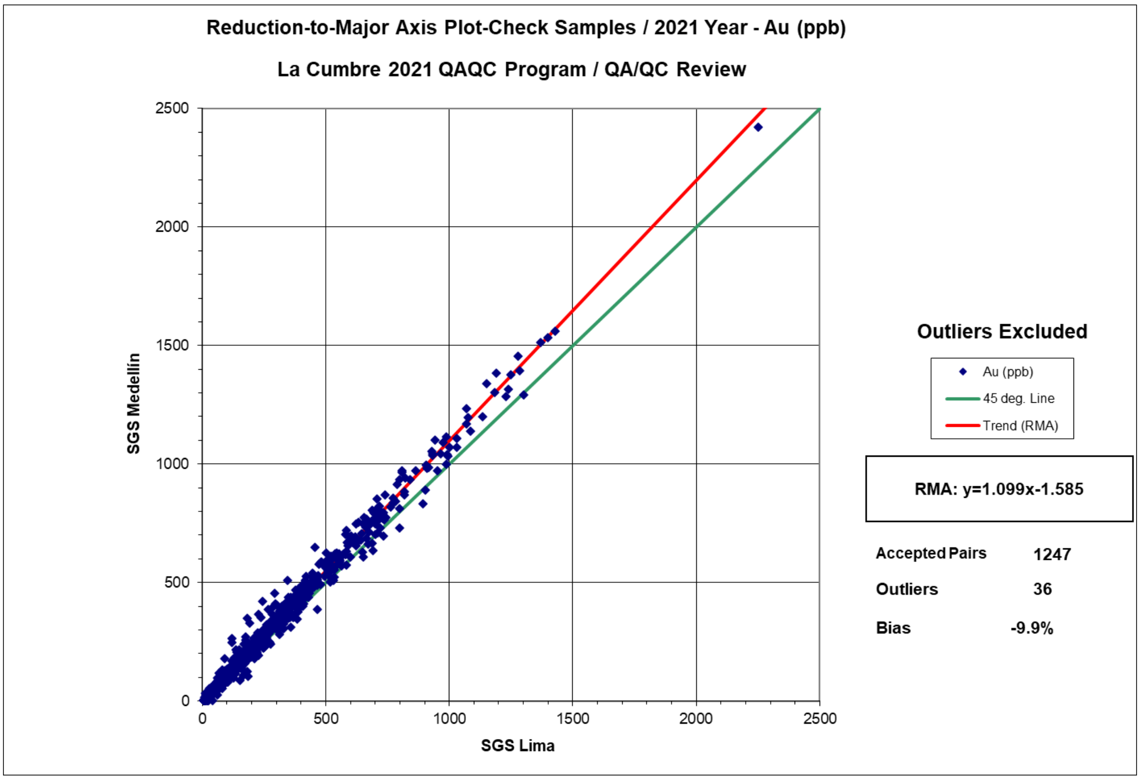


Figure 12-6:
RMA Plot Check Samples for Au in La Cumbre Deposit (Primary Zone)



13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Part of this information in this section has been extracted from Evans et al. (2013)

13.1 MINERAL METALLURGICAL EVALUATION BY G&T (MAY 2011)

The G&T Metallurgical Services Ltd. (G&T) carried out cyanidation, flotation and gravity separation tests on six samples weighing 515 kg (MMLC-004, MMLC-005, MMLC-010, MMLC-011, MMLC-012, and MMLC-014,) with calculated head assays of 1.17 g Au/t, 0.79 g Au/t, 0.66 g Au/t, 0.58 g Au/t, 0.65 g Au/t, and 0.78 g Au/t, respectively.

G&T completed scoping-level gravity, bottle roll cyanidation tests and rougher and cleaner flotation tests. They also conducted bottle roll cyanidation tests (BRT) on both the rougher and cleaner flotation tailings (G&T, 2011a, 2011b). Their tests were completed in two phases. Details of the test data are provided by RPA – Chamois & Evans (2012). During the first phase, gravity concentration, Bottle Roll Test (BRT), and rougher flotation tests were conducted on all six samples. During the second phase of testing, cleaner flotation tests with BRTs on rougher and cleaner flotation tailings were conducted. All bottle roll tests were conducted for 48 hours. Flotation tests used potassium amyl xanthate (PAX) and methyl isobutyl carbonyl (MIBC) under standard flotation conditions for pyritic material.

One sample, MMLC-004, was designated as oxide and, as expected, the results were different from the other five samples that had lower sulfur concentrations and higher gold content than the other five samples because the sample was more oxidized. The cyanide leach gold recoveries were higher and flotation recoveries were lower for the oxide sample.

A brief summary of the test results is presented in Table 13-1. Samples for the G&T work were identified by rock-type (saprolite, diorite, quartz, and breccias), pre-dating the current material type identification. Typically, saprolite represents the oxide material type, while the mixed and primary material types are more representative of the diorites and breccias.

Table 13-1:
Leach Test Data Summary of G&T Scoping Tests

	MMLC-004	MMLC-005	MMLC-010	MMLC-011	MMLC-012	MMLC-014
Material	**Saprolite**	**Saprolite**	**diorite + breccia**	**quartz diorite**	**Breccia**	**diorite**
Current Redox Domain	**Oxide**	**Mixed**	**Primary**	**Primary**	**Primary**	**Mixed**
Head Grade						
Au, g/t	1.17	0.79	0.66	0.58	0.65	0.78
Cu, %	0.11	0.14	0.17	0.13	0.12	0.15
S, %	0.03	1.12	0.79	2.06	3.37	0.88
Gravity Recovery	0.50%	2.40%	2.40%	1.60%	1.70%	1.80%
BRT Recovery, 100 µm	91.20%	72.20%	65.60%	66.30%	69.20%	74.70%
BRT Recovery, 70 µm Rougher Flotation	94.10%	76.10%	67.90%	82.40%	84.20%	75.90%
Gold Recovery	47.90%	66.60%	71.70%	79.30%	83.90%	70.20%
Sulfur Recovery	76.70%	94.00%	93.90%	96.80%	96.60%	92.20%
Cleaner Flotation plus Cyanide						
Copper Recovery			55.50%	59.80%	50.60%	
Gold Recovery			71.10%	76.60%	74.60%	
Silver Recovery			56.60%	62.30%	63.90%	

13.1.1 Conclusions of the G&T work

1. Gravity concentration recovered less than 3% of the gold so it was concluded that gravity concentration would not improve the metallurgical performance for Batero.
2. Whole ore cyanidation was successful at recovering gold and appears to be sensitive to the particle size:
3. Average gold recovery was approximately 73% for the tests conducted at a nominal particle size fraction of 80% passing (P80) of 100 µm.
4. Average gold recovery was 80% for a P80 of 70 µm.
5. The gold recovery for the oxide sample was 91% at 100 µm and 94% at 70 µm.
6. Rougher flotation gold recovery at a nominal P80 of 100 µm was 74% for the mixed and primary samples; the gold recovery was only 48% for the oxide sample.
7. Cyanidation extraction appears to be related to the gold and sulfur content of the samples.
8. There appears to be a strong correlation between sulfur concentration and rougher flotation gold recovery.
9. Using a combination of cleaner flotation and cyanide leaching of the flotation tailings increased the gold recovery by 5% to 10%.
10. There was no apparent relationship between gold recovery and sulfur concentration or gold head grade from this small data set.
11. Copper was also present in low quantities, measuring between 0.1% and 0.2% having chalcopyrite the dominating copper sulfide present, with only trace amounts of bornite and chalcocite measured.

13.2 MINERAL METALLURGICAL EVALUATION BY PLENGE (JULY 2012)

Batero commissioned C.H. Plenge & Cia. S.A. (Plenge) to carry out crush-leach testing on seven pulp reject samples with a head grade varies among 0.49 g/t Au to 1.3 g/t Au that had a particle size fraction of 93% passing 74 µm. BRT were conducted for 72 hours. These samples were selected by Batero to investigate the effects of sulfur, copper, and gold grades on the metallurgical results. The Plenge test data is summarized in Table 13-2.

Table 13-2:
Summary of Plenge Scoping Tests

	MMLC-004	MMLC-005	MMLC-010	MMLC-011	MMLC-012	MMLC-014	MMLC-014
Material Type	**Oxide**	**Mixed**	**Oxide**	**Mixed**	**Mixed**	**Oxide**	**Oxide**
Au g/t	0.87	0.67	0.58	1.30	0.66	0.75	0.49
Cu %	0.12	0.11	0.12	0.18	0.12	0.11	0.11
S %	0.03	0.62	0.02	0.71	0.75	0.03	0.59
Gold Recovery	**92.6%**	**83.9%**	**92.6%**	**86.0%**	**81.9%**	**93.9%**	**84.5%**

13.2.1 Conclusions of the Plenge Work

1. The gold recovery for the oxide samples was over 91%, which is consistent with the data from G&T.
2. The mixed ores that contained between 0.59% and 0.75% total sulfur achieved gold recoveries between 82% and 86%.
3. The test for MQ12-Comp-04 showed high cyanide consumption (i.e., 2.7 kg/t); the sample had the highest copper content (i.e., 0.18%).
4. There is no apparent relationship between gold recovery and gold grade, but the recovery clearly drops as the sulfur concentrate increases.

13.3 MINERAL METALLURGICAL EVALUATION BY SGS (FEBRUARY 2013)

Batero commissioned SGS Mineral Services (SGS-Lima-Peru) to carry out a 72-hour to heap leaching and milling of 16 samples and four composite samples. Batero personnel selected 16 samples to represent high, low, and average concentrations of gold, copper, and sulfur taken from across the resource. RPA confirmed that the samples were representative of material reported in the Quinchia Mineral Resource estimate.

The samples tested include 16 individual samples and 4 composite samples that were used to perform a number of tests including:

- BRT for 72 hours at nominal particle size distributions of 100% passing 2 mm and 80% passing 75 µm.
- Magnetic separation tests to determine if gold may be associated with magnetite.
- Column leach tests were conducted for 30 days including duplicate tests, one without agglomeration and one with agglomeration, for two samples containing saprolite.
- Analysis of +200 mesh (i.e., 74 µm) and -200 mesh size fractions was conducted to determine if the gold was concentrated in one of the size fractions.

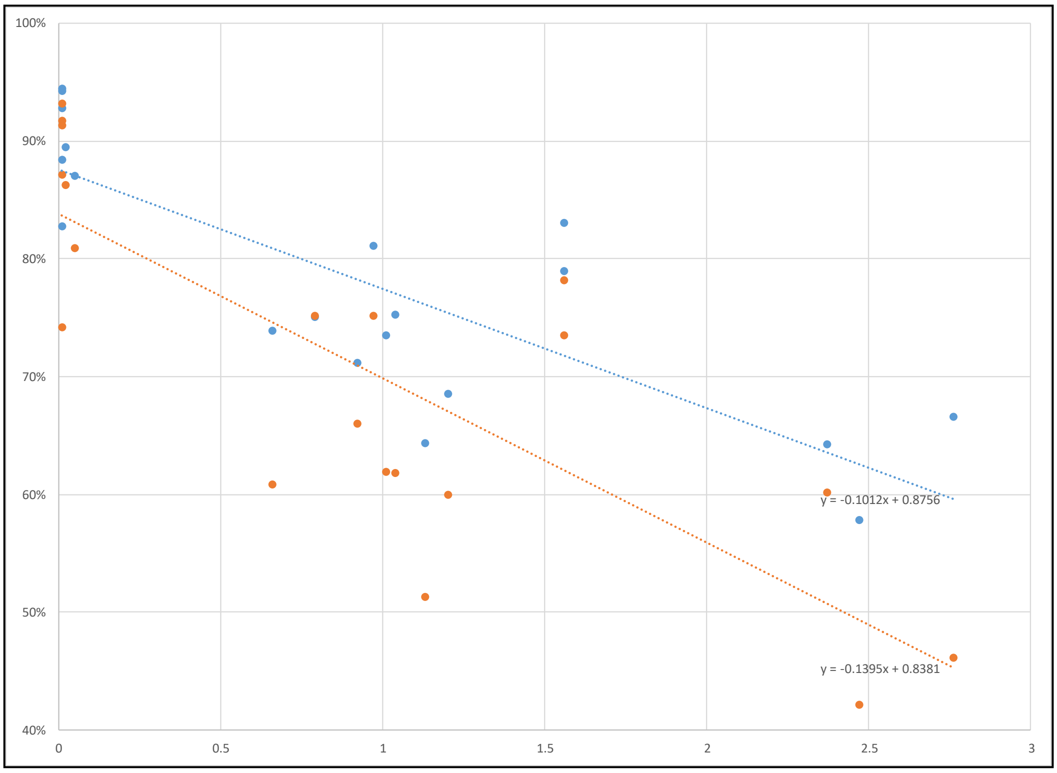
A summary of the results from the BRT reported by SGS (2013) are shown in Table 13-3. The tests were conducted on samples that were approximately one kilogram for 72 hours.

Table 13-3:
Summary of SGS Bottle Roll Test Data

Sample	Cu ppm	Fe %	Head Grade		S %	Gold Recovery		Silver Recovery	
			Au g/t	Ag g/t		2 mm	75 µm	2 mm	75 µm
CBR-1201	1845	3.14	0.548	2.44	1.04	61.8%	75.3%	70.23%	74.53%
CBR-1202	318	4.25	0.403	1.14	1.56	73.5%	83.1%	58.64%	58.85%
CBR-1204	1341	4.76	0.554	2.56	0.66	60.9%	73.9%	60.92%	70.25%
CBR-1205	1214	4.25	0.316	0.60	2.37	60.2%	64.3%	54.56%	62.85%
CBR-1206	1026	4.59	0.449	1.51	0.01	91.3%	92.8%	73.72%	82.00%
CBR-1207	1898	4.60	0.778	2.18	1.01	61.9%	73.5%	59.29%	59.67%
CBR-1208	119	4.34	0.845	1.08	1.56	78.2%	79.0%	58.00%	63.46%
CBR-1209	3110	2.89	0.694	1.92	0.92	66.0%	71.2%	54.92%	78.66%
CBR-1210	1447	5.67	0.802	0.32	0.01	87.2%	88.4%	18.33%	47.91%
CBR-1211	1209	4.49	0.859	2.06	2.47	42.2%	57.8%	40.51%	53.95%
CBR-1212	397	3.42	0.681	2.48	0.01	93.2%	94.5%	60.16%	71.44%
CBR-1213	1863	3.48	1.098	2.48	1.2	60.0%	68.6%	51.05%	52.26%
CBR-1214	532	3.74	1.429	2.20	0.97	75.2%	81.1%	61.38%	61.51%
CBR-1215	2307	2.71	1.369	3.84	0.79	75.2%	75.1%	73.63%	74.23%
CBR-1216	2132	6.39	1.262	<0.3	0.01	91.7%	94.3%	51.41%	52.11%
CBR-1218	568	5.44	1.321	1.44	0.01	74.2%	82.8%	57.27%	68.36%
BATCH-2	1673	4.30	0.694	2.62	1.13	51.3%	64.4%	69.24%	69.50%
BATCH-3	1157	5.43	0.715	1.57	0.05	80.9%	87.1%	60.87%	72.67%
BATCH-5	1516	4.69	1.343	1.90	2.76	46.1%	66.6%	52.27%	54.95%
BATCH-6	1006	4.46	1.286	1.83	0.02	86.3%	89.5%	63.73%	75.40%
Average						**70.9%**	**78.2%**	**57.5%**	**65.2%**

As shown in Figure 13-1, there appear to be weak correlations between gold recovery and sulfide sulfur concentrations of the samples. On average, the gold recovery was approximately 7% higher for the bottle roll tests conducted at 75 µm versus two millimeters (70.9% vs 78.2%), which confirms the previous observations that the gold recovery appears to be directly correlated with the particle size. The implication of this observation is that the recovery will be higher for finely crushed or ground material and the decision as to how to process the material will be dependent on trade-off studies to evaluate whether the increase in recovery will cover the higher capital and operating costs.

Figure 13-1:
Relationship between Gold Recovery and Sulfur Concentration



Similarly, silver recovery is an average of approximately 8% (57.5 vs 65.2) higher for the tests run at the smaller particle size distribution, but there does not seem to be a relationship between silver recovery and sulfide sulfur content. The data from the magnetic separation tests is shown in Table 13-4.

Table 13-4:
Summary of SGS Magnetic Separation Data

Sample	Initial Weight, g	Magnetic Fraction, g	Non-magnetic Fraction, g	Magnetic Au, g/t	Non-magnetic Au, g/t	% Au in Magnetic Fraction	% Au in: Non-magnetic Fraction
CBR-1201-MG	499.3	74.2	425.1	0.654	0.538	17.5%	82.5%
CBR-1202-MG	490.0	110.0	380.0	0.593	0.359	32.3%	67.7%
CBR-1204-MG	494.0	395.0	99.0	0.541	0.671	76.3%	23.7%
CBR-1206-MG	497.0	133.5	363.5	0.568	0.298	41.2%	58.8%
CBR-1207-MG	495.5	239.0	256.5	0.831	0.770	50.1%	49.9%
CBR-1208-MG	497.5	131.5	366.0	1.599	0.743	43.6%	56.4%
CBR-1209-MG	499.5	10.0	489.5	0.644	0.685	1.9%	98.1%
CBR-1210-MG	499.5	33.5	466.0	0.782	0.792	6.6%	93.4%
CBR-1211-MG	500.0	9.5	490.5	1.383	0.884	2.9%	97.1%
CBR-1212-MG	496.5	131.5	365.0	0.665	0.750	24.2%	75.8%
CBR-1213-MG	500.5	11.5	489.0	0.781	1.138	1.6%	98.4%
CBR-1214-MG	497.0	275.5	221.5	1.574	1.392	58.4%	41.6%
CBR-1215-MG	500.0	80.0	420.0	1.108	1.317	13.8%	86.2%
CBR-1216-MG	497.5	116.5	381.0	0.668	1.251	14.0%	86.0%

Sample	Initial Weight, g	Magnetic Fraction, g	Non-magnetic Fraction, g	Magnetic Au, g/t	Non-magnetic Au, g/t	% Au in Magnetic Fraction	% Au in: Non-magnetic Fraction
CBR-1218-MG	497.0	306.5	190.5	1.716	1.248	68.9%	31.1%
BACTH-2-MG	497.5	217.0	280.5	0.628	0.728	40.0%	60.0%
BACTH-3-MG	495.0	230.0	265.0	0.763	0.720	47.9%	52.1%
BACTH-5-MG	497.5	135.0	362.5	0.601	1.347	14.2%	85.8%
BACTH-6-MG	498.0	164.5	333.5	1.107	1.331	29.1%	70.9%
Average						**30.8%**	**69.2%**

Note: Sample CBR-1205 contained high percentage of magnetic fraction.

The analysis of magnetic separation data does not indicate that the presence of larger quantities of gold in the magnetic fraction had any effect on the gold recovery. Batero had expressed a concern that it might. Table 13-5 provides the data from the samples that were screened at 75 µm.

Table 13-5:
Summary of SGS Data Screened Samples

Sample	Initial Weight, g	+ 75 µm Weight, g	- 75 µm Weight, g	+ 75 µm Au, g/t	- 75 µm Au, g/t	% Au in +75 µm Fraction	% Au in - 75 µm Fraction
CBR-1205	498.60	423.83	74.77	0.253	0.549	72.3%	27.7%
CBR-1206	501.20	403.28	97.92	0.423	0.608	74.1%	25.9%
CBR-1212	500.50	342.95	157.55	0.618	0.777	63.4%	36.6%
CBR-1216	497.30	249.59	247.71	1.106	1.276	46.6%	53.4%
BATCH 2	1001.10	930.78	70.32	0.748	0.772	92.8%	7.2%
BATCH 3	999.10	714.31	284.79	0.756	0.771	71.1%	28.9%
BATCH 5	1000.70	930.57	70.13	1.320	1.095	94.1%	5.9%
BATCH 6	998.60	744.20	254.40	1.287	1.727	68.6%	31.4%
Average						**72.9%**	**27.1%**

This data shows that the quantity of gold contained in the plus 75 µm size fractions and the minus 75 µm size fractions does not indicate that it would be possible to upgrade the feed to a recovery process by washing the fine particles from the coarser particles prior to processing. Due to the presence of saprolite, RPA hoped that it might be possible to wash the fines from the material and process the material separately to avoid potential percolation problems in a heap leach operation. Since there are significant quantities of gold in both fractions, this does not appear to be a viable option.

Column leach tests were conducted in columns that were 150 mm diameter by 2.5 m high; material was crushed to 100% < ½ in. prior to loading. The gold and silver recoveries and reagent consumptions for the six column tests are summarized in Table 13-6.

Table 13-6:
Summary of SGS Column Leach Test Data (-12 Mm) Batero Gold Corp.

Test	Description	30-days Gold: Recovery	30-day Silver: Recovery	NaCN,: kg/t	CaO,: kg/t
Batch 2	Without Agglomeration	0.408	0.516	1.40	0.86
Batch 3	Without Agglomeration	0.792	0.580	2.17	2.36
Batch 3	With Agglomeration	0.795	0.462	2.49	2.36
Batch 5	Without Agglomeration	0.399	0.490	1.54	1.01
Batch 6	Without Agglomeration	0.855	0.766	2.47	1.48
Batch 6	With Agglomeration	0.858	0.689	3.02	1.48

13.4 MINERAL METALLURGICAL EVALUATION BY METTS (DECEMBER 2018)

Minera Quinchia S.A.S. commissioned Metallurgical Testing Services (METTS) to carry out mineral characterization tests (mineralogy), as well as dynamic heap leaching and/or conventional cyanidation on Oxide and Transition Head Ore from La Cumbre deposit in Colombia.

13.4.1 Experimental Tests on Oxide Ores

A total of 1686.35 kg of Oxide Head Ore Type material were homogenized in a composite sample, and by using a preparation and sampling protocol were distributed in samples based on the type of metallurgical evaluation/testing to be performed, including: Physical and Chemical Characterization, Permeability, Large Columns and Small Columns with counter-samples.

13.4.1.1 Physical and Chemical Characterization

From the sample obtained for chemical analysis from the head of the sampling protocol, 32 elements including Au and Ag were sent to the Certimin laboratory where they were analyzed by ICP method. The head ore grades were: 1.293 Au g/t and 0.50 g/t Ag. The parameters for physical characterization are shown in Table 13-7.

Table 13-7:
Physical Parameters for Oxide Head Ore Sample

Parameter	Value
Humidity (%)	4.35
Specific Gravity (g/cc) *	2.36
Natural pH	6.90
Angle of repose+ (°)	38.0

*Obtaining by using pycnometer method

13.4.1.2 Grain-Size Characterization of Head Ore

Sieving of the composited sample through sieves (Tyler): 2”, 1", ½", ¼" #10, #20, #40, #40, #80, #80, #100, #140, #200, #270, #325, - #325, all retained fractions were analyzed for gold and silver, the results of which are shown in Table 13-8.

Table 13-8:
Assayed Sieve Size Analysis of the Oxide Head Samples

Tyler Mesh	Weight (kg)	Weight Distr., %	Cumulative Retained weight	Accum. through weight	Assay (g/t)		Distribution (%)			
					Au	Ag	Partial Au Head	Cumulative Au Head	Partial Ag Head	Cumulative Ag Head
2	1,351	5,21%	5,21%	94,79%	1,22	0,6	4,51	4,51	3,61	3,61
1	1,757	6,78%	11,99%	88,01%	0,96	0,4	4,63	9,14	3,13	6,75
1/2	1,893	7,31%	19,30%	80,70%	0,83	0,6	4,29	13,43	5,06	11,81
1/4	3,090	11,92%	31,22%	68,78%	0,99	0,8	8,37	21,80	11,02	22,83
10 m	2,785	10,75%	41,97%	58,03%	1,26	1,3	9,62	31,42	16,14	38,98
20 m	2,890	11,15%	53,12%	46,88%	3,06	1,4	24,28	55,70	18,04	57,02
40 m	1,419	5,48%	58,60%	41,40%	1,64	1	6,37	62,07	6,33	63,34
80 m	1,130	4,36%	62,96%	37,04%	1,96	1,3	6,06	68,13	6,55	69,89
100 m	0,154	0,59%	63,55%	36,45%	1,29	0,9	0,54	68,67	0,62	70,51
140 m	0,514	1,98%	65,54%	34,46%	1,26	1,2	1,78	70,46	2,75	73,26
200 m	1,032	3,98%	69,52%	30,48%	0,95	0,6	2,68	73,14	2,76	76,02
270 m	0,644	2,49%	72,01%	27,99%	1,08	0,4	1,90	75,04	1,15	77,17
325 m	0,426	1,64%	73,65%	26,35%	0,65	0,8	0,76	75,80	1,52	78,69

Tyler Mesh	Weight (kg)	Weight Distr., %	Cumulative Retained weight	Accum. through weight	Assay (g/t)		Distribution (%)			
					Au	Ag	Partial Au Head	Cumulative Au Head	Partial Ag Head	Cumulative Ag Head
-325 m	6,828	26,35%	100,00%	0,00%	1,29	0,7	24,20	100,00	21,31	100,00
Total	**25.913**	**100.00%**			**1.41**	**0.87**	**100.00**		**100.00**	

The average grade obtained from the granulometric analysis of 1.41 g/t Au and 0.87 g/t Ag respectively, which has similar values to those obtained for the head grade of 1.293 Au g/t and 0.50 g/t Ag.

13.4.1.3 Head Ore Permeability Tests

The quantification of the passage of cyanide solutions through the material loaded in an acrylic column, having to prepare the material without agglomeration or with agglomeration (cement, lime and cyanide solution), to simulate the passage of cyanide solutions through a leaching heap. Those results are shown at Table 13-9 and the cyanidation conditions test in Table 13-10.

Table 13-9:
Results of Permeability Tests

Test	Condition	Permeability Coefficient (K; m/s)	Permeability Index (cm/d)	Result 1	Result 2
Test 1	Without Agglomeration	1.96E-05	6.29	Semi-permeable	Slow
Test 2	With Agglomeration (3 kg/t)	5.08E-06	43.9	Semi-permeable	Moderately slow
Test 3	With Agglomeration (5 kg/t)	7.63E-06	65.94	Semi-permeable	Moderate
Test 4	With Agglomeration (7 kg/t)	1.02E-05	87.81	Semi-permeable	Moderate
Test 5	With Agglomeration (8 kg/t)	2.39E-05	206.67	Semi-permeable	Moderately quickly
Test 6	With Agglomeration (9 kg/t)	3.40E-05	293.58	Permeable	Moderately quickly
Test 7	With Agglomeration (10 kg/t)	3.0E-05	6.29	Permeable	Moderately quickly
Test 8	With Agglomeration (15 kg/t)	3.55E-05	6.29	Permeable	Quickly

Table 13-10:
Cyanidation Conditions Test

Parameter	Value
Sample Weight (g)	2000
Water Volume (cc)	4000
Granulometry	80% -#200 Tyler
Testing time (hours)	72
pH	10.5 – 11.0
[CN-]	0.1%
Aliquot collection time (hours)	1, 2, 8, 12, 24, 48, and 72,
Aliquot volume	50 ml, for chemical analysis and 25 ml, for titration

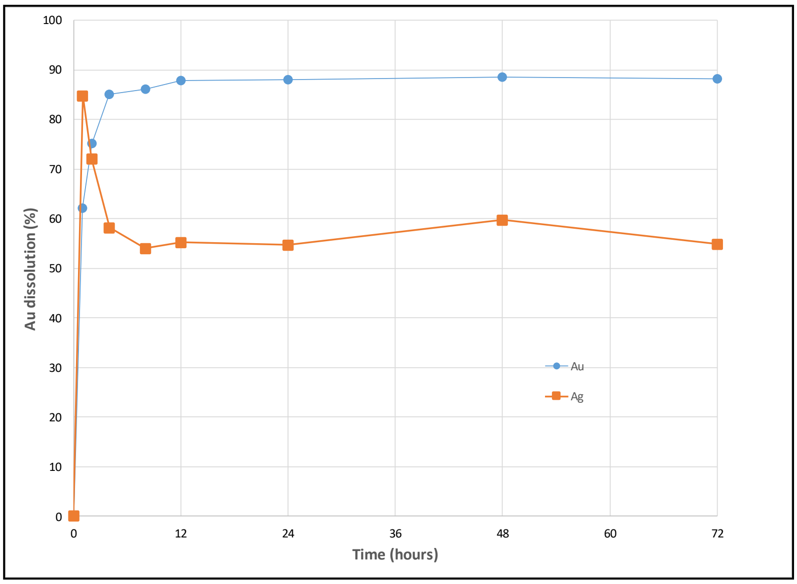
Table 13-11, shows the gold and silver dissolution and reagents consumption in bottle rolled test conducted by METTS Laboratory.

Table 13-11:
Gold & Silver Dissolution and Reagent Consumption in Bottle Rolled Tests

Head Grades				Solution Grade		Dissolved Grade		Tailing Grade		Reagent consumption (kg/t)		Recovery (%)	
Assayed		Calculated		Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Cyanide (NaCN)	Lime	Au	Ag
Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)							Au	Au		
1.293	0.5	1.466	3.213	0.553	1.039	1.294	2.513	0.172	0.70	2.82	1.25	88.27	54.84

The results indicate that there is fine-grained free gold due to the rapid gold dissolution kinetics achieved in the first 12 hours with dissolution values close to 90% for Au and 55% for Ag (See Figure 13-2). The reagent consumptions (lime and cyanide) are moderate for cyanide (2.82 kg/t) and low for lime (1.25 kg/t) which is interpreted that even the particle liberation is not optimal (80% - 75 microns), having to reduce the particle sizes to obtain gold dissolutions higher than 90%. The cyanide consumptions indicate that there are cyanides in the ore and there should be a higher liberation (85% - 75 microns).

Figure 13-2:
Gold and Silver Dissolution Curves in Bottle Rolled Tests



According to the results of the permeability tests on oxidized ore, it requires agglomeration to reach semi-permeability characteristics before the passage of cyanide solutions.

13.4.1.4 Tests on Columns

Three tests are carried out in PVC columns, one of them with ROM granulometry (100% -3") while the other two columns with 100% -1" granulometry, where the behavior of the mineral to heap cyanidation is evaluated, with variable granulometry of the fed mineral and the pre-treatment of agglomeration of the mineral with cement to fractions smaller than - ½", keeping other parameters constant (cyanide concentration of the leaching solution, irrigation rate). The test conditions are shown in Table 13-12.

Table 13-12:
Column Parameter Conditions

Number	Weight in kg (dry)	Granulometry (in)	Lime Addition (kg/t)	Irrigation Rate (l/h/m2)	[CN-] Leach Solution (ppm)
Column 1	631.29	100%-3”	10	6	1,000
Column 2	66.96	100%-1”	10	6	1,000
Column 3	66.96	100%-1”	7	6	1,000

The results of the oxide sample obtained a dissolution for gold between 87.8% to 93.4% and for silver between 38.5% to 45.8% with periods of 30 to 40 days of irrigation with variable granulometry from 3" to 1" but with agglomeration for the fine granulometry -½". Reagent consumptions are nearly similar in the order of 1 to 1.12 kg/t of cyanide and lime consumption from 1.2 to 1.25 kg/t, due to the near absence of cyanide elements in the oxides. Table 13-13 shows the results for Column Tests.

Table 13-13: Column Test Results for Au & Ag

Number	Head Grades				Grade of dissolved in PLS		Tailing Grade		Leaching Ratio (m^3/t)	Reagent consumption (kg/t)		Recovery (%)	
	Assayed		Calculated										
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)		Cyanide (NaCN)	Lime	Au	Ag
Column 1	1.41	0.5	1.726	0.899	1.516	0.399	0.210	0.50	1.84	1.00	1.25	87.83	45.81
Column 2			1.350	0.814	1.261	0.314	0.089	0.50	1.54	1.36	1.25	93.41	38.56
Column 3			1.357	0.734	1.254	0.334	0.103	0.40	1.49	1.12	1.20	92.41	45.47

Figure 13-3 and Figure 13-4 show the dissolution kinetics for the Oxide sample for Au and Ag respectively, noting that for Au, in a 15-day period, a dissolution of 90% associated with fine-grained free Au is accomplished. While for Ag, the dissolution is up to 45%, but improving the recovery periods when agglomerated (100% - 1") with higher cement consumption resulting in a better degree of percolation to the cyanide solutions.

Figure 13-3: Gold Dissolution Kinetics for Oxidized Sample

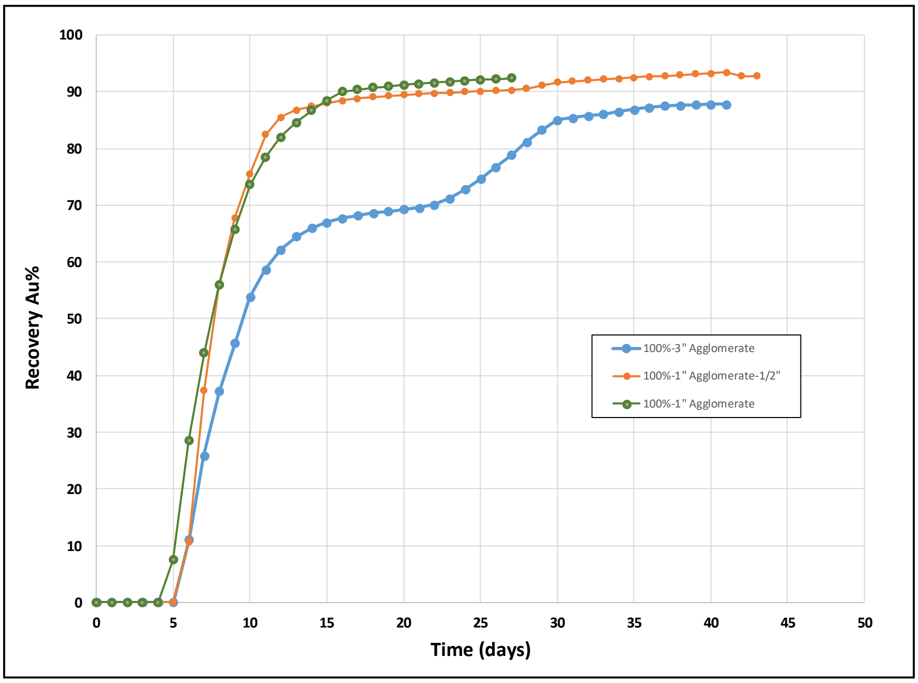
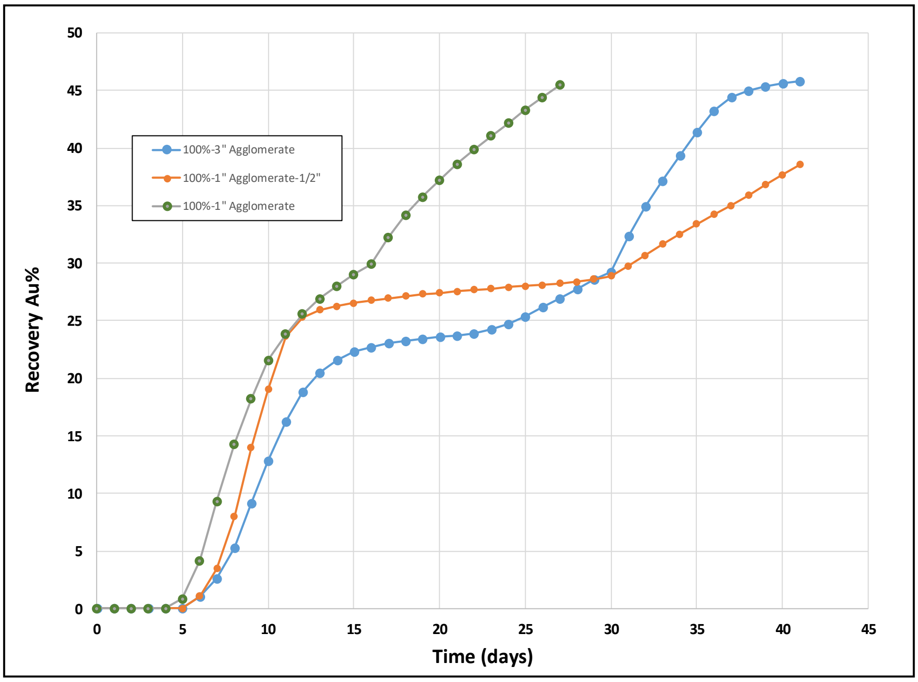


Figure 13-4:
Silver Dissolution Kinetics for Oxidized Sample



13.4.1.5 Analysis of Gold Performance in Tailing Columns

Comparison results of Au grades by particle size of head ore versus tailing waste materials in columns 1, 2 and 3 (100% -3" and 1" particle size) are shown in Figure 13-5 and Figure 13-6.

Figure 13-5:
Gold Grades by Particle Size (g/t)

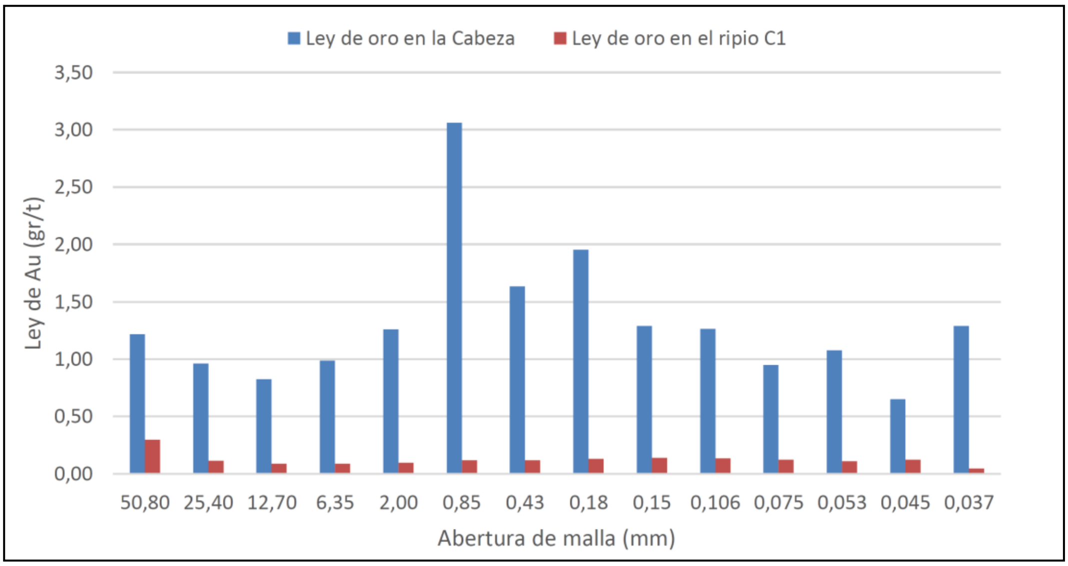
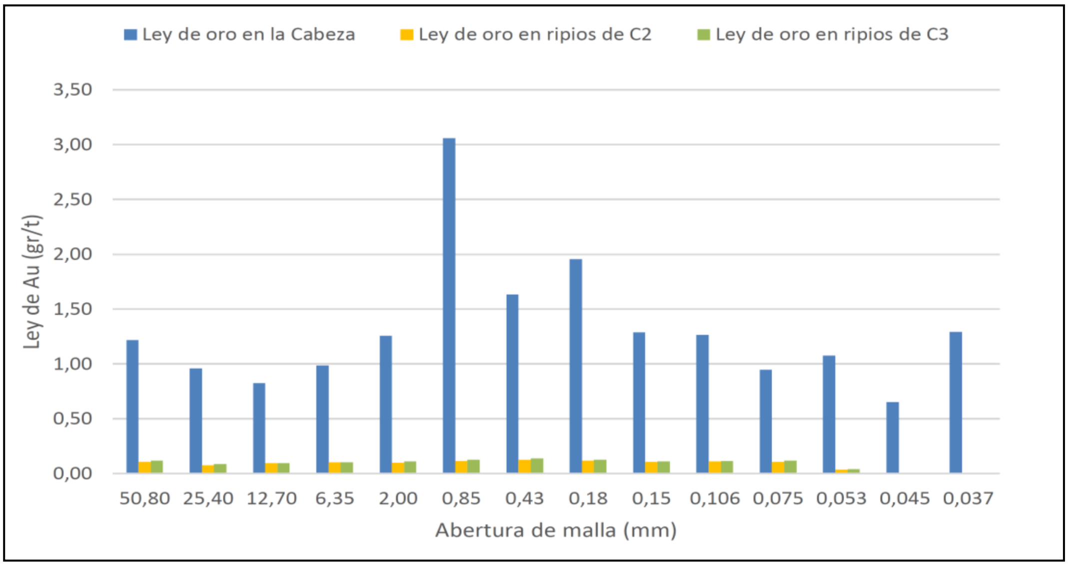


Figure 13-6:
Gold Grades by Particle Size (g/t)



13.4.2 Experimental Tests on Transition Head Ore

A total of 720.65 kg of Transition Head Ore Type material were homogenized in a composite sample, and by using a preparation and sampling protocol were distributed in samples based on the type of metallurgical evaluation/testing to be performed, including: Physical and Chemical Characterization, Permeability, Large Columns and Small Columns with counter-samples.

13.4.2.1 Physical and Chemical Characterization of Samples

By using sampling protocols to determine the sample sent to CERTIMIN Laboratory to be analyzed by the ICP method for 32 elements. The results for the main economic metals (Au, Ag) obtaining a head ore grade of: 1.08 Au g/t and 2.10 g/t Ag. Table 13-14, shows the physical parameters for oxide ore head.

Table 13-14:
Physical Parameters for Oxide Head Ore Sample

Parameter	Value
Humidity (%)	1.61
Specific Gravity (g/cc) *	2.65
Natural pH	6.77
Angle of repose⁺ (°)	38.1

*Obtaining by using pycnometer method

13.4.2.2 Grain-Size Characterization of Head Ore

Sieving of the composited sample through sieves (Tyler): 1", ½", ¼" #10, #20, #40, #40, #80, #80, #100, #140, #200, #270, #325, - #325, all retained fractions were analyzed for gold and silver, the results of which are shown in

Table 13-15.

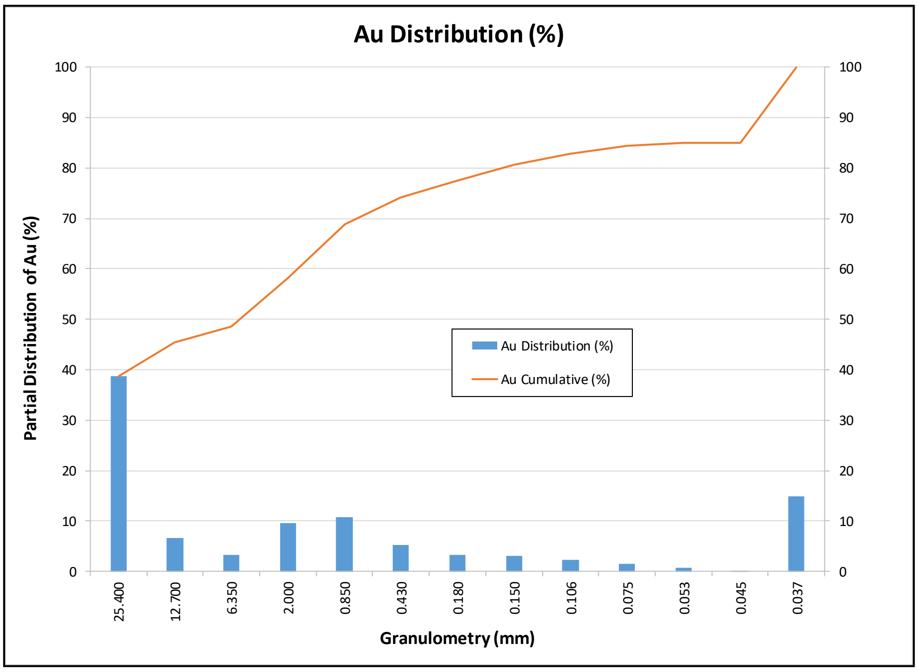
Table 13-15:
Assayed Sieve Size Analysis of the Transition Samples

Tyler Mesh	Weight. (kg)	Weight Distr. %	Cumulative Retained weight	Accum. through weight	Assay (g/t)		Distribution (%)			
					Au	Ag	Partial Au Head	Cum. Au Head	Partial Ag Head	Cum. Ag Head
1	10.210	35.99%	35.99%	64.01%	1.272	2.2	38.79	38.79	33.00	33.00
0.5	2.210	7.79%	43.78%	56.22%	1.007	1.6	6.65	45.44	5.19	38.19
0.25	0.960	3.38%	47.17%	52.83%	1.113	2.8	3.19	48.63	3.95	42.14
10M	3.100	10.93%	58.10%	41.90%	1.036	2.3	9.59	58.22	10.47	52.62
20M	3.510	12.37%	70.47%	29.53%	1.020	2.3	10.69	68.92	11.86	64.48
40M	1.730	6.10%	76.57%	23.43%	1.032	2.3	5.33	74.25	5.85	70.32
80M	1.190	4.20%	80.76%	19.24%	0.922	1.8	3.28	77.53	3.15	73.47
100M	0.907	3.20%	83.96%	16.04%	1.153	1.7	3.12	80.65	2.27	75.74
140M	0.550	1.94%	85.90%	14.10%	1.344	2.4	2.21	82.86	1.94	77.68
200M	0.436	1.54%	87.44%	12.56%	1.142	2.2	1.49	84.35	1.41	79.08
270M	0.166	0.59%	88.02%	11.98%	1.243	2.0	0.62	84.96	0.49	79.57
325M	0.018	0.06%	88.08%	11.92%	1.392	2.6	0.07	85.04	0.07	79.64
325M	3.380	11.92%	100.00%	0.00%	1.482	4.1	14.96	100.00	20.36	100.00
Totals	**28.367**	**100.00%**			**1.180**	**2.4**				

The average grade obtained from the granulometric analysis of 1.18 Au g/t and 2.4 g/t Ag respectively, which has similar values to those obtained for the head grade of 1.08 Au g/t and 2.10 g/t Ag.

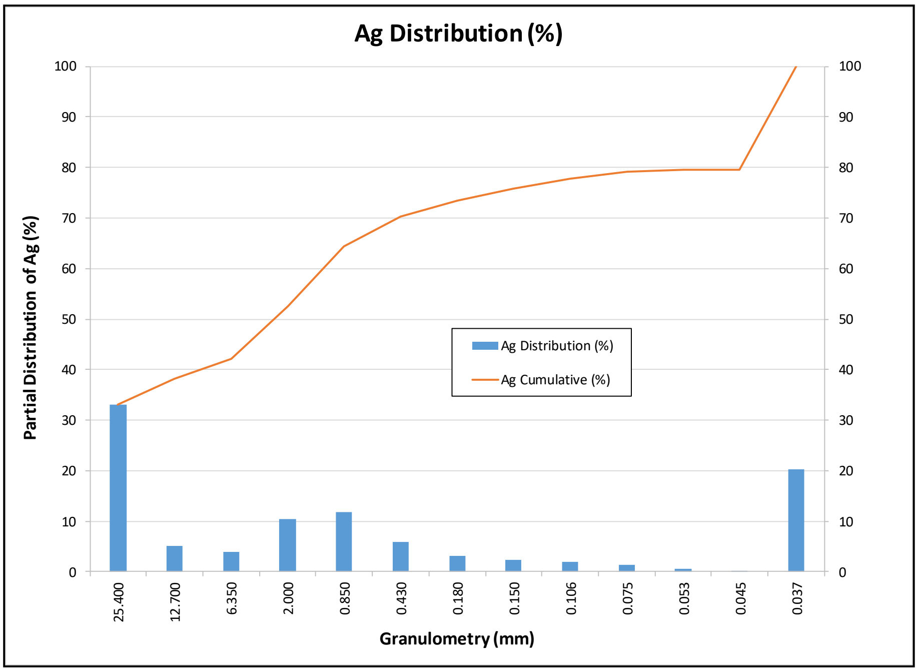
From the granulometric analysis assessed from the Transition sample, values up to 70% Au in coarse granulometry (> 2 mm), and 14.9% Au in fine to ultra-fine granulometry (<37 microns), and that due to contact with cyanide solutions might not be recovered by a heap leaching process. While the grades are evenly distributed from 1 to 1.5 Au g/t, having that wetting the ore adequately would dissolve the Au, see Figure 13-7.

Figure 13-7:
Distribution of Gold through the Mineral Particle Size



In the case of Ag, values up to 65% Ag in coarse grain size (>0.85 mm), and that 35% are distributed in fine mesh, being most of this value of 20% silver in very fine sizes less than 45 microns, and it is possible that this presence of silver in ultra-fine grain size with almost no recovery (see Figure 13-8).

Figure 13-8:
Distribution of Gold through Mineral Particle Size at Transition Ore



13.4.2.3 Head Ore Permeability Tests

Permeability tests for Transition ore material is shown in the Table 13-16.

Table 13-16:
Results of Permeability Tests

Test	Condition	Permeability Coefficient (K; m/s)	Permeability Index (cm/d)	Result 1	Result 2
Test 1	Without Agglomeration	0.00000458	39.58	Semi-permeable	Moderately slow
Test 2	Without Agglomeration	0.000171	1475.56	Permeable	Very quick
Test 3	Without Agglomeration	0.000223	1924.54	Permeable	Very quick

Transition ore does not require agglomeration because it has permeability properties to the passing of cyanide solutions.

13.4.2.4 Bottle Rolled Tests (BRT)

These processes for Transition Head Ore material are the same cyanidation conditions as those for Oxide Head Ore material (see Table 13-17).

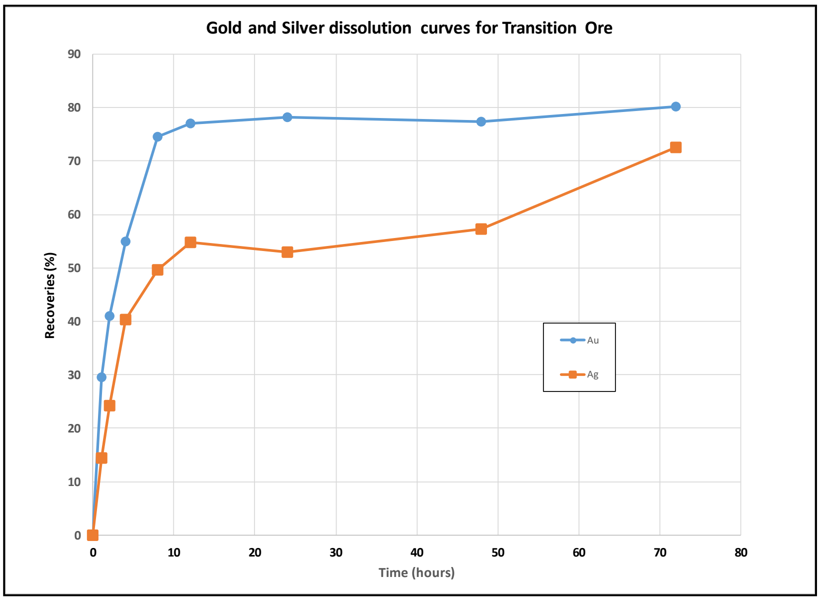
Table 13-17:
Gold & Silver Dissolution and Reagent Consumption in Bottle Tests (Transition Ore)

Head Grades				Solution Grade		Dissolved Grade		Tailing Grade		Reagent consumption (kg/t)		Recovery (%)	
Assayed		Calculated		Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Cyanide (NaCN)	Lime	Au	Ag
Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)							Au	Au		
1.08	2.1	1.213	3.275	0.431	1.081	0.972	2.375	0.241	0.90	3.53	3.56	80.1	72.5

High percentages of Au and Ag dissolutions (values close to 80% for Au and 72.5% for Ag) with high consumption of lime and cyanide reagents (3.56 kg/t and 3.53 kg/t respectively). Possibly, the particle liberation has not been adequate (80% - 75 microns), so, a higher reduction is required to obtain Au dilutions higher than 90%, but perhaps a higher initial cyanide concentration for the Transition Ore. Also, cyanide contents with sulfide contents (S°: 0.42% and S+2: 0.84%).

A rapid dissolution of Ag within 12 hours of leaching, then slowly increasing until the end of the test. The kinetic curves of gold and silver dissolution are shown in Figure 13-9.

Figure 13-9:
Gold and Silver Dissolution Curves in Bottle Tests



Similar to Oxidized Ore, presence of fine-grained free Au, with rapid dissolution of Au before 10 hours of leaching, then maintaining small increases until the end of the test.

13.4.2.5 Tests on Columns

Two tests are performed in PVC columns at 100% - 1" granulometry, where one test is irrigated without agglomeration and the second test evaluates the cyanidation behavior of the mineral in heaps, varying the agglomeration parameter as the pre-treatment with cement for fractions smaller than -½". The other parameters are kept constant. The results of this test can be seen in Table 13-18.

Table 13-18:
Column Parameter Conditions

Number	Weight in kg (dry)	Granulometry (in)	Lime Addition (kg/t)	Irrigation Rate (l/h/m2)	[CN-] Leach Solution (ppm)
Column 1	116.30	100%-1”	0	8	1,000
Column 2	111.77	100%-1”	0.5	8	1,000

The results of the gold and silver dissolution tests obtained by varying the above-mentioned parameters are shown in Table 13-19.

**Table 13-19:
Column Test Results for Au & Ag**

Number	Head Grades				Grade of dissolved in PLS		Tailing Grade		Leaching Ratio (m^3/t)	Reagent consumption (kg/t)		Recovery (%)	
	Assayed		Calculated										
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)		Cyanide (NaCN)	Lime	Au	Ag
Column 1	1.08	2.10	1.053	3.411	0.553	0.211	0.50	3.20	2.3	2.00	3.66	52.54	6.20
Column 2			1.027	3.359	0.657	0.659	0.37	2.70	2.1	1.62	3.66	63.97	19.61

Figure 13-10 and Figure 13-11 show the dissolution kinetics for the Transition samples for Au and Ag respectively, noting that for Au 52.54% to 63.9% and for Ag from 6.2% to 19.6% in a period of 35 days of irrigation, with different results in agglomeration or not for fine granulometry - ½". Consequently, low dilutions for Au and Ag, and an increase in the cyanide concentration and the use of higher doses of cement in the agglomeration than those used during these tests could be attempted.

**Figure 13-10:
Gold Dissolution Kinetics for the Transition Sample**

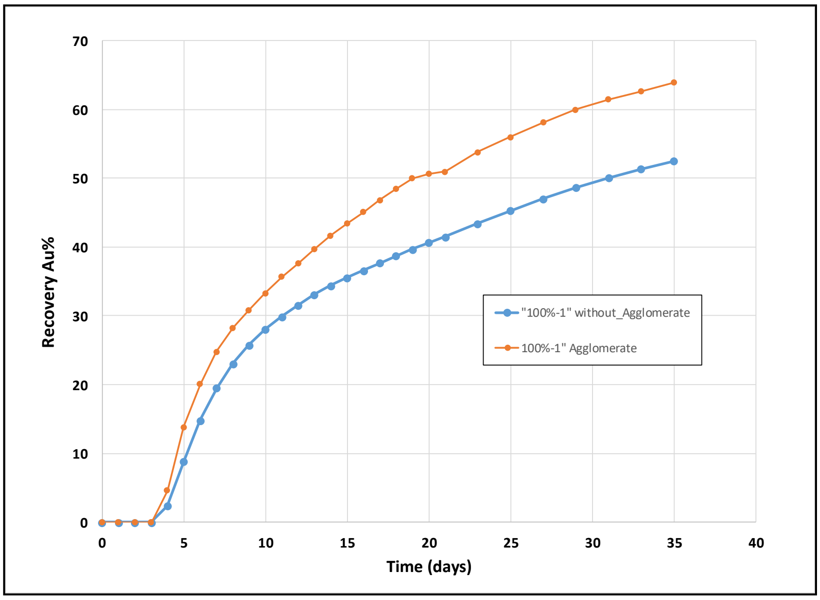
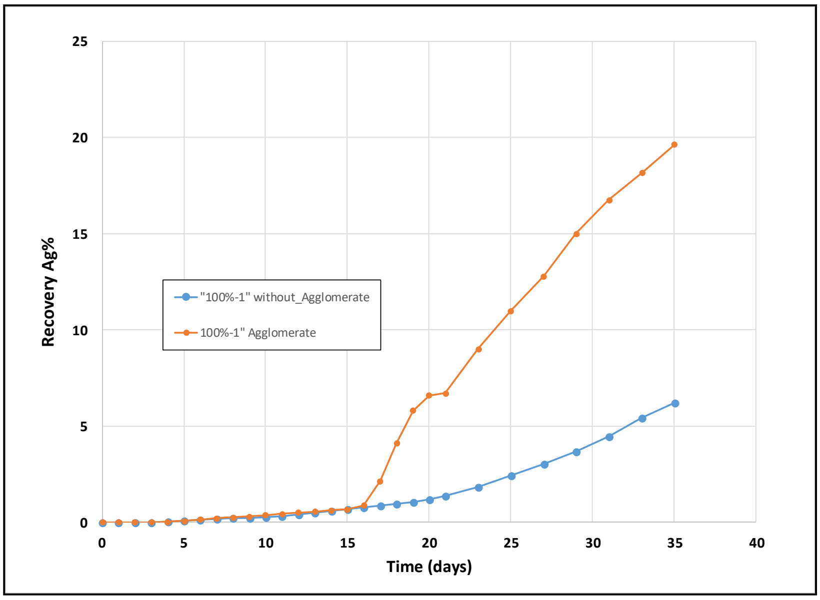


Figure 13-11:
Gold Dissolution Kinetics for the Transition Sample



13.4.3 Analysis of Gold Performance in the Leached Columns

Comparison of Au grades by particle size of head ore versus leached material in Column 1 and Column 2 (100% -1" particle size) for Transition head ore are shown in Figure 13-12 and Figure 13-13, respectively.

Figure 13-12:
Gold grades by particle size (g/t), Column 1

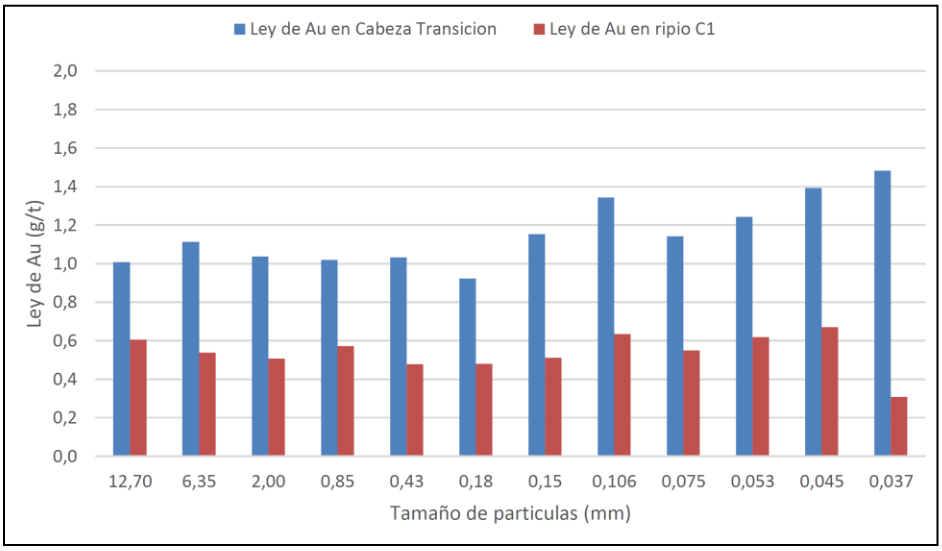
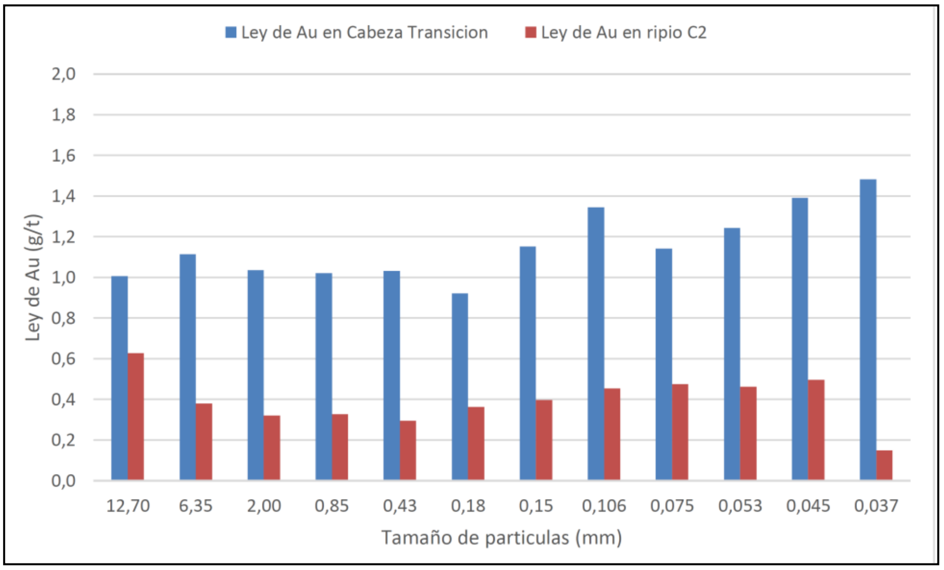


Figure 13-13:
Gold grades by particle size (g/t), Column 2



13.4.4 Conclusions

- Gold and silver grades for the Oxide Ore were 1.293 Au g/t and 0.5 g/t Ag with column leach recoveries of 88% Au and 45% Ag under agglomeration, grain size, irrigation rate and cyanide concentration parameters detailed in this report.
- Gold and silver grades for the Transition Ore were 1.08 Au g/t and 2.1 g/t Ag with column leach recoveries of 64% Au and 19% Ag under the parameters of particle size, agglomeration, irrigation rate and cyanide concentration detailed in this report.

For Oxide Ores, column tests are stated as follows:

- Reducing particle size to 100% -1" and agglomerating fine fractions less than ½" with cement.
- Agglomerate with 10 kg/t cement dosage and cured with lime and 0.05 g/L cyanide solution for 72 hours.
- Use an irrigation rate of up to 6 L/h/m2, a higher rate could break the agglomerates.

For Transition Ores, column tests are stated as follows:

- Reducing particle size to 100% -1" and agglomerating fine fractions less than ½" with cement.
- Agglomerate with 0.5 kg/t cement dosage and cured with lime and 0.1 g/L cyanide solution for 72 hours.
- Use an irrigation rate of up to 8 L/h/m2, a higher rate could break the agglomerates.
- Use longer irrigation periods (>35 days) to attain better gold dissolution and increase the cyanide concentration to values greater than 1000 ppm (0.1%), due to the presence of high Cu and Fe values in the Transition Ore.

13.5 MINERAL METALLURGICAL EVALUATION BY METTS (FEBRUARY 2019)

Minera Quinchia S.A.S. commissioned Metallurgical Testing Services (METTS) to carry out to tests to reduce cement consumption without detriment of Au and Ag dissolution rates, by using an F-1 setting reagent as an additive together with the cement in the Oxide Ore agglomeration processes from La Cumbre deposit in Colombia.

Previous works of pre-treatment of agglomeration with cement of the Oxide Ore obtaining acceptable recoveries of Au and Ag (80.1%) but with high cement consumptions between 7 kg/t to 10 kg/t being agreed to use a forge in the agglomeration, and with it to increase the strength of the agglomerate, to obtain a material permeable to the cyanide solutions.

After several tests, it was concluded that to achieve a cement consumption between 3 kg/t to 5 kg/t, a continuous curing time of 24 hours was required, with additions of forge accelerator (F-1) of 0.2 kg/t, a humidity of 20%, and maintaining low irrigation rates (6 l/h/m2), which avoids the formation of blind zones of difficult permeability. Also, a cyanide leach irrigation period (about 26 days) yielded recoveries for Au and Ag of 93.5% and 32.85%, correspondingly.

Figure 13-14 and Figure 13-15 show a rapid gold dissolution kinetics reaching values of 90% in 15 days of irrigation, which indicates that the cyanide solutions pass through the mineral dissolving the gold under an adequate agglomeration pre-treatment and avoids the formation of blind zones.

Figure 13-14:
Gold Dissolution Kinetics for the Oxide Sample

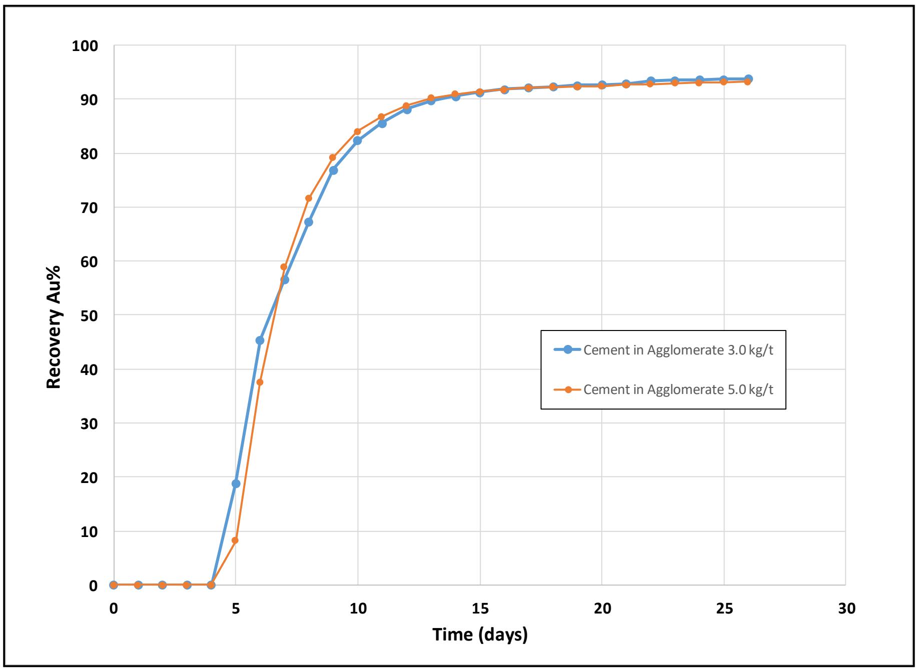
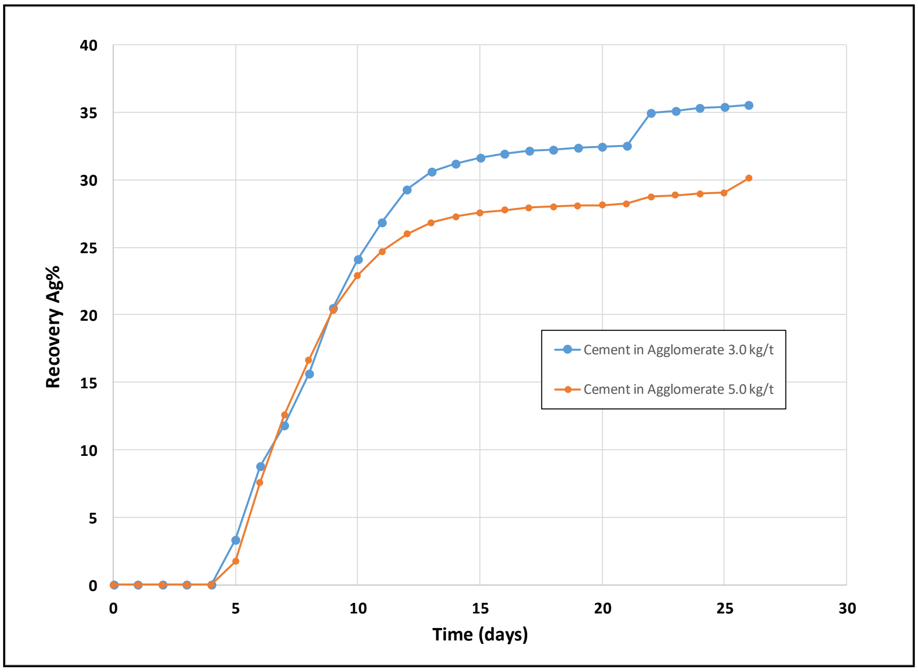


Figure 13-15:
Silver Dissolution Kinetics for the Oxide Sample



13.6 MINERAL METALLURGICAL EVALUATION BY METTS (MAY 2020)

Minera Quinchia S.A.S. commissioned Metallurgical Testing Services (METTS) to carry out metallurgical bottle leaching tests for gold ore and evaluate the behavior of a precious mineral solvent similar to sodium cyanide called "Jinchan", and that can be compared with that sodium cyanide solvent to establish gold and silver recovery rates and consumption of the reagent and soda. The results of this test can be seen in Table 13-20.

Equivalent results between both reagents, "Jinchan" versus sodium cyanide, were obtained, both in the coarse particle size (100% - #10) and in the fine particle size (100% - #200). In this sense, recoveries for Au were between 77% to 79.4%, while for Ag were between 70% to 72%, in both cases, Jinchan and sodium cyanide (see Figure 13-16 and Figure 13-17).

Regarding consumption, in the case of sodium cyanide it was 2.49 kg/t to 2.55 kg/t while for Jinchan it was 2.21 kg/t to 2.50 kg/t, coincidentally, the highest consumptions are reached in the fine grain size treatment. Finally, soda consumptions ranged from 3.0 kg/t to 3.95 kg/t, probably associated with sulfate components in the ore.

Table 13-20:
Leaching Test Results for Gold & Silver

Test Code	Head Grades				Grade of dissolved in PLS		Tailing Grade		Reagent consumption (kg/t)			Recovery (%)	
	Assayed		Calculated		Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Cyanide (NaCN)	Jinchan	Soda	Au	Ag
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)									
CN -10#	1.522	0.50	1.72	0.53	1.225	0.38	0.496	0.15	2.49	0	2.95	79.36	71.67
JC - 10#			1.81	0.53	1.341	0.38	0.469	0.15	0	2.21	3.00	77.37	71.60
CN -200#			1.82	0.51	1.200	0.36	0.616	0.15	2.55	0	3.45	78.05	70.86
JC -200#			1.76	0.55	1.266	0.40	0.496	0.15	0	2.50	3.95	77.08	72.82

Figure 13-16:
Gold Dissolution Kinetics Curves for Leaching Tests

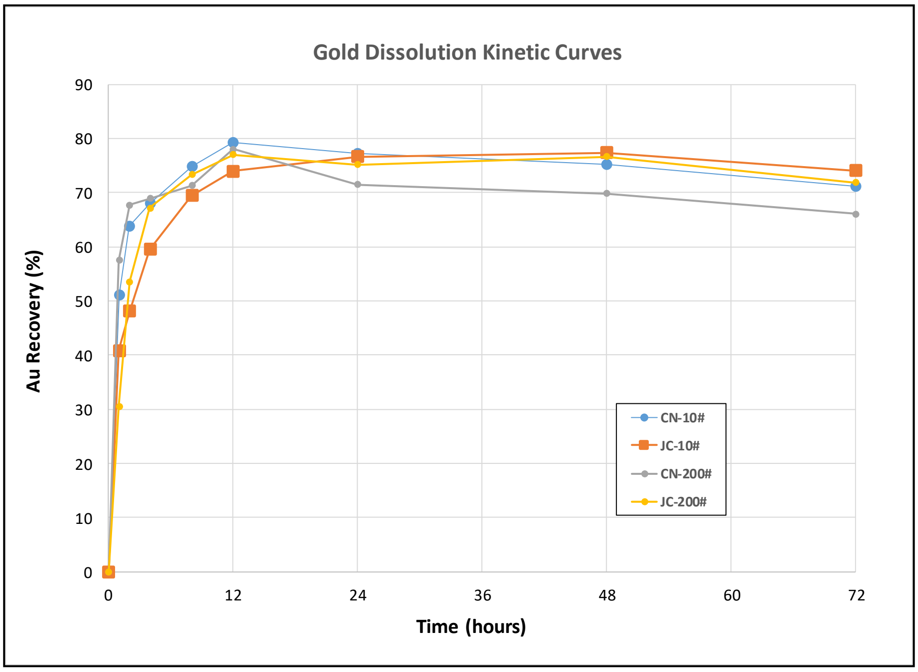
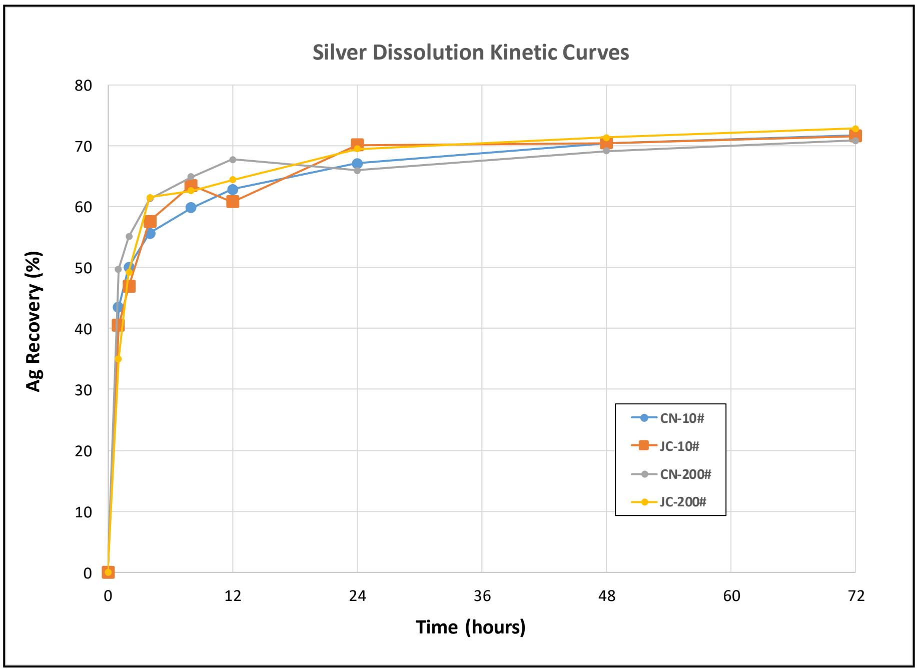


Figure 13-17:
Silver Dissolution Kinetics Curves for Leaching Tests



13.7 MINERAL METALLURGICAL EVALUATION BY METTS (JUNE 2020)

Minera Quinchia S.A.S. commissioned Metallurgical Testing Services (METTS) to perform bottle leach metallurgical tests on 18 samples from the Oxide, Transition and Primary zones, comparing both Au, Ag and Cu recovery yields, as well as soda consumption of the solvents, sodium cyanide and Jinchan.

13.7.1 Initial Conditions

Twelve samples were received from the zones of the deposit denominated TR, TR2, TR3 and TR4. The preparation of six composite samples, which are: two samples from the TR code samples, two samples from the TR2, TR3 and TR4 code samples; and two samples of Oxides totaling 18 samples.

- Grain size: 100% - #10
- PH: 10.5
- Solvent concentration 1: 0.1% of NaCN
- Solvent concentration 2: 0.1% of JINCHAN
- Controls in the test: c/24 hours PH, [CN-] and final aliquot of the test
- % Solids: 30%.
- Leaching time: 72 hours

13.7.2 Results

Table 13-21, Table 13-22 and Table 13-23 show the results obtained for the Oxide, Transition and Primary zones, respectively, using bottle roll metallurgical tests.

Table 13-21:
Results for Au, Ag & Cu from Bottle Roll Tests

Sample	Head Grade (g/t, %)						Solution (mg/L)			Residual (g/t, %)			%Extraction		
	Assayed			Calculated											
	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu
TR - ZA	0.929	2.26	0.129	0.798	1.844	0.132	0.200	0.576	223.7	0.331	0.5	0.080	58.5%	59.5%	39.5%
TR - ZM	0.635	2.82	0.183	0.306	2.059	0.193	0.050	0.668	271.9	0.189	0.5	0.130	38.2%	55.3%	32.8%
TR -ZB	1.800	2.780	0.273	1.165	2.836	0.317	0.204	0.744	441.2	0.689	1.1	0.214	40.9%	62.4%	32.5%
TR2 - ZA	0.869	1.31	0.0873	0.643	1.214	0.083	0.190	0.306	27.39	0.200	0.5	0.077	68.9%	54.5%	7.7%
TR2 - ZM	0.589	1.6	0.0924	0.447	1.870	0.096	0.137	0.33	35.56	0.127	1.1	0.088	71.6%	48.1%	8.6%
TR2 - ZB	2.040	2.940	0.161	1.764	2.840	0.178	0.469	0.703	73.26	0.67	1.2	0.161	62.0%	55.8%	9.6%
TR3 - ZA	1.540	2.830	0.208	1.495	1.837	0.209	0.431	0.573	72.92	0.489	0.5	0.192	67.3%	47.2%	8.1%
TR3 - ZM	0.596	3.02	0.108	0.495	2.012	0.113	0.099	0.391	65.78	0.264	1.1	0.098	46.7%	30.2%	13.5%
TR3 - ZB	1.490	3.020	0.234	1.215	2.101	0.232	0.339	0.686	195.5	0.424	0.5	0.186	65.1%	53.0%	19.7%
TR4 - ZA	1.295	1.37	0.185	1.064	1.725	0.172	0.227	0.225	89.15	0.534	1.2	0.151	49.8%	38.3%	12.1%
TR4 - ZM	0.767	1.52	0.0669	0.596	1.356	0.058	0.151	0.367	74.83	0.244	0.5	0.041	59.1%	56.3%	29.9%
TR4 - ZB	1.210	2.46	0.154	1.191	2.713	0.183	0.356	0.82	159.0	0.36	0.8	0.146	69.8%	77.8%	20.3%
TR - C - CN	1.033	2.80	0.295	0.816	3.414	0.229	0.225	0.649	302.2	0.291	1.9	0.158	64.3%	54.1%	30.9%
TR - C- JC	1.033	2.80	0.205	0.621	0.740	0.178	0.066	0.103	213.6	0.467	0.5	0.128	24.8%	8.6%	28.0%
TR234 - CN	1.125	2.00	0.155	0.935	2.662	0.159	0.266	0.498	103.5	0.314	1.5	0.135	66.4%	58.1%	15.2%
TR234 - JC	1.125	2.00	0.155	0.995	1.573	0.139	0.239	0.46	82.47	0.437	0.5	0.12	56.1%	53.7%	13.8%
OX - C- CN	1.301	1.00	0.103	1.068	1.987	0.099	0.433	0.123	18.15	0.058	1.7	0.095	94.6%	28.7%	4.3%
OX -C- JC	1.301	1.00	0.103	1.266	1.178	0.101	0.517	0.119	22.37	0.06	0.9	0.096	95.3%	27.8%	5.2%

Table 13-22:
Results of Reagent Consumption in the Cyanidation in Bottles Test

Sample	Consumptions (kg/t)	
	NaCN	Soda
TR - ZA	5.28	2.57
TR - ZM	5.20	1.75
TR -ZB	6.63	2.27
TR2 - ZA	3.51	0.93
TR2 - ZM	3.57	1.08
TR2 - ZB	4.07	1.37
TR3 - ZA	4.60	1.56
TR3 - ZM	3.92	1.52
TR3 - ZB	4.80	1.08
TR4 - ZA	4.22	1.15
TR4 - ZM	5.81	1.91
TR4 - ZB	4.46	0.91
TR - C	5.59	1.85
TR234 - C	4.23	1.45
OX -C	4.77	2.59

Table 13-23:
Results of Reagent Consumption in Leach Bottle Tests

Sample	Consumptions (kg/t)	
	Jinchan	Soda
TR - C	5.33	2.32
TR234 - C	5.00	1.39
OX -C	4.29	3.25

The ranges of dissolution values for Au between the Transition (TR) and Primary (TR2, TR3 and TR4) ore types vary between 40% to 70%, not showing homogeneity (see Figure 13-18).

Whereas, the ranges of dissolution values for Ag show certain uniformity for the Transition ore type (TR), and those samples from the Primary ore type (TR2), but in the case of the samples from the Primary ore type (TR4) there is an increase as it goes from High to Low zone (see Figure 13-19).

Figure 13-18:
Gold Recoveries from Bottle Roll Tests

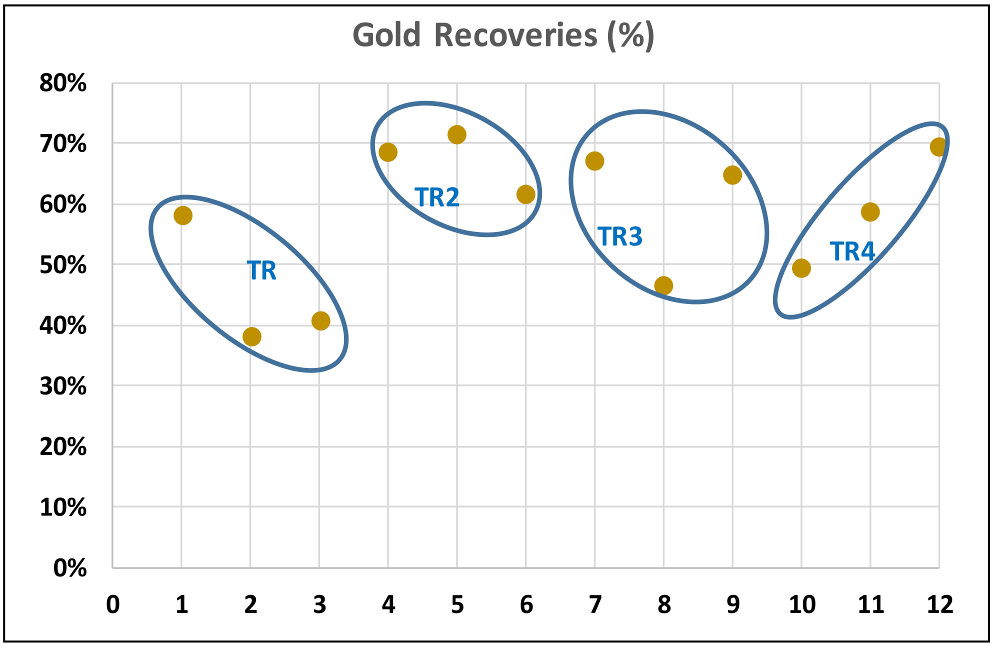
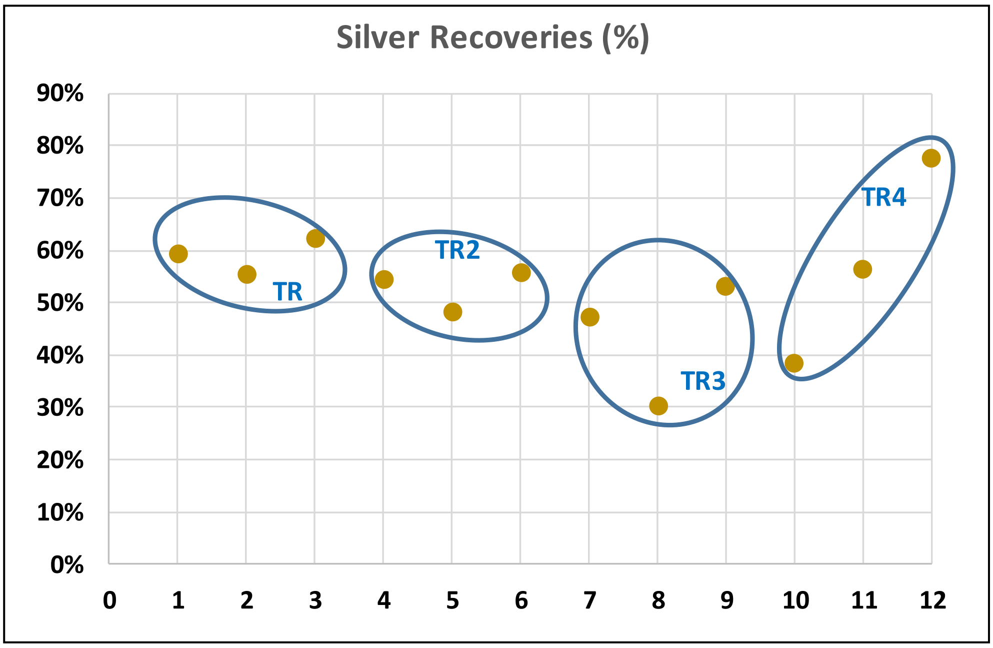


Figure 13-19:
Silver Recoveries from Bottle Roll Tests



The results of the tests show that the Au dissolutions vary between 40% and 70% in the different zones, while for Ag, they present uniform values in the Transition zones (samples TR and TR2) and in the primary zones (samples TR3 and TR4), with positive variations as the sample is taken deeper. Similarly, Cu dissolutions in the Transition zone reach values between 30% and 40, while in the Primary zone these values can drop to 5%.

The comparison between cyanide and Jinchan reagent shows that, in the Oxide Zone, Au dissolution values between 94% and 95% were obtained for both reagents. Likewise, in the

Transition and Primary zones, the dissolution with cyanide reached up to 64% on average, while, using Jinchan, dissolutions of 24% and 56% were reached in both zones, respectively.

High cyanide and soda consumption (3.5 kg/t to 6.5 kg/t) associated with the occurrence of cyanide elements (copper, arsenic, iron, etc.) present in the ore.

13.8 MINERAL METALLURGICAL EVALUATION BY METTS (JULY 2020)

Minera Quinchia S.A.S. commissioned Metallurgical Testing Services (METTS), to obtain pregnant solutions with gold, silver and copper contents. Gold leaching tests were carried out with cyanide and Jinchan (an alternative reagent to cyanide that dissolves metals such as gold, silver and copper).

For this purpose, two rolling bottle tests were carried out with coarse granulometry Batero ore (P100 - #10) and with 25% solids for both cyanide leaching and Jinchan leaching to obtain pregnant solutions of gold, silver and copper; these solutions will undergo adsorption tests with activated carbon to see the effect of the silver and copper elements in the adsorption process.

13.8.1 Initial Conditions for Bottle Roll Tests

Twelve samples were received from the zones of the deposit denominated TR, TR2, TR3 and TR4. The preparation of six composite samples, which are: two samples from the TR code samples, two samples from the TR2, TR3 and TR4 code samples; and two samples of Oxides totaling 18 samples.

13.8.2 Chemical Analysis of Head Ore

Table 13-24 shows the Au, Ag and Cu head grades obtained by chemical analysis.

Table 13-24:
Chemical Assay of Head Oxide Ore

Sample	Au (g/t)	Ag (g/t)	Cu (%)
Cyanide solution (mg/L) (*)	1.30	1.0	0.103

13.8.3 Cyanidation in Bottle Roll Tests

Prior to the cyanidation tests, alkalinity tests were carried out to determine the consumption of caustic soda, this is achieved by adding 0.1, 0.2 and 0.5 grams of caustic soda to the mineral until reaching a pH of 10.5.

The test conditions were as follows:

- Sample weight: 2 kg
- Grain size: 100% - #10
- pH: 10.5
- Solvent concentration 1: 0.1% of NaCN
- Solvent concentration 2: 0.1% of JINCHAN
- Controls in the test: c/24 hours PH, [CN-] and final aliquot of the test
- % Solids: 25%.
- Leaching time: 72 hours

Once the cyanidation tests in bottles are finished, the solutions rich in gold and silver are filtered and the pulp is washed with water. The results are used to determine the maximum extraction of gold and silver and their respective reagent consumption (cyanide, Jinchan and soda).

13.8.4 Experimental Results

Table 13-25 shows the results for the bottle roll leach tests for the elements Au, Ag and Cu.

Table 13-25:
Results for Au, Ag & Cu from Bottle Roll Tests

Sample	Head Grade (g/t, %)						Solution (mg/L)			Residual (g/t, %)			%Recovery		
	Assayed			Calculated											
	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu
BAT - CN	1.301	1.00	0.1	1.678	1.367	0.097	0.469	0.089	13.52	0.271	1.10	0.093	83.85	19.53	4.18
BAT - JC				1.698	1.170	0.101	0.466	0.090	16.10	0.300	0.90	0.096	82.33	23.08	4.79

13.8.5 Sodium Cyanide, Jinchan and Soda Reagents Consumption

The reagent consumptions: cyanide, Jinchan and soda reported in kg/t, are given in in Table 13-26. Reagent consumption controls were performed every 24, 48 and 72 hours. The consumption of sodium cyanide and Jinchan for the oxide zone composites show similar values between 4.21 and 4.78 kg/t as shown in Table 13-26. The soda consumptions are always high as in previous evaluations probably due to the presence of sulfates in the oxide zone.

Table 13-26:
Results of Reagent Consumption in Bottle Roll Tests

Sample	Consumptions (kg/t)		
	Cyanide	Jinchan	Soda
BAT-CN	4.21	0.0	2.40
BAT-JC	0.0	4.78	2.40

13.8.6 Activated carbon adsorption testing of leaching liquors

The liquors resulting from the leaching tests were analyzed to adsorption tests with activated carbon at the following conditions:

- Liquor volume: 1 liter
- Solvent strength: 1000 mg/l
- PH: >10
- Weight of virgin carbon: 5 grams
- Aliquot volume: 30 ml
- Aliquot run: 5, 10, 15, 15, 30, 60, 60, 120, 120, 240, 240, 360, 720, 1440 minutes
- Test time: 24 hours

The recovery of gold and silver using the activated carbon adsorption method for both cyanide and Jinchan liquors and the resulting kinetic curves for each test are shown in Figure 13-20 and Figure 13-21.

Figure 13-20:
Kinetic Curves of Adsorption of Au, Ag & Cu in Cyanide Solutions

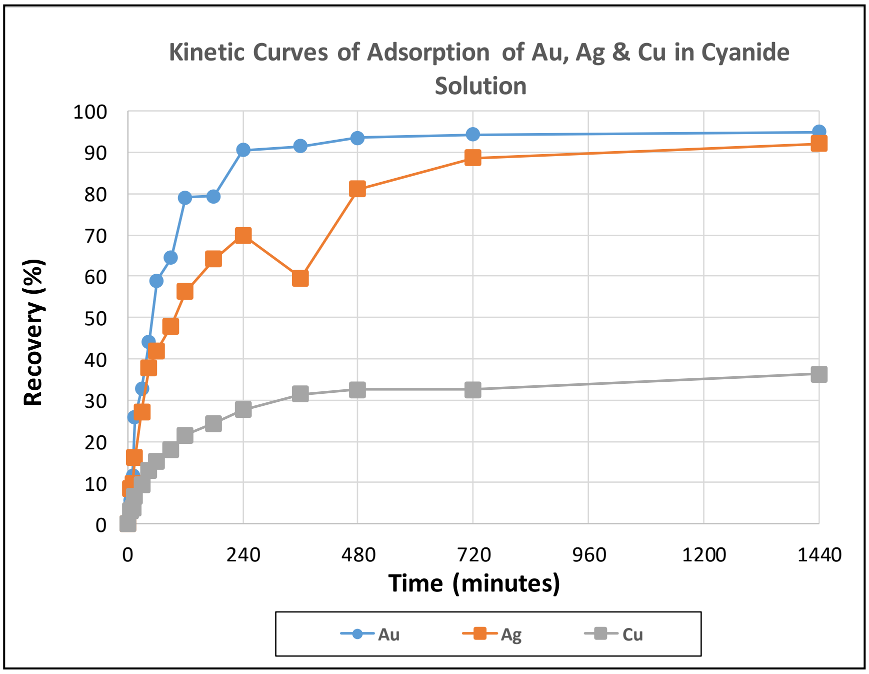
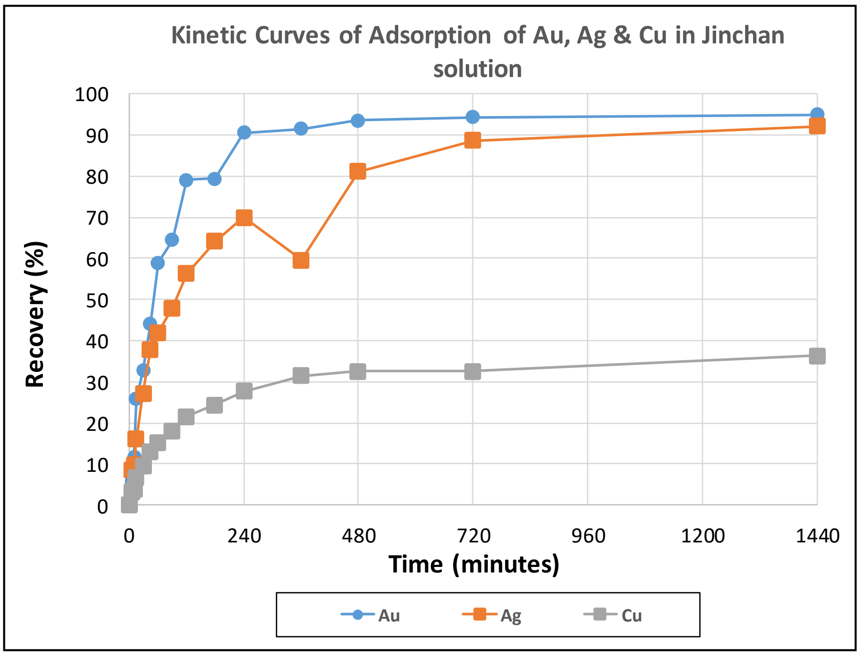


Figure 13-21:
Kinetic Curves of Adsorption of Au, Ag & Cu in Jinchan Solutions



13.8.7 Conclusions

1. Au recoveries in the oxide zone using cyanide and Jinchan solvents were lower than those obtained in the previous study (82% vs 94%: cyanide; 83% vs 95%: Jinchan, respectively) associated with a "pregnant-robbing" effect due to the effect of the clays, which could be avoided by taking aliquots throughout the test.
2. Cyanide and Jinchan reagent consumptions were in the range of 4.2 kg/t to 4.7 kg/t, due to the content of cyanide elements (copper, iron, etc.) in the ore.
3. The cyanide and Jinchan solvents in the Au adsorption process with activated carbon gave similar performances for the extraction and adsorption of gold for the oxide zone.

13.9 MICROSCOPY STUDY BY BIZALAB (JULY 2020)

Minera Quinchia requested BIZALAB to carry out a mineralogical study by optical microscopy and X-ray diffraction of twelve samples. For this purpose, BIZALAB used two techniques such as X Diffraction for non-metallic minerals and Reflected Light Optical Microscopy for metallic minerals.

13.9.1 X-Ray Diffraction

The X-ray diffraction test allows us to see the types of non-metallic minerals or gangue present in the sample. The twelve samples tested have quartz, plagioclase and feldspar as the main gangue. Likewise, phyllosilicate-type minerals were identified, which can cause negative effects of pregnant-robbing or pregnant-borrowing during gold extraction by cyanidation. Table 13-27 shows the quantification of the main phyllosilicates identified in the samples analyzed.

Table 13-27:
Critical Non-Metallic Mineralogy (Phyllosilicates)

Phyllosilicates	ZA	ZM	ZB
TR	chlorite (7%); laminar mineral (5%); muscovite (2%)	chlorite (8%); biotite (2%)	chlorite (7%); muscovite (8%)
TR2	chlorite (4%); muscovite (<LDL)	laminar mineral (3%); chlorite (5%); biotite (<LDL)	chlorite (7%); biotite (6%)
TR3	chlorite (8%); laminar mineral (5%); muscovite (7%); pyrophyllite (5%); kaolinite (3%)	chlorite (6%); laminar mineral (7%); muscovite (4%)	chlorite (5%); muscovite (4%)
TR4	pyrophyllite (6%); laminar mineral (5%); kaolinite (2%)	chlorite (8%); laminar mineral (13%) muscovite (19%)	chlorite (9%); muscovite (7%)

LDL: Lower detection Limit

13.9.2 Reflected Light Optical Microscopy

The metallic mineralogy is in trace proportions being the main minerals: pyrite, magnetite and chalcopyrite. Likewise, cyanicide minerals (cyanide-consuming elements) were identified such as secondary sulfides (covellite, digenite and bornite) and pyrrhotite. While minerals associated to "impurities" we have minerals of the gray copper family and galena. Table 13-28 shows the critical metallic mineralogy of the analyzed samples. The presence of visible gold was not detected in the samples tested.

Table 13-28:
Critical Metallic Mineralogy (Impurities)

Phyllosilicates	ZA	ZM	ZB
TR	covellite	covellite, digenite	covellite, bornite
TR2	covellite		
TR3	pyrrhotite, covellite, gray copper	covellite, digenite, bornite	
TR4		pyrrhotite, gray copper	covellite, digenite

13.10 MINERAL METALLURGICAL EVALUATION BY METTS (AUGUST 2020)

Minera Quinchia requested METTS to evaluate Au and Ag recoveries by precipitation with zinc powder (Merrill & Crowe Process) changing the molar ratio of Zn/Au and the dosage of lead salts (Acetate). The PLS solutions were obtained from bottle tests performed on oxidized ore, where liquors or cyanide solutions of Au and Ag were obtained by means of adsorption processes with activated carbon and precipitation with zinc powder, analyzing the technical viability of being able to be used at industrial scale.

The rich solutions from bottle tests on Oxide ore contain the values shown in Table 13-29.

Table 13-29:
Chemical Analysis of Solutions

Product	Au	Ag	Cu
Cyanide solution (mg/L) (*)	0.469	0.089	13.52

13.10.1 Test Results

Merrill & Crowe Process exploratory tests were encouraging, precipitating Au and Ag values from PLS solutions of intermediate grades (<1 mg/L) with the addition of zinc powder and lead salts in weight ratios between values around 100, for the evaluated conditions in the tests. It should be pointed out that zinc dosages would be reduced if gold grades are increased to values higher than 1 to 2 mg/L, increasing the capacity of zinc to precipitate gold and silver values, see Table 13-30 and Figure 13-22 for results.

Otherwise, at low to intermediate grades, precipitation is accomplished with higher zinc powder consumption, as obtained in the study.

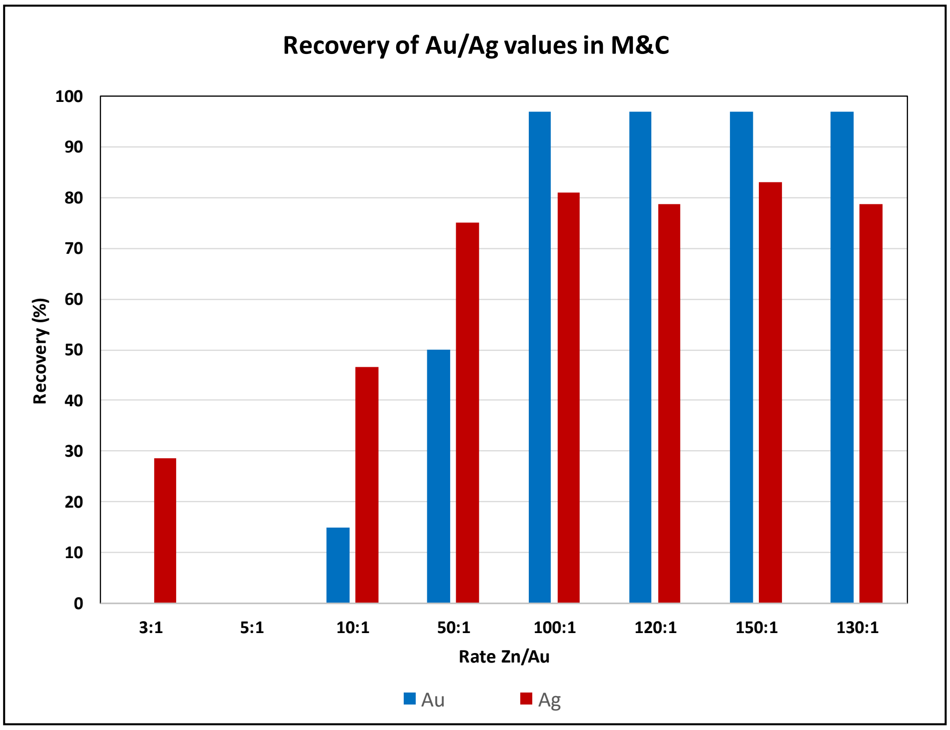
The addition of cyanide occurs when the solutions have low concentrations, in the case of the PLS of the La Cumbre Project have about 1000 ppm it is not required to add more cyanide, all the more that the copper grades are lower than 20 ppm.

Table 13-30:
Chemical Analysis of Solutions Test Product

Test	Description	Solution Grades (ppm)				Recovery (%)	
		Pregnant		Barren			
		Au	Ag	Au	Ag	Au	Ag
La Cumbre	Pregnant solution	0.443	0.137				
Zinc Dosage Tests							
Test 1	Free CN solution (3:1): Zinc powder (3:1): Lead acetate (0.5:1)	0.443	0.137	0.463	0.098	0	28.5
Test 2	Free CN solution (3:1): Zinc powder (5:1): Lead acetate (0.5:1)	0.443	0.137	0.553	0.213	0	0
Test 3	Free CN solution (3:1): Zinc powder (10:1): Lead acetate (0.5:1)	0.443	0.137	0.377	0.073	14.9	46.7
Test 4	Free CN solution (3:1): Zinc powder (50:1): Lead acetate (0.5:1)	0.443	0.137	0.221	0.034	50.1	75.2
Test 5	Free CN solution (3:1): Zinc powder (100:1): Lead acetate (0.5:1)	0.443	0.137	<0.013	0.026	97.1	81.0
Test 6	Free CN solution (3:1): Zinc powder (120:1): Lead acetate (0.5:1)	0.443	0.137	<0.013	0.029	97.1	78.8
Test 7	Free CN solution (3:1): Zinc powder (150:1): Lead acetate (0.5:1)	0.443	0.137	<0.013	0.023	97.1	83.2

Test 8	Free CN solution (3:1): Zinc powder (130:1): Lead acetate (0.5:1)	0.443	0.137	<0.013	0.029	97.1	78.8

Figure 13-22:
Recovery of Gold and Silver Values in M&C



13.10.2 Conclusions

M&C of the tests of precipitation with zinc powder to an alkaline cyanide solution product of the leaching of the oxide ore La Cumbre Project with addition of lead acetate in the ratio of 0.5:1 (Pb /Zn), it is summarized that:

- A Zn:Au weight ratio of 3 to 150 yields good precipitation from the 100:1 molar ratio, having to point out the requirement in excess zinc dosage to attain good precipitation of Au values in the zinc powder.
- An optimum zinc dosage for the conditions stated in the tests is between Zn: Au ratio values of 50 to 100, but these values can still be optimized in function of the Au grades and the other process variables.

13.11 MINERAL METALLURGICAL EVALUATION BY METTS (FEBRUARY 2021)

Minera Quinchia requested METTS to perform a series of metallurgical tests on two types of ores called "Oxides" and "Mixed" (70%Ox/30%Tx). These tests included cement addition rate in the agglomeration, leaching rate (l/h/m2), cyanide concentration in the leaching solution (ppm), required granulometry and granulometry to be agglomerated. Also, the dissolution of the copper element in the leaching of both types of ore (>0.1% Cu) analyzed and monitored both in the bottle tests and in the column tests. Finally, the calculation of the Work Index of both types of ore (Oxides & Mixed) was carried out.

13.11.1 Experimental Tests on Oxide Ores

A total of 2,410 kg of which 1730 kg was Oxide Ore and 680 kg was Transition Ore. This material was homogenized and each type of mineral (Oxidized and Transition) was successively quartered to obtain the weights for each type of metallurgical evaluation to be carried out.

- Samples for Physical and Chemical characterization for Oxide and Transitional ores.
- Samples for granulometric analysis of the samples (for Ox and Tx)
- Samples for Work Index tests (for Ox and Tx)
- Samples for bottle testing (Mixed: 70%Ox/30%Tx)

13.11.2 Physical and Chemical Characterization

The sample taken for chemical analysis of the head ore following the sampling protocol was analyzed for 32 elements, including Au and Ag. The chemical analyses were performed at the Certimin laboratory in Lima, Peru by the ICP method and the results are shown in Table 13-31. The physical parameters of the head ore sample from the oxide zone are shown in Table 13-32.

Table 13-31:
Sample Assay for Different Ore Zones

Description	Au (g/t)	Ag (g/t)	Cu (%)
Oxide Ore	1.1381	0.40	0.104
Transition Ore	1.345	3.0	0.108
Composite Ore: Mixed (70%Ox/30%Tx)	1.405	1.4	0.103

Table 13-32:
Physical Parameters for Oxide Head Ore Sample

Description	Oxides	Transition	Composite
Humidity (%)	27.48	12.56	14.0
Specific Gravity (g/cm^3) *	-	-	2.59
Natural pH	-	-	5.1

*Obtaining by using pycnometer method

13.11.3 Grain-Size Characterization of Head Ore

Sieving of the composited sample through sieves (Tyler): 2”, 1", ½", ¼" #10, all retained fractions were analyzed for gold and silver, the results of which are shown in Table 13-33 and Table 13-34, for oxide and transitional ores respectively.

Table 13-33:
Assayed Sieve Size Analysis of the Oxide Head Samples

Tyler Mesh	Weight (kg)	Weight Distr. %	Cumulative Retained weight	Accum. through weight	Assay			Distribution (%)		
					Au (g/t)	Ag (g/t)	Cu (%)	Au	Ag	Cu
2	0.88	2.0%	2.0%	98.0%	0.87	0.4	0.12	1.4	1	2
1	1.6	3.7%	5.8%	94.2%	1.14	0.7	0.12	3.4	4	4
1/2	2.11	4.9%	10.7%	89.3%	1.12	0.5	0.11	4.5	3	5
1/4	4.17	9.7%	20.4%	79.6%	1.34	0.8	0.10	10.5	11	9
10 m	9.95	23.2%	43.7%	56.3%	1.39	0.8	0.10	26.0	26	22
-10 m	24.10	56.3%	100.0%	0.0%	1.20	0.7	0.10	54.2	55	57
	42.80	100.0%	Calculated Head Grade		1.243	0.717	0.103	100.0	100	100
			Assayed Head Grade		1.38	0.4	0.104			

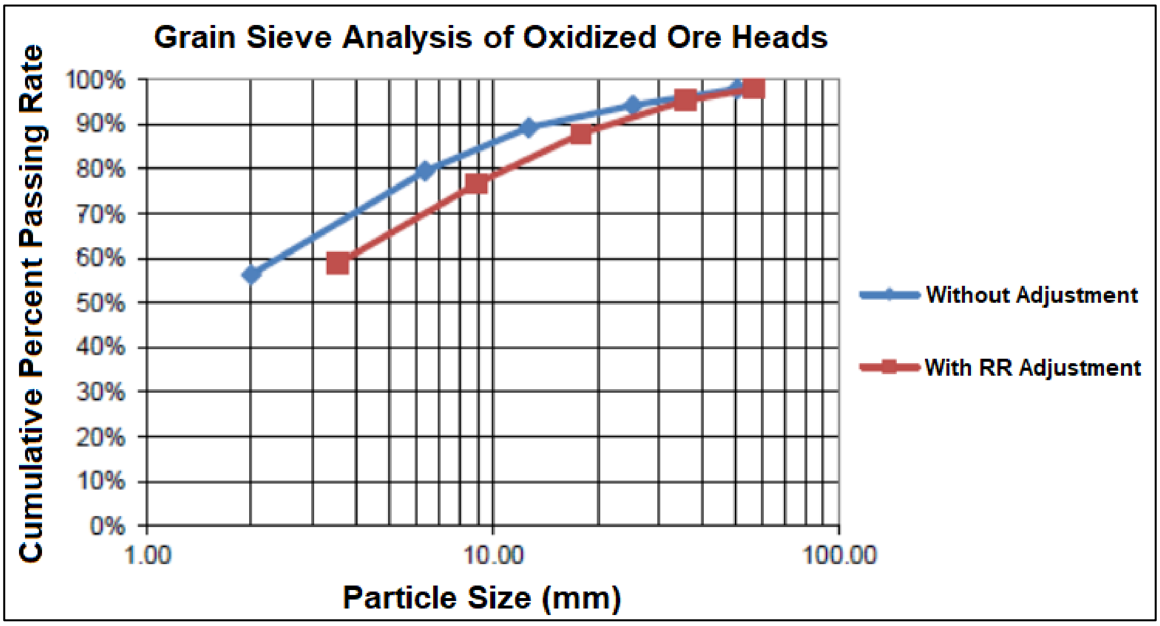
Table 13-34:
Assayed Sieve Size Analysis of the Transition Head Samples

Tyler Mesh	Weight (kg)	Weight Distr., %	Cumulative Retained weight	Accum. through weight	Assay			Distribution (%)		
					Au (g/t)	Ag (g/t)	Cu (%)	Au	Ag	Cu
2.5	5.00	13.4%	13.4%	86.6%	1.13	0.7	0.14	11.1	3	18
2	6.90	18.5%	31.9%	68.1%	1.19	3.3	0.12	16.2	20	21
1	13.2	35.4%	67.2%	32.8%	1.43	3.5	0.11	37.2	40	36
1/2	3.4	9.1%	76.4%	23.6%	1.48	4.8	0.08	9.9	14	6
1/4	1.99	5.3%	81.7%	18.3%	1.49	4.2	0.08	5.9	7	4
10 m	2.26	6.0%	87.7%	87.7%	1.55	3.7	0.08	6.9	7	4
-10 m	4.58	12.3%	100.0%	100.0%	1.42	2	0.10	12.8	8	11
	37.33	100.0%	Calculated Head Grade		1.36	3.1	0.11	100.0	100	100
			Assayed Head Grade		1.35	3.0	0.11			

The calculated head grade results for Au, Ag and Cu for the Oxide and Transition ores are similar to the assayed head grades shown in Table 13-33 and Table 13-34.

From the previous granulometric curve we can indicate that the P80 adjusted by R-R correlation is approximately 10,823 microns and that there is a 60% presence of fines below 3.5 mm. The distribution of gold through the particle sizes of the ore is shown in the Figure 13-23.

Figure 13-23:
Oxide Ore Head Grain Size Curve



From the granulometric curve in Figure 13-23, the adjusted P80 using the R-R correlation is approximately 10,823 microns with 60% of fines below 3.5 mm. Figure 13-24 illustrate the distribution of gold through the particle sizes of the ore which is obtained, while Figure 13-25 and Figure 13-26 are shown the distributions for silver and copper respectively.

Figure 13-24:
Distribution of Gold through Mineral Particle Size

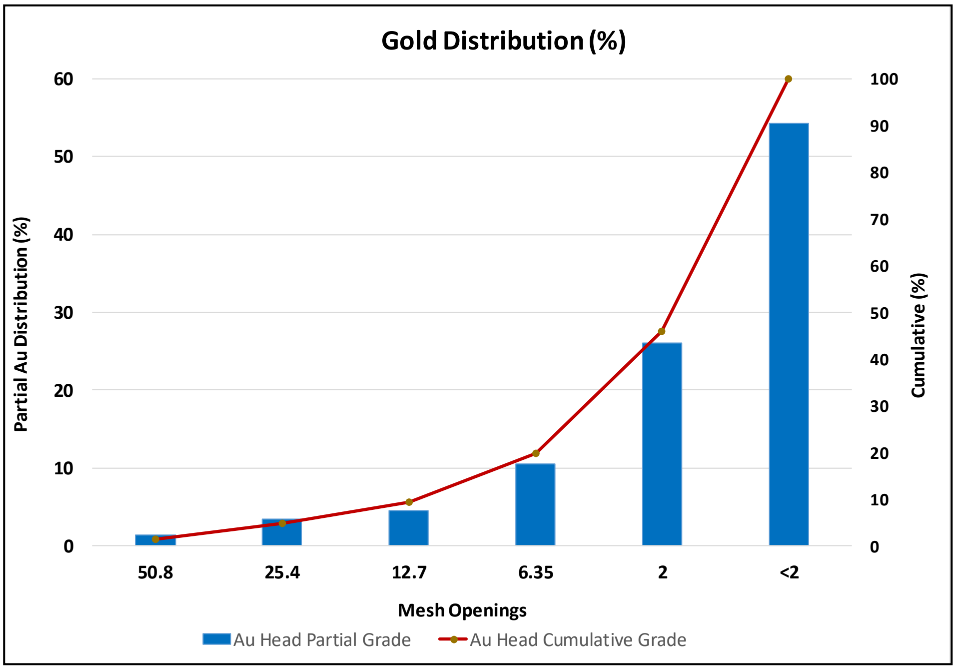


Figure 13-25:
Distribution of Silver through Mineral Particle Size

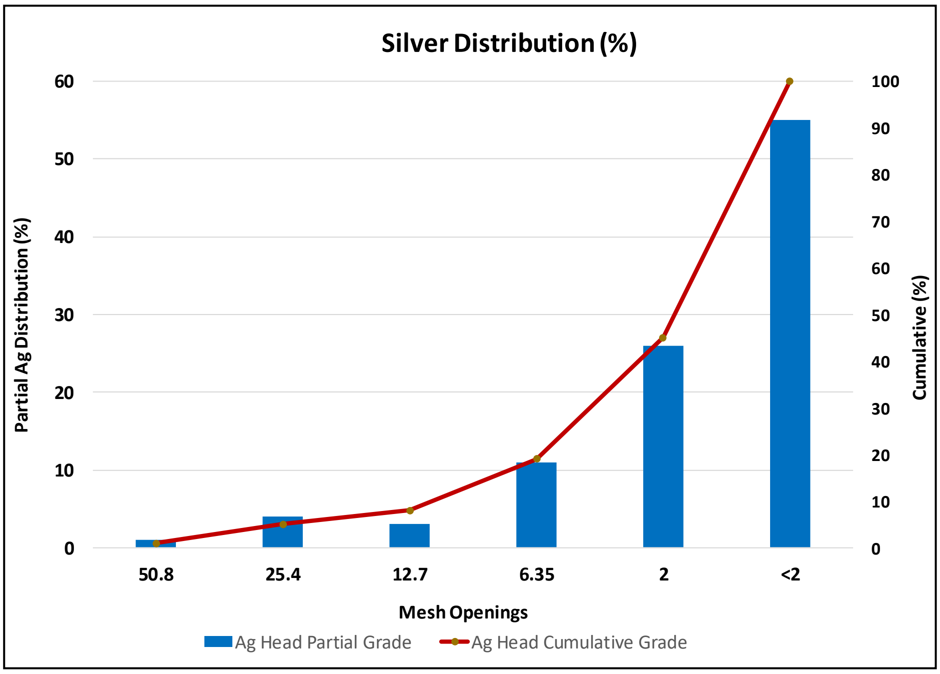
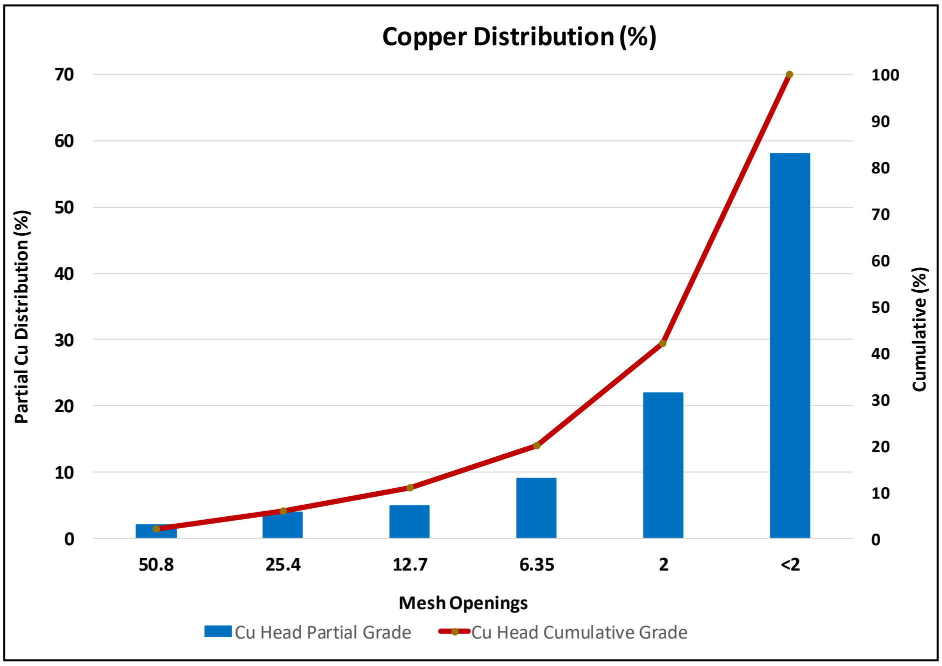
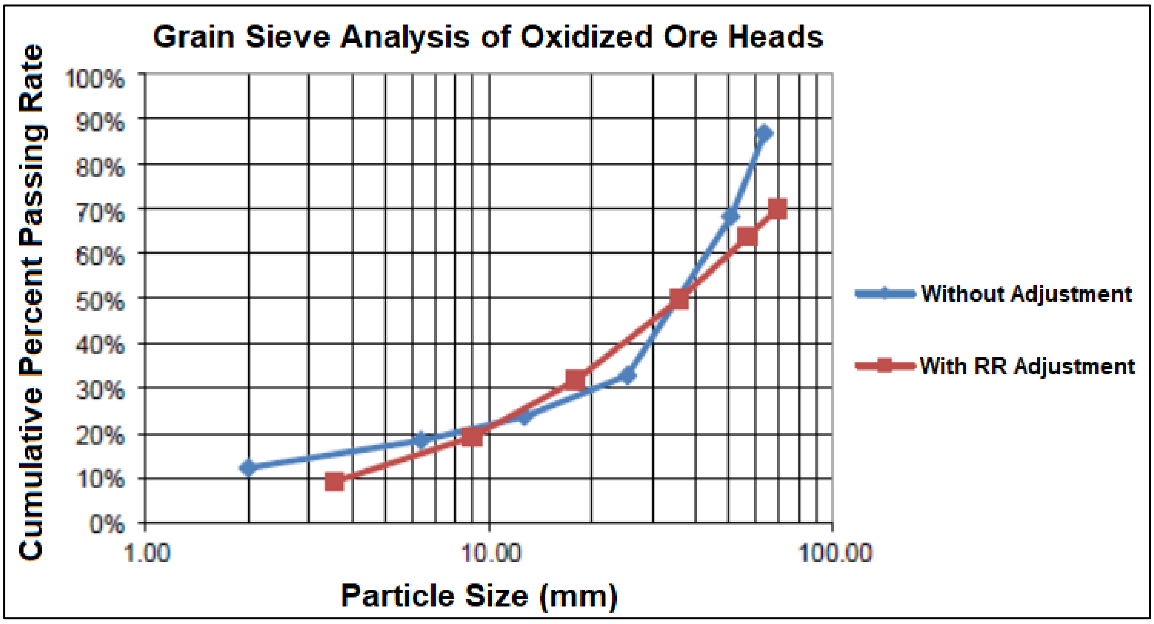


**Figure 13-26:
Distribution of Copper through Mineral Particle Size**



Based on the graded particle size distribution of the oxide ore for the gold, silver and copper, the contents of these elements increase as the particle size is reduced, indicating that 54% of the Au, 55% of the Ag and 57% of the Cu is less #10. Figure 13-27 shows the particle size versus cumulative passing percentage (%) granulometric curve.

**Figure 13-27:
Transition Ore Head Grain Size Curve**



Using the R-R function results in the curve adjustment, with P80 being 98.19 mm (approx. 4 inches). In Figure 13-28, Figure 13-29 and Figure 13-30 shows the distribution of gold, silver and copper respectively, across the ore particle sizes.

**Figure 13-28:
Distribution of Gold through Mineral Particle Size**

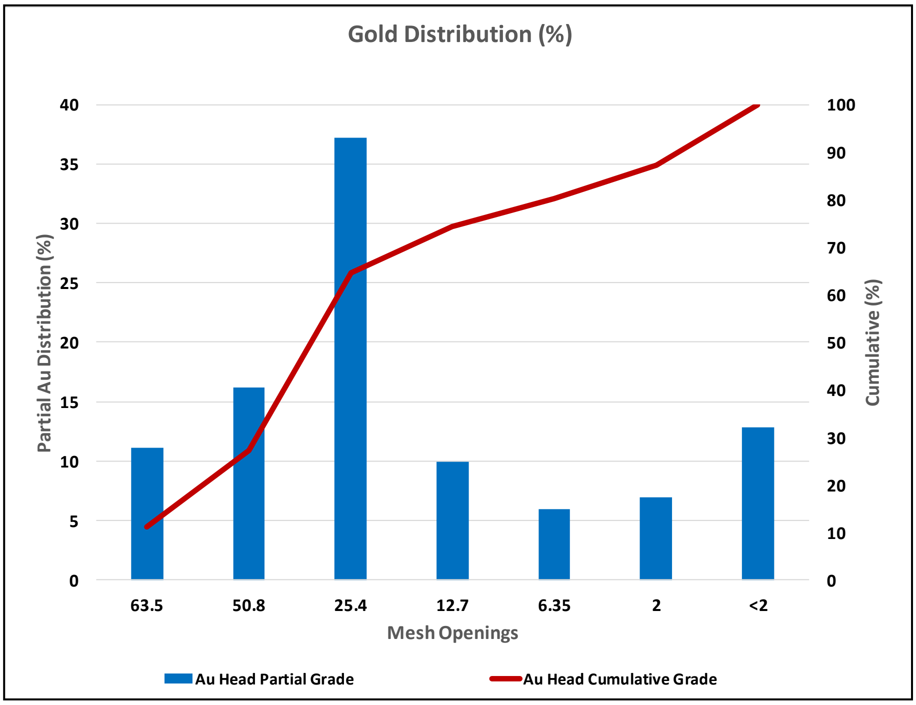


**Figure 13-29:
Distribution of Silver through Mineral Particle Size**

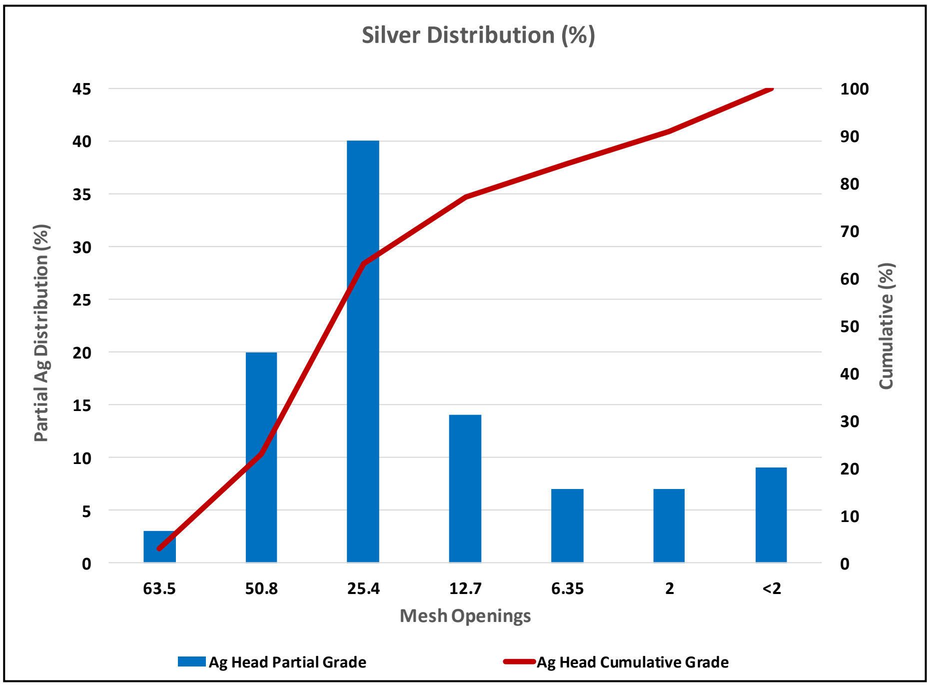
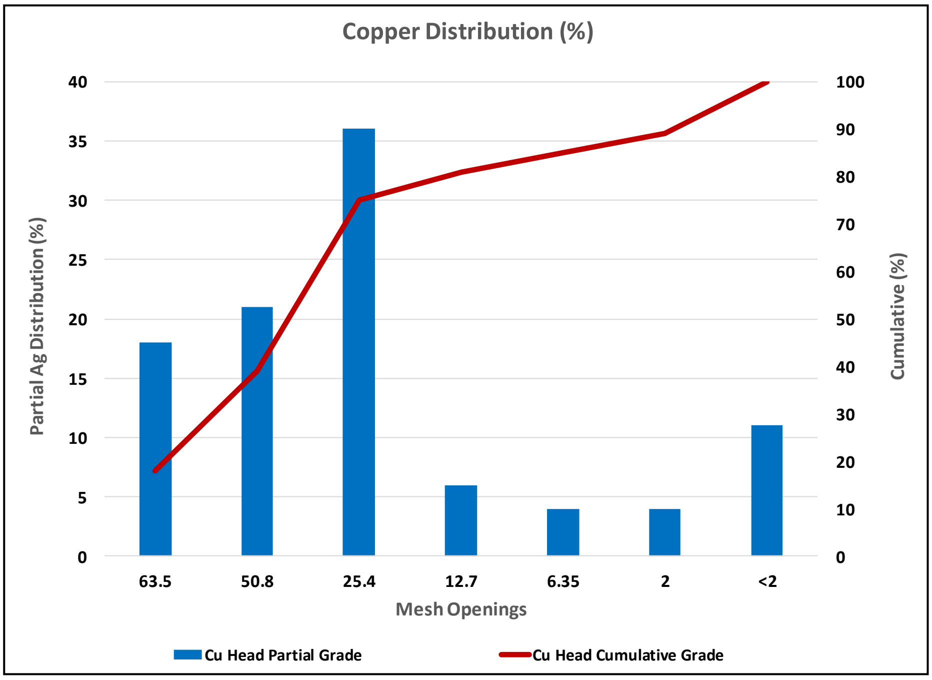


Figure 13-30:
Distribution of Copper through Mineral Particle Size



The granulometric analysis of the sample from the transition zone shows distributions of gold, silver and copper close to 64%, 63% and 75% for the coarse sizes (larger than 1"). It can be affirmed that most of the values are in larger mesh sizes and would have to be reduced in size to improve the exposure of the mineral to the cyanide action.

13.11.4 Work Index Tests of the Head Ore

As mentioned above (Section 13.4.1.1), composite samples from the Oxide and Transition zones were used to calculate the Work Index by the Bond method.

The results in the calculation of the Work Index by the Bond method for the Oxide and Transition ores are 8.06 and 8.26 Kwh/tc respectively, corresponding to a relatively soft ore according to the METSO scale for Iron and Magnetite ores, see Table 13-35 and Table 13-36.

Table 13-35:
Results of the Bond Index Calculation Test for Oxide Ore

Item	Value
Cutting Mesh, (Tyler)	100
Cutting Mesh, (µm)	150
Sample weight (700 cm3), (g)	912.87
%-150 µm in feeding	35.5
Weight for 250% circular load, (g)	260.82
Grindability, (g/rev)	2.54
F_{80}, (µm)	2801
F_{80}, (µm)	2801
Bond Work Index, kWh/tc	8.06

Table 13-36:
Results of the Bond Index Calculation Test for Oxide Ore

Item	Value
Cutting Mesh, (Tyler)	100
Cutting Mesh, (µm)	150
Sample weight (700 cm3), (g)	955.13
%-150 µm in feeding	27.66
Weight for 250% circular load, (g)	272.89
Grindability, (g/rev)	2.41
F80, (µm)	3599
F80, (µm)	101
Bond Work Index, kWh/tc	8.26

13.11.5 Bottle Rolled Tests (BRT) for Composite-Mixed Ore (70%Ox/30%Tx)

Composite samples were prepared in a proportion of 70% oxides and 30% transition, with these samples rolling bottle tests were carried out to evaluate the optimum cyanide concentration at values of 100, 250, 500 and 1000 ppm. Grindability tests are carried out to find the grinding time and obtain the appropriate particle size for all tests at 100%, -200 Tyler mesh. Representative samples of the composite were analyzed to determine the head ore grade. The initial test conditions are shown in Table 13-37.

Table 13-37:
Cyanidation Test Conditions

Parameter	Value
Sample Weight (g)	2000
Water Volume (cc)	4000
Granulometry	80% -#200 Tyler (P80)
Testing time (hours)	72
Natural pH	5.1
pH	10.5 to 11.0
[CN-]	0.1%
Aliquot collection time (hours)	0.01, 0.025, 0.05 and 0.1%
Aliquot volume	60 ml, for chemical analysis and: 25 ml, for titration

The results of the tests are shown in Table 13-38, Table 13-39 and Table 13-40 below.

Table 13-38:
Gold Dissolution and Reagent Consumption in Bottle Tests

[CN-] ppm	Head Grades Au (g/t)		Au Grade solution (g/t)	Au Dissolved Grade (g/t)	Tailing Grade (g/t)	Reagent Consump. (kg/t)		Gold Recovery (%)
	Assay	Calculated				Cyanide (NaCN)	Lime	
100	1.405	1.454	0.662	1.371	0.083	0.53	8.75	94.3%
250		1.498	0.641	1.367	0.13	0.98	8.75	91.3%
500		1.530	0.649	1.386	0.14	1.55	8.75	90.6
1000		1.465	0.656	1.399	0.07	2.814	8.75	95.5%

Table 13-39:
Silver Dissolution and Reagent Consumption in Bottle Tests

[CN-] ppm	Head Grades Ag (g/t)		Ag Solution Grade (ppm)	Ag Dissolved Grade (g/t)	Tailing Grade (g/t)	Reagent Consump. (kg/t)		Silver Recovery (%)
	Assay	Calculated				Cyanide (NaCN)	Lime	
100	1.4	1.402	0.472	1.002	0.40	0.526	8.75	94.3%
250		1.467	0.504	1.067	0.40	0.982	8.75	72.7%
500		1.47	0.503	1.072	0.40	1.548	8.75	72.8%
1000		1.53	0.531	1.133	0.40	2.814	8.75	73.9%

Table 13-40:
Silver Dissolution and Reagent Consumption in Bottle Tests

[CN-] ppm	Head Grades Ag (g/t)		Ag Solution Grade (ppm)	Ag Dissolved Grade (g/t)	Tailing Grade (g/t)	Reagent Consump. (kg/t)		Silver Recovery (%)
	Assay	Calculated				Cyanide (NaCN)	Lime	
100	1030	970.76	23.74	50.76	920	0.53	8.75	5.2%
250		990.34	19.05	40.34	950	0.98	8.75	4.1%
500		995.47	12.05	25.47	970	1.55	8.75	2.6%
1000		970.97	14.77	30.97	940	2.81	8.75	3.2%

The dissolution kinetic curves for Au, Ag and Cu are shown in Figure 13-31, Figure 13-32, and Figure 13-33.

Figure 13-31:
Gold Dissolution Curves for Bottle Roll Tests

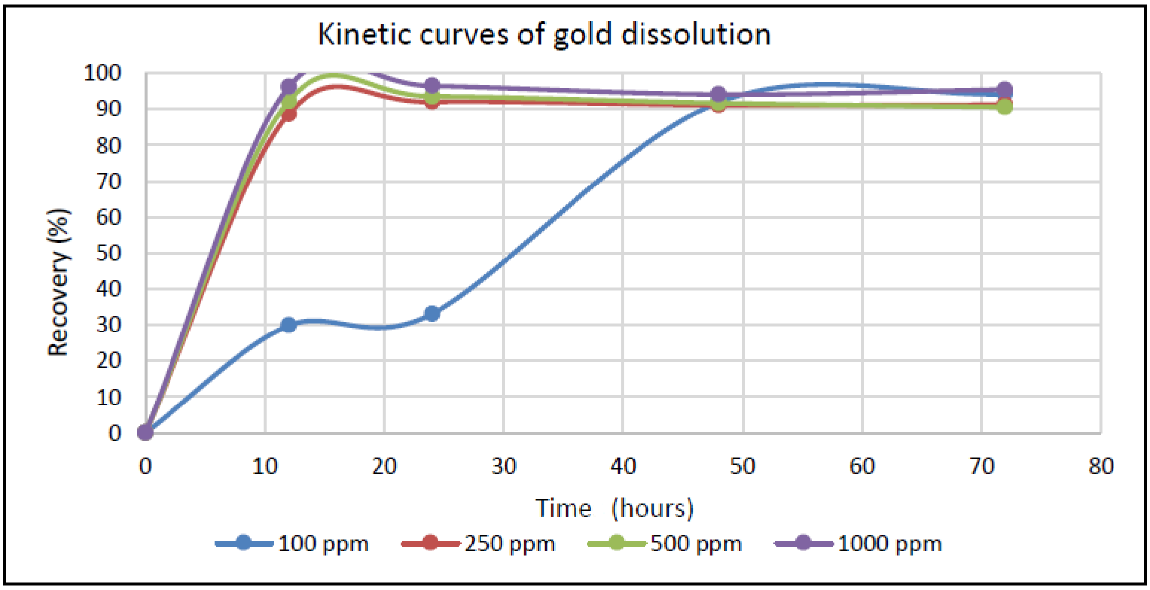


Figure 13-32:
Silver Dissolution Curves for Bottle Roll Tests

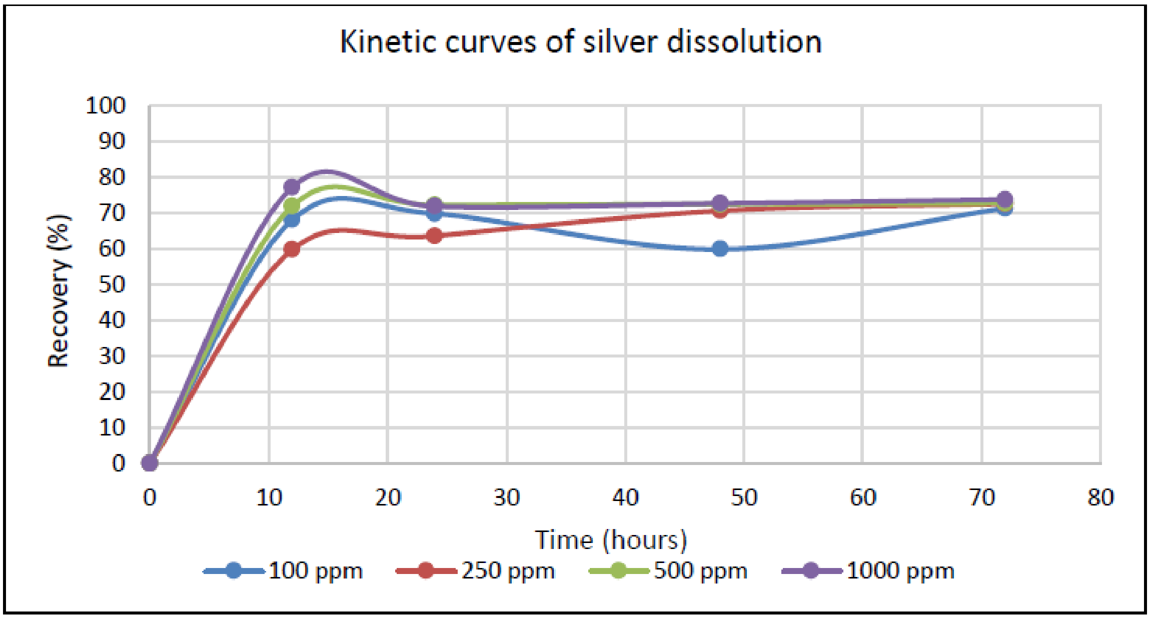
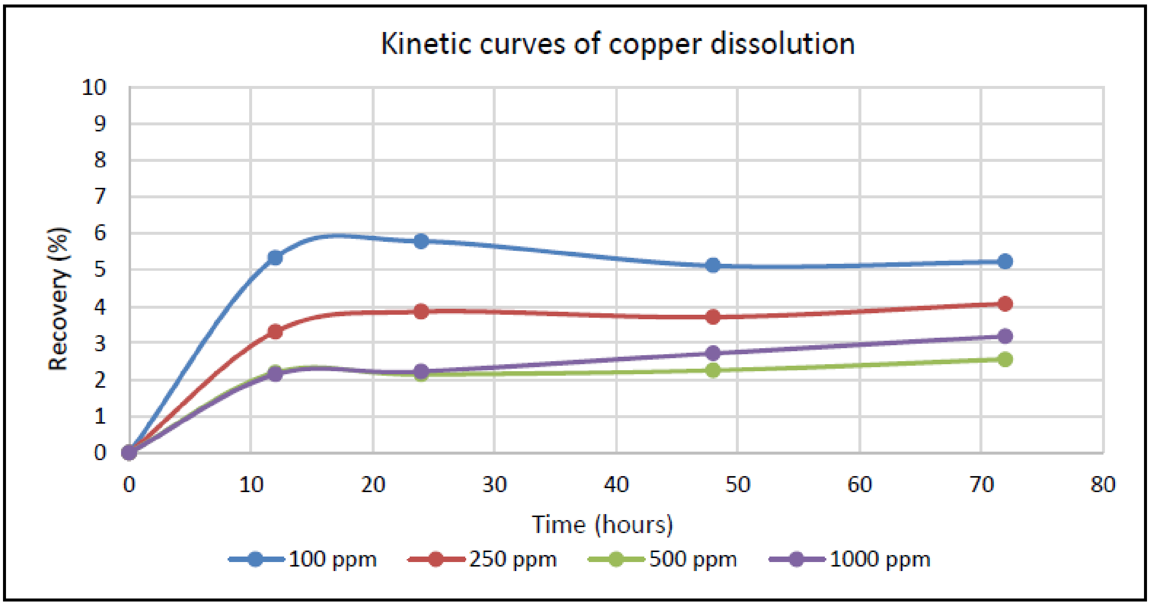


Figure 13-33:
Copper Dissolution Curves for Bottle Roll Tests



13.11.6 Tests on Columns

Two tests were conducted in PVC columns with a 12" diameter and a 5m height, where one of them is 100% Oxide ore and the other is a mixture of 70% Oxide ore and 30% transition ore. Both columns are evaluated for operating parameters such as: granulometry of the ore fed (100% <1") and the pre-treatment of agglomeration of the ore with cement to fractions smaller than ½". The other parameters are kept constant: cyanide concentration of the leaching solution (ppm), irrigation rate (l/h/m2), as those values are shown in Table 13-41.

Table 13-41:
Column Parameter Conditions

Condition	Weight in kg (dry)	Granulometry (inches)	Lime in a fraction <1/2" (kg/t)	Irrigation Rate (l/h/m2)	[CN-] Leach Solution (ppm)
70%Ox/ 30%Tx	333.50	100%-1"	12	9	1,000
100%Ox	296.20	100%-1"	23	9	1,000

The results of the oxide sample obtained a dissolution for gold between 87.8% to 93.4% and for silver between 38.5 to 45.8% with periods of 30 to 40 days of irrigation with variable granulometry from 3" to 1" but with agglomeration for the fine granulometry -½". Reagent consumptions are nearly similar in the order of 1 to 1.12 kg/t of cyanide and lime consumption from 1.2 to 1.25 kg/t, due to the near absence of cyanide elements in the oxides. Table 13-42, Table 13-43 and Table 13-44 shows the results for Column Tests.

Table 13-42:
Column Test Results for Gold

Condition	Head Grades Au (g/t)		PLS dissolved Au grade (g/t)	Tailing Grade (g/t)	Reagent consumption (kg/t)		Leaching Ratio (m3/ton)	Gold Recovery (%)
	Assay	Calculated			Cyanide (NaCN)	Lime		
70%Ox/ 30%Tx	1.405	1.567	1.406	0.161	1.07	7.0	1.87	89.7%
100%Ox	1.381	1.623	1.500	0.123	1.11	6.8	2.05	92.4%

Table 13-43:
Silver Dissolution and Reagent Consumption in Bottle Roll Tests

Condition	Head Grades Ag (g/t)		PLS dissolved Ag grade (g/t)	Tailing Grade (g/t)	Reagent Consumption (kg/t)		Leaching Ratio (m^3/ton)	Silver Recovery (%)
	Assay	Calculated			Cyanide (NaCN)	Lime		
70%Ox/ 30%Tx	1.400	1.31	0.81	0.5	1.07	7.0	1.87	58.1%
100%Ox	0.400	0.51	0.31	0.2	1.11	6.8	2.05	76.4%

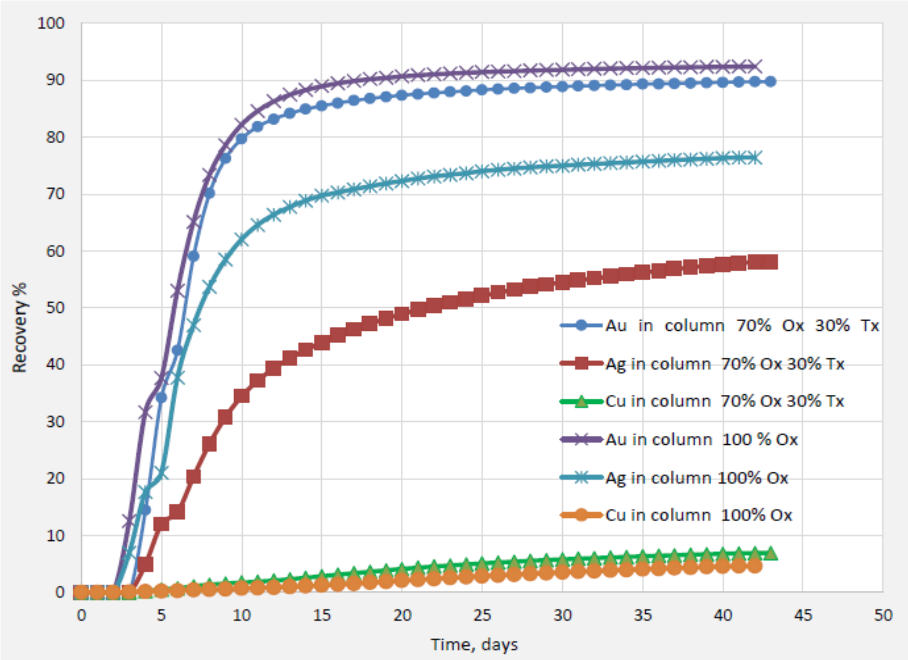
Table 13-44:
Silver Dissolution and Reagent Consumption in Bottle Roll Tests

Condition	Head Grades Cu (g/t)		PLS dissolved Cu grade (g/t)	Tailing Grade (g/t)	Reagent Consumption (kg/t)		Leaching Ratio (m3/ton)	Copper Recovery (%)
	Assay	Calculated			Cyanide (NaCN)	Lime		
70%Ox/ 30%Tx	1038	1009.84	69.84	940	1.07	7.0	1.87	6.9%
100%Ox	1037	953.99	43.99	910	1.11	6.8	2.05	4.6%

The column tests result in gold dissolution for the oxide ore of 92.4% and mixed ore of 89.7%, particle size <1" and cement addition of 23 and 12 Kg/t to the fine fraction <½" under a rate of 9 l/h/m2 and cyanide concentration of 1000 ppm. Silver dilutions are higher for the Oxide ore at 76.4% higher than for the Mixed ore at 58.1%; while, copper dilutions for the Oxide ore are lower than for the Mixed ore (these last values are almost similar in terms of cyanide soluble copper).

Reagent consumptions are almost similar in the order of 1.07 to 1.11 kg/t cyanide (similar values were found in past evaluations) and lime consumption are almost similar for 6.8 to 7.0 kg/t for both ore types. Because of the low cyanide consumption values, we estimate that there are no cyanicidal elements of consideration. The dissolution kinetic curves for gold, silver and copper are plotted for both columns, shown in Figure 13-34 where the 6 curves correspond to the Oxide and Transitional mineral zones ores.

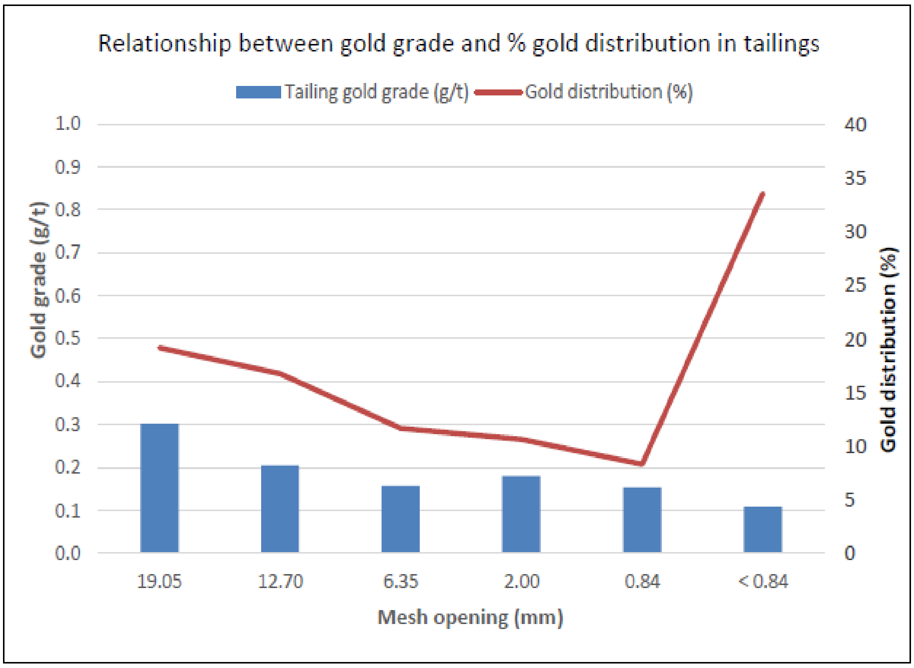
Figure 13-34:
Copper Dissolution Curves for Bottle Roll Tests



13.11.7 Analysis of Gold Performance in the Column Leach Test

The association between Au grades by particle size versus gold distributions in the leached material from the Mixed Ore column is shown in Figure 13-35.

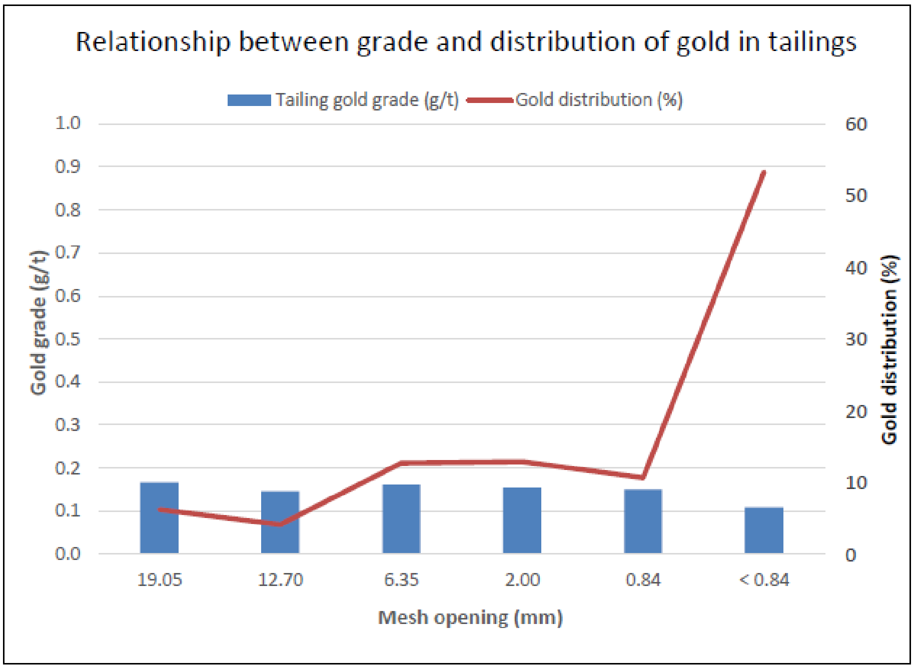
Figure 13-35:
Gold Grades and Distribution by Particle Size - Mixed Ore



13.11.7.1 Oxide Ore (100% Oxides)

The association between Au grades by particle size versus gold distributions in the leached material from the Oxide Ore column is shown in Figure 13-36.

Figure 13-36:
Gold Grades and Distribution by Particle Size - Oxide Ore



13.11.7.2 Grain-Size Characterization of the Leached Ore

Table 13-45:
Granulometric Analysis of the Tailings Sample - Mixed Ore

Tyler Mesh	Weight: (kg)	Weight Distr., %	Cumulative Retained weight	Accum. through weight	Chemical Assay			Distribution (%)		
					Au (g/t)	Ag (g/t)	Cu (%)	Au	Ag	Cu
3/4"	3.28	9.8%	9.8%	90.2%	0.30	1.30	0.11	19.1%	16%	11%
1/2"	4.23	12.7%	22.5%	77.5%	0.21	1.10	0.10	16.7%	17%	13%
1/4"	3.84	11.5%	34.1%	65.9%	0.16	0.80	0.09	11.6%	11%	11%
10 #	3.06	9.2%	43.3%	56.7%	0.18	0.70	0.09	10.6%	8%	8%
20 #	2.81	8.4%	51.7%	48.3%	0.15	1.10	0.09	8.3%	12%	8%
-20 #	16.09	48.3%	100.0%	0.0%	0.11	0.60	0.10	33.6%	36%	48%
	33.31	100%	Head Grade Calculated		0.155	0.81	0.095	100%	100%	100%
			Head Grade Assayed		0.161	0.50	0.094			

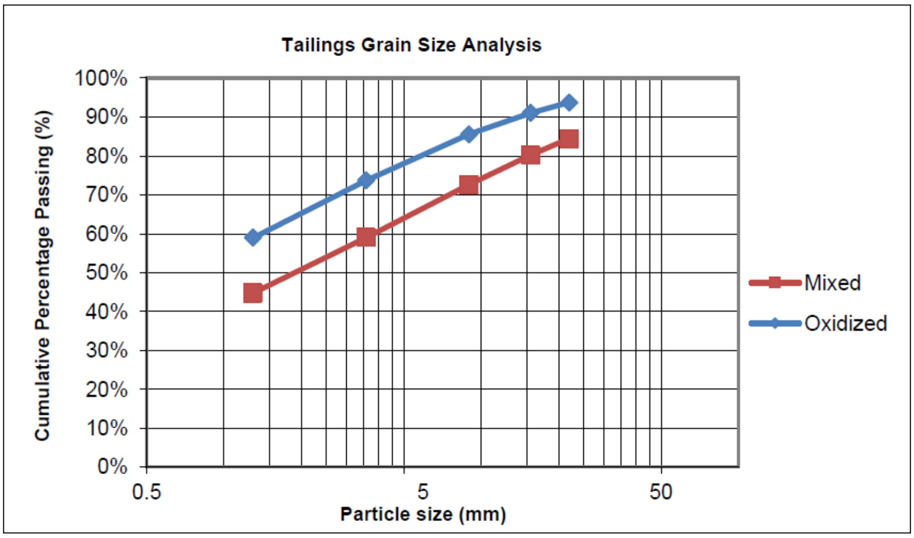
Table 13-46:
Granulometric Analysis of the Tailings Sample - Oxide Ore

Tyler Mesh	Weight: (kg)	Weight Distr.%	Cumulative Retained weight	Accum. through weight	Chemical Assay			Distribution (%)		
					Au (g/t)	Ag (g/t)	Cu (%)	Au	Ag	Cu
3/4"	1.57	4.8%	4.8%	95.2%	0.17	0.60	0.10	6.2%	6%	5%
1/2"	1.17	3.6%	8.3%	91.7%	0.15	0.40	0.10	4.1%	3%	4%
1/4"	3.28	10.0%	18.3%	81.7%	0.16	0.50	0.09	12.7%	11%	10%
10 #	3.47	10.6%	28.9%	71.1%	0.16	0.70	0.09	12.9%	16%	10%

20 #	2.95	9.0%	37.9%	62.1%	0.15	0.40	0.09	10.7%	8%	9%
-20 #	20.35	62.1%	100.0%	0.0%	0.11	0.40	0.09	53.3%	55%	62%
	32.77		Head Grade Calculated		0.127	0.45	0.093	100%	100%	100%
			Head Grade Assayed		0.123	<0.5	0.091			

The particle size curve of particle size versus cumulative passing percentage are shown in Figure 13-37.

Figure 13-37:
Gold Grades and Distribution by Particle Size - Oxide Ore



In Figure 13-37, it can be seen that the P80 adjusted by the R-R correlation is approximately 15.37 mm for the Transition ore column and 5.66 mm for the Oxide ore column, this shows that the Mixed ore has a coarse grain size compared to the Oxide ore.

13.11.8 Conclusions

1. Gold grades for Oxide, Transition and Mixed ore were 1.381 g/t, 1.345 g/t, and 1.405 g/t respectively. In the case of Ag, grades for the Oxide, Transition and Mixed ores were 0.4 g/t, 3.0 g/t, and 1.4 g/t respectively. While for Cu, they are close to 0.1% for the three types of material.
2. The particle size distribution of the Oxide and Transition ores of Au, Ag and Cu are opposite, having that in the Oxide ore the contents of the three elements increase when the particle size is reduced, while in the Transition ore the contents of the three elements increase for particles larger than 1".
3. The P80 for Oxide and Transition ores were values of 10.8 mm (< ½") and 98.19 mm (< 4") respectively.
4. The Work Index for Oxide and Transition ores were 8.06 Kwh/tc and 8.26 Kwh/tc respectively, classified on METSO scale as soft ore.
5. The bottle tests performed on the Mixed ore, we have with the increase of cyanide concentration in the LIX solutions, high recoveries are achieved for gold (around 95%), and a fast dissolution in the first hours of leaching related to the presence of free and fine gold.
6. The column tests for Oxide and Transition ore with different cement additions (23 Kg/t for Ox and 12 Kg/t for Tx), obtaining dissolutions for Au were 92.4% and 89.7%, with high cyanide and lime consumptions of 1.1 Kg/t and 7 Kg/t respectively. In METTS's opinion, this is due to the fact that the sample would have come from an acid contact zone with contents of soluble salts, sulfates, etc.

7. The evaluation carried out on the residual (tailings) column indicates that the highest gold values in the tailings for the transitional ore are in the coarse particles, while for the oxide ore, the grade distribution is uniform regardless of the particle size. Likewise, the P80 adjusted by the R-R function (Rosim Ramler) of the tailings for the Mixed and Oxide ores were 15.37 mm and 5.66 mm correspondingly.

13.12 MINERAL METALLURGICAL EVALUATION BY METTS (MARCH 2021)

MQ requested METTS to evaluate copper precipitation using the SART methodology prior to Au recovery by Merrill & Crowe and Activated Carbon Adsorption. The SART process is applied for complex cyanide solutions with Au-Cu values, allowing the cyanide Cu complexes to be precipitated as copper sulfides, while allowing cyanide to be recycled to heap leach solutions once the copper and gold have been removed from the solutions.

The Cu precipitation tests by the SART method applied to the A and B sample composites resulting from the irrigation of the Oxide and Mixed ore columns gave promising results, reducing the Cu contents to lower values of 3.98 and 3.3 mg/l for both composites with copper grades of 48 and 40 mg/l, obtaining copper recovery efficiencies of 87% and 92.5%. Moreover, the application of the SART method did not affect the initial gold values in both composites maintaining them at values of 5.56 and 2.17 mg/l.

Subsequently, the precipitation values of the Zn-Au ionic pair by the Merrill & Crowe process had efficiencies of 98.8% and 96.3% for composites A and B, respectively, with leaching liquors with values of 5.56 and 2.17 mg/l, and with cyanide concentrations contained in the solutions close to 330 ppm.

Finally, the adsorption of gold from the cyanide solutions of composites A and B reached values greater than 99% in two to four hours of residence time, being accompanied by the copper element that adsorbs on the activated carbon in values of 62.5 to 96.5% depending on the type of composite. It is important to note that silver was not evaluated due to its low concentration (<0.005 mg/l).

13.13 MINERAL METALLURGICAL EVALUATION BY BIZALAB (SEPTEMBER 2021)

Minera Quinchia requested BIZALAB to carry out a mineralogical analysis with emphasis on clays by X-ray diffraction (ADRX) on two fine-powder samples.

In mineralogical analyses for clays by X-Ray diffraction (ADRX), by determining the global mineralogy (random powder preparation) with differentiations between the types of lamellar minerals by applying processes such as granulometric separation and glycolation (oriented sheet preparation).

The results of those samples (Transition and Oxide) are shown in Table 13-47.

Table 13-47:
Mineralogical Analysis with Emphasis on Clays by X-Ray Diffraction

Sample	Mineral	Formula	Approx. Percentage (%)
Oxide	Halloysite	$Al_2Si_2O_5(OH)4$	29
	Quartz	SiO_2	27
	Gibbsite	$Al(OH)_3$	20
	Goethite	FeO(OH)	12
	Chlorite (clinochlorine)	$(Mg, Fe)_5Al(Si_3Al)O_{10}(OH)_8$	6

Sample	Mineral	Formula	Approx. Percentage (%)
	Magnetite	Fe_3O_4	3
	Amphibole (Actinolite)	$Ca_2(Mg, Fe)_5Si_8O_{22}(OH)_2$	< L. D.
Transitional	Halloysite	$Al_2Si_2O_5(OH)_4$	35
	Quartz	SiO_2	29
	Gibbsite	$Al(OH)_3$	15
	Goethite	$FeO(OH)$	12
	Magnetite	Fe_3O_4	4
	Chlorite (clinochlorine)	$(Mg, Fe)_5Al(Si_3Al)O_{10}(OH)_8$	3

13.14 RECOVERY

In order to estimate recovery for the Project, Minera Quinchia personnel first classified the samples using the material type domains from the resource model. Then, the recovery data from the BRT (78.2% at 75 µm vs 68.5%) was compared to the recoveries achieved in the column tests. Based on this difference, the heap leach recoveries were estimated for all samples tested.

Table 13-48 shows the data used to calculate the recovery deductions from the BRTs. Even though the samples labelled BATCH-3 and BATCH-6 were classified as mixed samples in the resource model, they were used to estimate the recoveries for oxide material because the sulfur content was very low. In order to be conservative, LINAMEC used the higher difference between the 200 mesh BRT gold recovery (87.1% & 89.5%) and the column test gold recovery (79.2% & 85.5%) for these samples because the differences were similar (i.e., 7.9% difference and 3.9% difference). For silver, the average of the differences for the two samples was used (i.e., the average of 3.3% and 17.6%). For the primary domain, data from BATCH-2 and BATCH-5 samples was used. For both gold recovery and silver recovery, the average of the differences was used. Since no column tests were performed using pure mixed domain sample material; the recoveries for both gold and silver were estimated using the averages of the oxide recoveries and the primary recoveries.

Finally, after the deductions were calculated, the heap leach recovery was estimated by using the data from the 200 mesh (i.e., 74 µm) BRTs and deducting these differences to account for the larger particle size. The results are shown in Table 13-48. The average recoveries are summarized in Table 13-49.

Table 13-48:
Summary of SGS Magnetic Separation Data

Composite	Type	Au Rec 200 Mesh	Au Rec 2 mm	Au Rec Column	Difference/ Deductions	Ag Rec 200 Mesh	Ag Rec 2 mm	Ag Rec Column	Difference/ Deduction
BATCH-3	Mixed (Oxide/ mixed/primary)	87.1%	80.9%	79.2%	7.9%	72.7%	60.9%	58.0%	14.7%
BATCH-6	Mixed (Oxide/ mixed)	89.5%	86.3%	85.5%	3.9%	75.4%	63.7%	68.9%	6.5%
	Oxide				7.9%				10.6%
	Mixed				16.5%				12.6%
BATCH-2	Primary	64.4%	51.3%	40.8%	23.6%	69.5%	69.2%	51.6%	17.6%
BATCH-5	Primary	66.6%	46.1%	39.9%	26.7%	54.9%	52.3%	49.0%	3.3%
TOTAL	**Average Primary**				**25.2%**				**10.5%**

Table 13-49:
Estimated Heap Leach Recovery Data Batero Gold Corp.

Sample	Type	Au Rec 200 Mesh (BRT-13-3)	Estimated	Ag Rec 200 Mesh	Estimated
CBR-1211	Primary	57.8%	32.7%	52.3%	41.8%
BATCH-2	Primary	64.4%	40.8%	54.0%	43.5%
BATCH-5	Primary	66.6%	41.4%	59.7%	49.2%
CBR-1213	Mixture (mostly Primary)	68.6%	43.5%	61.5%	51.0%
CBR-1207	Mixture (mostly Primary)	73.5%	48.4%	54.9%	49.0%
CBR-1204	Primary	73.9%	48.7%	69.5%	51.6%
CBR-1208	Primary	79.0%	53.9%	63.5%	53.0%
CBR-1214	Primary	81.1%	56.0%	70.2%	59.8%
	Average Primary		**46.0%**		**50%**
CBR-1205	Mixture (Primary-Mixed)	64.3%	47.7%	36.4%	25.9%
CBR-1209	Mixture (Primary-Mixed)	71.2%	54.6%	58.9%	48.4%
CBR-1215	Mixed	75.1%	58.6%	84.6%	59.8%
CBR-1201	Mixed	75.3%	58.7%	74.2%	63.8%
MQ12COMP-05	Mixed	81.9%	65.4%	74.5%	64.1%
CBR-1202	Mixture (Primary-Mixed)	83.1%	66.6%	75.2%	64.7%
MQ12COMP-02	Mixed	83.9%	67.4%	78.7%	68.2%
MQ12COMP-04	Mixed	86.0%	69.5%	79.0%	68.5%
	Average Mixed		**61.0%**		**58%**
CBR- 1218	Mixture (mostly Oxide)	82.8%	74.9%	85.9%	0.0%
MQ12COMP-07	Oxide	84.5%	76.6%		
	Mixture				
BATCH-3	(Oxide/Mixed/Primary)	87.1%	79.2%	68.4%	57.9%
CBR-1210	Mixture (mostly Mixed)	88.4%	80.5%	72.7%	58.0%
BATCH-6	Mixture (Oxide/Mixed)	89.5%	85.5%	75.4%	68.9%
MQ12COMP-01	Oxide	92.6%	84.7%	82.4%	71.9%
MQ12COMP-03	Oxide	92.6%	84.7%	69.6%	59.0%
CBR-1206	Oxide	92.8%	84.8%	71.4%	60.9%
MQ12COMP-06	Oxide	93.9%	86.0%	89.9%	73.7%
CBR-1216	Mixture (mostly Oxide)	94.3%	86.4%	84.0%	73.4%
CBR-1212	Mixture (mostly Oxide)	94.5%	86.6%	89.2%	78.6%
TOTAL	**Average Oxide**		**83%**		**60%**

13.14.1 Sampling

The samples were selected to be representative of the material that would be processed in the heap-leaching scenario. As the Project progressed, the model was changed and some of the material was re-classified, no metallurgical data directly correlates to the particular material types. LINAMEC is of the opinion that the samples and the metallurgical data are representative of the results that are expected for the La Cumbre Project.

14.0 MINERAL RESOURCE ESTIMATE

14.1 KEY ASSUMPTIONS/BASIS OF ESTIMATE

The mineral resource statement presented herein represents the updated mineral resource estimation prepared by LINAMEC for the La Cumbre Gold Project in accordance with the Canadian Securities Administrators National Instrument 43-101.

The Mineral Resource Estimate (MRE) prepared by LINAMEC utilized 143 drillholes, including 40 drillholes drilled by Minera Quinchia in 2016-2017. The resource estimation work was completed by Mr. Fernando Linares Geo, (MAusIMM), Principal Resource Geologist with Linares Americas Consulting S.A.C. and Mr. Walter La Torre Geo MAusIMM (CP), who have reviewed pertinent geological information in sufficient detail to support the data incorporated in the mineral resource estimate. Mr. La Torre is an Independent Qualified Person and Chartered Professional as defined under NI 43-101. The effective date of the mineral resource statement is December 31, 2021.

This estimation approach was considered appropriate based on a review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralization and the style of mineralization. The estimation was constrained within mineralized geological-grade interpretations that were created with the assistance of Minera Quinchia geologists.

14.2 GEOLOGICAL MODELS

For the geological modeling of the La Cumbre deposit, mineral zone, lithology, and the gold grades were taken into account. Using these data, a first set of sections interpreted by Minera Quinchia geologists and a second set of 18 sections created and interpreted by LINAMEC's modeller, see Figure 14-1 it was possible to define four domains.

Using Leapfrog Geo 4.0, it was possible to build the 3D geological solids corresponding to each of the domains used for the MRE of the Quinchía project. (Figure 14-2).

Ash Domain, this domain corresponds to a last post-mineral volcanic event, therefore barren, it is partially eroded and covers the oxide domain and transitional domain in several locations.
Oxide Domain, consisting mainly of diorite hydrothermally and supergenically altered with the formation of saprolite, it is mineralized with gold and silver and presents iron and copper oxides and sulfates.
Transitional Domain, shows a diorite less affected by weathering but altered by hydrothermal processes, containing iron oxides and primary sulfides with gold and silver mineralization.
Primary Domain, the rocks in this domain are almost free of iron oxides and present only mineralization of primary sulfides and hydrothermal magnetite with potassium alteration, mainly fine brown color biotite and retrograde propylitic alteration.

The interpretation of the boundary between the oxide and transitional domains was made by the Minera Quinchia geologists taking into account the sulfur content reported in the multi-element analysis.

The update resource estimate disclosed in this report on the La Cumbre deposit now considers the resources of the oxide, transitional and primary domains. Using a grade shell greater than 0.22 Au g/t, high-grade and low-grade sub-domains have been defined (Figure 14-26).

14.3 SOLID MODELLING

For modelling of the four new 3D solid domains, LINAMEC used Leapfrog Geo 4.0 and Gemcom GEMS 6.5 (Figure 14-2). Minera Quinchia geologists use SURPAC to produce interpreted sections. The modelling was based upon information obtained from drill hole databases, which compiles the different lithological, mineralogical, structural and alteration characteristics in the Ash, Oxide, Transitional and Primary domains. The attributes modelled were the gold, silver and copper grades inside of each mineralized zone. This methodology allows an adjustment of the mineralized zones and avoids an overestimation of the volumes.

Leapfrog uses implicit modelling to create a 3D geological solid. An Implicit Model is a continuous mathematical representation of an attribute across a volume. It has an infinitely fine resolution. Creating tangible surfaces from this model is a separate and secondary step and is independent of the creation of the implicit model. Implicit modelling uses radial basis functions (RBF's) to model grade shells, lithology boundaries, faults or surfaces.

Implicit modelling generally has three distinct parts:

1. Organize the data into an appropriate format (error free database).
2. Generate a continuous, volumetric model (the implicit model).
3. Output one or more surfaces contained in the model.

The Leapfrog solids created for each domain were used as hard contacts in the interpolation process. Ore zones were defined within the solids, considering composited gold values of the intercepts of the DDHs within the modeled structures. The oxide, transitional and primary domains were encoded for use in Leapfrog Geo and Gemcom GEMS. Additionally, a grade shell was created with Leapfrog Geo that encompasses grades above 0.22 Au g/t, which allowed for the separation of low-grade and high-grade populations, with high grade being understood as grades above the established cutoff of 0.22 Au g/t, see Figure 14-26 for the solid created with Leapfrog Geo. The following 3D domains at La Cumbre were modelled with Leapfrog Geo software, see **Error! Reference source not found.** and Figure 14-1.

Table 14-1:
Domains Modelled with Leapfrog Geo

Domain	Rock Code	Rock Type	DDH Samples	Total Composites
ASH	ASH	700	100	153
OXIDE	ZOX	710	2,400	2,400
TRANSITIONAL	ZTR	720	1,491	1,290
PRIMARY	ZSP	730	18,983	16,585
TOTAL			**22,974**	**20,428**

A set of 18 cross-sections were used for modelling, review and editing the solids in the La Cumbre deposit using traditional interpretation on sections (see Figure 14-1 and Figure 14-3).

The topographic surface is based on a LIDAR (*Laser Imaging Detection and Ranging*) survey provided by Airborne Solutions International to Batero.

Figure 14-1:
La Cumbre – Sections Used to Model 3D Solids

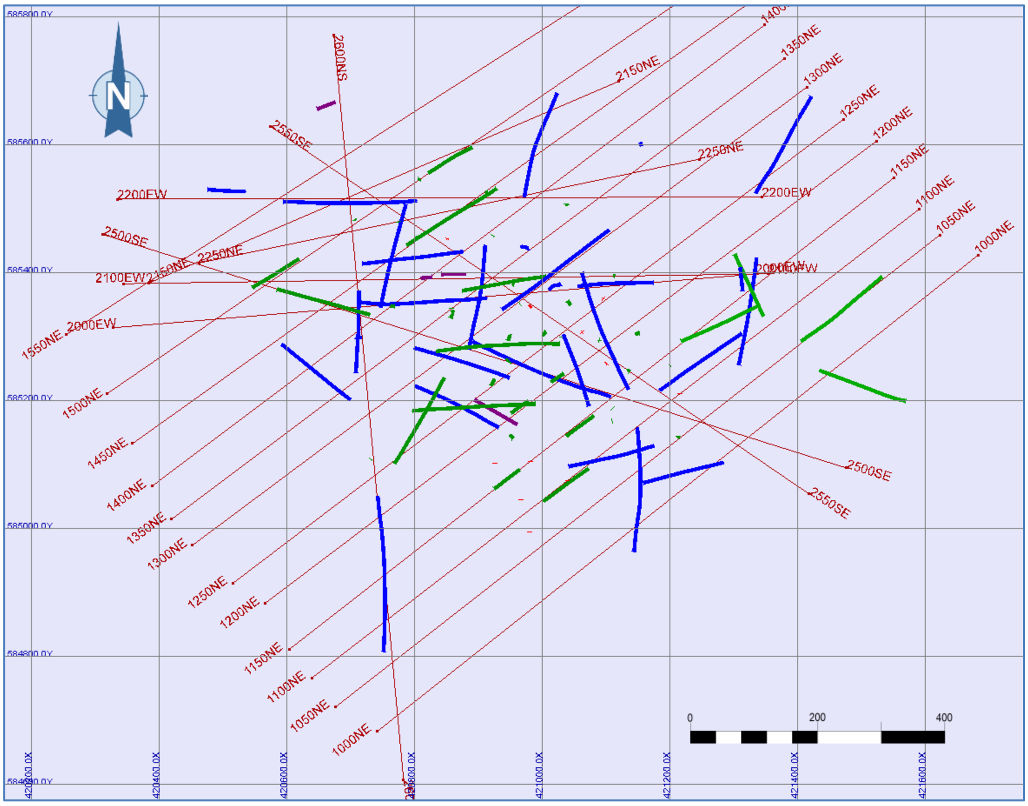


Figure 14-2:
3D Modelled Mineral Zones at La Cumbre

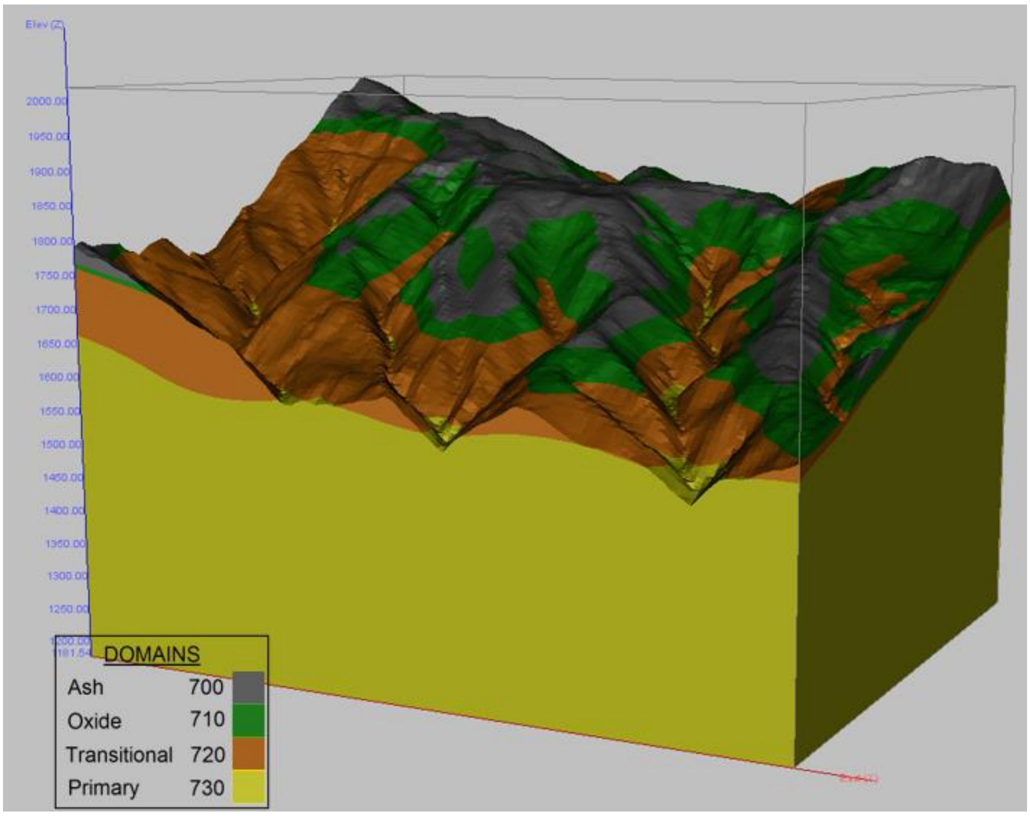
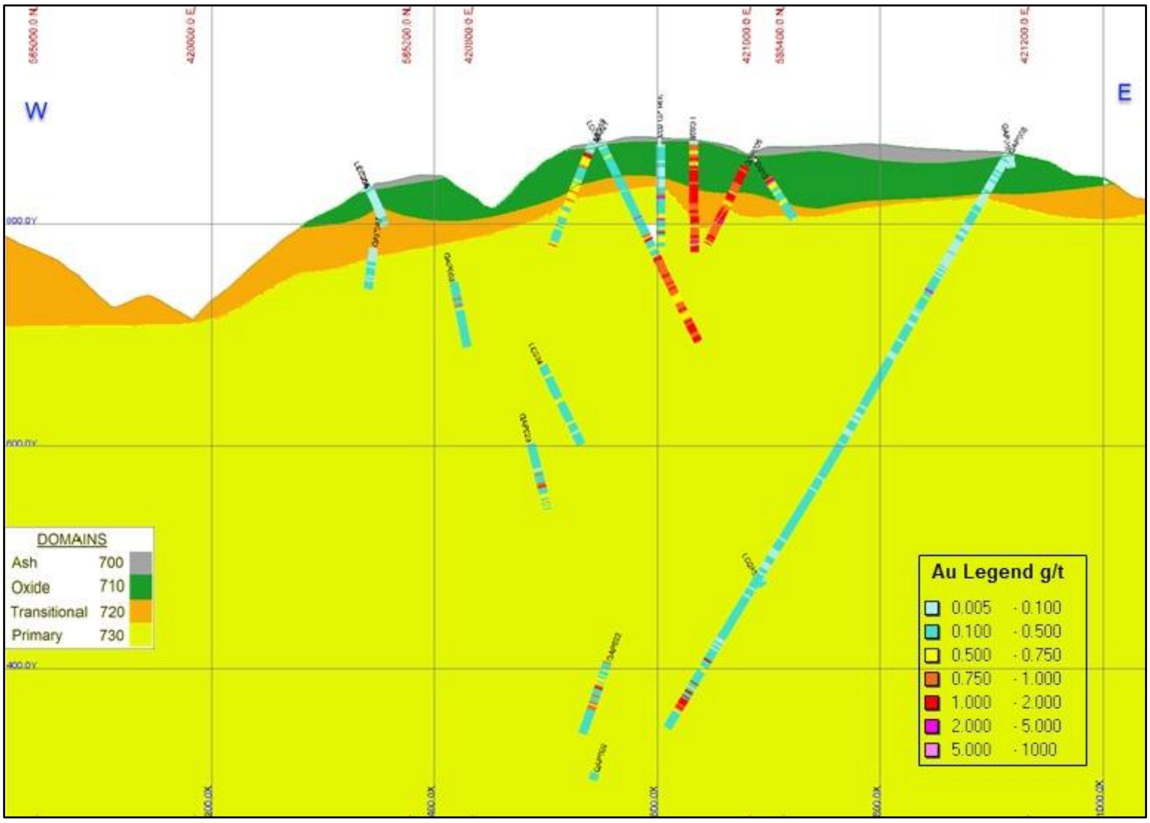


**Figure 14-3:
Modelled Section 1300 at La Cumbre Looking to the North**



14.4 EXPLORATORY DATA ANALYSIS (EDA)

LINAMEC did a complete statistical analysis of the La Cumbre Deposit data for assays, composites and capped composites. This statistical analysis of the composites was used to set the capping value by mineral zones (domains) with enough data to produce reliable statistics. The capped composite data were utilized in the interpolation process and resource estimation.

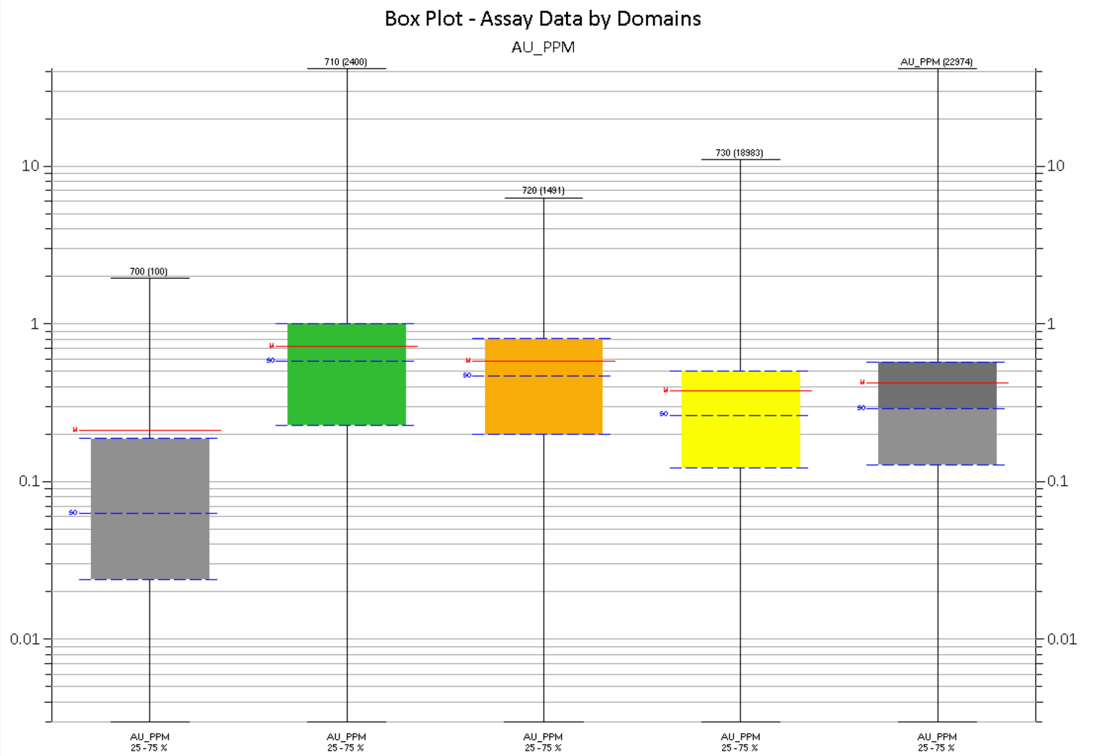
The EDA results are useful to validate the resource estimate by comparing the average of gold and silver grades of composites to block model values. The statistical tools used were histograms, probabilistic plots and box plots. The histograms and boxplot are accompanied by descriptive statistics, which provide the mean and coefficient of variation. The coefficient of variation (CV) is the standard deviation divided by the mean and is a measure of relative variability. Typically, most disseminated gold deposits show coefficients of variation around 1.0 to 2.0. Where higher values occur, they may indicate a mixture of populations with widely varying means.

14.5 SUMMARY STATISTICS – ASSAYS

Raw data (assays) statistics based on the mineralized zones are showed in the boxplot of Figure 14-4 for Ash (700), Oxide (710), Transitional (720) and Primary (730) zones.

The CV for La Cumbre Deposit domains have values less than 2.00 (see Figure 14-4), these values indicate that gold values came from one or more ore population and are adequate for interpolation process and MRE.

Figure 14-4:
Boxplot for La Cumbre Deposit – Raw Data (Assays)



Points	100	2400	1491	18983	22974	Points
Mean	0.211	0.715	0.579	0.374	0.422	Mean
Std Dev	0.359	1.056	0.508	0.387	0.520	Std Dev
Variance	0.129	1.115	0.258	0.150	0.270	Variance
CV	**1.699**	**1.477**	**0.876**	**1.035**	**1.231**	**CV**
Skewness	2.690	25.412	2.296	4.487	24.585	Skewness
Kurtosis	7.548	949.544	13.300	61.920	1756.134	Kurtosis
Maximum	1.955	41.600	6.310	11.000	41.600	Maximum
75%	0.188	1.005	0.807	0.504	0.572	75%
50% (median)	0.063	0.576	0.467	0.264	0.290	50% (median)
25%	0.024	0.227	0.200	0.122	0.128	25%
Minimum	0.003	0.003	0.003	0.003	0.003	Minimum

*Coefficient of variation (CV) = standard deviation divided by the mean.

Figure 14-5 shows the histogram of gold grades for La Cumbre deposit Oxide zone. The histogram has a log-normal distribution with a mean of 0.714 g/t Au and CV=1.482. Figure 14-6 shows a cumulative probability plot with a remarkable inflexion of about 0.90 g/t Au.

Figure 14-7 and Figure 14-8 show histogram and probability plot for La Cumbre deposit Transitional zone. The curve of the probability plot shows clearly two populations, with an inflection point around of 0.50 g/t Au. At this point, the low-grade population is apart of the high-grade population. Each of these populations shows a log-normal distribution.

Figure 14-9 shows the histogram of gold grades for La Cumbre deposit Primary zone. The histogram has a log-normal distribution with a mean of 0.368 g/t Au and CV=1.033. Figure 14-10 shows a cumulative probability plot with an inflexion of about 0.40 g/t Au.

Figure 14-5:
Weighted Gold Assay Histogram, Oxide Zone

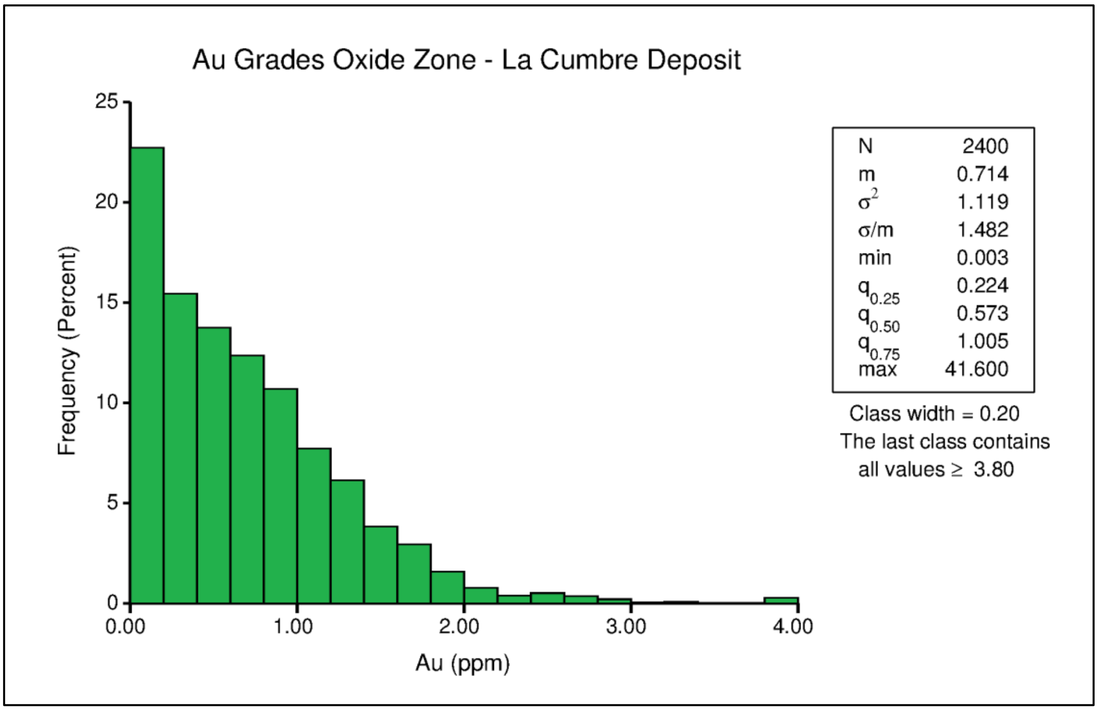


Figure 14-6:
Cumulative Probability Plot Gold Assays, Oxide Zone

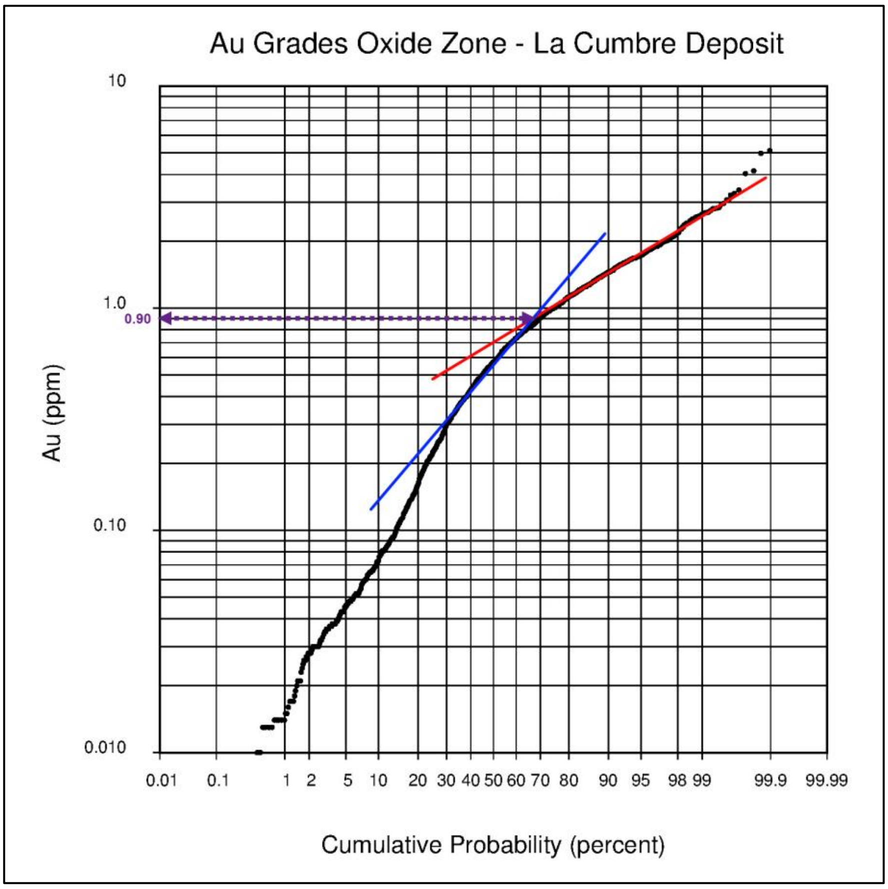


Figure 14-7:
Weighted Gold Assay Histogram, Transitional Zone

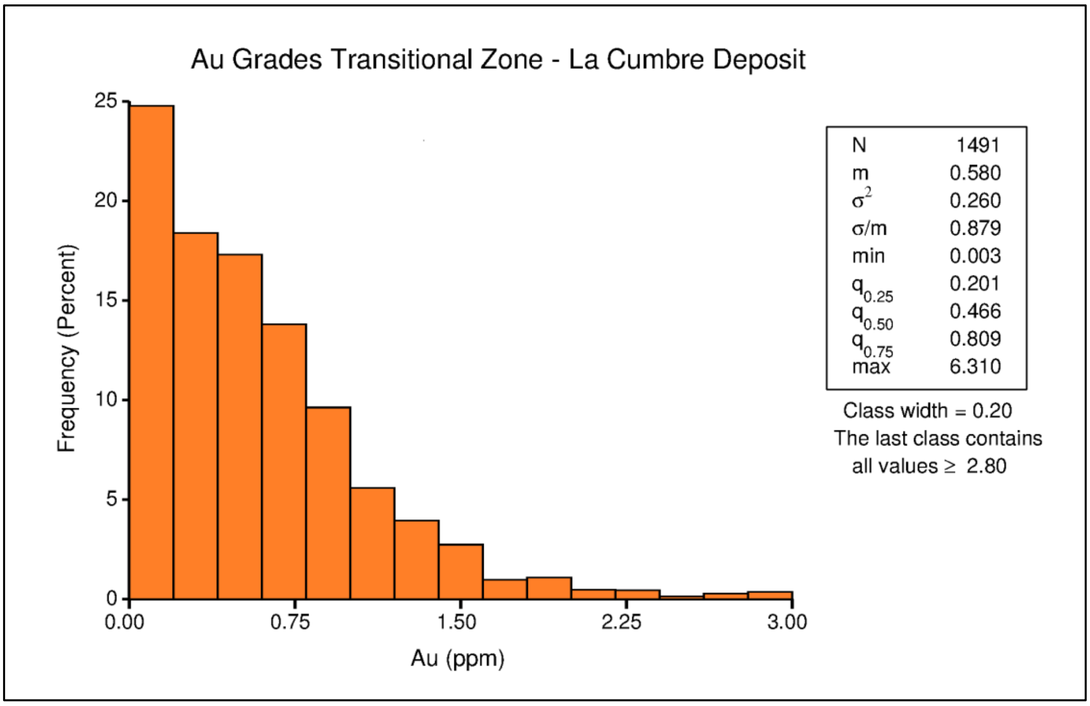


Figure 14-8:
Cumulative Gold Probability Plot Gold Assays, Transitional Zone

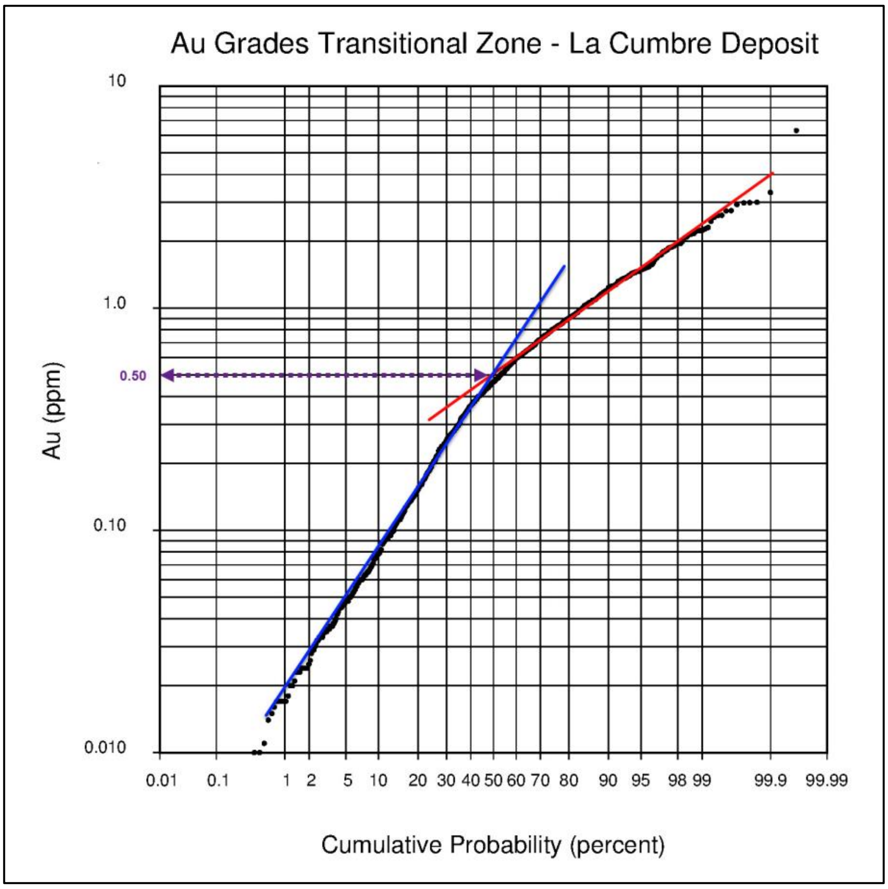


Figure 14-9:
Weighted Gold Assay Histogram, Primary Zone

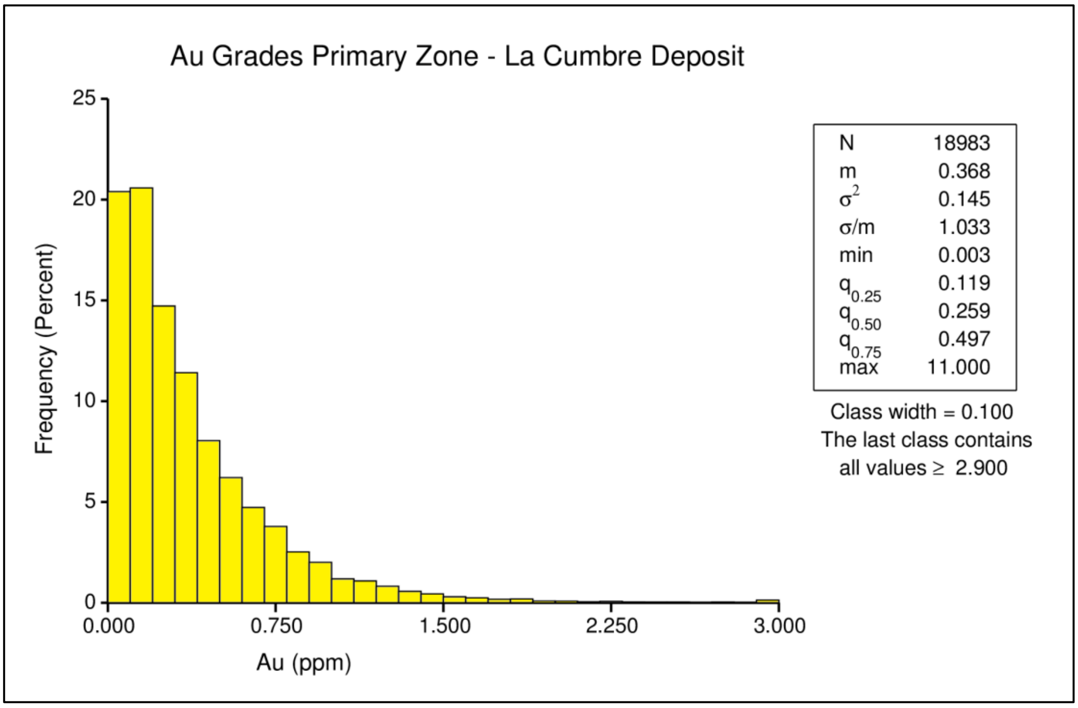
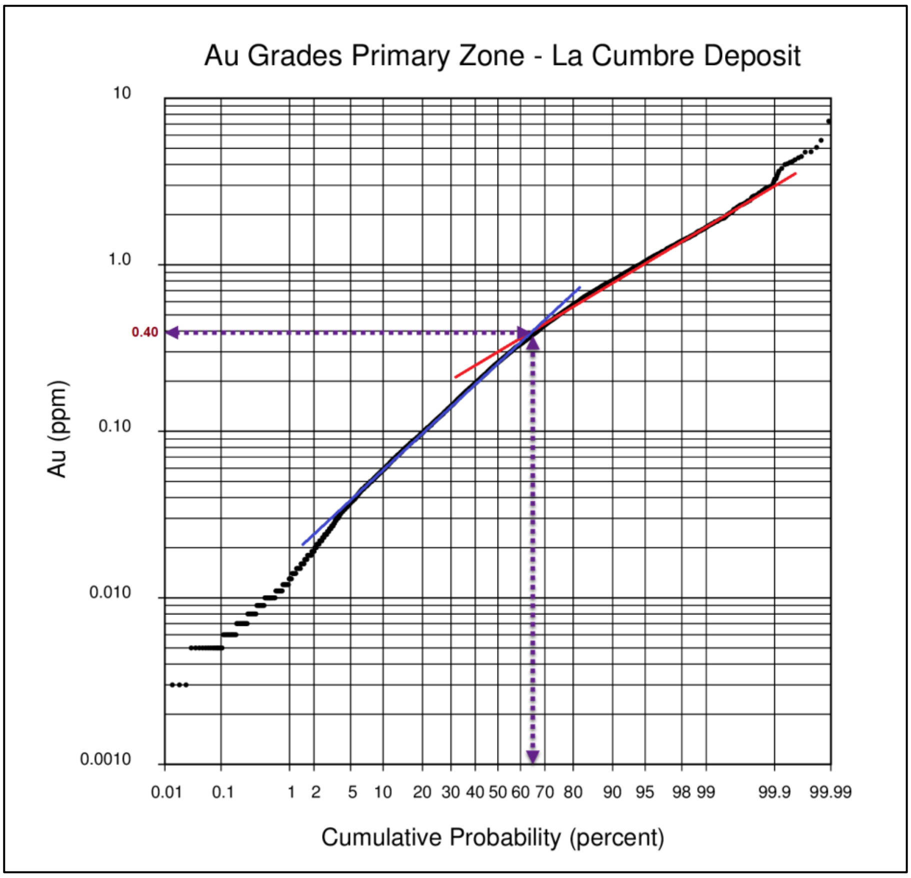


Figure 14-10:
Cumulative Gold Probability Plot Gold Assays, Primary Zone



14.6 SUMMARY STATISTICS – COMPOSITES

LINAMEC composited the assays into 2 m intervals for grade interpolation and subsequent exploratory data analysis and variographic analysis. The composite datasets were completed using GEMS mining software package.

The global effect of the compositing produces negligible effect to the total length and mean grade. A decrease in the sample variance is noted as a natural effect of compositing. The 2 m composite files were used for all statistical, geostatistical and grade estimation studies. The majority of the sampling used 2.0 m sample intervals (87.8%), with a small number of samples with lengths ranging from 0.6 m to 5.9 m and mean lengths equal to 1.97 m. See Figure 14-11 for a histogram of sample lengths for the La Cumbre Deposit.

This statistical analysis of the composite data was used to set the capping value for the oxide and transitional zones with sufficient data to produce reliable statistics. The capped composite data were utilized in the variographic analysis, interpolation process and resource estimation. The statistical tools used were histograms, probabilistic plots and box plots.

Composite statistics for the updated mineral zones of the La Cumbre deposit, are summarized in the boxplot of Figure 14-12, that displays graphically the statistics of each zone and permits a comparison of all zones. The composite statistics were calculated for gold and silver values.

Figure 14-11:
Sample Length Histogram – La Cumbre Deposit

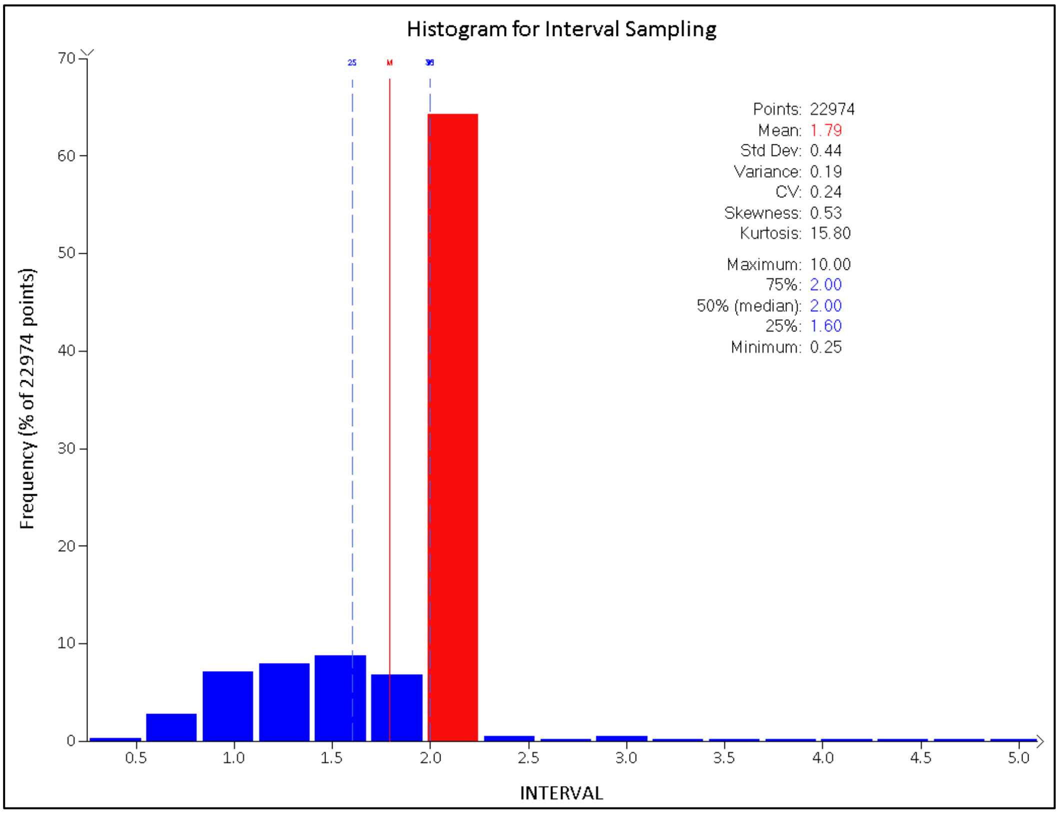
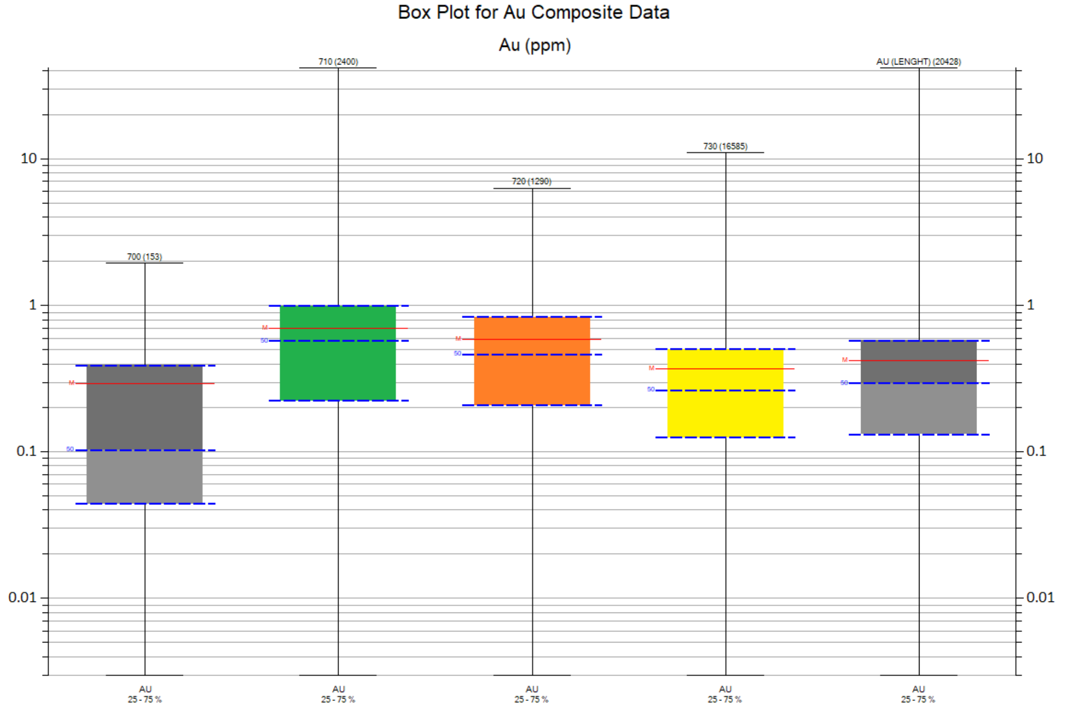


Figure 14-12:
Boxplot for La Cumbre Deposit – Composite Data



Points	153	2400	1290	16585	20428	Points
Mean	0.293	0.703	0.592	0.370	0.423	Mean
Std Dev	0.399	1.029	0.521	0.362	0.512	Std Dev
Variance	0.159	1.059	0.272	0.131	0.262	Variance
CV	**1.360**	**1.464**	**0.881**	**0.976**	**1.211**	**CV**
Skewness	2.621	37.757	3.510	5.363	38.782	Skewness
Kurtosis	8.969	2088.030	34.328	113.420	4133.570	Kurtosis
Maximum	1.955	41.600	6.310	11.000	41.600	Maximum
75%	0.392	0.996	0.833	0.506	0.579	75%
50% (median)	0.102	0.573	0.467	0.266	0.296	50% (median)
25%	0.044	0.222	0.206	0.124	0.131	25%
Minimum	0.003	0.003	0.003	0.003	0.003	Minimum

14.7 CONTACT PLOT ANALYSIS

A contact grade profile is a type of graph that helps to visualize grade relationships near geological boundaries. Contact plots were generated to explore the relationship between grade and domains for gold and silver. The plots were constructed with Supervisor 8.5 software that searches for data with a given code, and then searches for data with another specified code and bins the grades according to the distance between the two points. This allows for a graphical representation of the grade trends away from a “contact”.

Where there is a marked discontinuity of the grade profile at the boundary between two domains, a strong control (usually lithological) on the grade is probably present, and data selected for interpolation should not come from across the boundary with respect to the domain in which a block resides. Such boundaries are referred to as “hard” boundaries or contacts. Where the change in grade is relatively slight when crossing a boundary, no limitation on the domain in selecting samples for interpolation is indicated, and the boundary is referred to as “soft”. In some cases, the change in grade occurs over an interval of a few tens of meters; such boundaries are termed “firm”. During interpolation, samples from a limited distance across a boundary may be selected. If average grades are reasonably similar near a boundary and then diverge as the distance from the contact increases, the particular boundary should probably not be used as a grade constraint. In fact, if a hard boundary is imposed where grades tend to change gradually, grades may be overestimated on one side of the boundary and underestimated on the opposite side.

For La Cumbre deposit contact plots were constructed between each mineralization zone. LINAMEC notes that contact between the oxide zone and transitional zone for gold is soft, which indicates that composites in the oxide zone could have been used in the estimation of the transitional zone and vice versa. Figure 14-13 shows the soft contact profile between oxide and transitional zones.

The contact plot between the transitional and primary zones is firm. Figure 14-14 shows that the red line (mean values) decreases smoothly as it moves from the transitional to the primary zone.

Figure 14-13:
Contact Profile for Oxides vs. Transitional Zones Composite Au Grades – La Cumbre Deposit

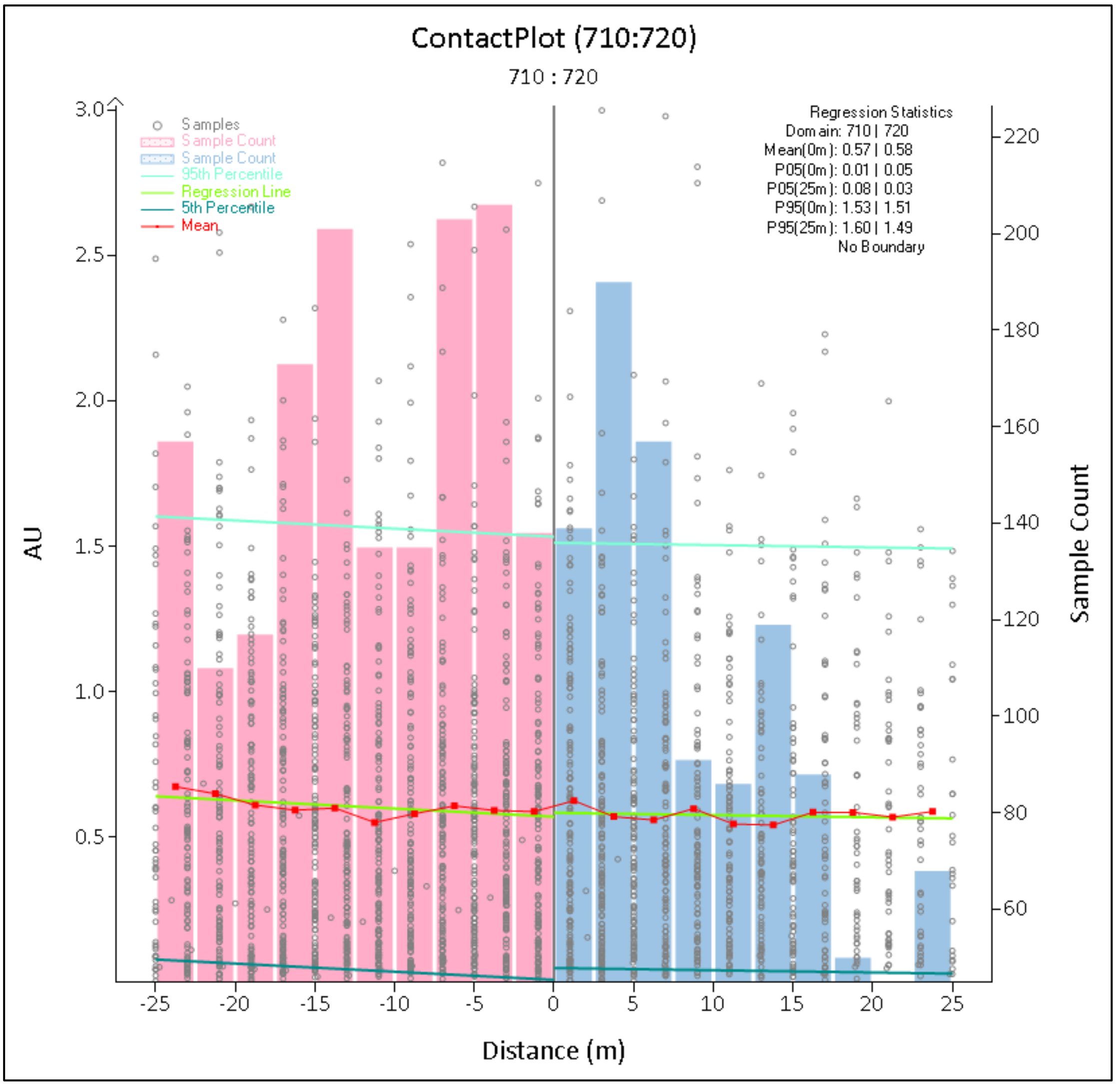
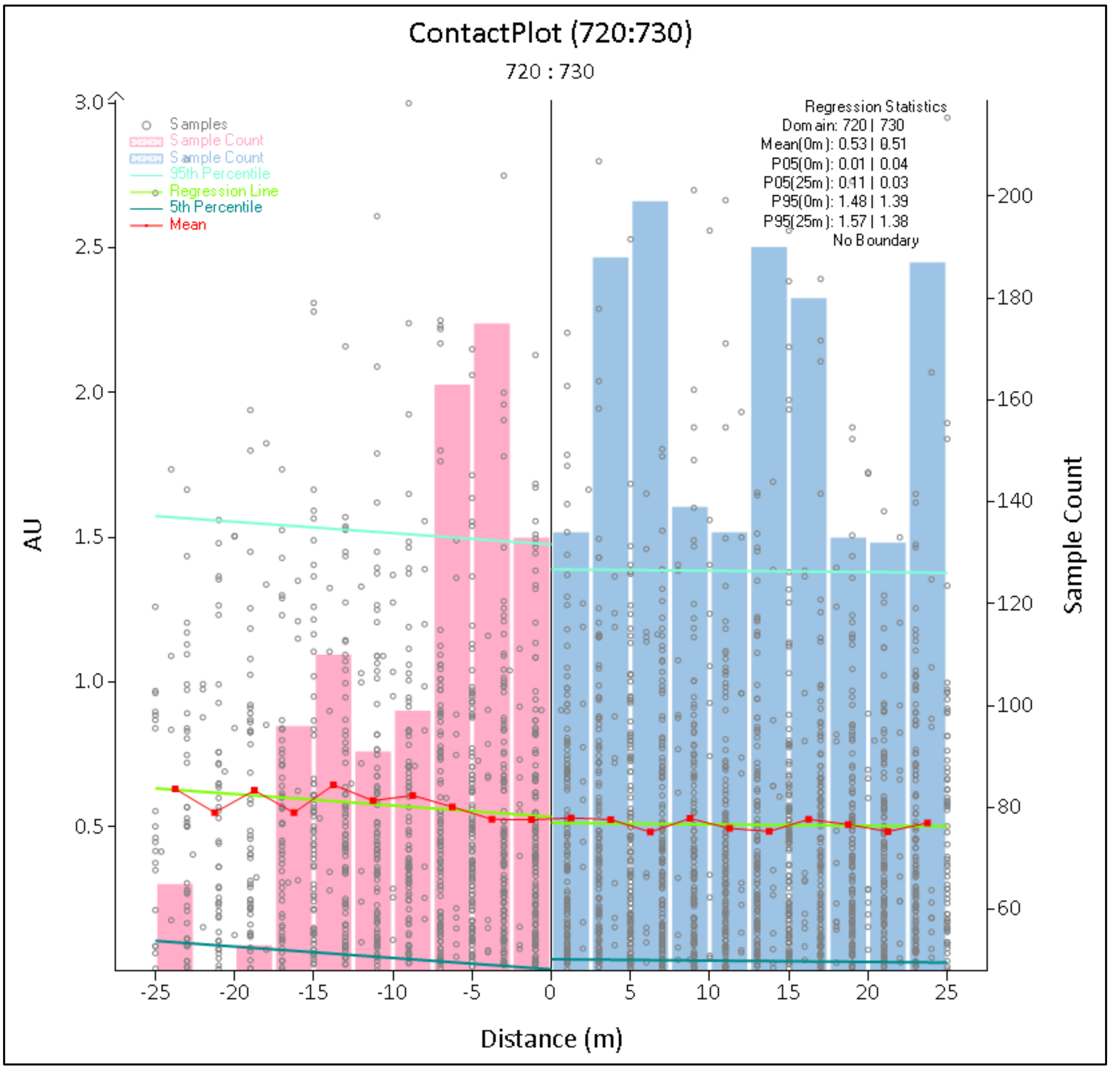
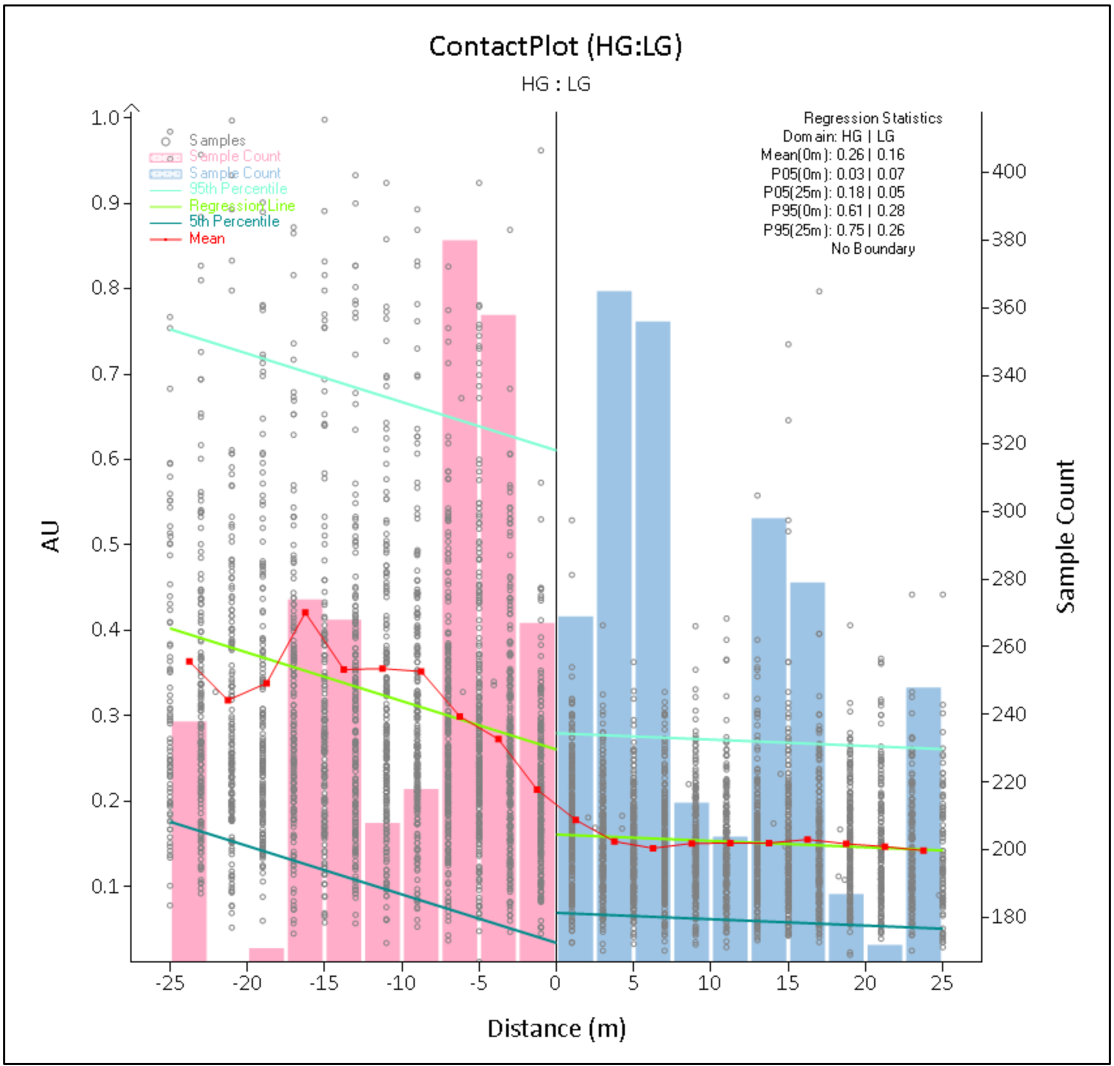


Figure 14-14:
Contact Profile for Transitional vs. Primary Zones Composite Au Grades – La Cumbre Deposit



Also, contact profiles were generated to evaluate the change in gold and silver grades across the boundary of the high-grade shell domain. The results for gold are shown in Figure 14-15. The change in gold grade at this contact is hard, the average gold grade is almost three times higher inside the shell grade, there is evident the great change in gold grade at this contact.

Figure 14-15:
Contact Profile of Gold Inside (HG) vs. Outside (LG) Shell Domain



14.8 GRADE CAPPING/OUTLIER RESTRICTIONS

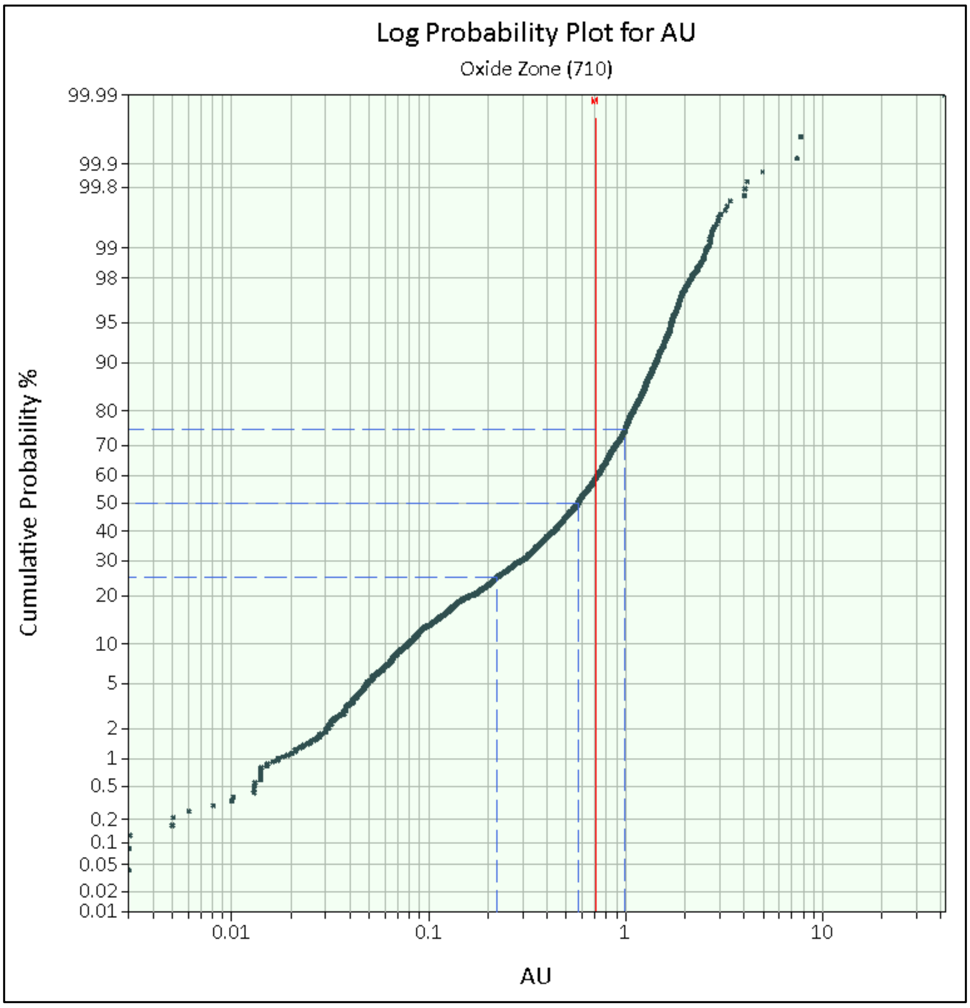
High-grade capping (cutting) was determined for each mineral zone. The composite data for each zone generally had a positively skewed grade distribution characterized by differences between mean and median grades, and moderate to high coefficients of variation (CV = standard deviation/mean). The CV is a relative measure of skewness and values greater than one can often indicate distortion of the mean by outlier data.

The requirement for high-grade caps was assessed via a number of steps to ascertain the reliability and spatial clustering of the high-grade composites. The steps completed as part of the high-grade cap assessment included:

- A review of the composite data to identify any data that deviate from the general data distribution. This was completed by examining the cumulative distribution function.
- A review of summary statistics comparing the percentage of metal and change in CV caused by the high-grade cuts.
- A visual 3D review to assess the clustering of the higher-grade composite data.

Based on the review, appropriate high-grade caps were selected for each zone. The application of high-grade caps resulted in relatively few data being capped with only 6 outlier values for the oxide zone (see Figure 14-16), 7 outlier values for the transitional zone (see Figure 14-17) and 7 outlier values for primary zone (see Figure 14-18). The capping was required to reduce the amount of metal, which would be artificially added during the estimation process in these zones and resulted in a minor reduction in the mean grade.

Figure 14-16:
Composite Au Grades Cumulative Probability Plot, Oxide Zone

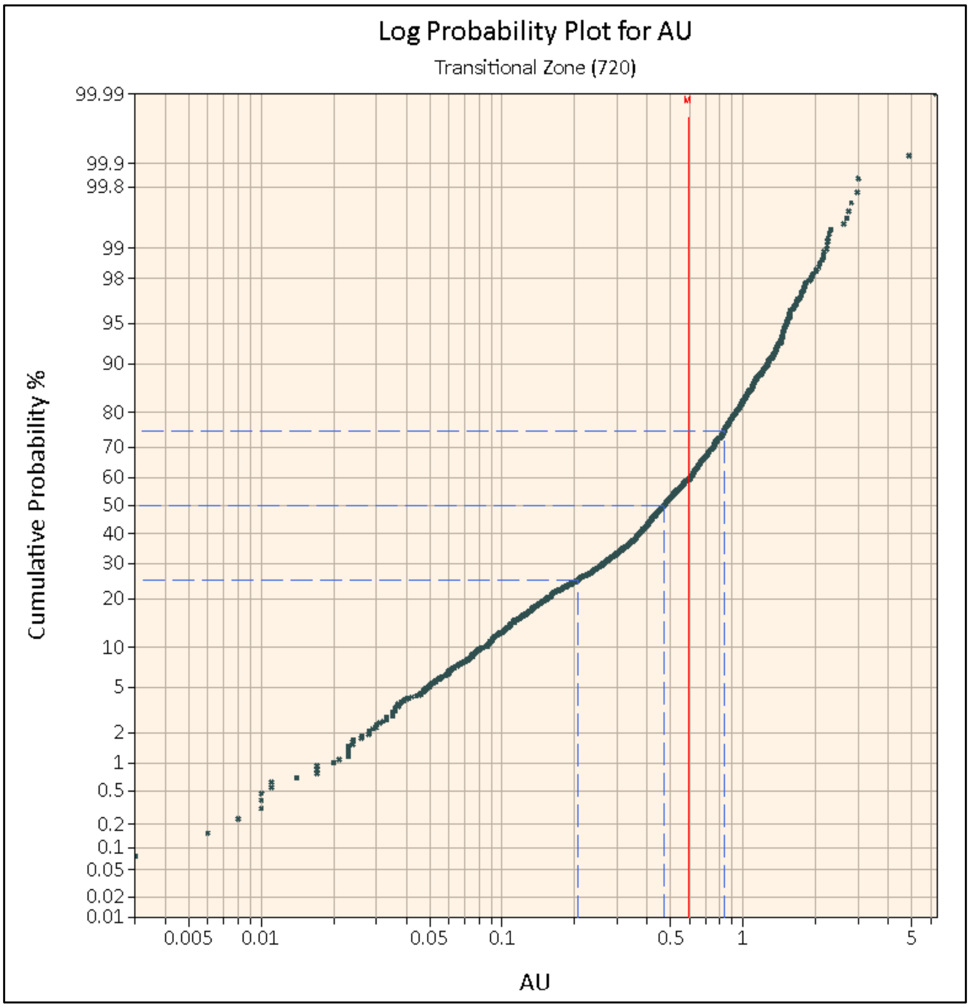


Graphical analysis of Figure 14-16, shows at least four mixed populations with log normal distributions, possibly due to multi-pulse mineralization or erratic mineralization with a high nugget effect. Evidence of this variability is shown by the CV of 1.46. The capping value for the oxide zone, was set at 3.0 Au g/t, corresponding to the upper inflexion of the probability curve.

The cumulative probability plot for composited gold values of the transitional zone shows a log normal distribution with an inflexion at 2.3 Au g/t. The transitional zone shows at least three mixed populations of gold grades. The transitional zone has a low CV value of 0.88. The capping value for the transitional zone was set at 2.3 Au g/t (see Figure 14-17).

Similar treatment was used for silver composite values and the capping value for the oxide zone was set at 12.0 g/t Ag and for the transitional zone at 5.0 g/t Ag.

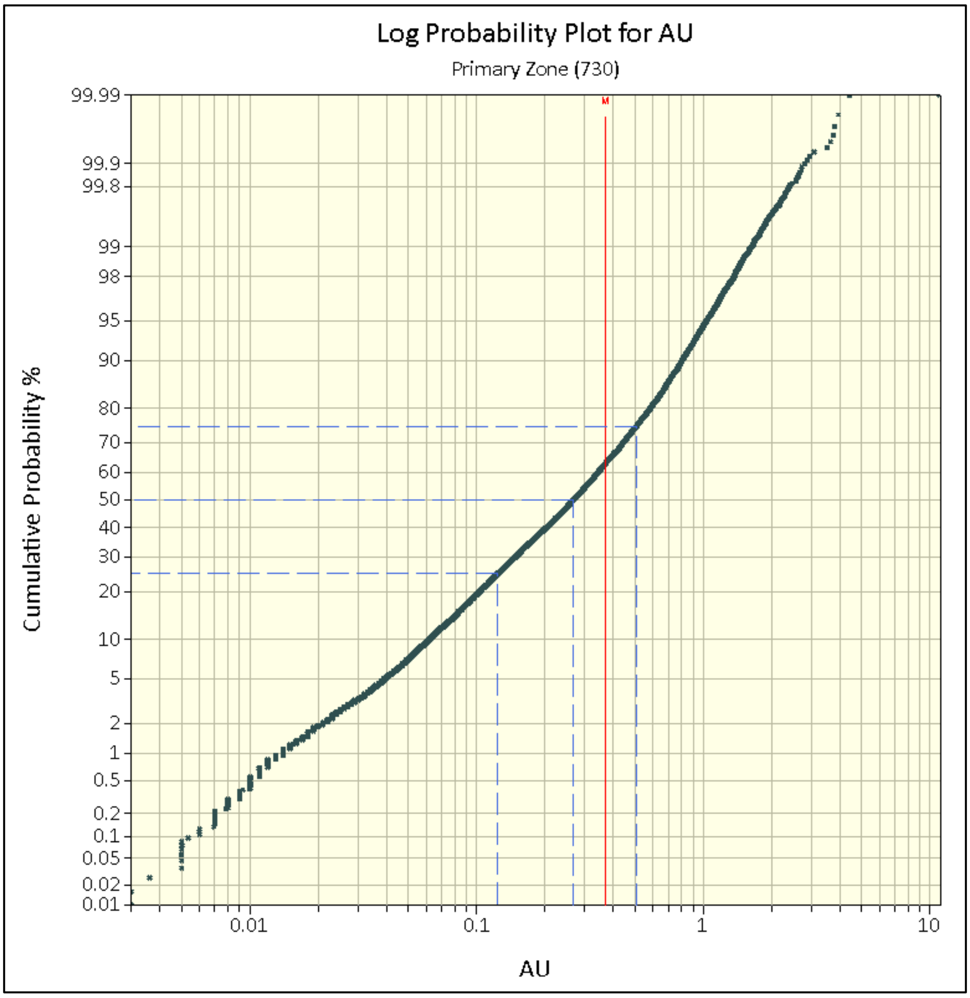
Figure 14-17:
Composite Au Grades Cumulative Probability Plot, Transitional Zone



The cumulative probability plot for composited gold values of the primary zone shows a log normal distribution with an inflexion at 3.0 Au g/t. The primary zone shows at least two mixed populations of gold grades. The primary zone has a low CV value of 0.98. The capping value for the transitional zone was set at 2.3 Au g/t (see Figure 14-18).

Similar treatment was used for silver composite values. The capping value for the oxide zone was set at 12.0 g/t Ag, for the transitional zone at 5.0 g/t Ag and for the primary zone at 11.0 g/t Ag.

Figure 14-18:
Composite Au Grades Cumulative Probability Plot, Primary Zone



14.9 CORRELATION ANALYSIS FOR GOLD VS. SILVER

Correlation analysis is performed to identify the strength of relationships between a pair of variables, in this case the grades of gold and silver. The correlation coefficient r varies between −1 and +1 where a perfect correlation is ±1 and 0 is the absence of correlations. Values of r between 0 and 1 reflect a partial correlation, which can be significant or not. For example, r=0.80 indicates that variable 1 is related to variable 2 at 80%. In some cases, the squared value of r is applied to always have a positive value and is defined by R or r2.

LINAMEC has compared the relationship between gold and silver inside oxide zone, transitional zone and, primary zone. From the analysis of Figure 14-19 it is concluded that there is a linear and weak positive correlation of the gold grades with respect to silver in the oxide zone with a correlation coefficient of R = 0.371. The transitional zone has a linear and positive correlation with a better correlation coefficient of R = 0.613, see Figure 14-20.

The primary zone has a linear and positive correlation, between gold and silver, with a correlation coefficient of R = 0.504, see Figure 14-21.

Figure 14-19:
Scatter Plot for Oxide Zone – La Cumbre Deposit

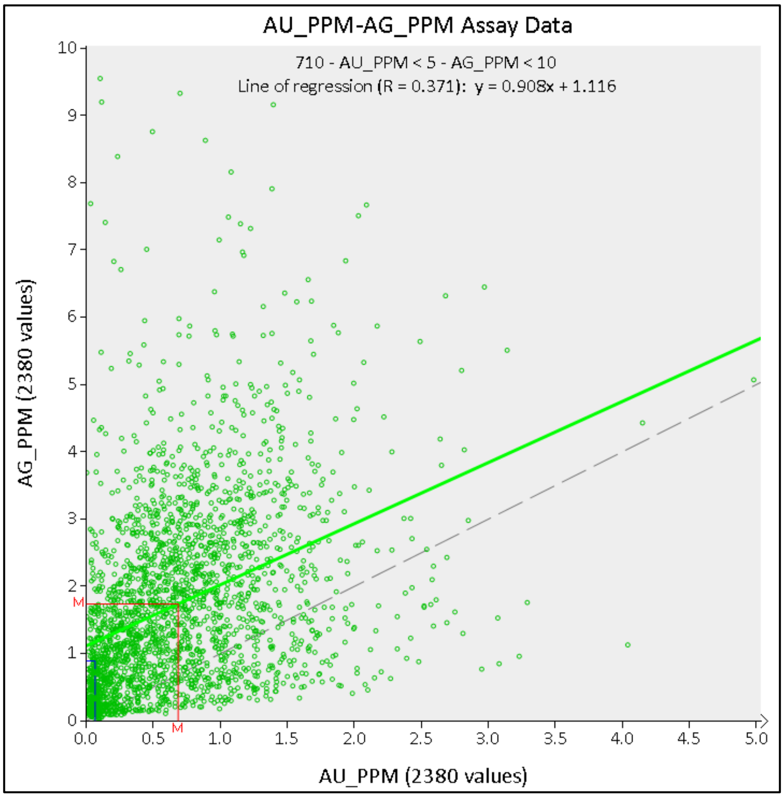


Figure 14-20:
Scatter Plot for Transitional Zone – La Cumbre Deposit

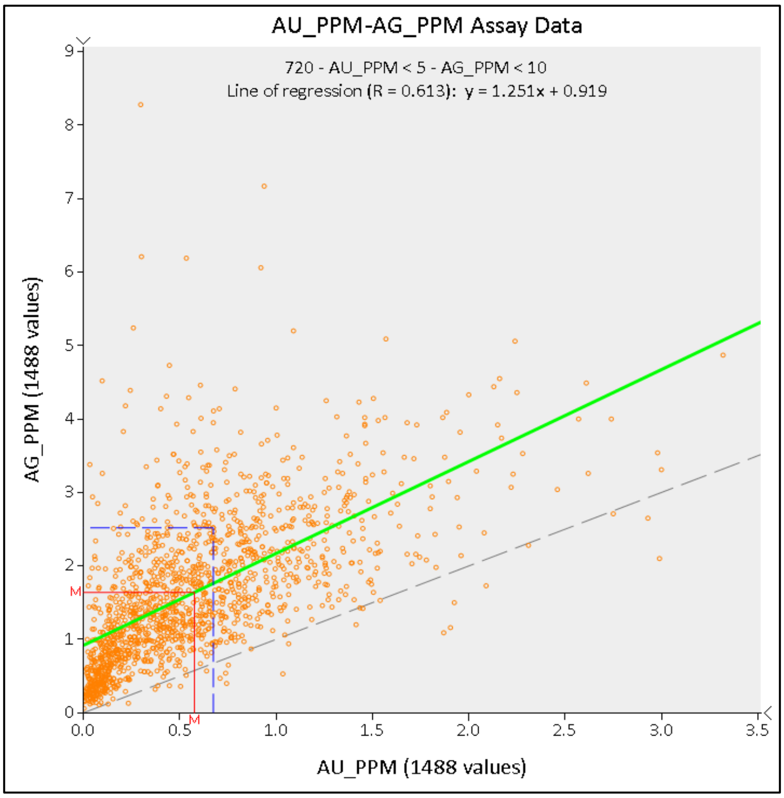
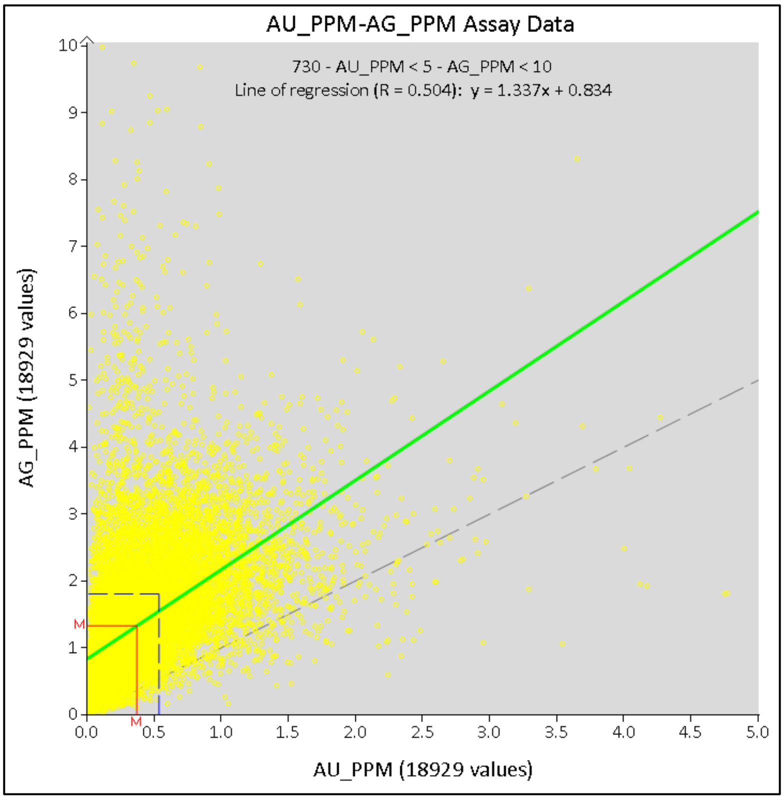


Figure 14-21:
Scatter Plot for Primary Zone – La Cumbre Deposit



14.10 DENSITY ASSIGNMENT

Density determinations, as described in Section 11.2, were used in mineral resource estimation. La Cumbre database contains 76 bulk density measurements and 39 absolute density measures, see Table 14-2.

Table 14-2:
Density Measures Used in Resource Estimation – La Cumbre Project

Method	Mineral Zone	Samples	SG g/cm3	Type
Pycnometer (saprolite)	AHS, OXD & ZTR	39	2.590	Absolute
Paraffin	OXD & ZTR	20	2.390	Bulk
Waxed Core	ZPR	56	2.645	Bulk
Total Bulk/weighted average	OXD, ZTR & ZPR	76	2.577	Bulk
Total Oxide Zone	OXD & ZTR	59	2.522	Abs. & Bulk
Total/weighted average	AHS, OXD, ZTP & ZPR	115	2.582	Abs. & Bulk

The average of all bulk density is 2.577 g/cm^3, the average of density measures for oxides and transitional zone was 2.522 g/cm^3 and this value was used to calculate the tonnage of the oxide zone and the transitional zone.

To determine the density of the primary zone, 56 diamond drill core samples were taken and submitted to the Bureau Veritas laboratory in Medellin. After eliminating the maximum and minimum values, the average of the results was 2.645 g/cm^3. This value was used to report the tonnes of this mineral zone which has now been included in the present mineral resource estimation for La Cumbre deposit.

The importance of dry bulk density as one of the three key parameters in the estimation of resources and reserves should not be overlooked. Poor estimates of density can easily have the same impact on resource tonnage as the errors inherent in the interpretation and modelling of the geometry of mineralized zones.

LINAMEC considers that the use of different density values for each mineral zone domain and rock type is good practice and recommends taking more measurements of densities to estimate tonnage in future mineral resource estimates.

14.11 VARIOGRAPHY

The degree of spatial variability in a mineral deposit depends on both the distance and direction between points of comparison. Typically, the variability between samples increases as the distance between those samples increases. If the degree of variability is related to the direction of comparison, then the deposit is said to exhibit anisotropic tendencies, which can be summarized with a search ellipse. The semi-variogram is a common function used to measure the spatial variability within a deposit.

The components of the variogram include nugget, sill, and range parameters. Often samples compared over very short distances, and even samples compared from the same location, show some degree of variability. As a result, the curve of the variogram often begins at some point on the y-axis above the origin, this point is called the nugget. The nugget is a measure of not only the natural variability of the data over very short distances, but also a measure of the variability which can be introduced due to errors during sample collection, preparation, and the assay process.

Experimental variograms were calculated and modelled using the GEMS geostatistical tool for gold and silver inside oxide, transitional and primary zones. Also, for a high-grade zone defined by a grade shell > 0.22 Au g/t. General aspects of the variography are:

- Experimental variograms were calculated from capped 2 m composite data.
- Down hole and directional correlograms were generated.
- Variogram orientations reflected obvious trends for strike, dip and thickness in the data.
- Variograms were modelled with a nugget effect and one spherical structure.
- The modelled variogram for the Au oxide high grade zone with a range of 70 m at 135.00° direction and 0.097-nugget effect is shown in Figure 14-22.
- The modelled variogram for the Au transitional high-grade zone, with a range of 72.87 m at 112.5° main direction and 0.052 nugget effect is shown in Figure 14-23.
- The modelled variogram for the Au primary high grade zone, with a range of 76.81 m at 90.0° main direction and 0.226 nugget effect is shown in Figure 14-24.
- The modelled variogram for the Au high grade shell, all mineral zones, with a range of 110.40 m at 90.0° main direction and 0.545 nugget effect is shown in Figure 14-25.

Figure 14-22:
Modelled Variogram for High Grade Au in Oxide Zone

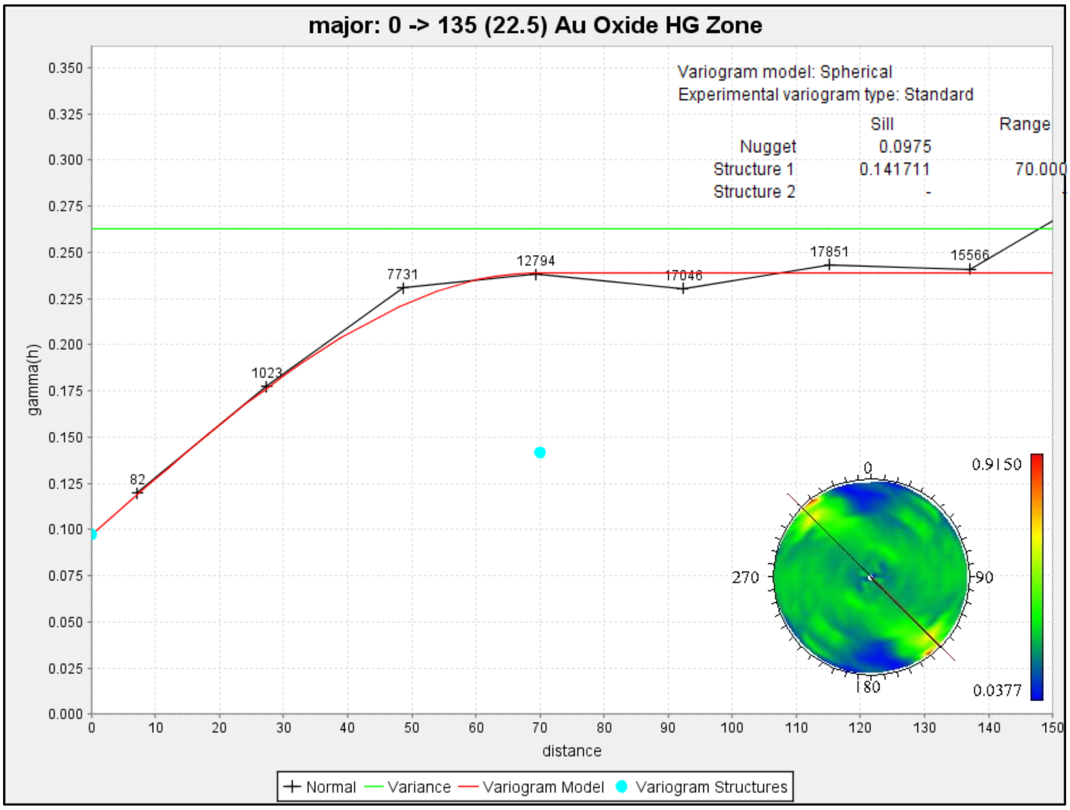


Figure 14-23:
Modelled Variogram for High Grade Au in Transitional Zone

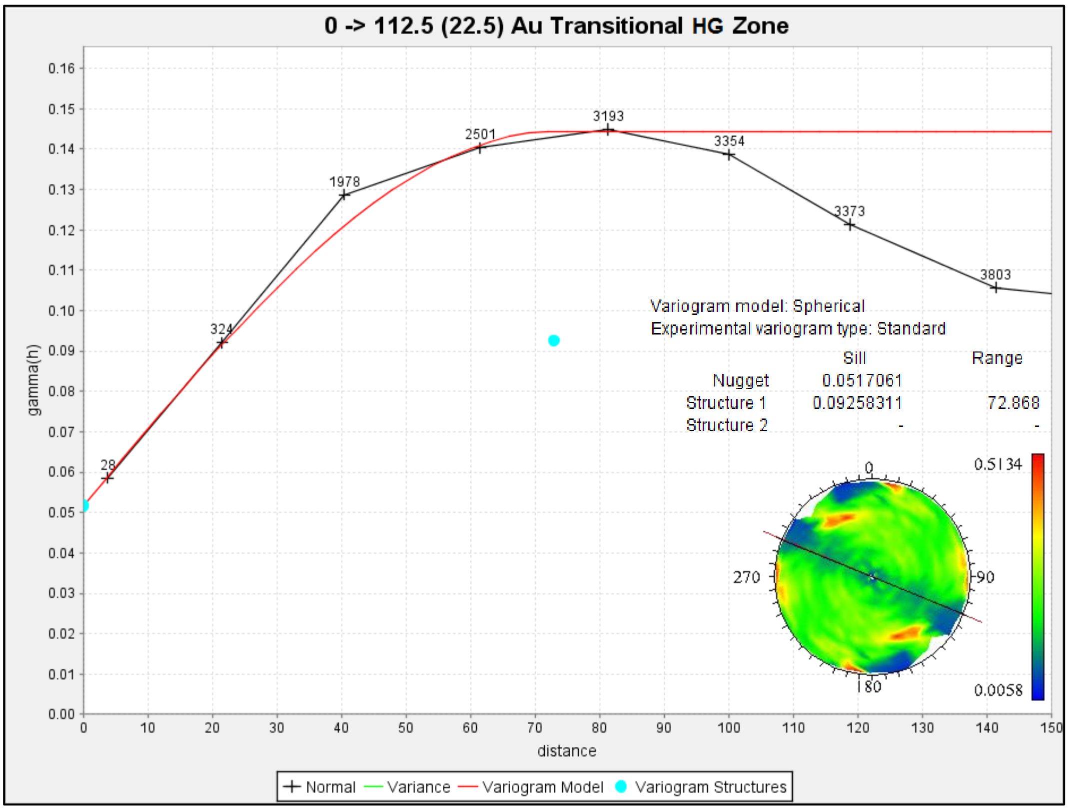


Figure 14-24:
Modelled Variogram for High Grade Au in Primary Zone

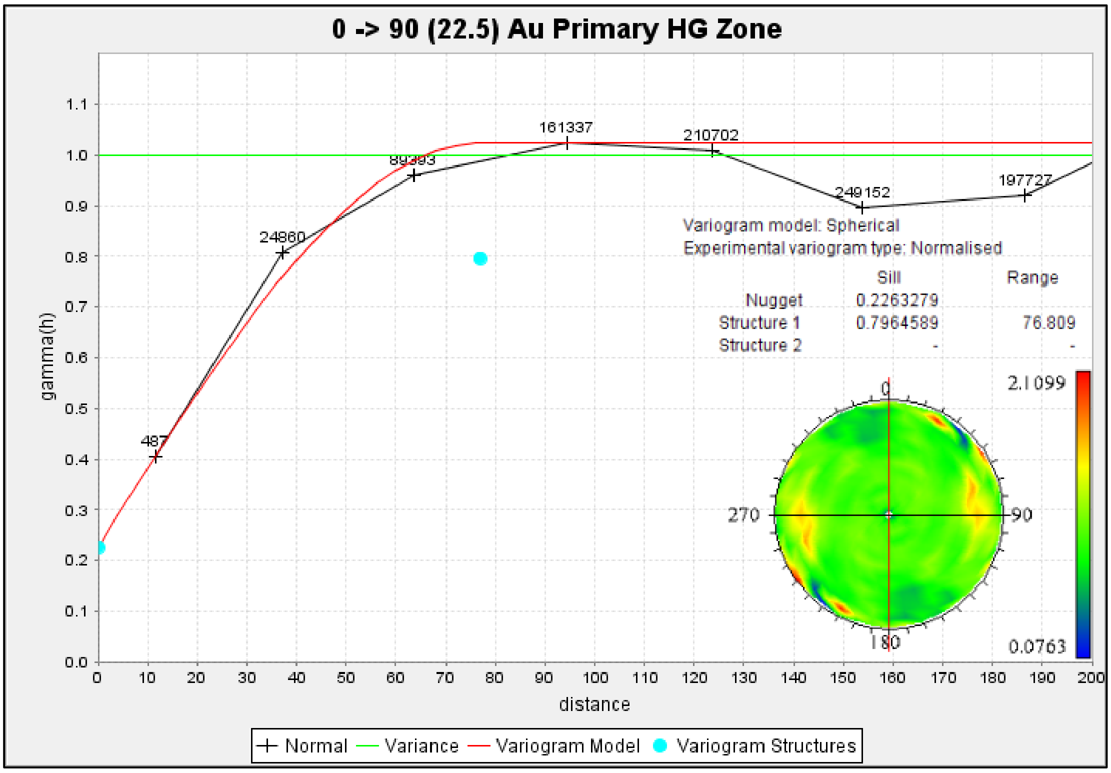
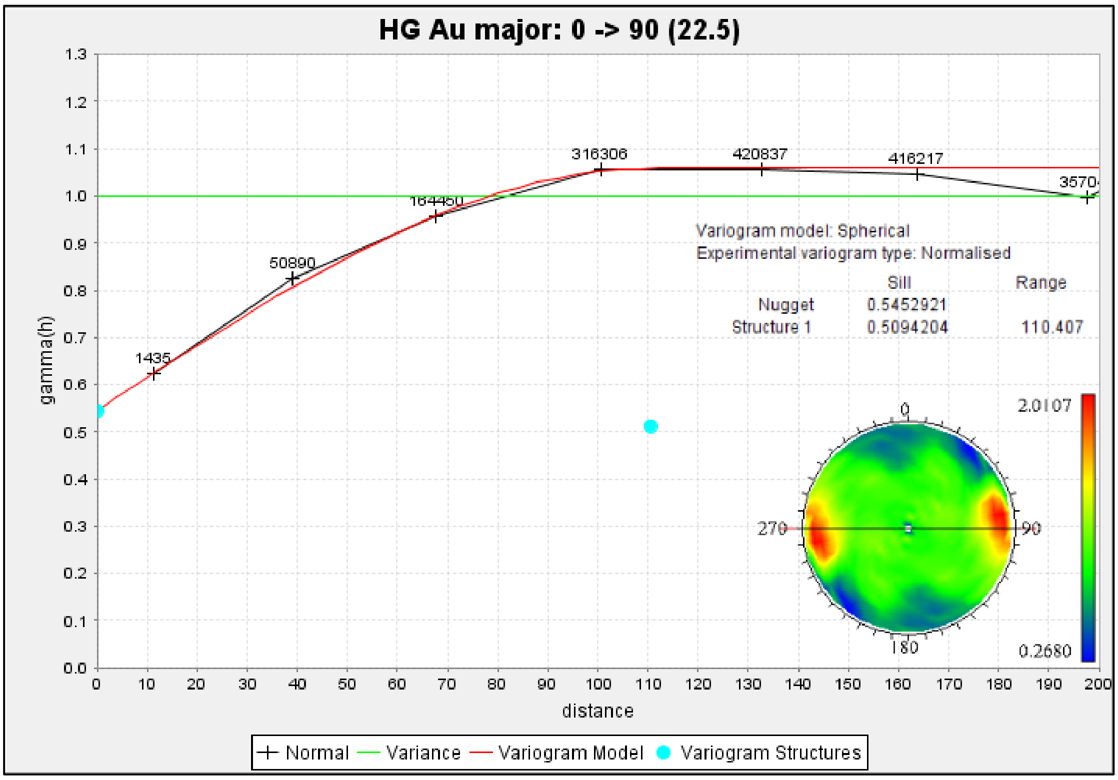


Figure 14-25:
Modelled Variogram for Au High Grade All Zones



14.12 ESTIMATION AND INTERPOLATION METHODS

Two block models were created in GEMS to enable grade estimation, mine planning and mine design. A first block model, with block size of 6m x 6m x 6m, was selected to adjust the oxide, transitional and primary zones with volume calculations using the Leapfrog Geo wireframe models. A second high-grade block model, with block size of 6m x 6m x 6m, was selected to adjust the volume calculations of Leapfrog Geo 0.22 Au g/t grade-shell (see Table 14-3 for block model geometries and Figure 14-26 for the solid grade-shell).

Ordinary kriging (OK) was the method selected for interpolation of each oxide zone, transitional zone and primary zone blocks. The nearest neighbor (NN) method and inverse distance squared (ID2) were used for verification and validation of the block model.

The sample search strategy was based upon analysis of the variogram model anisotropy, mineralization geometry and data distribution for the oxide zone, transitional zone, primary zone and high-grade shell.

During estimation runs, each block model was coded with the number of composites selected, kriging variance, and block variance, which were later used in the determination of the resource classification.

LINAMEC used GEMS to build a block model with the attributes listed in Table 14-4. The rock type model was coded taking into account the percentage of the volume of each block that is inside the solid, this percentage must be equal to or greater than 50% to ensure that all of the gold and silver mineralization blocks have at least 50% of their volumes in the mineralization wireframes for the oxide domain and transitional domain, respectively. The tonnage factors were assigned directly based on the rock type model. A block model with the distance from block centroids to the nearest composite was created to help develop the resource classification criteria.

Table 14-3:
Block Model Geometry for La Cumbre Deposit

Number of blocks		Block size *m		Origin and rotation	
Columns:	192	Column size:	6	X:	420,452
Rows:	153	Row size:	6	Y:	584,800
Levels:	174	Level size:	6	Z:	2,002
				Rotation:	0°

Figure 14-26:
Grade-Shell Solid (>0.22 Au g/t) Used for High Grade Au Estimation

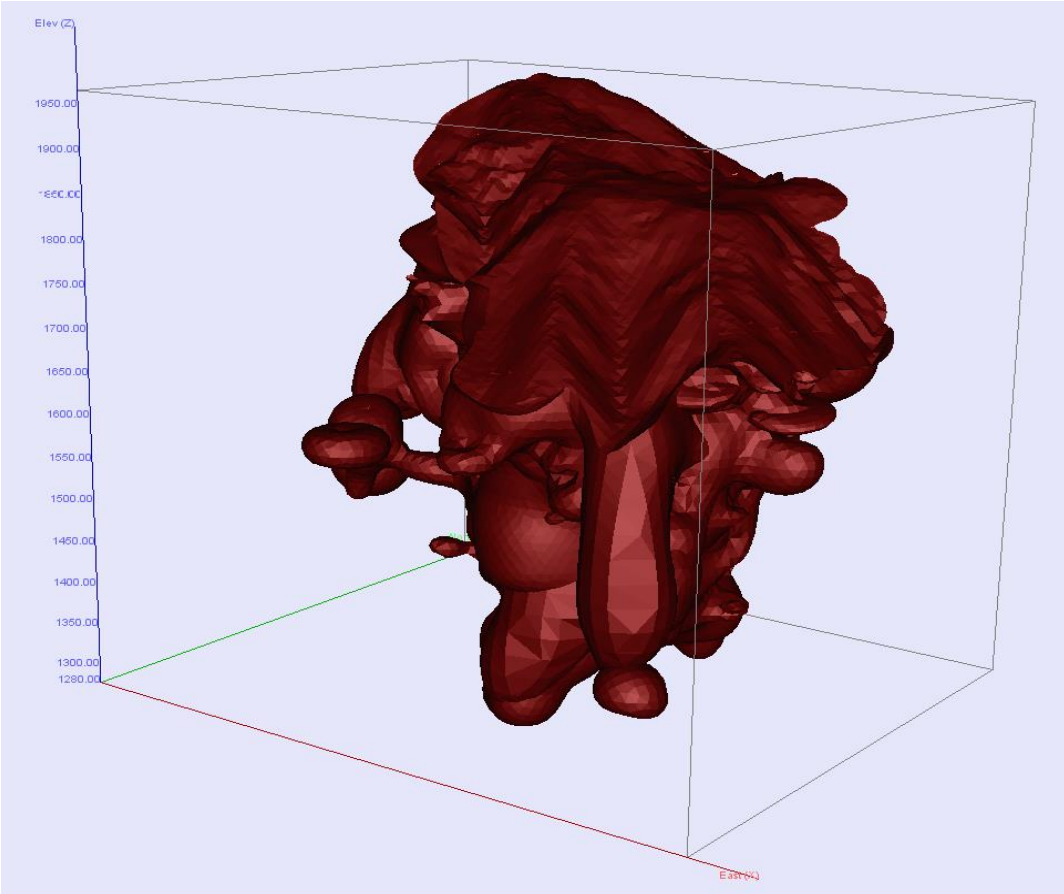


Table 14-4:
Block Model Folder Descriptions – La Cumbre Project

Name	Data Type	Decimals	Product Factor	Description
Rock Type	Integer	0	1	ASH=700, ZOX=710, ZTR=720, ZSP=730, HG=780
Density	Double	3	1.000	Bulk Density
Percent	Single	2	1.000	Percent of block inside wireframe solid
Economic	Double	3	1.000	Not used
Material	Single	3	1.000	Not used
Elevation	Single	3	1.000	Not used
AU	Double	3	1.000	Interpolated gold capped grades
AG	Double	2	1.000	Interpolated silver capped grades
CU	Double	2	1.000	Interpolated copper capped grades
S	Double	2	1.000	Interpolated sulfur capped grades
RESOURCE	Single	3	1.000	Measured=501, Indicated=502, Inferred=503
NN_AU	Double	3	1.000	Interpolated Nearest Neighbor Au grade
NN_AG	Double	2	1.000	Interpolated Nearest Neighbor Ag grade
ID2_AU	Double	3	1.000	Interpolated Inverse Distance Squared Au grade
ID2_AG	Double	2	1.000	Interpolated Inverse Distance Squared Ag grade
Variance	Double	2	1.000	Kriging block variance

The interpolation parameters are summarized in Table 14-5 for Oxide Zone, Table 14-17 for Transitional Zone, Table 14-18 for Primary Zone and Table 14-19 for Grade-Shell Domine.

Table 14-5:
Interpolation Parameters for Au Oxide Mineral Zone

Oxide Zone	1st run	2nd run	3rd run
Estimation method	Ordinary Kriging	Ordinary Kriging	Ordinary Kriging
Samples used 2m compos	OXIDES	OXIDES	OXIDES
Minimum sample quantity	3	2	1
Maximum sample quantity	10	10	10
Az first direction	124	124	124
Dip first direction	0	0	0
Az second direction	34	34	34
Nugget / Sill (spherical)	0.275 / 0.367	0.275 / 0.367	0.275 / 0.367
Search radius [m]	2/3 range	range	range + 1/2 range
first/second/third radius[m]	50/25/12	75/50/25	100/75/50
High grade treatment			
High grade transition value	3.0 Au g/t	3.0 Au g/t	3.0 Au g/t
High grade search radius [m]	25/12.5/6	37.5/25/12.5	50/37.5/25
Categorization	Measured	Indicated	Inferred

Table 14-6:
Interpolation Parameters for Au Transitional Mineral Zone

Transitional Zone	1ª run	2ª run	3ª run
Estimation method	Ordinary Kriging	Ordinary Kriging	Ordinary Kriging
Samples used 2m compos.	TRANS	TRANS	TRANS
Minimum sample quantity	3	2	1
Maximum sample quantity	10	10	10
Az first direction	124	124	124
Dip first direction	0	0	0
Az second direction	34	34	34
Nugget / Sill (spherical)	0.049 / 0.167	0.049 / 0.167	0.049 / 0.167
Search radius [m]	2/3 range	range	range + 1/2 range
first/second/third radius[m]	50/25/12	75/50/25	100/75/50
High grade treatment			
High grade transition value	2.5 Au g/t	2.5 Au g/t	2.5 Au g/t
High grade search radius [m]	25/12.5/6	37.5/25/12.5	50/37.5/25
Categorization	Measured	Indicated	Inferred

Table 14-7:
Interpolation Parameters for Au Primary Mineral Zone

Primary Zone	1ª run	2ª run	3ª run
Estimation method	Ordinary Kriging	Ordinary Kriging	Ordinary Kriging
Samples used 2m compos.	Primary	Primary	Primary
Minimum sample quantity	3	2	1
Maximum sample quantity	10	10	10
Az first direction	90	90	90
Dip first direction	0	0	0
Az second direction	0	0	0
Nugget / Sill (spherical)	0.226 / 0.796	0.226 / 0.796	0.226 / 0.796
Search radius [m]	2/3 range	range	range + 1/2 range
first/second/third radius[m]	50/25/12.5	76/38/20	114/57/28.5
High grade treatment			
High grade transition value	3.0 Au g/t	3.0 Au g/t	3.0 Au g/t
High grade search radius [m]	25/12.5/6	37.5/25/12.5	57/28.5/14
Categorization	Measured	Indicated	Inferred

Table 14-8:
Interpolation Parameters for Au High-Grade Shell

Transitional LG Inside Shell	1ª run	2ª run	3ª run
Estimation method	Ordinary Kriging	Ordinary Kriging	Ordinary Kriging
Samples used 2m compos.	HG	HG	HG
Minimum sample quantity	3	2	1
Maximum sample quantity	10	10	10
Az first direction	90	90	90
Dip first direction	0	0	0
Az second direction	34	34	34
Nugget / Sill (spherical)	0.545/ 0.509	0.545/ 0.509	0.545/ 0.509
Search radius [m]	2/3 range	range	range + 1/2 range
first/second/third radius[m]	74/37/19	110/55/28	165/83/41
High grade treatment			
High grade transition value	2.5 Au g/t	2.5 Au g/t	2.5 Au g/t
High grade search radius [m]	37/19/10	55/28/14	83/41/21
Categorization	Measured	Indicated	Inferred

Similar parameters were used to estimate the silver grades and populate the block models.

14.13 BLOCK MODEL VALIDATION

Block model validation is the process of confirming that the model produced is an accurate reflection of the data entered into the system. The ideal situation occurs when the model can be validated (at least in part) with a measure of production compliance by reconciling a grade control model and actual production.

14.13.1 Volumetric Validation

A comparison between the measured volumes of the solids generated during the geological modelling and the volume of mineralization in the block model was carried out and indicated that the volume of mineralized blocks in the block model corresponds well with the volume of the mineralized wireframes.

14.13.2 Block Model Comparison against Drill Data

A detailed validation of the OK, ID2 and NN estimate was completed for each zone and included both an interactive 3D and statistical review. The validation included a visual comparison of the input data against the block model's grade in plan and cross sections. It also included a review of the distribution of recorded estimation controls including search pass, average sample distance, number of contributing samples and number of drill holes.

A spatial comparison of the mean grade of the input composites against the block model's grade was also made. The models were divided into slices by directions (Northing, Easting and Elevation) and average grades calculated into each domain. Similarly, the composite averages were also computed. Examination of these plots indicated that the models were appropriately honoring the input data and trends.

14.13.3 Visual Checks

Estimated block grades and composite grades were compared visually in a plan views, longitudinal sections and transversal sections, show a good agreement (see Figure 14-27 and Figure 14-28). The updated model was also visually compared with the 2018 model, oxides and transitional domains. The two models agreed very closely with each other in areas where no new drill hole composites had been included. In the primary zone, now included, the observed changes in grade were consistent with the new composite grades.

Figure 14-27:
Gold Composites, Mineral Zones and Block Model on 1804 Level

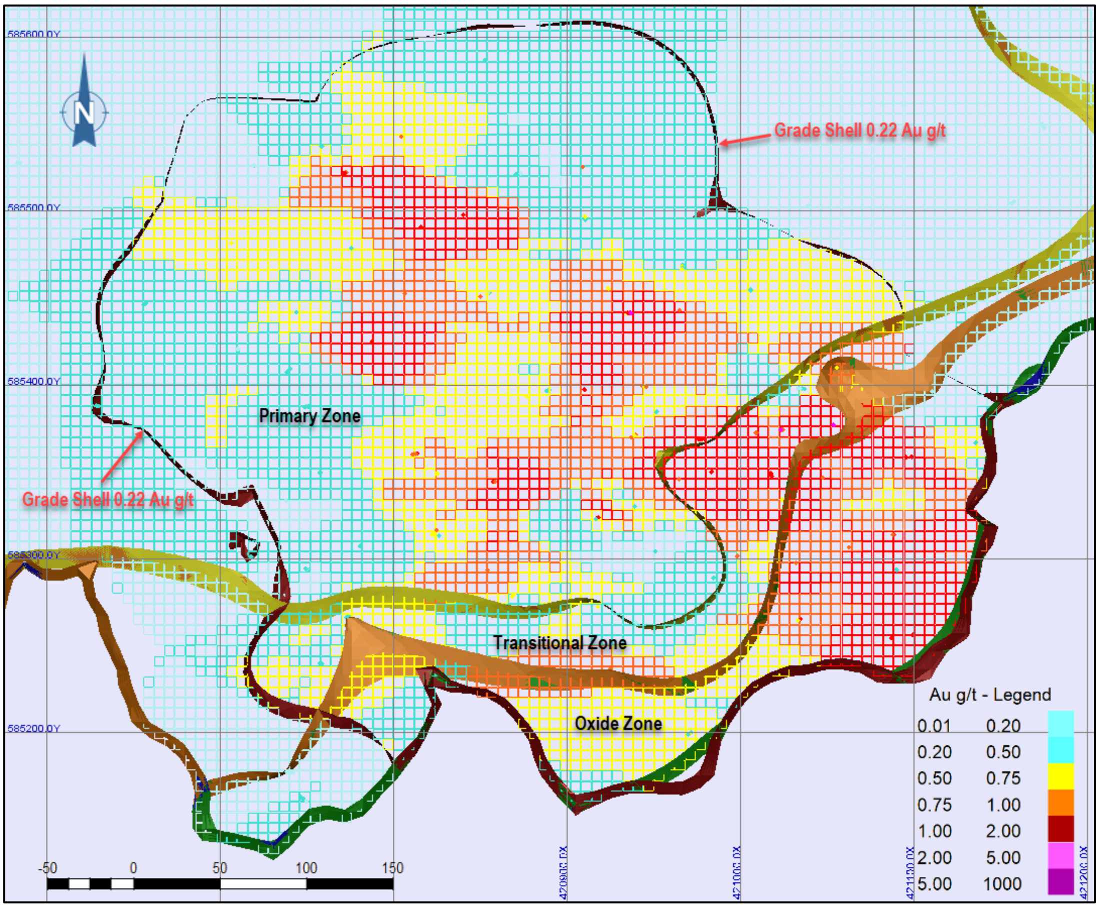
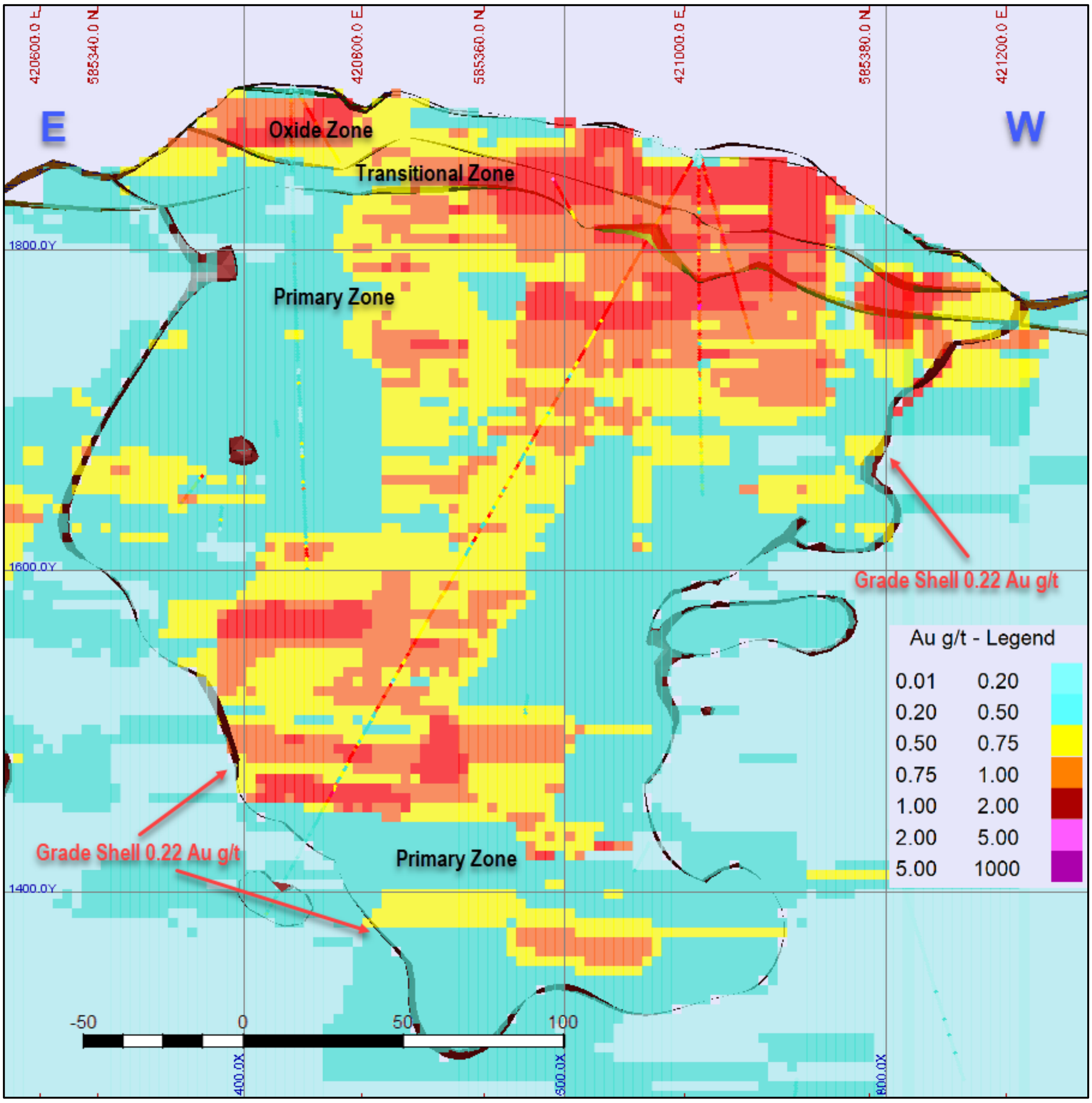


Figure 14-28:
Gold Composites, Mineral Zones and Block Model on Section 2000-EW



14.13.4 Inverse Distance Block Models

LINAMEC created inverse distance squared (ID2) block models as a validation tool for the OK resource models. Using the same composite files and search parameters that were utilized for the OK models, ID2 blocks were populated with the numerical weighted value for each single block. The comparison shows that the OK model grades and the ID2 model grades are quite similar and that no grade bias is present (see Figure 14-36, Grade and Tonnage Distribution for All Mineral Zones).

LINAMEC is of the opinion that the OK resource models show negligible bias as compared to the ID2 models for the oxide zone, transitional zone and primary zone in the La Cumbre deposit.

14.13.5 Nearest Neighbor Block Models

LINAMEC created nearest neighbour (NN) block models as a validation tool for the OK resource models. Using the same composite files and search parameters that were utilized for the OK models, NN blocks were populated with the closest composite to each centroid block.

The comparison shows that the OK model grades and the NN model grades are similar and that no grade bias is present.

LINAMEC is of the opinion that the OK resource models show negligible bias as compared to the NN models for the oxide zone, transitional zone and primary zone in the La Cumbre deposit.

14.13.6 Swath Plots (Trend Analysis)

A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions throughout the deposit. Using the swath plot, grade variations from the OK model are compared to the distribution derived from the ID2 grade model and NN grade model. These are obtained by plotting the mean of the estimated OK, ID2 and NN grades in East-West, North-South, and vertical slices. The estimated swath grades should normally follow each other, although the estimated OK swath grades generally appear smoother, due to the increased amount of averaging, than the ID2 and NN swath grades.

On a local scale, the NN model does not provide reliable estimations of grade, but on a much larger scale, it represents an unbiased estimate of the grade distribution based on the underlying data. Therefore, if the OK model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the NN distribution of grade.

Figure 14-29, shows the swath plots of gold for the La Cumbre deposit along the easting, Figure 14-30, shows the swath plots of gold for the La Cumbre deposit along the northing, Figure 14-31, shows the swath plots of gold for the La Cumbre deposit by the elevation, Figure 14-32, shows the swath plots of gold for the La Cumbre deposit along the 135° strike cross section.

The analysis of the swath plots for the La Cumbre deposit shows an acceptable correlation of ordinary kriging, inverse distance squared and nearest neighbour estimates, but better resolution might be achieved with the definition of improved lateral limits for the mineralization. The swath plots show a relatively consistent mean grade comparison across the deposit, but exceptions to the relative agreement in mean grade do occur as follows:

- On the east side of the La Cumbre deposit the mean OK grade is higher than the NN mean grade, see Figure 14-29.
- On the north side the mean OK grade is again higher than the mean NN grade, see Figure 14-30.
- On the elevation the mean of OK grade is remarkable lower than the NN grade, see Figure 14-31, due to the increased amount of averaging of OK swath grades (extra smoothing), than the NN swath grades.
- On the 135° cross section the mean of OK grade is slightly greater than ID2 and NN average grades, see Figure 14-32, showing good grade distribution in the main mineralization direction.

LINAMEC is of the opinion that the OK resource models are adequate based on the current information and show very little bias as compared to the ID2 and NN validation models.

Figure 14-29:
Swath Plots for Au, All Domains – Easting, La Cumbre Deposit

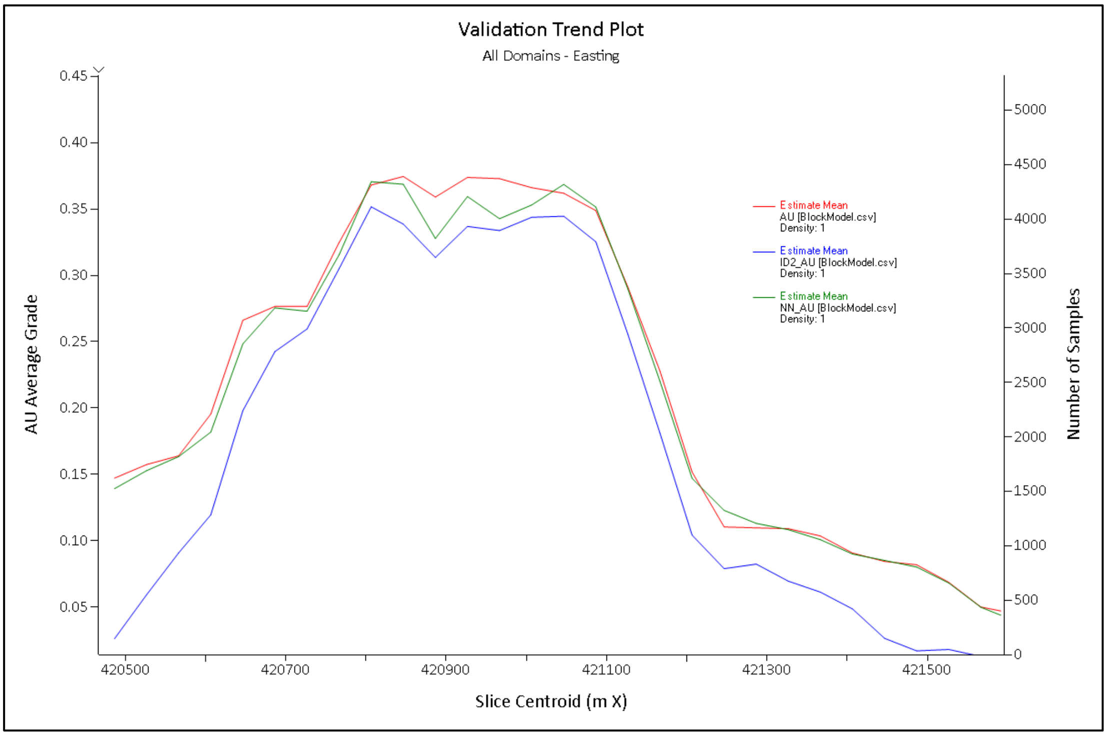


Figure 14-30:
Swath Plots for Au, All Domains – Northing, La Cumbre Deposit

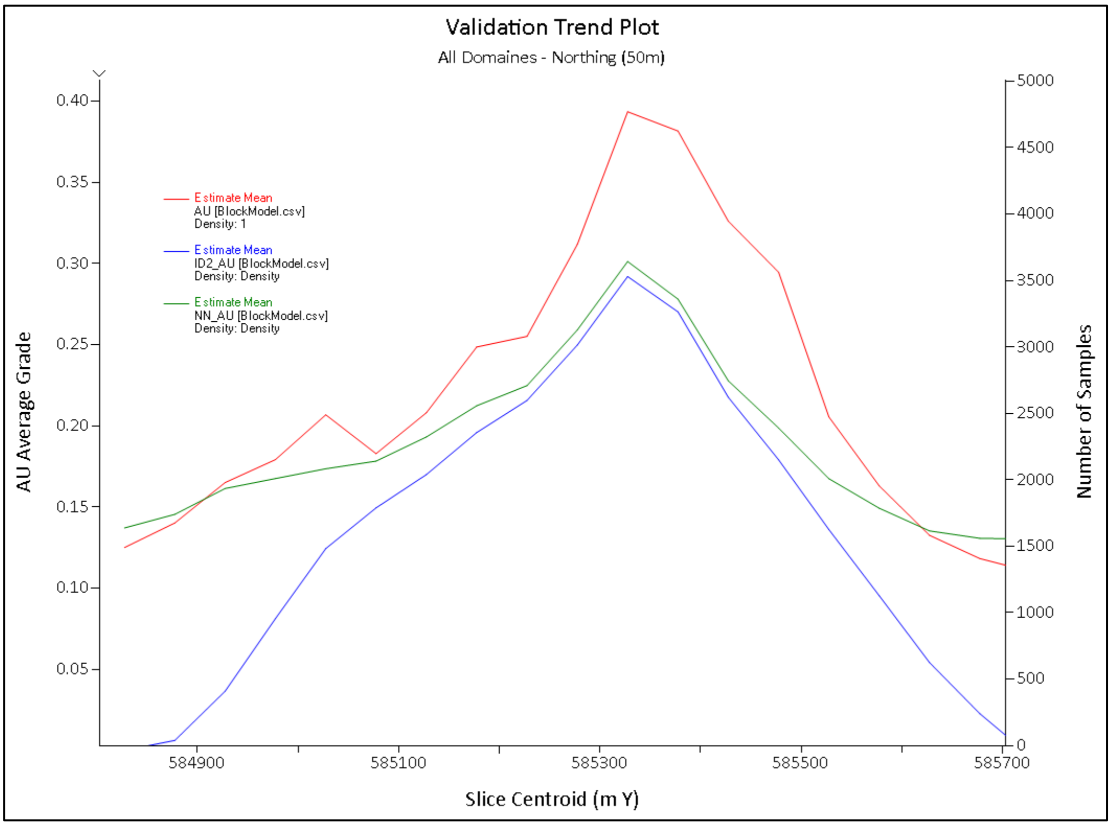


Figure 14-31:
Swath Plots for Au, All Domains – Elevation, La Cumbre Deposit

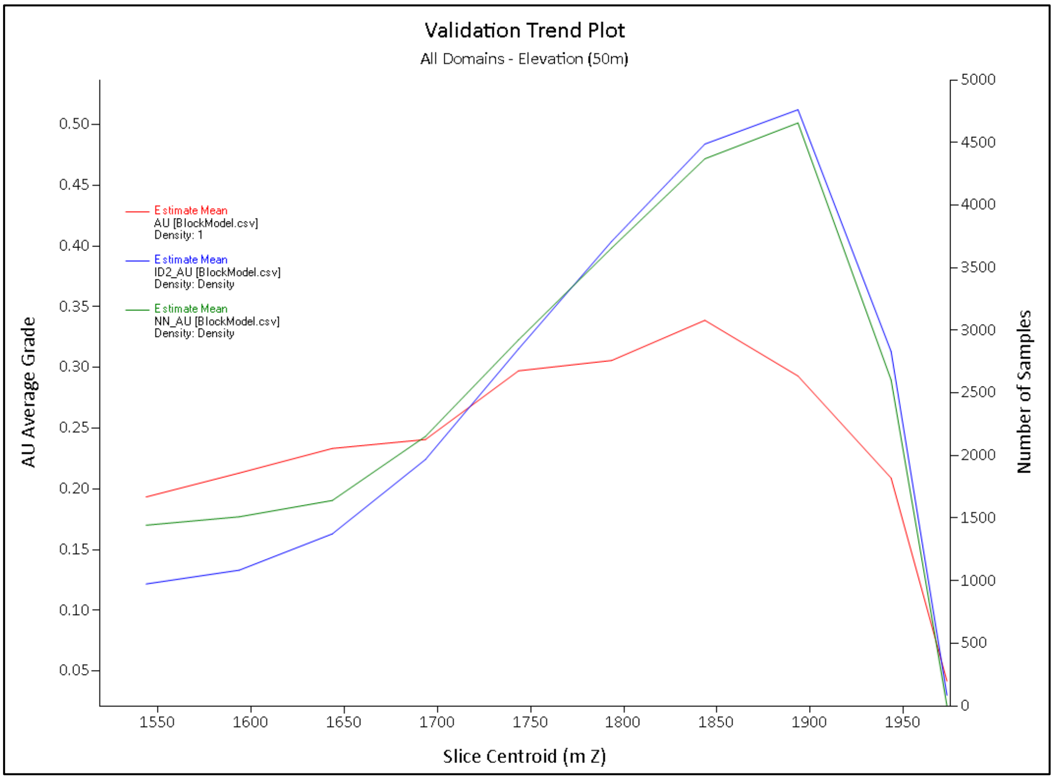
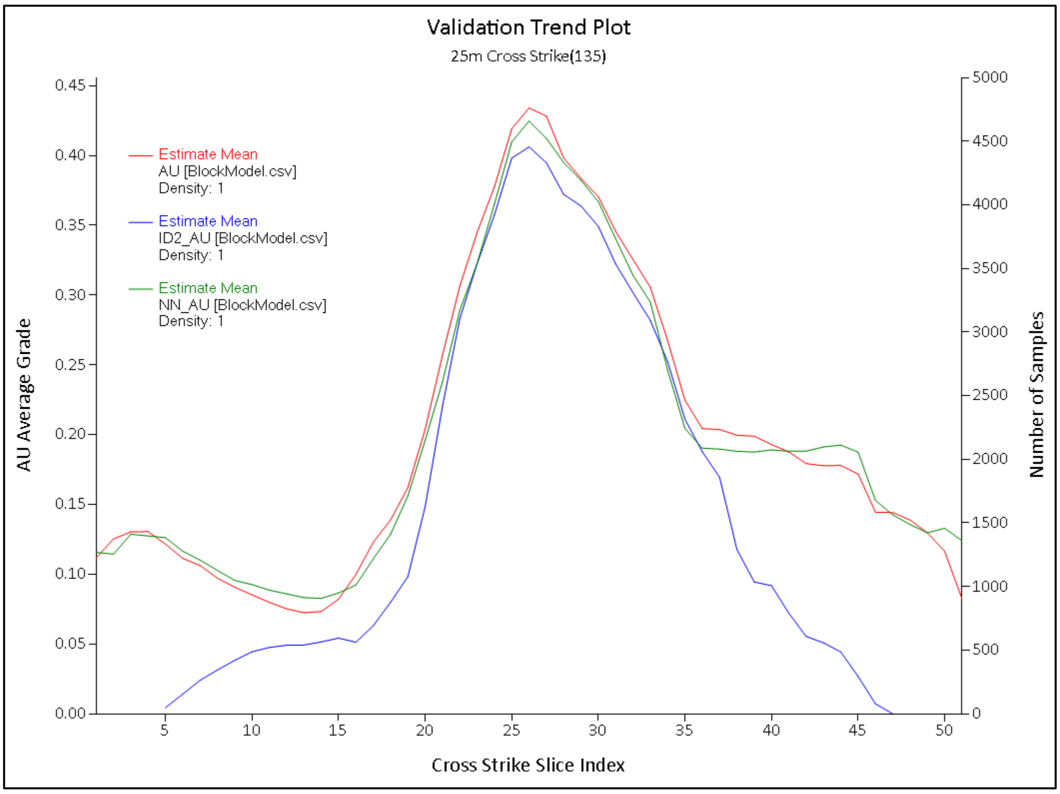


Figure 14-32:
Swath Plots for Au, All Domains – 135° Cross Section, La Cumbre Deposit



14.13.7 Cross Validation Scatterplot

This method is based on successively removing a point data from the data array and estimating the given location, which has become unsampled, using the remaining data (Davis 1987). The procedure is repeated many times with the different data points and the estimates are compared with the sample which has been removed. Plotting the estimates values against the true (sample) values allows to diagnose and quantify the global and conditional biases.

The method is popular among the resource geologists and is widely used for validating mineral resource estimation. The procedure of cross validation is in particular useful for comparing results by several estimation methods, which can be ranked by the degree of their conditional biases (Sinclair and Blackwell 2002). The more the slope departs from unity the greater is the conditional bias of the estimate.

Cross validation takes the form of a scatterplot with the original sample or composite value (denoted by “z”) plotted against the estimated value for the same sample location (denoted by “z*”) plotted on the Y-axis. The original sample value is plotted on the X-axis with the estimated value on the Y-axis (see Figure 14-33).

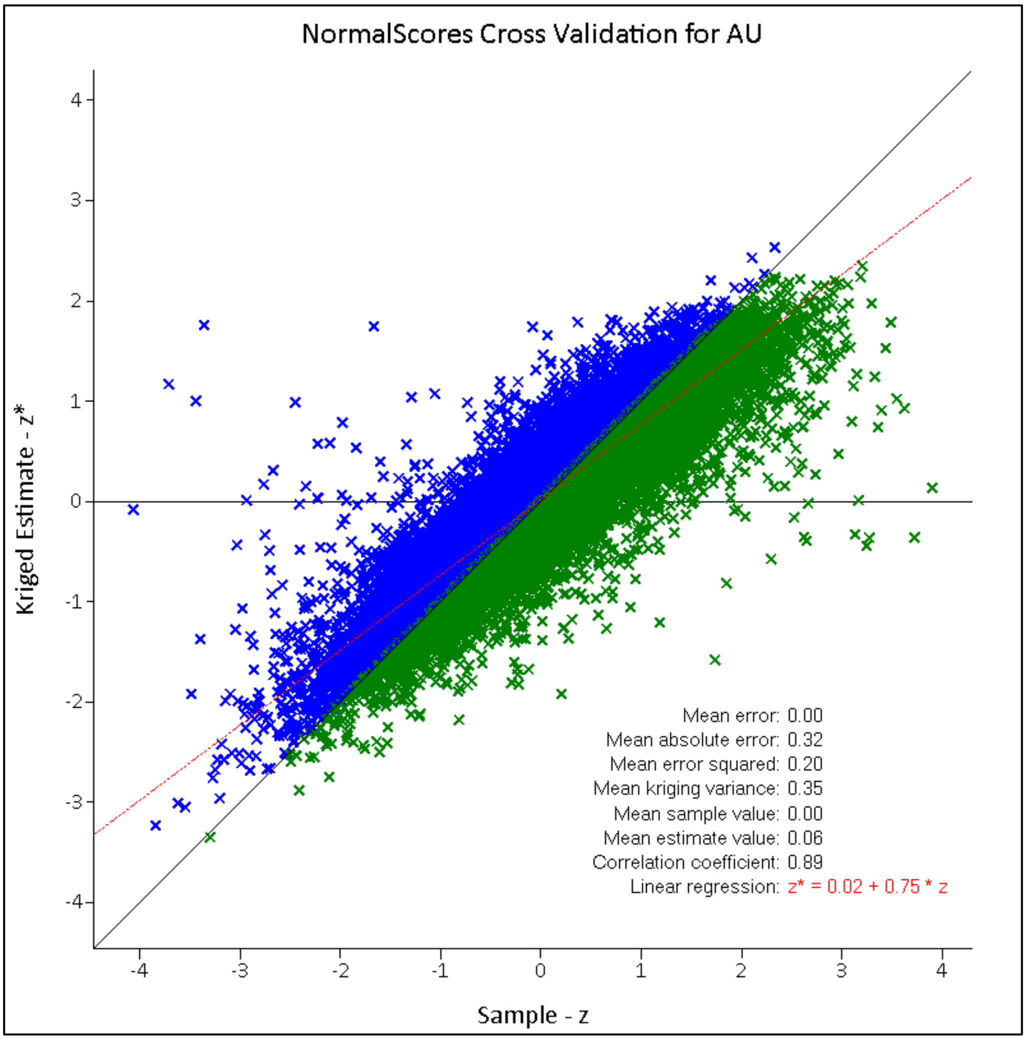
Estimates at each sample location are carried out using Ordinary Kriging. A default search is set up using the modelled variogram ranges and directions and a minimum of 2 and maximum of 100 samples. Samples from the same drillhole are also used for estimation by default.

With autocorrelation and a good kriging model, the doted red line should be closer to the 1:1 black line, the individual points on the scatterplot are color coded according to whether they plot above or below this line. Error statistics and a linear regression line and equation are calculated and plotted and are displayed in orange.

When a cross validation is inserted below a continuity model that has a transform applied to the data, the cross validation will be plotted in the transformed data units, not original data units. Figure 14-33 shows the cross-validation plot using normal score transformed data.

The kriging estimate (z*) shows a good correlation coefficient (0.89) when is compared with real sample values (z). The mean absolute error is equal to 0.32, the mean error squared is equal to 0.20, and the mean kriging variance is equal to 0.35, these are acceptable values that indicate that the conditional bias, which occurs when the interpolation process is ordinary kriging, is not significant. This is also corroborated by the slope of the straight line of the estimated data (red dotted line) which does not deviate too much from the 1:1 or 45° line, in black in Figure 14-34.

**Figure 14-33:
Cross Validation Scatterplot, All Domains – La Cumbre Deposit**



14.14 CLASSIFICATION OF MINERAL RESOURCES

The mineral resource estimate for the La Cumbre Deposit conforms to the requirements of CIM Definition Standards (2014) and comply with the codes of Canadian National Instrument NI 43-101. The criteria used to categorize the mineral resources include the robustness of the input data, the confidence in the geological interpretation, including the continuity of gold and silver grades within the mineralized zones, the distance from data and the amount of data available for block estimates within the respective mineralized zones.

For the resource classification, the number of composites used to evaluate each block was considered. These parameters are evaluated and the result is recorded in a field named "RESOURCE" and used for resource classification.

Measured, indicated and inferred mineral resource categories have been assigned to blocks in the block model using criteria generated during validation of the grade estimates, with detailed consideration of the CIM (2014) categorization guidelines. A summary of the criteria considered and confidence level are listed in Table 14-9, Table 14-10 and Table 14-11.

Measured resources take 2/3 of the range; indicated resources reach the whole range. Finally, inferred resources take the whole range plus 50% of the range. The criteria for resource categorization are summarized in Table 14-9, Table 14-10 and Table 14-11.

Table 14-9:
Summary of Criteria Categorization of Resource for Oxide Zone

Distance (m)	No. of samples	Resource Code	Resource Category
0 – 50	>=3	501	Measured
0 – 75	>=2	502	Indicated
0 - 100	1	503	Inferred

Table 14-10:
Summary of Criteria Categorization of Resource for Transitional Zone

Distance (m)	No. of samples	Resource Code	Resource Category
0 – 50	>=3	501	Measured
0 – 75	>=2	502	Indicated
0 - 100	1	503	Inferred

Table 14-11:
Summary of Criteria Categorization of Resource for Primary Zone

Distance (m)	No. of samples	Resource Code	Resource Category
0 – 120	>=3	501	Measured
0 – 180	>=2	502	Indicated
0 - 270	1	503	Inferred

14.15 REASONABLE PROSPECTS OF ECONOMIC EXTRACTION

CIM Definition Standards for Mineral Resources and Mineral Reserves (May, 2014) provides the following definition:

"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."

Mineral resources are reported above a cut-off grade of 0.218 Au g/t for the oxide zone and 0.218 Au g/t for transitional zone, within three-dimensional geological wireframes constructed with Leapfrog Geo and edited by LINAMEC to constrain the gold mineralization in the mineral resource estimate to zones defined by mineralized diamond drill core intersections. Mineral resources above the cut-off grade have reasonable prospects for economic extraction, based on mineralization continuity, shape and distribution inside each mineral zone.

The "reasonable prospects for eventual economic extraction" requirement generally imply that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade which takes the extraction scenarios and processing recovery into account.

The "reasonable prospects for eventual economic extraction" were tested using a series of floating cone pit shells generated by Whittle software, based on projected technical and economic assumptions. The resource-limiting pit shells were generated using recoverable gold grades that include contributions from silver and adjustments for projected metallurgical recoveries.

In constructing the pit shell, the following technical and economic parameters were assumed:

Table 14-12:
Operating Costs

Cost	US$
Mining open pit:	1.95/t.
Processing (leach + ADR):	9.08/t.
Processing (milling + flotation)	7.14/t.
G&A	1.12
Gold cost of sales:	47.0/oz.

Table 14-13:
Pit Slope Angles

Mineral Zone	Pit Slope Angles
Overburden (OVB, ASH)	39°
Oxide (OXD)	39°
Transitional (ZTR)	43°
Primary (ZPR)	38°

Table 14-14:
Metallurgical Recoveries for Gold

Mineral Zone	Recovery	Processes
Oxide	85.5%	Leach
Transitional	85.5%	Leach
Primary	84.1%	Mill

Table 14-15:
Metal Prices

Gold Price US$/oz	Solver Price US$/oz
1,750.00	22.00

Using the projected operating costs listed here, the base case cut-off grade for the La Cumbre deposit resources is estimated to be 0.218 Au g/t for leachable material and 0.179 Au g/t for flotation and gravimetric method. See Figure 14-34, for the optimized Open Pit Shell for leachable material and Figure 14-35, for the optimized Open Pit Shell for the primary sulfide materials.

Metallurgical gold recoveries for oxide and transitional zones are considered same value for economic estimations based on results of mixed columns performed leaching both materials.

Figure 14-34:
Optimized Open Pit Shell at 0.218 Au g/t plus Block Model (Leach Process)

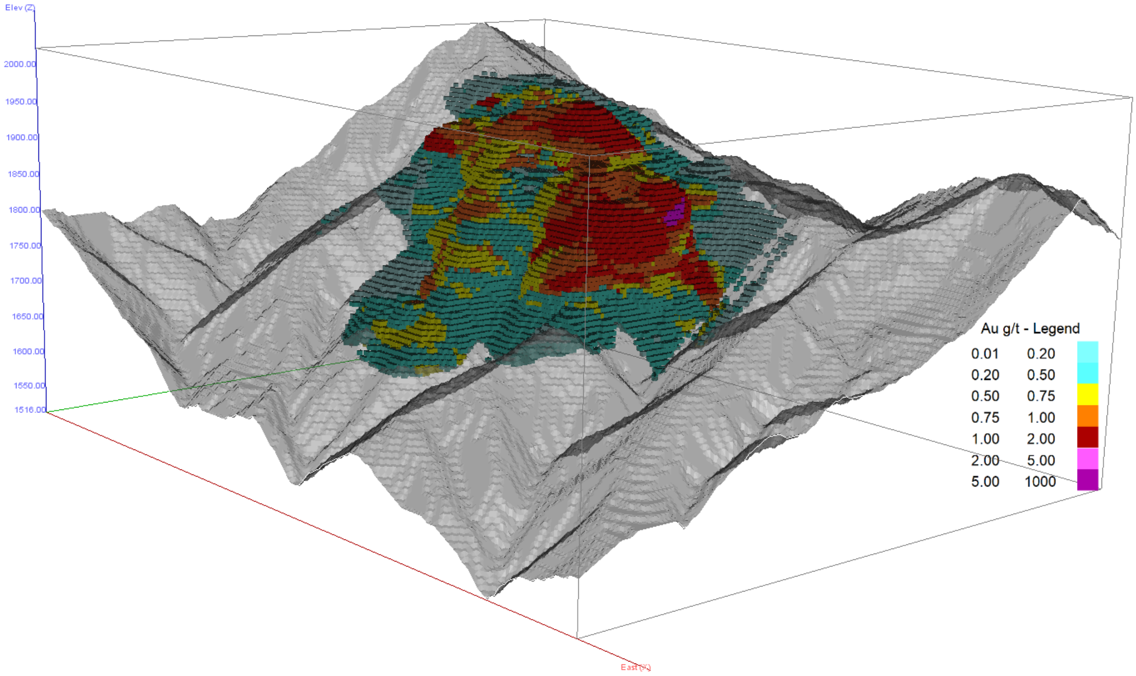
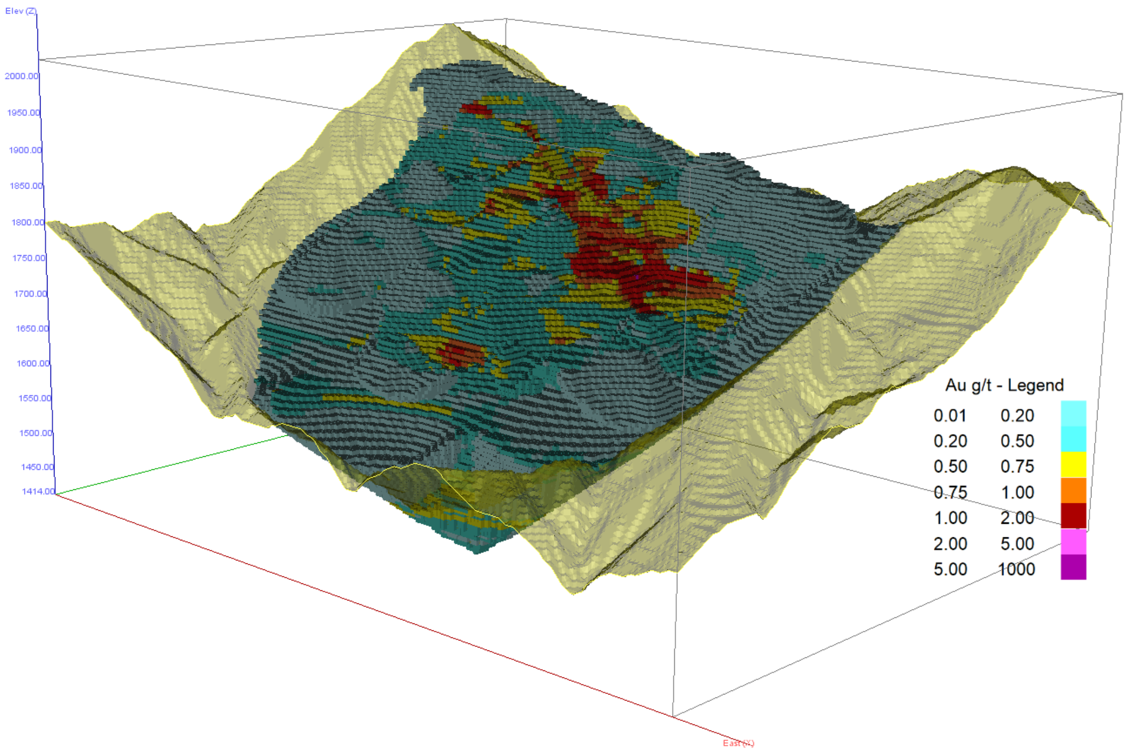


Figure 14-35:
Optimized Open Pit Shell at 0.179 Au g/t plus Block Model (Flotation Process)



14.16 MINERAL RESOURCE STATEMENT

The new resource classification for this update consists of first estimating the resources within the high-grade shell, followed by estimating the resources classified by mineral zone outside this grade shell, taking into account the ore to be leached and the ore to be flotation processed. The total resource estimate for the La Cumbre deposit is shown in Table 14-19.

A new grade shell, generated in Leapfrog Geo, with a cut-off grade of 0.22 Au g/t, called "High Grade", has been used to disclose the mineral resources estimated for the La Cumbre Project in this NI 43-101 Technical Report and Preliminary Economic Assessment. The variographic analysis within this high-grade domain have shown that there is a strong correlation and spatial continuity in gold grades, as demonstrated during structural analysis of gold grades with ranges up to 110 m in the main direction, see Figure 14-25.

Updated mineral resources above cut-off grade 0.218 Au g/t for the Oxide Zone of the La Cumbre Deposit measured and indicated resources consist of 14,260,640 tonnes with an average grade of 0.745 Au g/t and 31,471 tonnes of inferred mineral resources with an average grade of 0.412 Au g/t (see Table 14-16). For the Transitional Zone of the La Cumbre Deposit, measured and indicated resources for this zone, consist of 8,288,826 tonnes with an average grade of 0.644 Au g/t and 777,827 tonnes of inferred mineral resources with an average grade of 0.422 Au g/t (see Table 14-17).

This present updated mineral resource estimate now includes the Primary Zone. The ore from this mineral zone will be processed by gravimetry and flotation. The La Cumbre Deposit measured and indicated resources for this zone consists of 113,156,067 tonnes with an average grade of 0.462 Au g/t and 94,789 tonnes of inferred mineral resources with an average grade of 0.337 Au g/t (see Table 14-18).

All mineral resources were estimated by Mr. Fernando Linares, principal geologist with Linares Americas Consulting S.A.C. using the Canadian Institute of Mining (CIM) Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions. The resource estimate was reviewed and validated by Mr. Walter La Torre, independent consultant and has an effective date of December 31, 2021.

Table 14-16:
Mineral Resource Statement for Oxide Zone – December 31, 2021:
Batero Gold Corp. – La Cumbre Gold Project

Total Resources for Oxide Zone (Cutoff = 0.218 Au g/t)							
Resource	**Volume**	**Density**	**Tonnage**	**Au g/t**	**Au oz**	**Ag g/t**	**Ag oz**
Measured	5,304,580	2.520	13,367,541	0.768	330,230	1.62	694,795
Indicated	354,404	2.520	893,099	0.403	11,565	1.38	39,713
Meas. + Ind.	**5,658,984**	**2.520**	**14,260,640**	**0.745**	**341,795**	**1.60**	**734,507**
Inferred	12,489	2.520	31,471	0.412	417	1.31	1,321

Notes to accompany La Cumbre Mineral Resource tables:

1. Mineral Resources have an effective date of December 31, 2021. The Qualified Person for the estimate is Mr. Walter La Torre, CP and MAusIMM.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
3. Mineral Resources are reported within a conceptual optimized pit that uses the following input parameters: Au price: US$1,750/troy oz and US$22.0/troy oz Ag, mining cost: US$1.95/t, process cost (including G&A): US$10.20/t processed, gold selling cost: US$47.00/troy oz and overall slope angle of 39°.
4. Gold recovery for leachable materials were fixed at 85.5%.
5. Mineral Resources (leachable materials) are reported using a 0.218 Au g/t cut-off grade.
6. Totals may not sum due to rounding as required by reporting guidelines.

Table 14-17:
Mineral Resource Statement for Transitional Zone – December 31, 2021:
Batero Gold Corp. – La Cumbre Gold Project

Total Resources for Transitional Zone (Cutoff = 0.218 Au g/t)							
Resource	**Volume**	**Density**	**Tonnage**	**Au g/t**	**Au oz**	**Ag g/t**	**Ag oz**
Measured	2,887,300	2.520	7,275,997	0.670	156,665	1.62	378,460
Indicated	401,916	2.520	1,012,829	0.461	15,000	1.43	46,552
Meas. + Ind.	**3,289,217**	**2.520**	**8,288,826**	**0.644**	**171,665**	**1.59**	**425,012**
Inferred	308,662	2.520	777,827	0.422	10,561	1.35	33,667

Notes to accompany La Cumbre Mineral Resource tables:

1. Mineral Resources have an effective date of December 31, 2021. The Qualified Person for the estimate is Mr. Walter La Torre, CP and MAusIMM.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
3. Mineral Resources are reported within a conceptual optimized pit that uses the following input parameters: Au price: US$1,750/troy oz and US$22.0/troy oz Ag, mining cost: US$1.95/t, process cost (including G&A): US$10.20/t processed, gold selling cost: US$47.00/troy oz and overall slope angle of 43°.
4. Gold recovery for leachable materials was fixed at 85.5%.
5. Mineral Resources (leachable materials) are reported using a 0.218 Au g/t cut-off grade.
6. Totals may not sum due to rounding as required by reporting guidelines.

Table 14-18:
Mineral Resource Statement for Primary Zone – December 31, 2021:
Batero Gold Corp. – La Cumbre Gold Project

Total Resources for Primary Zone (Cutoff = 0.179 Au g/t)							
Resource	**Volume**	**Density**	**Tonnage**	**Au g/t**	**Au oz**	**Ag g/t**	**Ag oz**
Measured	41,126,022	2.645	108,778,327	0.466	1,630,754	1.50	5,263,075
Indicated	1,655,100	2.645	4,377,739	0.362	50,911	0.04	5,168
Meas. + Ind.	**42,781,122**	**2.645**	**113,156,067**	**0.462**	**1,681,665**	**1.45**	**5,268,243**
Inferred	35,837	2.645	94,789	0.337	1,026	1.14	3,484

Notes to accompany La Cumbre Mineral Resource tables:

1. Mineral Resources have an effective date of December 31, 2021. The Qualified Person for the estimate is Mr. Walter La Torre, CP and MAusIMM.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
3. Mineral Resources are reported within a conceptual optimized pit that uses the following input parameters: Au price: US$1,750/troy oz and US$22.0/troy oz Ag, mining cost: US$1.95/t, process cost (including G&A): US$8.26/t processed, gold selling cost: US$47.00/troy oz and overall slope angle of 38°.
4. Gold recovery in the primary zone was fixed at 84.1%.
5. Mineral Resources for primary material are reported using a 0.179 Au g/t cut-off grade.
6. Totals may not sum due to rounding as required by reporting guidelines.

Table 14-19:
Total Mineral Resource Statement for All Mineral Zones - December 31, 2021:
Batero Gold Corp. – La Cumbre Gold Project

Total Resources All Mineral Zones							
Resource	**Volume**	**Density**	**Tonnage**	**Au g/t**	**Au oz**	**Ag g/t**	**Ag oz**
Measured	49,317,902	2.624	129,421,866	0.509	2,117,649	1.52	6,336,330
Indicated	2,411,421	2.606	6,283,667	0.383	77,476	0.45	91,432
Meas. + Ind.	**51,729,322**	**2.623**	**135,705,533**	**0.503**	**2,195,124**	**1.47**	**6,427,763**
Inferred	356,987	2.533	904,088	0.413	12,005	1.32	38,472

Notes to accompany La Cumbre Mineral Resource tables:

1. Mineral Resources have an effective date of December 31, 2021. The Qualified Person for the estimate is Mr. Walter La Torre, CP and MAusIMM.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
3. Mineral Resources are reported within a conceptual optimized pit that uses the following input parameters: Au price: US$1,750/troy oz and US$22.0/troy oz Ag, mining cost: US$1.95/t, process cost (including G&A): US$9.08/t processed, gold selling cost: US$47.00/troy oz and overall slope angle of 38°.

4. Gold recovery in the oxide and transitional zones was fixed at 85.5%. Gold recovery in the primary zone was fixed at 84.1%.
5. Mineral Resources (Oxide) are reported using a 0.218 Au g/t cut off grade.
6. Mineral Resources (Transitional) are reported using a 0.218 Au g/t cut off grade.
7. Mineral Resources (Primary) are reported using a 0.179 Au g/t cut off grade.
8. Totals may not sum due to rounding as required by reporting guidelines.

14.17 SENSITIVITY OF MINERAL RESOURCES TO CUT-OFF GRADE

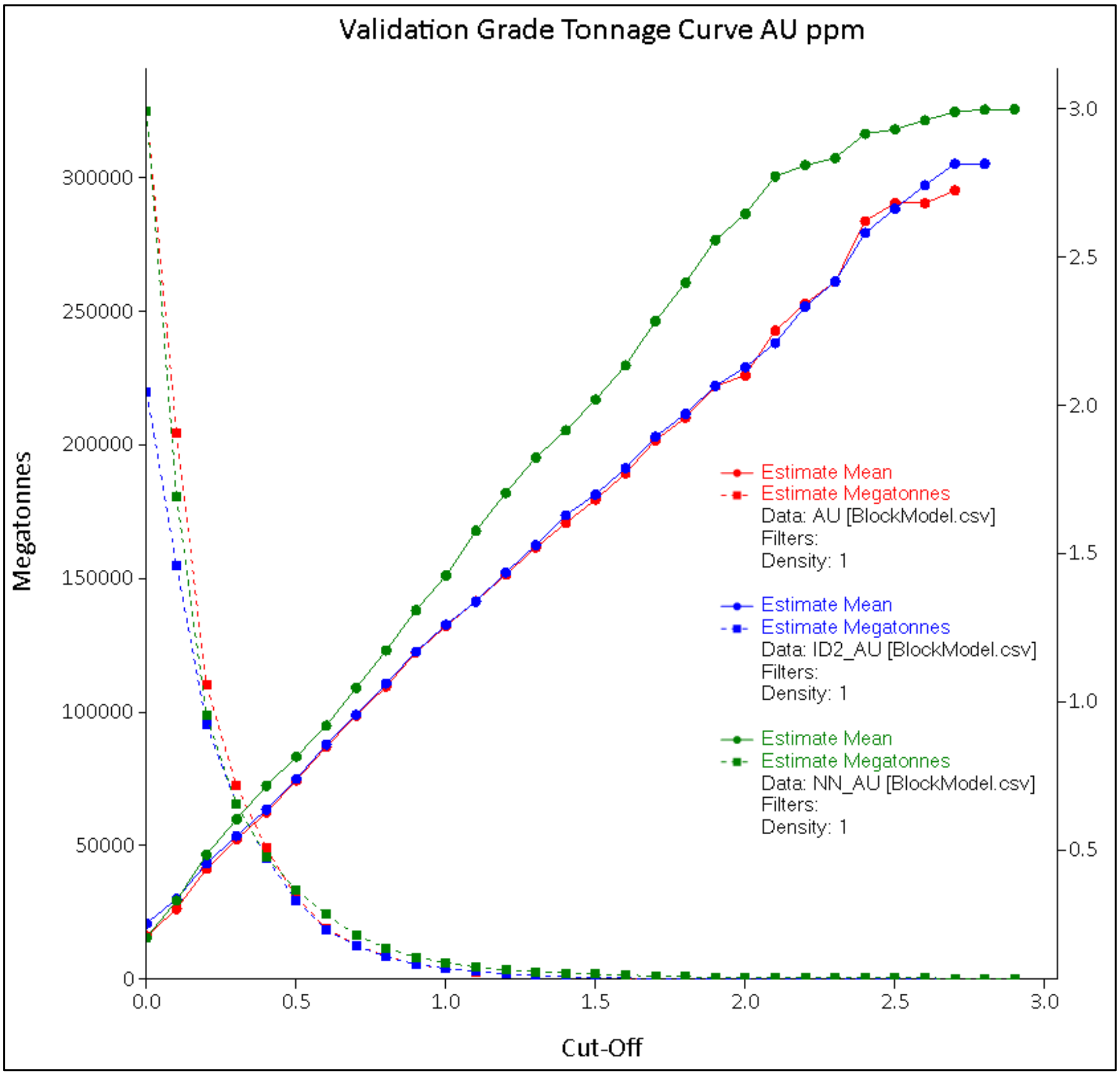
Table 14-20 summarizes the mineral resource at different cut-off grades for all mineral zones. The corresponded tonnage – grade curve is shown in Figure 14-36.

Table 14-20:
All Mineral Zones – Sensitivity of Mineral Resources to Cut-off Grade

Cut-Off	Mega Tonnes	Assay Name	Assay Mean
0	325,071,986	OK_AU	0.211
0.1	204,577,636	OK_AU	0.303
0.2	110,002,860	OK_AU	0.438
0.3	72,485,849	OK_AU	0.538
0.4	49,034,806	OK_AU	0.628
0.5	30,590,082	OK_AU	0.736
0.6	19,088,885	OK_AU	0.85
0.7	12,604,210	OK_AU	0.954
0.8	8,571,812	OK_AU	1.052
0.9	5,508,419	OK_AU	1.168
1	3,937,471	OK_AU	1.256
1.1	2,805,678	OK_AU	1.34
1.2	1,904,379	OK_AU	1.429
1.3	1,265,520	OK_AU	1.521
1.4	853,808	OK_AU	1.604
1.5	567,007	OK_AU	1.683
1.6	348,326	OK_AU	1.771
1.7	178,850	OK_AU	1.883
1.8	116,668	OK_AU	1.959
1.9	57,272	OK_AU	2.065
2	42,617	OK_AU	2.102
2.1	12,391	OK_AU	2.252
2.2	6,674	OK_AU	2.343
2.3	3,893	OK_AU	2.418
2.4	1,131	OK_AU	2.622
2.5	884	OK_AU	2.682
2.6	884	OK_AU	2.682
2.7	298	OK_AU	2.725
0	219,980,255	ID2_AU	0.254
0.1	154,901,382	ID2_AU	0.337
0.2	95,304,905	ID2_AU	0.455
0.3	65,733,202	ID2_AU	0.549
0.4	45,449,152	ID2_AU	0.637
0.5	29,362,418	ID2_AU	0.741
0.6	18,419,525	ID2_AU	0.857
0.7	12,490,957	ID2_AU	0.957
0.8	8,283,459	ID2_AU	1.063
0.9	5,551,150	ID2_AU	1.17
1	3,965,690	ID2_AU	1.26

Cut-Off	Mega Tonnes	Assay Name	Assay Mean
1.1	2,883,847	ID2_AU	1.339
1.2	1,903,239	ID2_AU	1.437
1.3	1,292,058	ID2_AU	1.529
1.4	826,544	ID2_AU	1.63
1.5	598,671	ID2_AU	1.7
1.6	374,738	ID2_AU	1.789
1.7	215,675	ID2_AU	1.895
1.8	140,396	ID2_AU	1.972
1.9	81,307	ID2_AU	2.066
2	53,746	ID2_AU	2.129
2.1	24,435	ID2_AU	2.211
2.2	9,212	ID2_AU	2.333
2.3	4,821	ID2_AU	2.418
2.4	1,458	ID2_AU	2.582
2.5	960	ID2_AU	2.663
2.6	589	ID2_AU	2.743
2.7	296	ID2_AU	2.815
2.8	296	ID2_AU	2.815
0	325,071,902	NN_AU	0.206
0.1	180,801,161	NN_AU	0.331
0.2	98,696,031	NN_AU	0.486
0.3	65,528,701	NN_AU	0.605
0.4	45,625,701	NN_AU	0.718
0.5	33,451,210	NN_AU	0.816
0.6	24,165,411	NN_AU	0.921
0.7	16,488,723	NN_AU	1.048
0.8	11,625,771	NN_AU	1.174
0.9	8,215,401	NN_AU	1.31
1	6,207,126	NN_AU	1.427
1.1	4,413,797	NN_AU	1.578
1.2	3,400,840	NN_AU	1.705
1.3	2,691,163	NN_AU	1.825
1.4	2,250,155	NN_AU	1.916
1.5	1,841,169	NN_AU	2.02
1.6	1,471,083	NN_AU	2.136
1.7	1,125,601	NN_AU	2.284
1.8	903,889	NN_AU	2.414
1.9	718,704	NN_AU	2.558
2	627,374	NN_AU	2.646
2.1	519,202	NN_AU	2.772
2.2	488,802	NN_AU	2.81
2.3	469,088	NN_AU	2.834
2.4	401,240	NN_AU	2.916
2.5	388,519	NN_AU	2.931
2.6	358,866	NN_AU	2.962
2.7	330,245	NN_AU	2.99
2.8	319,784	NN_AU	2.998
2.9	317,366	NN_AU	2.999

Figure 14-36:
Grade and Tonnage Distribution for All Mineral Zones



15.0 MINERAL RESERVE ESTIMATES

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Prefeasibility or Feasibility level as appropriate that include application of modifying factors.

A Mineral Reserve has not been estimated for the La Cumbre Project as part of this PEA.

Chapter 16 presents an estimate of potential economic mineral resources which could be an advance estimate of the potential Mineral Reserves of the La Cumbre Deposit.

16.0 MINING METHODS

The preferable mining and processing methods for La Cumbre is conventional open pit mining followed by heap leaching process for oxide and transitional zone and flotation-gravimetry process for sulfide mineral recovering gold as the primary product and silver as secondary.

The term “leach” material is used to refer in this section to oxide and transitional mineralization and the term “sulfide” is used to refer to the primary sulfide mineralization. The two mineralization types - leach and sulfide - are treated independently in the designed pit and for mine planning purposes. Each of these materials are assigned a different metallurgical recovery according to their metallurgical process.

The current PEA is relatively advanced because the forward-looking information available from Minera Quinchia is at development stage, mainly for its leaching material from a preliminary pit internally evaluated by Batero staff. Sulfide materials are in the initial stage of evaluation and are included in this PEA. The forward-looking information relates to to mining method, ore processing alternatives, permit to implement a mining operation and environmental studies.

This section outlines the parameters and procedures used to estimate a subset of the Mineral Resources contained in an optimized pit shell, designing the open pit mine and scheduling. This subset is referred to as the potentially Economic Mineral Resource, which is based on the economic evaluations and considers Measured and Indicated Mineral Resources only. No Inferred Mineral Resource was included in the potentially Economic Mineral Resource, in order to generate an approximation of a Mineral Reserve.

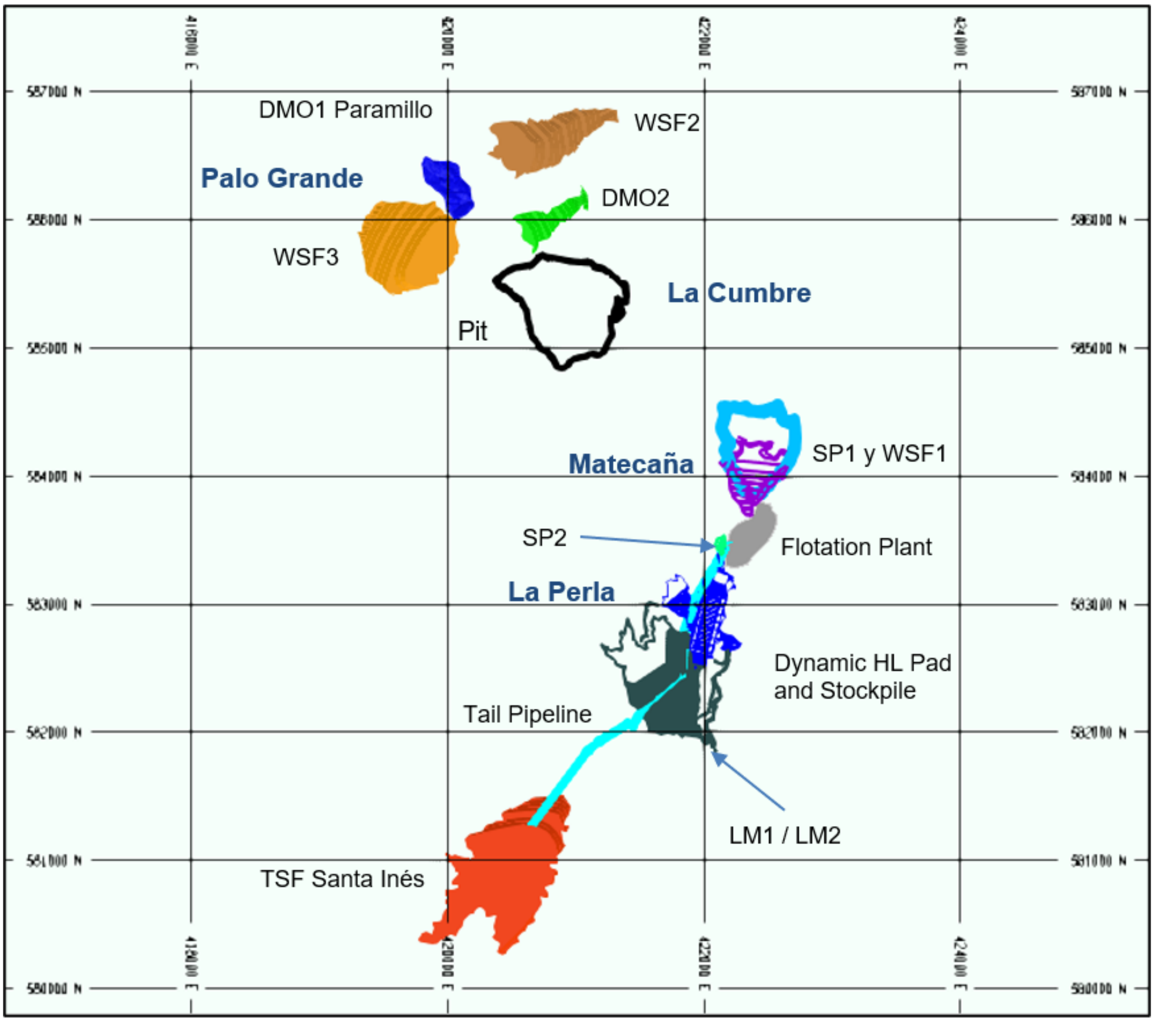
The potentially Economic Mineral Resource is used to define the features of mining method, mine design and scheduling as part of this section.

The mine schedule to feed the process plant is initially at a rate of 5.4 Mtpy (leaching only), increasing to 10.8 Mtpy in a second phase (including flotation and gravimetry). The present study does not consider mining dilution into the block model.

The waste storage facilities (WSF) and tailings storage facility (TSF) for Batero are under conceptual design, in contrast to the heap leaching area which is at an advanced stage of design (can be considered as PFS).

The general layout of the site is shown in Figure 16-1, General Site Layout.

Figure 16-1:
General Site Layout



Source: Minera Quinchia, 2021

16.1 METHODOLOGY OF ECONOMIC MINERAL RESOURCE

The Economic Mineral Resource for La Cumbre is constrained for an ultimate pit designed in base of a pit optimization according to the following:

- Review the geological information and resource block model and initial topography.
- Selected applicable mining method for La Cumbre is open pit with initial pit for leachable materials like oxide and transitional material and second pit extended to cover primary sulfides materials under milled-flotation.
- Economic factors like commodities price, mining cost, processing cost, selling cost and general expenses were based on forecasting bench marking, quotation and budgeting and internal reports of Batero.
- The economic cut-off value is based on gold grade and considering price, mining and processing cost, general and administration expenses, metal recovery for each process and selling cost input. The resulting cut-off for leachable materials is 0.243 g/t Au and for mill process is 0.20 g/t Au.
- Metal recovery for gold and silver is assumed depending on the metallurgical process.
- Zero metal recovery for copper is assumed for both leach and flotation processes. Metal recovery under laboratory test suggest no commercial option at the present is possibly caused by the low grade of copper.

- Although copper mineral is well known as cyanided material under heap leaching process no impact in recovering gold or silver was assumed for the PEA.
- Materials from primary sulfide domain are classified to be processed under gravimetry-flotation plant with crushing and milling activities to recover gold and silver.
- Commercial products are gold and silver doré bars for leaching process and gold concentrate with silver contents for mill process.
- Mining cost is assumed on average 1.95 $/t ore or waste mined. Incremental haulage cost of 0.01 $/t/bench.
- The Lerchs and Grossman pit optimization was used to estimate the ultimate Pit.
- A block size of 6 m by 6 m and 6 m and inter-ramp angle (IRA) of 38° as predominant was used for Pit optimization and designing.
- Final Pit design using a bench was used 6-meter, 12-meter ramp width two way only.

16.2 PIT OPTIMIZATION

Pit optimization was developed using NPV Scheduler ® software and was based on Measured and Indicated Mineral Resources. The mineralization considered in the optimization was limited to the oxide, transitional and primary sulfide zone. Other domain types are considered to be waste materials for PEA purpose.

16.2.1 Parameter for Pit Optimizations

Pit optimization considered the following metal prices: US$ 1,575.0/oz for gold, and US$20.0/oz for silver. Parameters for pit optimization is summarized in the Table 16-1.

The overall slope angle (OSA) used in the pit optimization was 39°. This angle was based on an initial slope analysis available for La Cumbre in a preliminary designed pit before this PEA. This pit only covered leachable material and it is recommended that a more detailed geomechanical study should be conducted in the next stage of the Project as results of the ultimate pit estimated under this PEA.

Table 16-1:
Parameters for Pit Optimizations

Item	Value	Units
Gold Price	1575	$/oz
Silver Price	20	$/oz
Gold selling cost	47	$/oz
Silver selling cost	0	$/oz
Average Mining Cost	1.95	US $/t mined
Haulage incremental cost	0.01	US$/t/bench
Base level for haulage cost	1900	masl
	Leach process	
Gold recovery	85.5	%
Silver recovery	47	%
Materials	Oxide, Transitional	
	Mill process	
Gold recovery	84.1	%
Silver recovery	51.9	%
Materials	Primary Sulfide	
Leach Processing Cost	9.08	US$/t proc
Mill Processing Cost	7.14	US$/t proc

Item	Value	Units
General and administrative expenses	1.12	US$/t proc
IRA by Geotechnical Domains		
Ash	39	°
Oxide	39	°
Transitional	43	°
Primary	38	°
	Pit Geometry	
Bench Height	6	m
Bench Face angle	70	°
Ramp Width	12	m

Note: Gold selling cost of 47 $/oz is 2.71% to include TC & RC, transport cost. It is a gross estimated from LINAMEC preliminary internal job for La Cumbre.

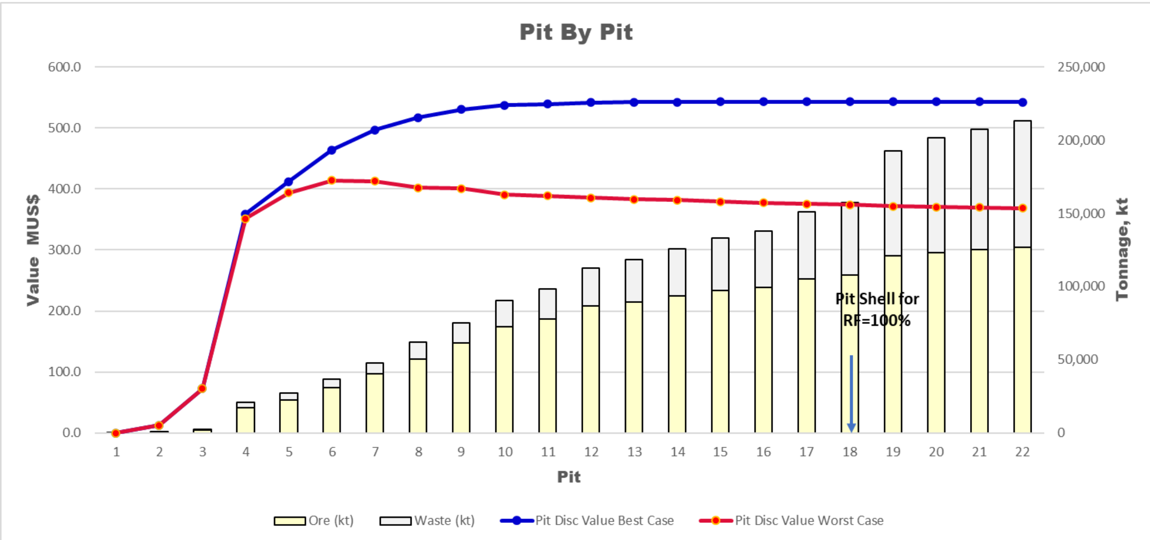
An optimal pit shell for leaching material was first estimated considering sulfide zone as waste material. In a second stage, sulfide materials were incorporated as an economic ore source and the result was an optimal pit shell which supports the ultimate pit for La Cumbre.

16.2.2 Pit Optimizations Results

The logic used for block valuation and mineralized material/waste classification in NPV Scheduler was of the "Marginal Net Benefit", which establishes that a particular block goes to the plant if its value covers the processing, selling cost and marginal mining cost (defined as the mining cost when treated as ore minus mining cost when treated as waste); otherwise, it will be send to the Waste Storage Facility (WSF).

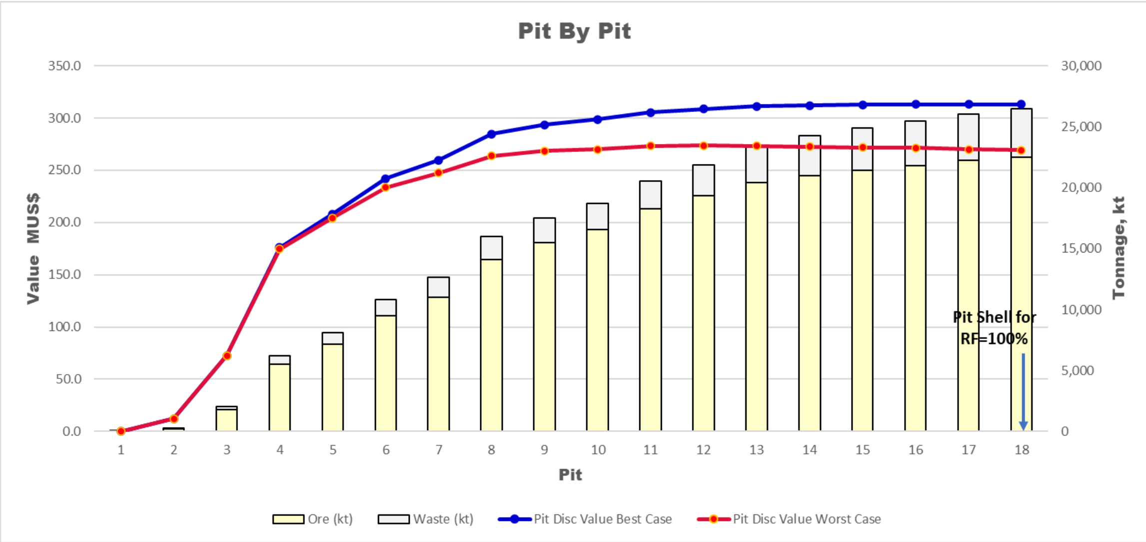
A total of 22 nested pit shells were created in the pit optimization process for different Revenue Factors (RF) which varied from 15 to 120% in 4% steps. Figure 16-2 shows the result of this process. The optimal pit corresponds to 100% of RF. These pits consider sulfide materials as a source of ore.

Figure 16-2:
Pit by Pit for La Cumbre - Ultimate Pit



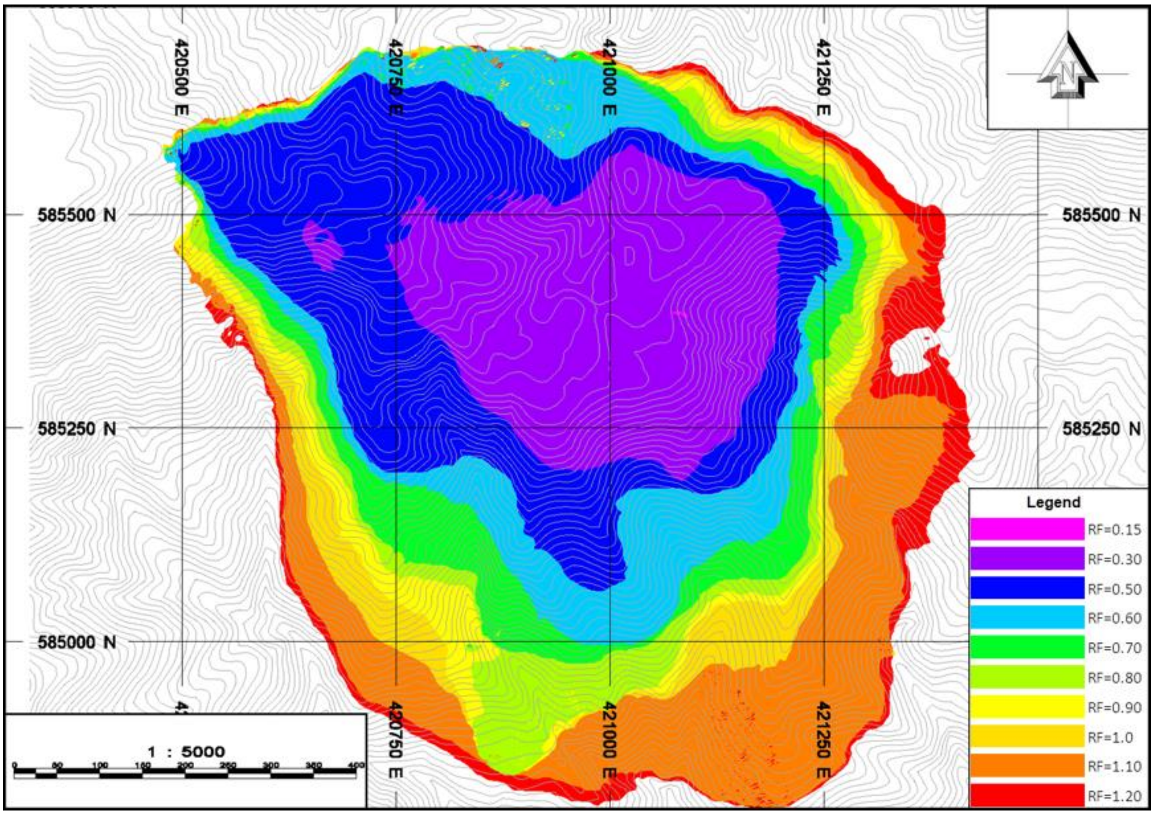
In order to estimate the case of optimized pit, when is limited of leachable ore results show in Figure 16-3 and optimal pit correspond to pit 18 or 100% of RF.

Figure 16-3:
Pit by Pit for La Cumbre – Restricted to Leachable Materials



Comparison of these results La Cumbre Pit increase from 22 Mt of ore to 108 Mt caused by considering the opportunities to process sulfide materials. The plan view of the nested pit shell for ultimate pit optimized in NPV Scheduler are showed in Figure 16-4. RF=1.0 is the shell used for ultimate pit design purpose.

Figure 16-4:
Nested Pit Shells for La Cumbre Project – Ultimate Pit

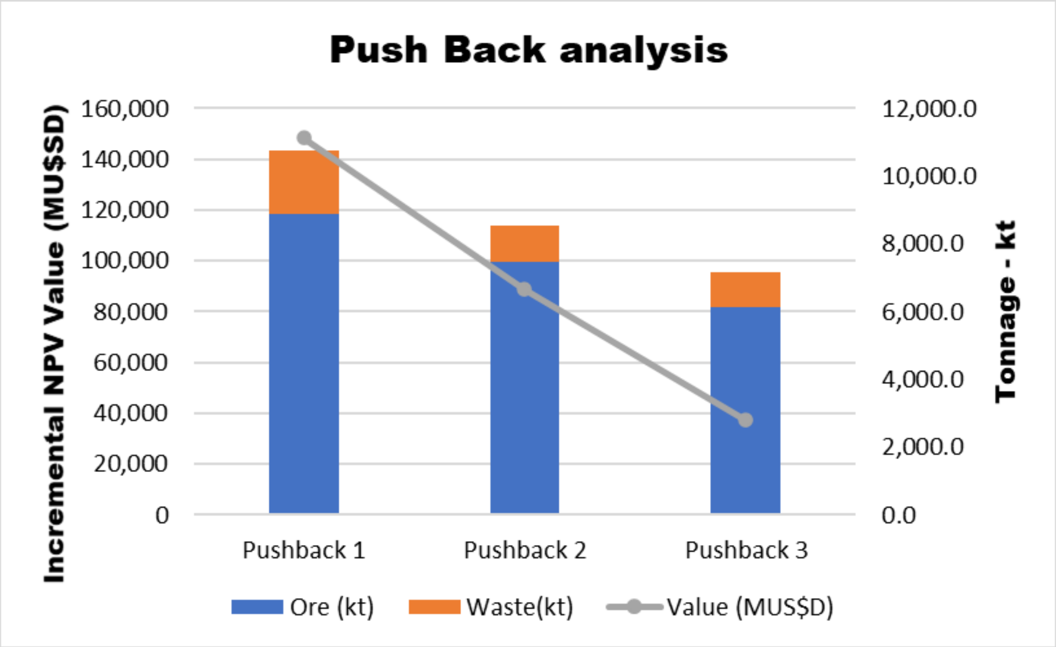


16.3 ULTIMATE AND PUSHBACK PIT SHELL SELECTION

The ultimate pit for La Cumbre corresponds to 100% of RF pit shell and has 108 Mt of ore and 49 Mt of Waste resulting in an overall stripping ratio (SR) of 0.5 (waste: mineralization) with gold grade value of 0.56 g/t and silver grade of 1.57 g/t. Mining phases or pushback pit shells was created in two separate dependencies: a) One controlled by leachable materials pit only called starter pit, intermediate pit or leachable pit here, and b) Second as additional to cover ultimate pit. A total of eight push backs were generated for La Cumbre pit. The minimum width used to generate pit stages was 40 m and it depends of the minimum space required for the selected equipment in the pit design process, which it is 35 m.

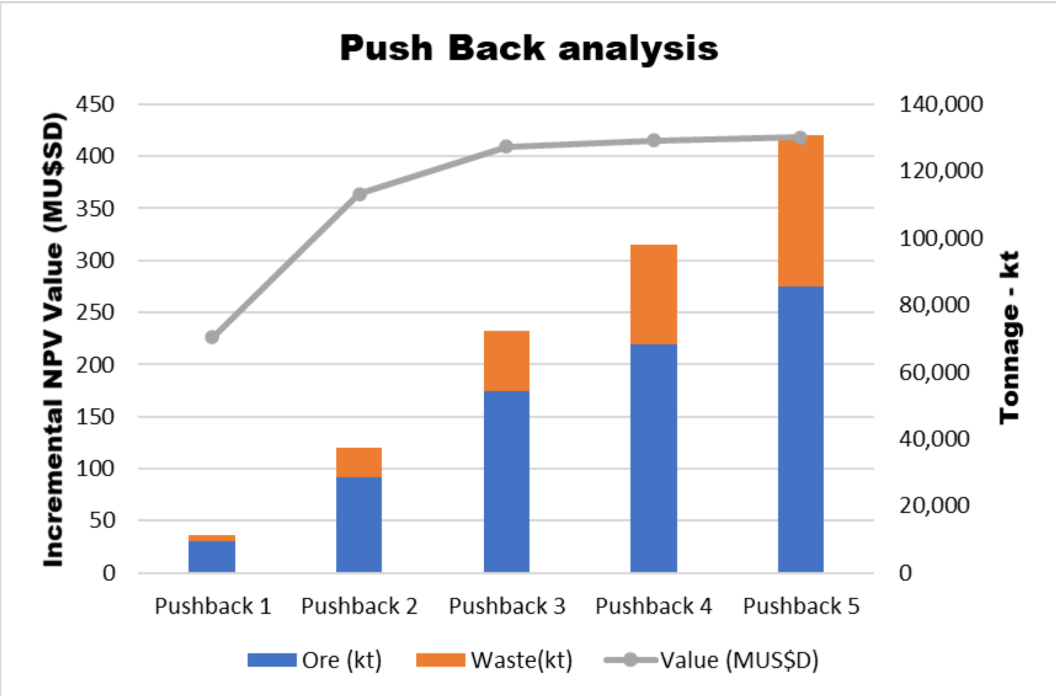
Skin analysis in the nested pit for leachable materials indicates that pit 5 with 8 Mt of ore could be an initial stage. Then pushback task was used in NPV Scheduler to generate three (3) mining phases conditioned to the starter pit as maximum (pit 18 at Figure 16-3) with initial phase of 8 Mt ore size and minimum width as input parameters. Figure 16-5 shows the resulted three push back as pit shell from this process.

Figure 16-5:
Initial Push Back for La Cumbre Pit – Restricted to Leachable Materials



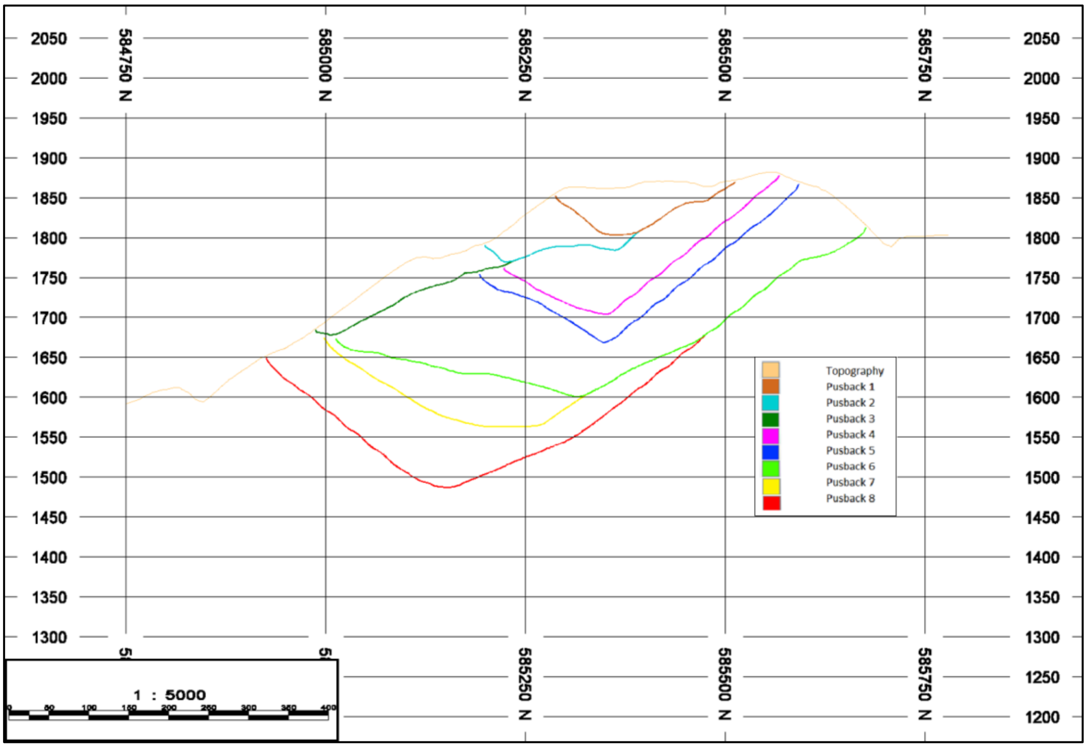
Subsequently, LINAMEC has evaluated a push back analysis having depleted the starter pit and considering the final pit those which size is the 108 Mt of ore (Pit 18 on Figure 16-3) to generate nested pit shell and identify the best five (5) mining stages as additional. This analysis suggests that pit size with 9.5 Mt, 18 Mt, 23.5 Mt, 13.5 Mt and 22 Mt of ore is good criteria to generate the additional phases. Then the push back task was processed with a minimum width 40 m as input and limited to five pushbacks. Figure 16-6 shows the resulted push back.

Figure 16-6:
Additional Push Back for La Cumbre Pit



Using the NPV Scheduler Push back tool to create a pit shell that represents a mining phase ensuring the best NPV pit is selected for designing and scheduling purposes. The resulting pit shells for mining phases design purposes are shown in a vertical section in Figure 16-7.

Figure 16-7:
Push Back Pit Shell – Vertical Section North-South 421 004 E



16.4 PIT DESIGN

Staged pit designs were developed by LINAMEC staff using the pushback pit shells. A total of eight pit shell were selected as stages for La Cumbre project, three to get the leachable pit and five additional pit stages to get the ultimate pit, those that include sulfide materials.

Parameters for the pit design such as minimum mining width, ramp slope angle, bench height, inter-ramp slope angle and others were based on the mine equipment selected. Table 16-2 shows the design parameters used in the final pit and pit phase designs

Table 16-2:
Pit Geometry parameters

Inter-Ramp Angle by Domains	
Ash	39°
Oxide	39°
Transitional	43°
Primary Sulfides	38°
Pit Geometry	
Bench Height	6m
Bench Face angle	70°
Ramp Width	12m
Maximum Ramp gradient	10%
Minimum mining width	35m

The final pit design and the phase design were developed with Datamine Studio OP® software. All the staged pit was designed as single bench and ramps. Ramps allocation was designed to avoid access interference between each stage or allow connection in the scheduling process. All the designed pit incorporates crest, toe, and ramp line.

The bottom level of the final pit is at 1510 masl, with a maximum overall wall height of 550 m at the north west side, 336 m at the north east wall and a minimum height of 120 m at the south wall. The IRA of 38° degree is predominant angle as minimum. The main pit includes only one access as PEA status. It is recommended that after this PEA additional study must be followed for IRA and second ramp allocation in the final pit to decrease the risk of production stoppages if one ramp is blocked.

The intermediate and final designed pits are shown in Figure 16-8 and Figure 16-9 The intermediate designed pit corresponds to pushback 3 and the ultimate corresponds to pushback 8.

Figure 16-10 shows the pit design and pushbacks for leachable material.

Figure 16-11 shows the final pit design and all pushbacks (eight in total), including primary material. Figure 16-12 is a vertical section showing all designed pushbacks.

Figure 16-8:
La Cumbre Intermediate Pit Based on Leachable Ore

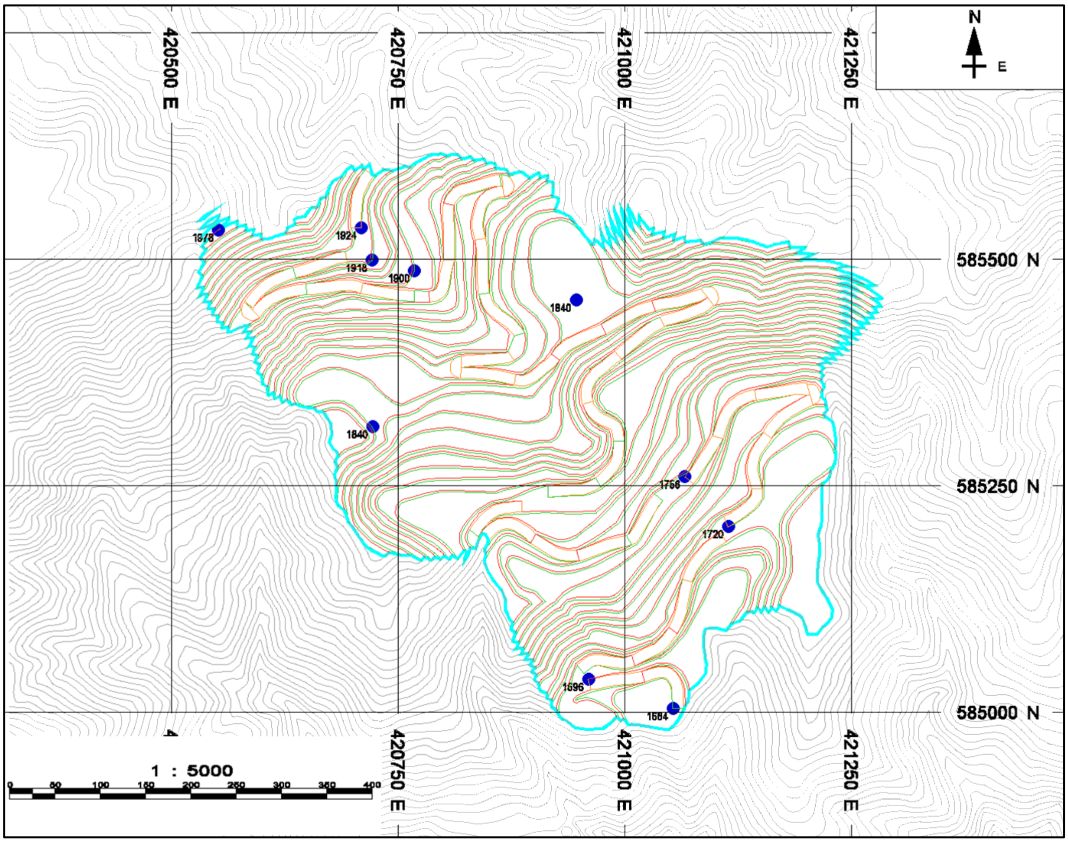


Figure 16-9:
La Cumbre Final Pit - Sulfide and Leachable Materials

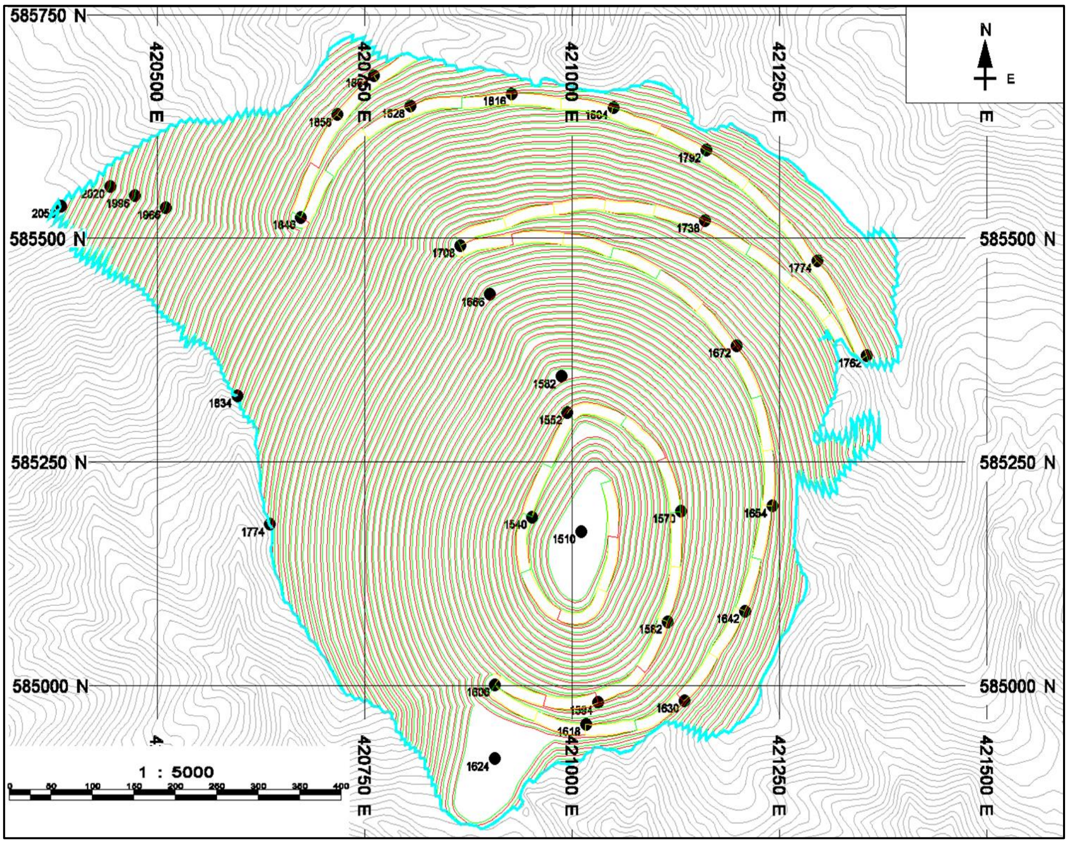


Figure 16-10:
Designed Pushback for Leachable Pit

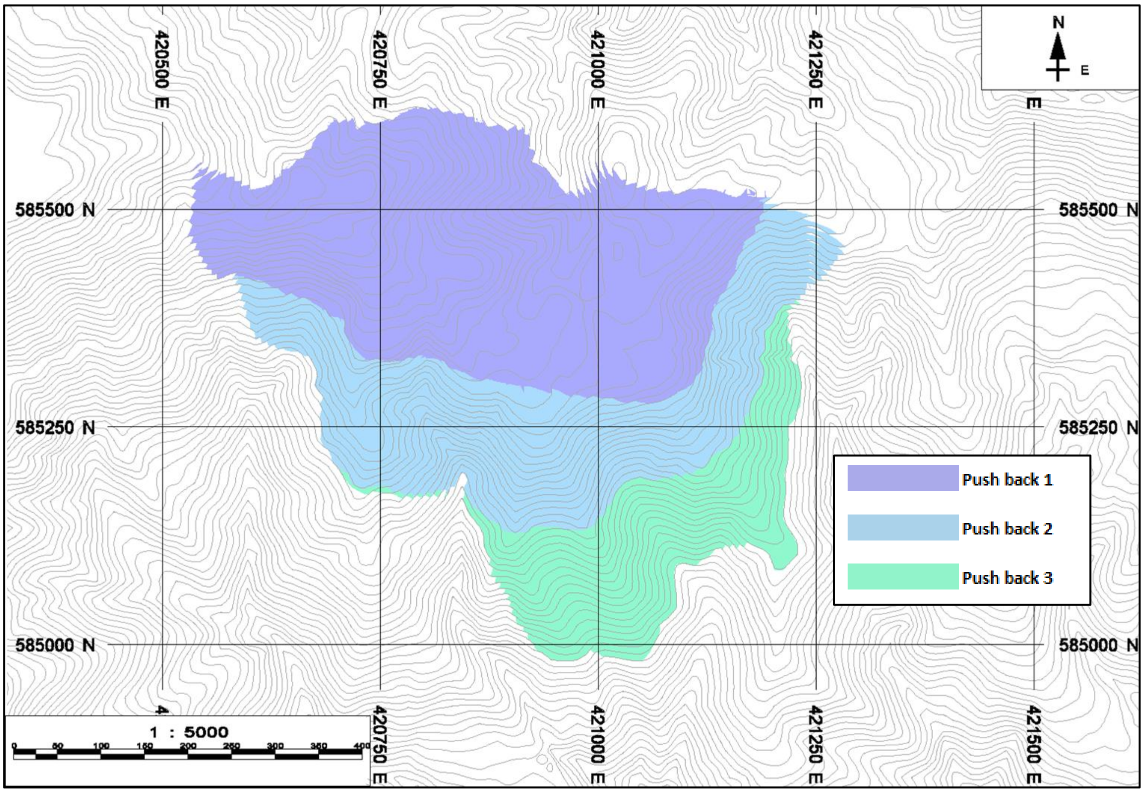


Figure 16-11:
Designed Pushback up to Final Pit

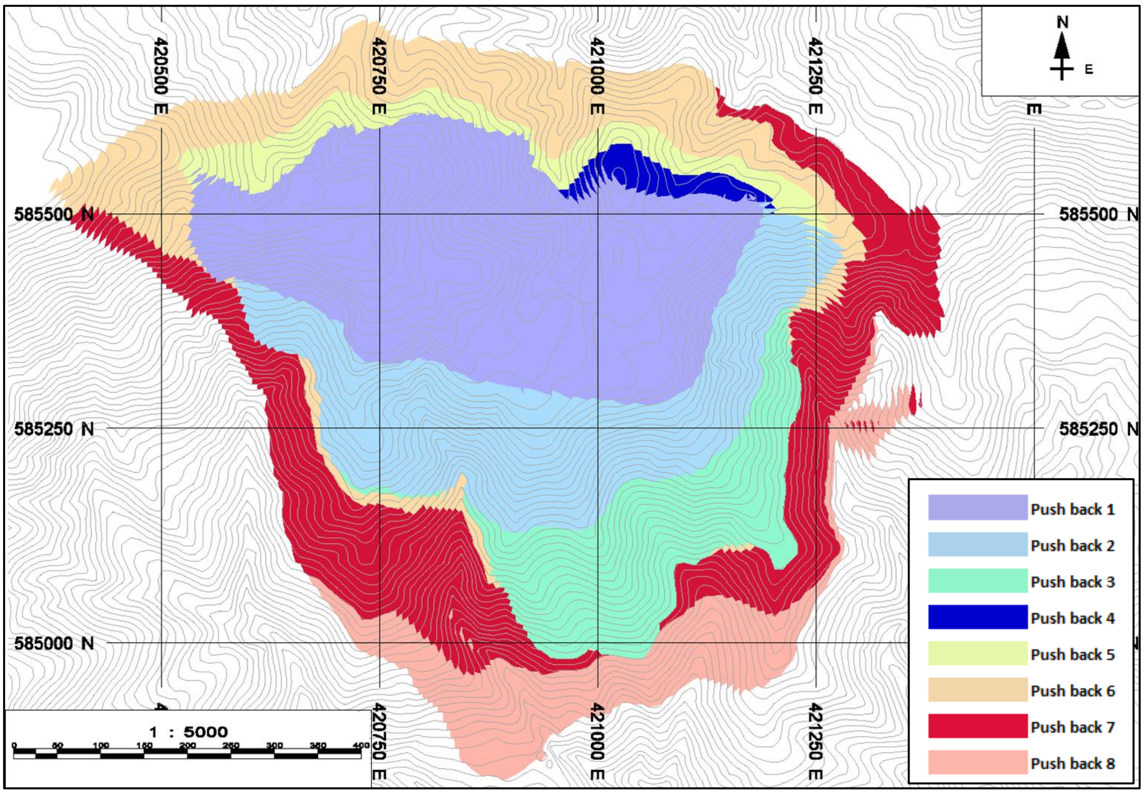
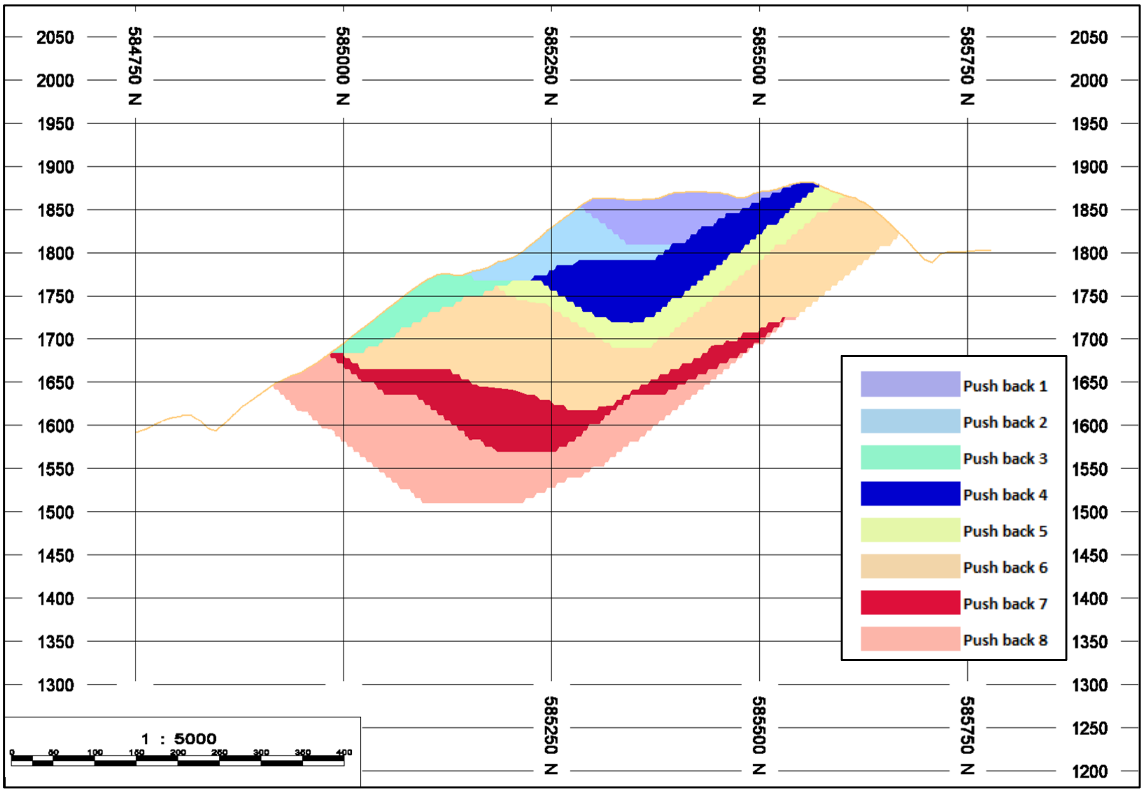


Figure 16-12:
Vertical Section 421004-E (Looking to the West) with Designed Pushback



The designed final pit contains 98.5 percent of the Measured and Indicated Mineral Resource subset of the optimized pit shell. Similarly, the quantity of waste rock increased by 6.3%, which resulted in an overall increase in total rock of 1%. All the volumetric calculations for La Cumbre project were made using Minesight® MS3D software including MS Reserve® and MSSO® applications, see Table 16-3 below.

Table 16-3:
Total Material inside the Final Pit for La Cumbre Deposit

Material	Tonnes (kt)	Au (g/t)	Au oz	Ag (g/t)	Ag oz
Subset of MEASURED Resource	103,772	0.569	1,898,397	1.579	5,268,134
Subset of INDICATED Resource	2,750	0.400	35,366	1.294	114,409
TOTAL Measure + Indicated	**106,522**	**0.565**	**1,933,763**	**1.572**	**5,382,543**

Notes:

1. Economic Mineral Resource is not the same as an NI 43-101 Mineral Reserve.
2. The Economic Mineral Resource estimation is a Mineral Estimate output under PEA study.
3. Subset of the measured and indicated resource include oxide, transitional and primary sulfide ore.
4. Cut-off 0.243 Au g/t were used for oxide and transitional materials and cut-off 0.20 Au g/t for primary sulfides.
5. Gold spot price 1,575 US$/oz and silver spot price 20.0 US$/oz were used.

This reported material is named Economic Mineral Resource and it is used for scheduling purposes and economic evaluation for the La Cumbre project. Disaggregated materials by mining phases are showed in Table 16-4 and Table 16-5. The 85% of the leachable material is in the main leachable pit – 1st to 3rd pushback - and remaining material comes with the material mined with sulfide ore. The gold grade is in decrease either for leachable or sulfide pit, evidencing the best practice in the pit design considering the value of the pit. The waste material by mining phases is in Table 16-6.

**Table 16-4:
Resource Material by Mining Phase for La Cumbre Deposit**

	Leachable Ore	Grades		Milled Ore	Grades	
Pushback	Tonnes (kt) Mea + Ind	Au g/t	Ag g/t	Tonnes (kt) Mea + Ind	Au g/t	Ag g/t
PH1	8,289	0.77	1.55	177	0.63	1.47
PH2	8,085	0.75	1.70	211	0.68	1.46
PH3	4,551	0.58	1.58	38	0.68	1.56
PH4	531	0.88	1.85	7,068	0.96	1.81
PH5	732	0.44	1.40	16,558	0.67	1.78
PH6	672	0.41	1.57	23,192	0.48	1.53
PH7	1,191	0.30	1.10	15,970	0.43	1.49
PH8	158	0.29	0.75	19,099	0.38	1.41
Total	**24,209**	**0.68**	**1.58**	**82,313**	**0.53**	**1.57**

**Table 16-5:
Resource Material by Mining Phases for La Cumbre Deposit**

	Leach Ore				Mill Ore	
	Oxide		Transitional		Primary	
Pushback	Measured	Indicated	Measured	Indicated	Measured	Indicated
PH1	6,405	297	1,504	83	174	3
PH2	4,752	94	2,940	298	211	1
PH3	2,285	307	1,730	230	38	0
PH4	16	6	491	17	7,065	3
PH5	57	44	552	79	16,540	18
PH6	45	8	413	206	23,132	60
PH7	307	275	202	407	15,912	58
PH8	0	20	30	108	18,973	127
Total	**13,867**	**1,052**	**7,861**	**1,429**	**82,044**	**269**

**Table 16-6:
Waste Material by Mining Phases for La Cumbre Deposit**

Pushback	Ash	Oxide	Transitional	Primary	Other Waste
PH1	1,126	1,208	274	80	17
PH2	268	678	414	12	20
PH3	398	351	117	1	13
PH4	9	725	124	451	1
PH5	26	1,304	759	1,932	3
PH6	292	1,093	2,974	7,601	91
PH7	194	2,271	5,061	7,576	53
PH8	92	486	3,067	11,020	10
Total	**2,404**	**8,115**	**12,790**	**28,675**	**208**

16.5 GEOTECHNICAL CONSIDERATIONS

To date a comprehensive analysis study of the geostructural characteristics of the orebody has not been developed by MQ. However, In the property, various field research campaigns were carried out including geological mapping, geotechnical investigations through diamond drilling with SPT and permeability tests, installation of pipe piezometers open, pits logging and on-site density testing, geophysical investigations such as electrical resistivity tests, refraction seismic and MASW (Anddes Asociados S.A.C., 2020).

The result of those researches, a conceptual geotechnical design included the design of the leaching pile and waste dump, and a geometric configuration of the operating accesses (from pit to dump and from pit to heap leach), internal accesses (of the plant), the crushing area platform, the plant platform, and the process ponds.

16.5.1 Deposit Characteristics

The Dos Quebradas, El Centro and La Cumbre porphyry gold deposits are associated with three Miocene intrusive centers in a north-south trend that have a strike extension of over two kilometers at elevations between 1,600 masl and 1,050 masl. These intrusive centers are composed of dikes and stocks separated into three groups: i) early inter-mineral, ii) late inter-mineral, and, iii) post-mineral. These dioritic phases were emplaced into intermediate to felsic volcanic rocks of the Miocene Combia Formation and Cretaceous basalts of the Barroso Formation. The three porphyry gold deposits on the Batero-Quinchia property are described as follows: 1. Dos Quebradas: the mineralized area occurs in the incised Dos Quebradas valley and covers an area of approximately 700 m by 700 m trending north of Batero's Concession Contract 22270. 2. El Centro: a lower grade, possibly deeper deposit covering an area of approximately 800 m by 500 m. The La Cumbre deposits are copper-poor porphyry gold systems in which intermediate argillic alteration locally extensively overprints an early potassic assemblage and its associated quartz veinlet stockwork.

At the La Cumbre deposit, ore will be mined from the oxide zone, the transition zone and, to a greater extent, from the primary zone. The oxide zone due to leaching processes generated on the rock, the action of late mineralizing fluids and later supergene processes, have allowed the development of an oxidation zone within the saprolitized rock. This oxidation zone directly affects the mineralization contained in the porphyry system, and the surrounding wall rock mineralization. These late and supergene processes allow the oxidation of the sulfides, releasing the metals in a natural way by dissolution generated by meteoric water. The dissolution releases the gold, silver and copper contained within the sulfide matrix, and generates an enrichment within the superficial layer of the rock.

16.5.2 Geotechnical Characterization of Materials

A geotechnical characterization of the different lithological components obtained from a variety of information (exploratory drilling, field observations and laboratory results) are displayed in Table 16-7.

Table 16-7:
Geotechnical Characterization of Materials

Material	Lithological Unit	Zone
Competent basalt	Barroso Formation	Foundations
Fractured & altered basalt	Barroso Formation	Foundations
Competent andesitic porphyry	Hypabyssal rocks	Foundations
Fractured & altered andesitic porphyry	Hypabyssal rocks	Foundations
Competent monzonite	Plutonic intrusive	Foundations
Fractured & altered monzonite	Plutonic intrusive	Foundations
Residual soil	Residual quaternary deposit	Foundations
Structural backfilling	-	Dykes & embankments
Surplus material	-	Material Storage Surplus

Material	Lithological Unit	Zone
Oxides & Transition	-	Stockpile
Tailings	-	Leached material deposit
Geomembrane oxide interface	-	Deposition of leached ore, stockpiles of oxides and transition
Geomembrane transition interface	-	Deposition of leached ore, stockpiles of oxides and transition

16.5.3 Geotechnical Fieldwork

- A geological mapping of the project area comprising a total area of approximately 724 ha made it possible to identify and record geomorphological features, main lithological types, structural settings and geodynamic agents.
- A drilling campaign in 27 drill holes with depths between 3.90 m and 100.30 m, having carried out Standard Penetration Test (SPT), geological logging, samples for laboratory tests. Additionally, open tube piezometers were installed in holes PZ-09, GEO-LP-021 (PZ-10), GEO-LP-027 (PZ-11), LP-PZ-12 (PZ-12) and GEO- LP-026 (PZ-13) of 20 m, 23 m, 30 m, 30 m and 40 m depth, respectively.
- Nineteen (19) trenches were done for evaluating stratigraphic profile, quality of material, density in situ and collecting samples for laboratory assays.
- An electrical resistivity test was tested out covering a length of 400 m reaching depth of up to 60 m.
- Nine (09) seismic refraction profiles were made that add up to a length total of 3,697 m and reach depths of up to 70 m.
- Twenty-one (21) MASW (Multichannel Analysis of Surface Waves) tests were performed with a maximum of 30 m depth.

16.5.4 Geotechnical Laboratory Tests

16.5.4.1 SPT and Diamond Drilling at La Perla Zone

For the geotechnical-geomechanical terrain characterization in the La Perla area, 10 SPT were carried out in 2018 and 17 diamond drill holes in 2020 that cut through soil deposits and some of them the bedrock. Table 16-8 and Table 16-9 show general information of those workings.

During 2020, two investigation campaigns were carried out where the first campaign was conducted by the company Risk and Design Consulting S.A.S. in which seven diamond drill holes were executed, while the second campaign was carried out by MQ executing 10 diamond drill holes. The diamond drillings recorded the stratigraphic profile of the subsoil (soil and rock) describing its main characteristics. In addition, to monitoring the water table in La Perla zone, five Casagrande type piezometers were installed in drill holes PZ-09, PZ-10, PZ-11, PZ-12 and PZ-13.

Table 16-8:
Location of SPT DDH Tests

Material	Coordinates		Depth (m)	Component
	East (WGS 84)	North (WGS 84)		
SPT-LP-01	422,096	582,991	4.50	Dynamic Pad
SPT-LP-02	422,090	582,660	4.95	Dynamic Pad
SPT-LP-03	422,012	582,729	3.90	Dynamic Pad
SPT-LP-04	421,956	582,761	3.90	Dynamic Pad
SPT-LP-05	421,886	582,549	4.90	Tailing deposit
SPT-LP-06	421,801	582,443	4.85	Tailing deposit

Material	Coordinates		Depth (m)	Component
	East (WGS 84)	North (WGS 84)		
SPT-LP-07	421,854	582,250	5.85	Tailing deposit
SPT-LP-08	422,057	583,093	4.50	Dynamic Pad
SPT-LP-09	422,121	583,187	5.85	Dynamic Pad
SPT-LP-10	421,976	583,084	5.35	Dynamic Pad

Source: SRK Consulting (Peru) S.A.

Table 16-9:
Location of SPT Soil Tests

Hole ID	Coordinates		Inclination	Azimuth	Depth (m)	Component
	East (WGS 84)	North (WGS 84)				
2020-1 / 2019						
GEO-LP-008	421,892	582,393	-90°	-	26	Tailing deposit
GEO-LP-009	421,961	582,671	-90°	-	24	Dynamic Pad
GEO-LP-010	421,807	582,724	-90°	-	25	Dynamic Pad
CLP-01	421,803	582,728	-45°	50°	100	-
CLP-02	421,957	582,671	-50°	45°	86	-
CLP-03	421,894	582,392	-50°	45°	60	-
MQ-PZ-09	421,893	581,998	-90°	-	32	Tailing deposit
2020-2						
GEO-LP-019	421,926	582,891	-90°	-	26	Stockpile
GEO-LP-020	421,793	583,006	-90°	-	35	DME
GEO-LP-021	421,300	583,030	-90°	-	25	-
GEO-LP-022	421,132	582,674	-90°	-	42	-
GEO-LP-023	421,546	582,287	-90°	-	35	Tailing deposit
GEO-LP-024	421,785	582,130	-90°	-	33	Tailing deposit
GEO-LP-025	421,701	582,393	-90°	-	31	Tailing deposit
GEO-LP-026	421,660	582,697	-90°	-	40	Tailing deposit
GEO-LP-027	422,055	582,904	-90°	-	30	Dynamic Pad
LP-PZ-012	421,757	582,471	-90°	-	30	Tailing deposit

Source: SRK Consulting (Peru) S.A.

16.5.4.2 Soil Mechanic

To test geotechnical characteristics of the soils, these tests were performed:

- Granulometric analysis by sieving (INV E-213, 214, ASTM D-422)
- Granulometric analysis by sieving and hydrometer (ASTM D7928-16)
- Liquid limit and plastic limit (INV E-125, 126, ASTM D-4318)
- Moisture content (INV E-122, ASTM D-2216)
- Specific gravity of solids and moisture absorption (INV E-128, 222, 223, ASTM D-854)
- Standard Proctor Compaction (ASTM D-698)
- Modified Proctor Compaction (INV E-142; ASTM D-1557)
- Flexible wall hydraulic conductivity test (ASTM D-5084)
- CU undrained consolidated triaxial compression test (INV E-153, ASTM D-4767)
- Unconsolidated undrained triaxial compression test UU
- CD Drained Consolidated Direct Shear Test (INV E-152)
- Undrained consolidated direct cut test CU (INV E-154)

- Large scale direct shear test (ASTM D-5321B)
- Simple compression (INV E-152)
- Consolidation (INV E-1517, ASTM D-2435)
- Minimum density (ASTM D-4254)
- Maximum density (ASTM D-4253)
- Relative density (ASTM D-4253)

16.5.4.3 Rock Mechanic

The physical-mechanical properties of the main lithological units in the study area were tested by using:

- Point load (ASTM D-5731-08)
- Schmidt's hammer

16.5.4.4 Laboratory Assays

Those samples evaluated for static tests need to be analyzed.

- ABA test (Acid-Base Accounting)
- NAG test (Net Acidity Generation)

16.5.5 Geotechnical Domains

Geological-geotechnical units were defined based on geological mapping (Anddes 2020), being mostly: colluvium deposit (geological-geotechnical unit I or GGU-I), alluvial deposit (geological geotechnical unit II or GGU-II), residual soil 1 and residual soil 2 - saprolite (geological geotechnical unit III or GGU-III), bedrock (geological-geotechnical unit IV or GGU-IV).

The geological-geotechnical units are described below:

16.5.5.1 GGU-I (Colluvium Deposit)

Deposit originated by the action of both gravity and weathering, distributed over the study area, with an approximate strength of 3 m, and in accordance with the Unified Soil Classification System (SUCS) classified as: poorly graded gravel with silt and sand (GP-GM), silty gravel (GM), poorly graded sand with gravel (SP), silty sand (SM), low to medium plasticity, medium dense to dense compactness, slightly moist to moist, dark brown color, homogeneous structure, subangular to sub rounded gravel, presence of bollards TM=5.5".

16.5.5.2 GGU-II (Alluvial Deposit)

Deposit originated due to the transport and sedimentation by the action of water flows distributed in the creeks, approximately 3 m thick, and equivalent to the SUCS as poorly graded gravel (GP), poorly graded sand with gravel (SP), of null plasticity, medium loose to medium dense compactness, humid to very humid, yellowish-brown color, homogeneous structure, sub rounded gravel, presence of blocks of TM=30.5".

16.5.5.3 GGU-III (Residual Soil)

Deposit generated by in situ weathering processes of the rocky basement. This unit was divided into two subunits (Residual Soil 1 and 2), differentiated only in that in Residual 2 the original rock structure is observed while in Residual 1 it is completely weathered. Considering that:

Residual Soil 1

They do not preserve the texture and structure of the original bedrock, with approximate thickness of 16 m in Matecaña and 7 m in La Perla, classified according to SUCS as: poorly graded gravel (GP), silty sand with gravel (SM), clayey sand with gravel (SC), clay, clay with sand (CL), silt (ML), elastic silt (MH), with low to very high plasticity, very soft to very high consistency, very soft to very soft consistency, and very soft to very soft consistency.

high, very soft to firm consistency (NSPT=0 to 8), loose to dense compactness (NSPT=8 to 35), slightly moist to saturated, yellowish brown to yellowish white color, homogeneous structure, angular to subangular gravel.

Residual Soil 2 (Saprolite)

It preserves the texture and structure of the bedrock, with approximate thickness of 12 m in Matecaña and 8 m in La Perla, classified according to SUCS as: poorly graded gravel, poorly graded gravel with sand (GP), silty-clay gravel with sand (GC-GM), silty sand with gravel (SM), silt (ML), elastic silt (MH), low to high plasticity, soft to hard consistency (NSPT>50), very loose to very dense compactness (NSPT>50), yellowish brown to reddish brown, dry to very wet, heterogeneous structure, angular gravel TM=3".

16.5.5.4 GGU-IV (Bedrock)

Both drillhole information and geomechanical stations on rock outcrops establish this unit.

Basalt

Volcanic rock of aphanitic texture, belonging to the Barroso formation, with weak to very high strength (R2 - R5), fresh to very altered, extremely fractured to fractured (RQD from 0 to 75%), slightly wet to very wet, dark gray to greenish gray color, presents very altered, slightly rough to rough joints, filled with quartz, sand and oxides. Poor to good geomechanical quality (RMR89 29 to 70) according to International Society for Rock Mechanics and Rock Engineering (ISRM) standards.

Andesite

Subvolcanic igneous rock, porphyritic texture, weak to very high strength (R2 - R5), fresh to much altered, extremely fractured to moderately fractured (RQD 0 to 60%), slightly humid, greenish-gray color with much altered to slightly altered, soft, sand and clay-filled cleavages. Very poor to good geomechanical quality (RMR89 17 to 76) according to ISRM standards.

Monzonite

Intrusive igneous rock of phaneritic texture, belonging to the Irra Stock. It presents weak to very high strength (R3 - R5), fresh to moderately altered, extremely fractured to moderately fractured (RQD from 0 to 80%), slightly wet to dry, light gray color, presents slightly altered, rough to very rough diaclases, filled with sand, oxides, feldspars. Very poor to good geomechanical quality (RMR89 from 17 to 78) according to the ISRM standards.

16.5.6 Lithological Domains

This information is based on SRK analysis (April 2021).

16.5.6.1 Competent Basalt

Pertaining to the Barroso formation, it is a fine-grained, dark gray, slightly to moderately fractured (RQD<75%) aphanitic textured rock with slightly rough to rugose cleavage

(JRC=1.5), slightly to intensely weathered, fractures filled with hard oxides (< 5 mm) and soft clay (fault zone), with a geological strength index (GSI) value equal to 50.

The intact rock (sample taken from the GEO-MAT-20 borehole) has a point load index (Is50) of 12.75 MPa, which implies a simple compressive strength of 318.75 MPa. In addition, the intact rock constant (mi) was assigned a value equal to 25 and a disturbance factor equal to zero in order to determine the rock mass parameters according to the generalized Hoek-Brown failure criterion.

The estimated parameters for the rock mass are reduced value of the rock mass constant (mb) equal to zero (mb) equal to 4.192 and the rock mass constants ***s*** and ***a*** equal to 0.00387 and 0.506; respectively. In addition, it is worth noting that the natural density assigned to this material was 25 kN/m³.

16.5.6.2 Fractured and Altered Basalt

In the case, superficial horizons of highly altered and fractured basalt rock, with thicknesses ranging from 5 m to 15 m depth. A geological strength index (GSI) value of 35 has been estimated for these horizons.

The intact rock of these horizons was assigned a simple compressive strength of 318.75 MPa. In addition, the intact rock constant (mi) was set to 25 and the disturbance factor to zero in order to determine the parameters of the rock mass according to the generalized Hoek-Brown failure criterion.

The estimated parameters for the rock mass are reduced value of the rock mass constant (mb) equal to zero (mb) equal to 2.453 and the rock mass constants ***s*** and ***a*** equal to 0.00073 and 0.516 respectively. In addition, it is worth noting that the natural density assigned to this material was 24 kN/m³.

16.5.6.3 Competent Andesite Porphyry

Hypabyssal intrusive, fine to medium grained, greenish-gray, slightly to moderately fractured (RQD<60%) andesitic porphyry-type rocks. It has soft, slightly rough (JRC=1.5), slightly to moderately weathered and filled with hard oxide (< 5 mm), with a geological strength index (GSI) equal to 55.

The intact rock (MQ-GEO-09) has a simple compressive strength of 50.15 MPa. In addition, the intact rock constant (mi) was assigned a value equal to 25 and an alteration factor equal to zero in order to determine the parameters of the rock mass according to the generalized Hoek-Brown failure criterion.

The estimated rock mass parameters are reduced value of the rock mass constant (mb) equal to 5.011 and the rock mass constants "***s"*** and "***a"*** equal to 0.00674 and 0.504; respectively. Additionally, a natural density of 25 kN/m³ is determined.

16.5.6.4 Fractured and Altered Andesitic Porphyry

Outcrops of highly altered and fractured andesitic porphyry with thickness between 4 m and 20 m depth, with a geological strength index (GSI) of 35.

The intact rock of this horizon was assigned a simple tensile strength of 50.15 MPa. In addition, a value for the intact rock constant (mi) equal to 25 and an alteration factor equal to zero were established in order to determine the parameters of the rock mass according to the generalized Hoek-Brown failure criterion.

The estimated rock mass parameters are with a reduced value of the rock mass constant (mb) equal to 2.453, while the rock mass constants s and a, equal to 0.00073 and 0.516; respectively. Additionally, it has a natural density of 24 kN/m³.

16.5.6.5 Competent Monzonite

The monzonite rock of the Irra stock has a white phaneritic texture and a white color with pinkish tones, it has point load indices (Is50) from 3.26 to 5.42; resulting in simple tensile strength values of the intact rock from 81.50 to 135.0 MPa for a conversion factor of 25.

The monzonite is slightly fractured with an RQD of 0 to 80%, located between 9 to 12 m depth intensely fractured (<20%); and from 17 m to 27 m depth, it is moderately fractured (20-40%) at greater depths it is medium to slightly fractured (40 to 80%).

The geological characterization of the competent monzonite assigned a strength index (GSI) value of 65 and a simple tensile strength of 108.25 MPa. Also, a value to the intact rock constant (mi) equal to 29 and a disturbance factor equal to zero in order to determine the rock mass parameters based on the generalized Hoek-Brown failure criterion.

The parameters estimated for the rock mass with a reduced value of the rock mass constant (mb) equal to 8.309 and the rock mass constants s and a, equal to 0.0205 and 0.502; respectively. Additionally, the natural density was 25 kN/m³.

16.5.6.6 Fractured and Altered Monzonite

This variety is presented with pinkish hues, has point load indices (Is50) from 3.26 to 5.42 with simple compressive strength values of the intact rock from 81.50 to 135.0 MPa for a conversion factor of 25.

These monzonites are slightly fractured with an RQD of 0 to 80%, located from 9 m to 12 m depth to intensely fractured (<20%) from 17 m to 27 m it is moderately fractured (20%- 40%), and at greater depths it is medium to slightly fractured (40% to 80%).

This variety has been assigned a geological strength index (GSI) value of 35 and a simple tensile strength of 108.25 MPa. In addition, a value to the intact rock constant (mi) equal to 29 and a disturbance factor equal to zero were defined in order to determine the rock mass parameters according to the generalized Hoek-Brown failure criterion.

The estimated parameters for the rock mass are reduced value of the rock mass constant (mb) equal to 2.846 and the rock mass constants s and a equal to 0.00073 and 0.516; respectively. Moreover, it should be noted that the natural density assigned to this material was 25 kN/m³.

See Table 16-10 for estimation of rock parameter of diferents unit rocks.

16.5.6.7 Residual Soil

The results of direct shear and triaxial shear has been specific for each type of residual soil and are detailed in Table 16-11, Table 16-12 and Table 16-13.

Residual Soil 1.

Conformed by inorganic clays (CL) and high plasticity silts (MH), of yellowish-brown color, slightly humid. The texture and original structure of the protolith are not visible, with average strengths from 1.5 m to 12.0 m.

Table 16-10:
Estimation of Rock Mass Parameters

Unit	Hoek Brown Classification				Hoek Brown Criterion			Rock Mass Parameters				Failure Range Envelope		Mohr Coulomb Fit	
	UCS of intact rock (Mpa)	GSI	mi	Intact Modulus (Mpa)	mb	s	a	Tensile strength (Mpa)	Uniaxial Comp. strength (Mpa)	Global Strength (Mpa)	Modulus of Deformation (Mpa)	Application	Sig3max (Mpa)	Cohesion (MPa)	Friction angle D
Basalt	318.75	50	25	111562.5	4.192	0.00387	0.506	-0.294	19.197	85.643	34270.427	General	79.688	20.920	38.447
Fractured Basalt	318.75	35	25	111562.5	2.453	0.00073	0.516	-0.095	7.676	63.148	12651.969	General	79.688	16.826	33.894
Andesitic Porphyry	50.15	55	25	20060	5.011	0.00674	0.504	-0.067	4.034	15.079	8190.109	General	12.538	3.520	39.953
Fractured Andesitic Porphyry	50.15	35	25	20060	2.453	0.00073	0.516	-0.150	1.208	9.935	2274.915	General	12.538	2647	33.984
Competent Monzonite	108.25	65	29	43300	8.309	0.02050	0.502	-0.267	15.368	42.844	27353.451	General	27.063	9.049	44.201
Fractured & Altered Monzonite	108.25	35	29	43300	2.845	0.00073	0.516	-0.028	2.607	23.138	4910.523	Custom	10.000	2997	43.448

UCS = Uniaxial Compressive Strength
Source SRK, 2020.

Residual Soil 2.

Located between the zone of the excess material deposit and the leaching platform, corresponding to a saprolite, characterized by a lower degree of weathering of the bedrock compared to that of residual soil 1, some of the texture and structure of the protolith rock is still visible, with an average thickness of 6.0 m. It is made up of high silts, yellowish brown, slightly wet. It is made up of high plasticity silts (MH), yellowish brown to reddish in color and presents angular gravels of 3" maximum size.

Table 16-11:
Results of Direct Shear and Triaxial Shear Tests - Residual Soil – Leached Material Deposit

Sample	Sample Data			Results	
				Effective Parameters	
	Depth (m)	w (%)	P_d (g/cm³)	c´ (kg/cm²)	Φ (°)
CAL-LP-08	2.00	30.27	1.233	12.1	46.4
CAL-LP-09	1.70	31.52	1.372	93.4	24.6
CAL-LP-10	1.16-1.56	5.10	1.857	65.5	36.9
CAL-LP-12	3.20	36.10	1.258	34.5	27.3
CAL-LP-14	4.00	14.20	1.262	13.7	35.1
CAL-LP-15	1.00	24.57	1.417	40.8	35.1
CAL-LP-15	2.00	26.82	1.179	9.8	28.6
CAL-LP-16	2.40	27.66	1.051	21.4	24.6
GEO-LP-021	0.60	22.50	1.571	54.1	32.5
CAL-LP-14(1)	4.00	17.05	1.637	25.9	27.96
CAL-LP-16(1)	2.40	29.10	1.435	49.2	33.74
GEO-LP-022(1)	1.00-1.60	30.70	1.420	64.5	19.13

P_d Dry density w: humidity content c: Effective cohesion Φ: angle of effective friction:
(1) Triaxial test CU

Table 16-12:
Results of Direct Shear and Triaxial Shear Tests - Residual Soil – DME

Sample	Sample Data			Results	
				Effective Parameters	
	Depth (m)	w (%)	P_d (g/cm³)	c´ (kg/cm²)	Φ (°)
CAL-LP-04	4.00	17.85	1.33	24.9	29.1
CAL-LP-019	5.40	29.43	1.431	43.0	27.4
CAL-LP-020	1.50-2.10	35.03	1.296	15.2	29.7
SPT-LP-04	1.80-2.25	26.30	1.213	18.0	29.5
GEO-LP-020(1)	4.00	14.2	1.262	13.7	35.1

Pd Dry density w: humidity content c: Effective cohesion Φ: angle of effective friction:
(1) Triaxial test CU

Table 16-13:
Results of Direct Shear and Triaxial Shear Tests - Residual Soil – Leaching Pad

Sample	Sample Data			Results	
				Effective Parameters	
	Depth (m)	w (%)	P_d (g/cm³)	c´ (kg/cm²)	Φ (°)
CAL-LP-05	2.80-3.10	34.5	1.70	38.5	28.9
CAL-LP-06	4.00	21.5	1.77	124.0	22.9
SPT-LP-02	2.70-3.15	31.1	1.68	29.0	26.8
SPT-LP-03	2.25-2.70	18.7	1.61	41.0	15.4
SPT-LP-04	1.80-2.25	26.6	1.53	18.0	29.5

16.5.7 Ore Materials (Oxide and Transition Material)

These materials will be provisionally disposed of in the stockpiles planned for the project, in proportions that vary according to the needs of the mining operation. The individual geotechnical characterization of each material available for this study is presented below.

16.5.7.1 Oxides

The oxides are classified, following the Unified Soil Classification System (USCS), as silty sands (SM), with medium plasticity, slightly wet, composed of 10% gravel, 60% sand and 30% fines; having a specific gravity for solids (SGs) of 2.73.

The data and results of the soil classification modified by Proctor, maximum and minimum densities, and hydraulic conductivity tests are shown in Table 16-14. The results of triaxial compression test for oxides is in Table 16-15.

Table 16-14:
Results of Tests Performed on Oxides Material

Sample	USCS	SGs	OMC(1) (%)	MDD(1) (g/cm³)	OCMc (%)	MDDc (g/cm3)	ρ_{dmin} (2) (g/cm³)	ρ_{dmax} (2) (g/cm³)	k(3) (cm/s)
OX-CM	SM	2.73	19.3	1.687	18.0	1.718	1.149	1.415	4.45E-6

USCS: Unified Soil Classification System
OMC: optimum moisture content;
MDD: maximum dry density;
OCMc: Optimum corrected moisture content;
MDDc: Maximum corrected dry density;
SGs: Specific Gravity of solids;
ρ_{dmin}: minimum dry density;
ρ_{dmax}: maximum dry density;
k: permeability.

(1) Material passing the ¾" mesh.
(2) Dry material passing through the 1" mesh.
(3) Sample remolded at pd=1.50 g/cm³ and w=14%. Effective confinement equal to 300 kPa.

Table 16-15:
Results of Triaxial Compression Test CU - Oxides

Sample	USCS	w (%)	ρ_d (g/cm³)	Effective Parameters		Total Parameters	
				c´ (kg/cm²)	Φ (°)	c´ (kg/cm²)	Φ (°)
OX-CM	SM	14.0	1.50	8.0	35.0	61.0	18.0

ρ_d: dry density; **w**: moisture content; **c**: cohesion; **Φ:** friction angle

16.5.7.2 Transitional

The transition material is classified, following the Unified Soil Classification System (USCS), as a poorly graded gravel (GP) with angular, fine, slightly wet, low plasticity, slightly moist particles of low plasticity, slightly wet. It is made up of 80% gravel, 10% sand and 10% fines; and, it has a specific gravity of 2.61 solids.

The data and results of the soil classification tests, modified Proctor, maximum and minimum densities and hydraulic conductivity are shown in Table 16-16. The results of triaxial compression test on oxide material is showing in Table 16-17.

Table 16-16:
Results of Tests Performed on Transition Material

Sample	USCS	SGs	OMC(1) (%)	MDD(1) (g/cm^3)	OCMc (%)	MDDc (g/cm^3)	ρ_{dmin} (2) (g/cm^3)	ρ_{dmax} (2) (g/cm^3)	k(3) (cm/s)
TX-CM	SM	2.61	14.8	1.81	8.1	1.942	1.39	1.647	5.7E-6

USCS: Unified Soil Classification System; OMC: optimum moisture content;
MDD: maximum dry density; OCMc: Optimum corrected moisture content;
MDDc: Maximum corrected dry density; SGs: Specific Gravity of solids;
ρ_{dmin}: minimum dry density; ρ_{dmax}: maximum dry density; k: permeability.

(1) Material passing the ¾" mesh.
(2) Dry material passing through the 1" mesh.
(3) Sample remolded at pd=1.50 g/cm^3 and w=14%. Effective confinement equal to 300 kPa.

Table 16-17:
Results of Triaxial Compression Test on Oxide Material

Sample	USCS	w (%)	ρ_d (g/cm^3)	Effective Parameters		Total Parameters	
				c´ (kg/cm^2)	Φ (°)	c´ (kg/cm^2)	Φ (°)
TX-CM	SM	10.0	1.65	17.0	39.5	11.0	24.5

ρ_d: dry density; **w**: moisture content; **c**: cohesion; **Φ:** friction angle

16.6 DESIGN CRITERIA

16.6.1 Leaching Pad

The design criteria defined for the development of the study at the PEA of the dynamic leaching pad and its associated components are listed in Table 16-18.

Table 16-18:
Leach Pad Design Criteria

Item	Criteria	Unit	Value	Source Reference
1.00	**Operational Criteria**			
1.01	Daily ore production	t/d	15,000	MQ
1.02	Operation time	month	54	MQ
1.03	Mineral type	-	Oxides, transition and sulfides	MQ
1.04	Total oxide production	t	11,954,538	MQ
1.05	Total transition production	t	6,065,732	MQ
1.06	Total sulfide production	l/h/m2	10	MQ
2.00	**Leach Pad**			
2.01	Total area	ha	8.0	MQ
2.02	Effective leaching area	ha	4.4	MQ
2.03	Total number of cells	units	22	MQ
2.04	Number of operational cells	units	21	MQ
2.05	Number of standby cells	units	1.0	MQ
2.06	Cell length	m	90	MQ
2.07	Cell crest width	m	18	MQ

Item	Criteria	Unit	Value	Source Reference
2.08	Stacking height	m	5.0	MQ
2.09	Operating width for belts feeding	m	25	MQ
2.10	Impermeable coating	-	LLDPE geomembrane double textured 2 mm	SRK
2.11	Collector pipes	-	HDPE double wall, corrugated exterior	SRK
3.00	**PLS Pond**			
3.01	Minimum volume of storage	m^3	10,000	SRK
3.02	Primary impermeable coating	-	LLDPE geomembrane smooth 1.5 mm	SRK
3.03	Secondary impermeable coating	-	22	SRK
4.00	**PME Pond**			
4.01	Minimum volume of storage	m^3	10,000	SRK
4.02	Primary impermeable coating	-	LLDPE geomembrane smooth 1.5 mm	SRK
4.03	Secondary impermeable coating	-	LLDPE Geomembrane Smooth 1.5 mm	SRK
5.00	**Agglomeration plant**			
5.01	Platform length	m	107	MQ
5.02	Platform width	m	85	MQ
6.00	**Process plant**			
6.01	Platform length	m	65	MQ
6.02	Platform width	m	30	MQ
7.00	**Operation access**			
7.01	Minimum width	m	5.0	SRK
7.02	Maximum slope	m	15.0	SRK
8.00	**Physical Stability of Slopes**			
8.01	Return period of the earthquake design	years	475	SRK
8.02	Seismic coefficient	-	0.5 x PGA	SRK
8.03	Minimum safety factor for static conditions	-	1.4 -1.5	-1
8.04	Minimum safety factor for pseudostatic conditions	-	1	CDA
9.00	**Hydraulic Criteria**			
9.01	Return period of the design avenue	years	100	MQ-SRK
9.02	Minimum free edge pools	m	1	SRK
9.04	Minimum free edge channels	m	0.3	SRK

CDA: Canadian Dam Association; PGA: peak ground acceleration.
(1) For the analysis of the heap leaching the value of 1.4 will be used; meanwhile, for the leach pad platform (or embankment) 1.5 will be used.

16.6.2 Leached Material Deposit

The defined tailings deposit design criteria and associated components are shown in Table 16-19, at the PEA level.

Table 16-19:
Leached Material Deposit Design Criteria

Item	Criteria	Unit	Value	Source Reference
1.00	**Operational Criteria**			
1.01	Daily tailing production	t/d	15,000	MQ
1.02	Operation time	month	54	MQ
1.03	Tailing type	-	Oxides, transition and sulfides	MQ
1.04	Total amount of tailings	t	21,369,273	MQ

Item	Criteria	Unit	Value	Source Reference
1.05	Total amount of leached oxides	t	11,954,538	MQ
1.06	Total amount of transition leached	t	6,065,732	MQ
1.07	Total amount of leached sulfides	t	3,349,003	MQ
2.00	**Impermeability and Basal Drainage**			
2.01	Impermeable coating	-	LLDPE geomembrane double textured 2 mm	SRK
2.02	Collector pipes	-	HDPE double wall, corrugated exterior	SRK
3.00	**Groundwater Monitoring Pond**			
3.01	Material	-	Reinforced concrete	SRK
3.02	Length	m	2,0	SRK
3.03	Width	m	2,0	SRK
3.04	Depth	m	1,5	SRK
4.00	**Contact Water Pool**			
4.01	Dam material	-	Compacted fill	SRK
4.02	Primary impermeable coating	-	LLDPE geomembrane smooth 1.5 mm	SRK
4.03	Secondary impermeable coating	-	LLDPE geomembrane smooth 1.5 mm	SRK
5.00	**Operation Access**			
5.01	Minimum width	m	5,0	SRK
5.02	Maximum slope	%	15,0	SRK
6.00	**Physical Stability of Slopes**			
6.01	Return period of the earthquake design	Years	475	SRK
6.02	Seismic coefficient	-	0,5 x PGA	SRK
6.03	Minimum safety factor for static conditions	-	1,5	CDA
6.04	Minimum safety factor for pseudostatic conditions	-	1,0	CDA
7.00	**Hydraulic Criteria**			
7.01	Design avenue return period	Years	100	MQ-SRK
7.02	Minimum free edge pools	m	1,0	SRK
7.04	Minimum free edge channels	m	0,3	SRK

CDA: Canadian Dam Association; PGA: peak ground acceleration.
Source: SRK Consulting (Peru) S.A.

16.6.3 Stockpiles of Oxides and Transition

The design criteria defined for the development of the study at the PEA level of the oxide stockpiles, transition and their associated components are listed in Table 16-20.

Table 16-20:
Design Criteria for Oxide and Transition Stockpiles

Item	Criteria	Unit	Value	Source Reference
1.00	**Operational Criteria**			
1.01	Operation time	month	54	MQ
1.02	Type of ore going to the stockpile	-	Oxides and transition	MQ
1.03	Total amount of ore going to the stockpile	t	958,335	MQ
1.04	Total amount of oxides going to the stockpile	t	677,548	MQ
1.05	Total amount of transition going to the stockpile	t	280,787	MQ
2.00	**Impermeability and Basal Drainage**			

Item	Criteria	Unit	Value	Source Reference
2.01	Impermeable coating	-	LLDPE geomembrane double textured 2 mm	SRK
2.02	Collector pipes	-	HDPE double wall, corrugated exterior	SRK
3.00	**Operation access**			
3.01	Minimum width	m	5.0	SRK
3.02	Maximum slope	%	15.0	SRK
4.00	**Physical Stability of Slopes**			
4.01	Design earthquake return period	Years	475	SRK
4.02	Seismic coefficient	-	0.5 x PGA	SRK
4.03	Minimum safety factor for static conditions	-	1.5	CDA
4.04	Minimum safety factor for pseudostatic conditions	-	1.0	CDA
5.00	**Hydraulic Criteria**			
5.01	Design avenue return period	Years	100	MQ-SRK
5.02	Minimum free edge pools	m	1.0	SRK
5.04	Minimum free edge channels	m	0.3	SRK

CDA: Canadian Dam Association; PGA: peak ground acceleration.
Source: SRK Consulting (Peru) S.A.

16.6.4 Deposit of Surplus Material

The design criteria defined for the development of the study at the PEA level of the surplus material deposit and its associated components are indicated in Table 16-21.

Table 16-21:
Surplus Material Storage Design Criteria

Item	Criteria	Unit	Value	Source Reference
1.00	**General criteria**			
1.01	Materials to store	-	Surplus cuts of natural land and soil with organic content	MQ
1.02	Foot dam material	-	Compacted fill	SRK
1.03	Basal drainage system	-	Double wall HDPE pipes, corrugated outer	SRK
3.00	**Leakage monitoring pond**			
3.01	Material	-	Reinforced concrete	SRK
3.02	Length	m	2.0	SRK
3.03	Width	m	2.0	SRK
3.04	Depth	m	1.5	SRK
5.00	**Operation access**			
5.01	Minimum width	m	5.00	SRK
5.02	Maximum slope	%	15	SRK
6.00	**Physical stability of slopes**			
6.01	Return period of the earthquake design	years	475	SRK
6.02	Seismic coefficient	-	0.5 x PGA	SRK
6.03	Minimum safety factor for static conditions	-	1.50	CDA
6.04	Minimum safety factor for pseudostatic conditions	-	1.00	CDA
7.00	**Hydraulic criteria**			
7.01	Return period of the design avenue	years	100	MQ-SRK

7.02	Minimum free edge channels	m	0.3	SRK

CDA: Canadian Dam Association; PGA: peak ground acceleration.
Source: SRK Consulting (Peru) S.A.

16.7 ECONOMIC MINERAL RESOURCE ESTIMATION

The assumptions for the potentially Economic Mineral Resource estimation procedure for La Cumbre project is based on the development of leached and sulfide pit. Total material is showed in Table 16-22, for the Oxide Zone in Table 16-23, for Transitional Zone in Table 16-24 and for Primary Zone in Table 16-25.

Table 16-22:
Total Economic Mineral Resource Estimates for La Cumbre Deposit

Classification	Tonnage (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
MEASURED	103,772	0.569	1.579	1,898	5,267
INDICATED	2,750	0.400	1.294	35	114
TOTAL M & I	**106,522**	**0.564**	**1.571**	**1,933**	**5,381**

Notes:

1. Economic Mineral Resource is not the same as an NI 43-101 Mineral Reserve.
2. The Economic Mineral Resource estimation is a Mineral Estimate output under PEA study.
3. Gold spot price 1575 US$/oz and silver spot price 20 US$/oz.
4. Cut-off for leachable materials is 0.243 Au g/t and for mill process is 0.20 Au g/t
5. The economic mineral resource disaggregated by domain are showed in the Table 16-23, Table 16-24 and Table 16-25.

Table 16-23:
Economic Mineral Resource Estimates for Oxide Zone

Classification	Tonnage (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
MEASURED	13,867	0.745	1.610	332	718
INDICATED	1,052	0.399	1.322	13	45
TOTAL M & I	**14,919**	**0.721**	**1.589**	**346**	**762**

Table 16-24:
Economic Mineral Resource Estimates for Transitional Zone

Classification	Tonnage (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
MEASURED	7,861	0.666	1.599	168	404
INDICATED	1,429	0.401	1.386	18	64
TOTAL M & I	**9,290**	**0.625**	**1.566**	**187**	**468**

Table 16-25:
Economic Mineral Resource Estimates for Primary Zone

Classification	Tonnage (kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
MEASURED	82,044	0.530	1.571	1,397	4,145
INDICATED	269	0.397	0.695	3	6
TOTAL M & I	**82,313**	**0.529**	**1.569**	**1,401**	**4,151**

16.8 CUT-OFF ESTIMATION

Factor used for cut-off grade estimation for La Cumbre is according to the following assumptions.

16.8.1 Price

The prices assumptions for Au and Ag in this document for both the Resource Estimate and the Financial Evaluation are US$ 1,750/oz and US$ 22/oz respectively. On the other hand, the prices for Au and Ag for the Cut-off Estimation are US$ 1,575/oz and US$ 20/oz respectively (Minera Quinchia & LINAMEC, 2022).

The market consensus of 30 banks expects the gold price to hold between US$ 1,600 and US$ 1,962 in 2022, while long-term price forecasts largely range between US$ 1,360/oz and US$ 2,030/oz.

16.8.2 Metal Recovery

The metal recovery for each commodity and leach and mill treatment process, are in Table 16-26.

**Table 16-26:
Metal Recovery**

Gold Recovery for Leach process	%
Oxide	85.5
Transitional	85.5
Sulfide	36.1
Silver Recovery for Leach process	**%**
Oxide	47.0
Transitional	47.0
Sulfide	0.0
Maximum Copper Grade under leach process for sulfide materials	0.1
Leach Copper recovery	
Oxide/Transitional/Sulfides	0.0
Gold Recovery for Mill Process	**%**
Oxide	0.0
Transitional	0.0
Sulfide	84.1
Silver Recovery for Mill Process	**%**
Oxide	0.0
Transitional	0.0
Sulfide	51.9
Copper Recovery for Mill Process	**%**
Oxide/Transitional/Sulfide	0.0

Note: Cut-off grade for sulfide material of leach process for sulfide material is 0.406 and not used and only to demonstrate why mill process is better than Leach for sulfide material.

16.8.3 Mining Cost, Processing Cost

The detailed mining and processing cost for La Cumbre proyect are in Table 16-27

**Table 16-27:
Mining Cost and Processing Cost**

Mining Cost as Ore/Waste	Value	Units
Oxide	1.95	US$/t
Transitional	1.95	US$/t
Sulfide	1.95	US$/t

Haulage cost (raise or deep)	0.01	US$/t/bench
Base level for haulage cost	1900	masl
Processing Cost	**Value**	**Units**
Leach and ADR Processing Cost		
Oxide	9.08	US$/t
Transitional	9.08	US$/t
Sulfide	6.08	US$/t
General and Administrative expenses	**Value**	**Units**
Oxide	1.12	US$/t
Transitional	1.12	US$/t
Sulfide	1.12	US$/t
Total Leach and ADR Processing Cost	**Value**	**Units**
Oxide	10.20	US$/t
Transitional	10.20	US$/t
Sulfide	7.20	US$/t
Mill and Flotation Processing Cost	**Value**	**Units**
Oxide	0.00	US$/t
Transitional	0.00	US$/t
Sulfide	7.14	US$/t
General and Administrative expenses	**Value**	**Units**
Oxide	0.00	US$/t
Transitional	0.00	US$/t
Sulfide	1.12	US$/t
Total Mill and Flotation Processing Cost	**Value**	**Units**
Oxide	0.00	US$/t
Transitional	0.00	US$/t
Sulfide	8.26	US$/t

Note: Processing cost and metal recovery are in case of sulfide material by leaching are decision cost information. Analysis shows the cost opportunity to treat sulfide materials by mill and flotation is better than heap leach process.

16.8.4 Smelter Terms

Table 16-28:
Smelter Terms

Selling Cost		
Gold	47	US$/oz
Silver	0	US$/oz
Net Spot Price		
Gold	1528	US$/oz
Silver	20	US$/oz
Net Spot Price		
Gold	49.118	US$/gr
Silver	0.643	US$/gr

Note: Gold selling cost of 47 $/oz is 2.71% to include TC & RC, transport Cost. It is a gross estimated from LINAMEC preliminary internal job for La Cumbre.

16.8.5 Geometry and Geotechnical Factors

Table 16-29:
Pit Geometry Parameters

IRA* by Geotechnical Domains	
Ash	39°
Oxide	39°
Transitional	43°
Sulfide	38°
Pit Geometry	
Bench Height	6 m
Bench Face angle	70 m
Ramp Width	12 m

*IRA = Inter-Ramp Angle

16.8.6 Cut-off Estimation

The cut-off grade (COG) at La Cumbre is mainly controlled by the gold content as the primary product, representing 98% of the Net Smelter Return (NSR) value. Silver content is a by-product and is not included in the cut-off grade estimation, Table 16-30 shows the cut-off value estimation.

Table 16-30:
Gold Cut-off Value Estimation

Item	Value	Units
Mining cost	1.95	US$/t
Leach and ADR Processing Cost	9.08	US$/t proc
Mill Processing Cost	7.14	US$/t proc
General and Administrative expenses	1.12	US$/t proc
Gold recovery as Leach	85.5	%
Gold recovery as Mill	84.1	%
COG for Leach Process		
Oxide	0.243	g/t
Transitional	0.243	g/t
Sulfide	0.406	g/t
COG for Mill Process		
Sulfide	0.200	g/t

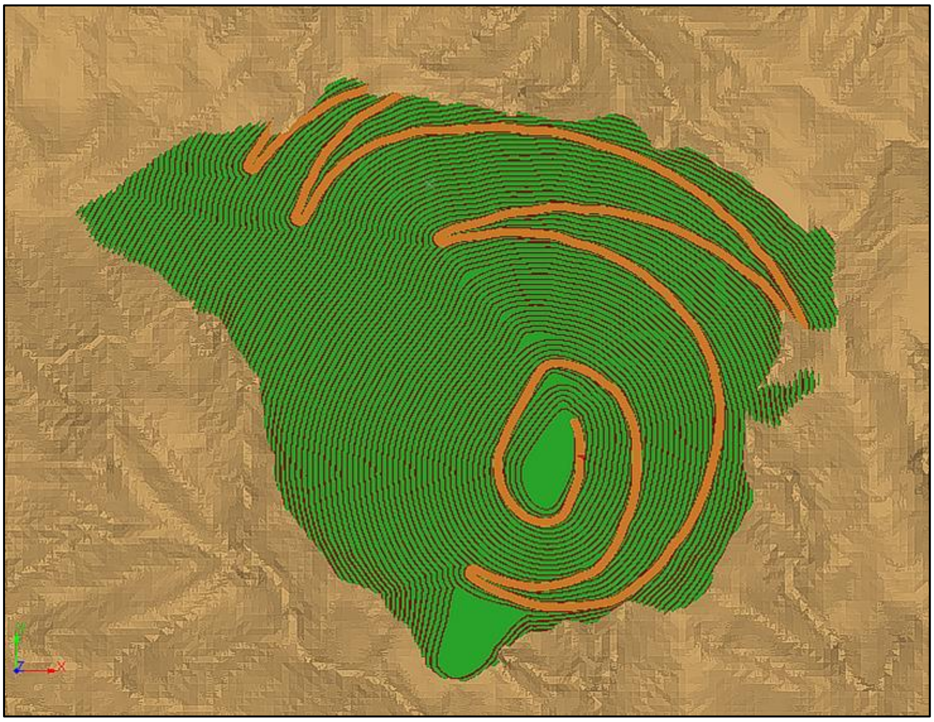
Note: Cut-off grade for sulfide material of leach process for sulfide material is 0.406 and not used and only to demonstrate why Mill process is better than Leach for sulfide material.

16.9 MINE SCHEDULING

Mine scheduling for La Cumbre project was made using MineSight®, MS3D, MS Reserve and MSSO applications. The designed ultimate pit, the subset of mineral resource estimation and designing mining phases was used as input. Material type in three (3) mineral types was coded leach, mill or waste for scheduling purpose.

No destinations scheduling was made for La Cumbre project, either it is an ore/waste stockpile, waste dump or tailing storage facilities because they are at early designing stage. However, a total balance from these were considered as requirements for infrastructure. Figure 16-13 shows the final pit for La Cumbre.

Figure 16-13:
La Cumbre Final Pit with Ramp Access



Mine scheduling consists of a total eight (8) mining phases, to be mined using 6 m benches. The mine plan is for heap leaching process to be implemented first, and in a second stage will start the mill-flotation-gravimetry plant process and it is allowed using the first three mining stages as priority of scheduling.

The production schedule was developed with the following criteria:

- The rate capacity is 5.4 Mty for heap leaching process and 10.8 Mty for mill process
- The mine development is rated at average ten (10) bench mined per year and maximum 20 in case of limited tonnes bench size.
- The ramp-up for mill-flotation plant processes assumed to be the year 5 will after finishing leachable pit when sulfide pit is planned to be mined.
- Mine capacity is thought to be with 35 tonne truck capacity or almost four working mine areas at the same time of scheduling at minimum.
- Mining capacity is restricted by deepening rate.
- Leachable ore from 3 to 8 mining phases is planned to be processed after managed by stockpiling caused by the tonnage size when it is mined.

The outputs of the mine scheduling and ore processing are presented in Figure 16-14, and Figure 16-15, two section with final pit and block model for gold are in Figure 16-16 and Figure 16-17.

Figure 16-14:
Proposed Mining Plan

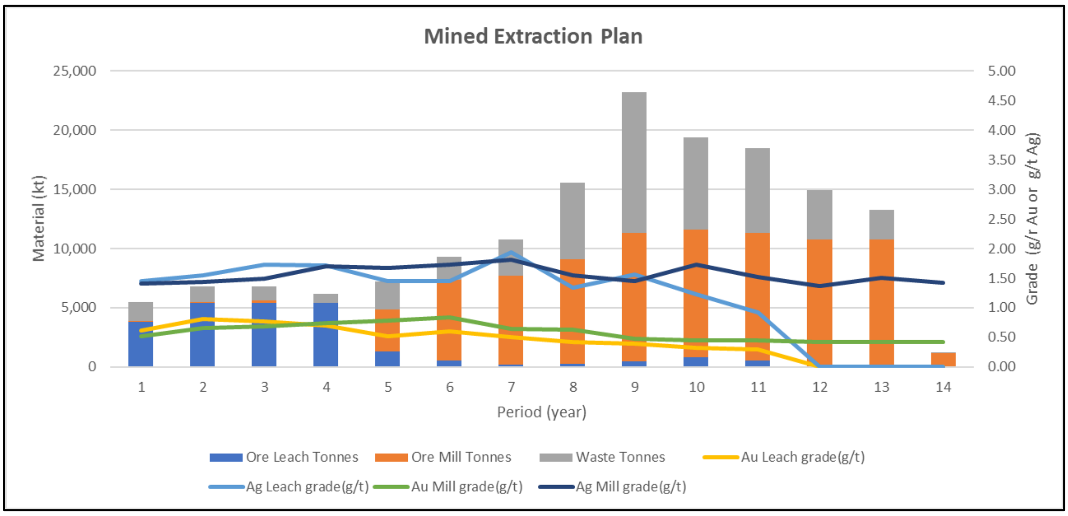


Figure 16-15:
Feed to Plant

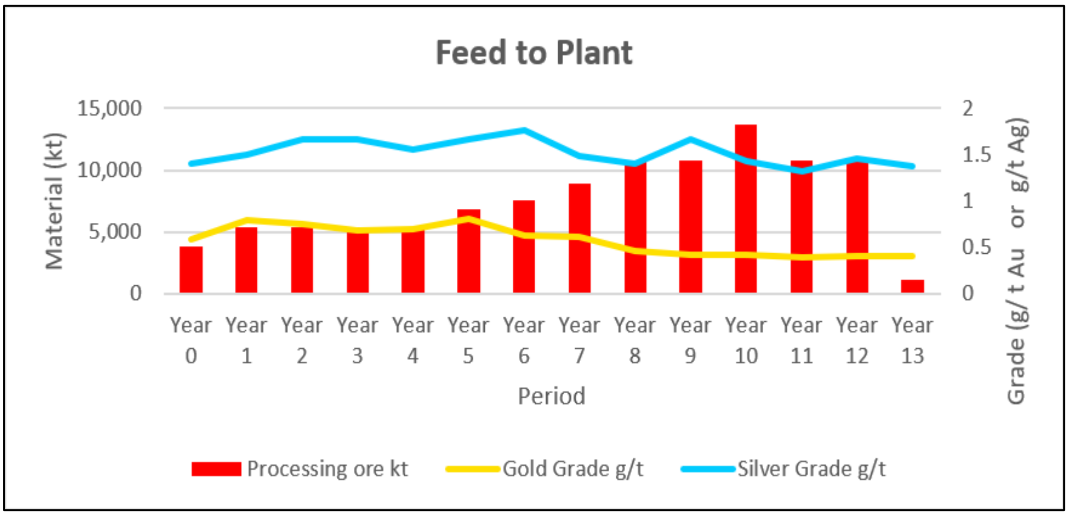


Figure 16-16:
Final Pit Sections North-South at 420752 E

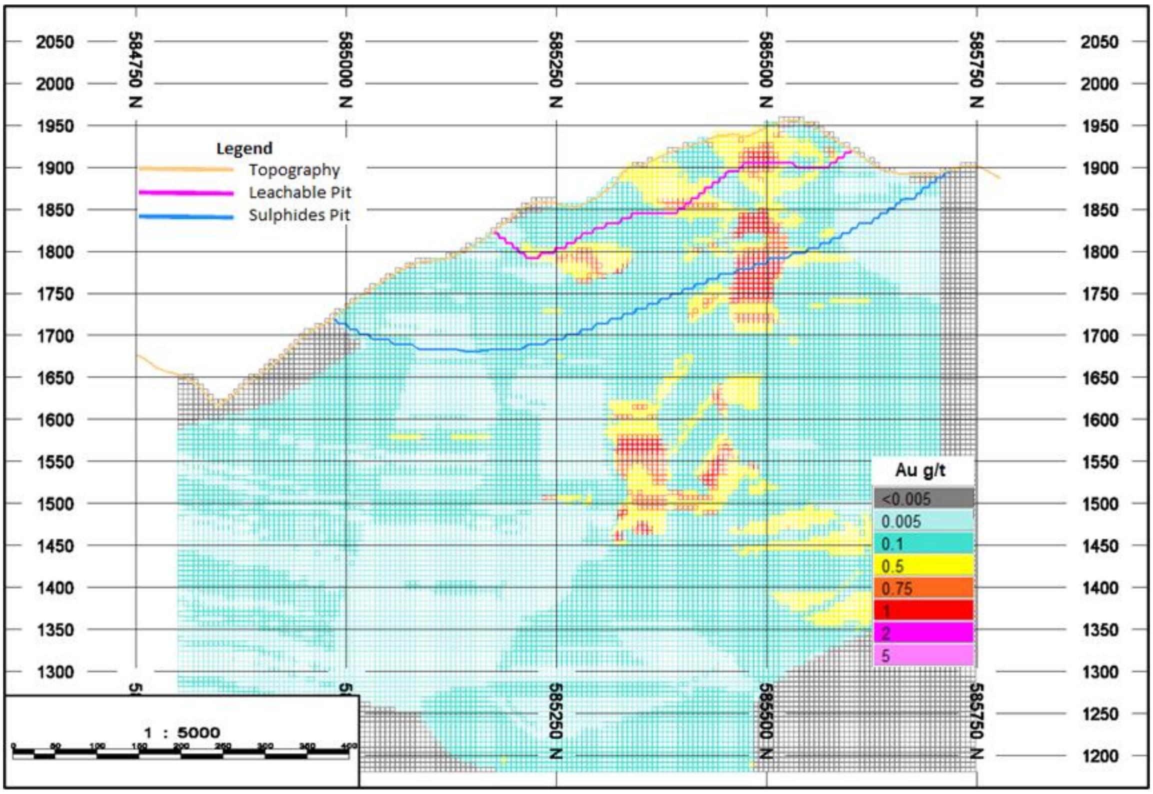


Figure 16-17:
Final Pit Sections North-South at 421004 E

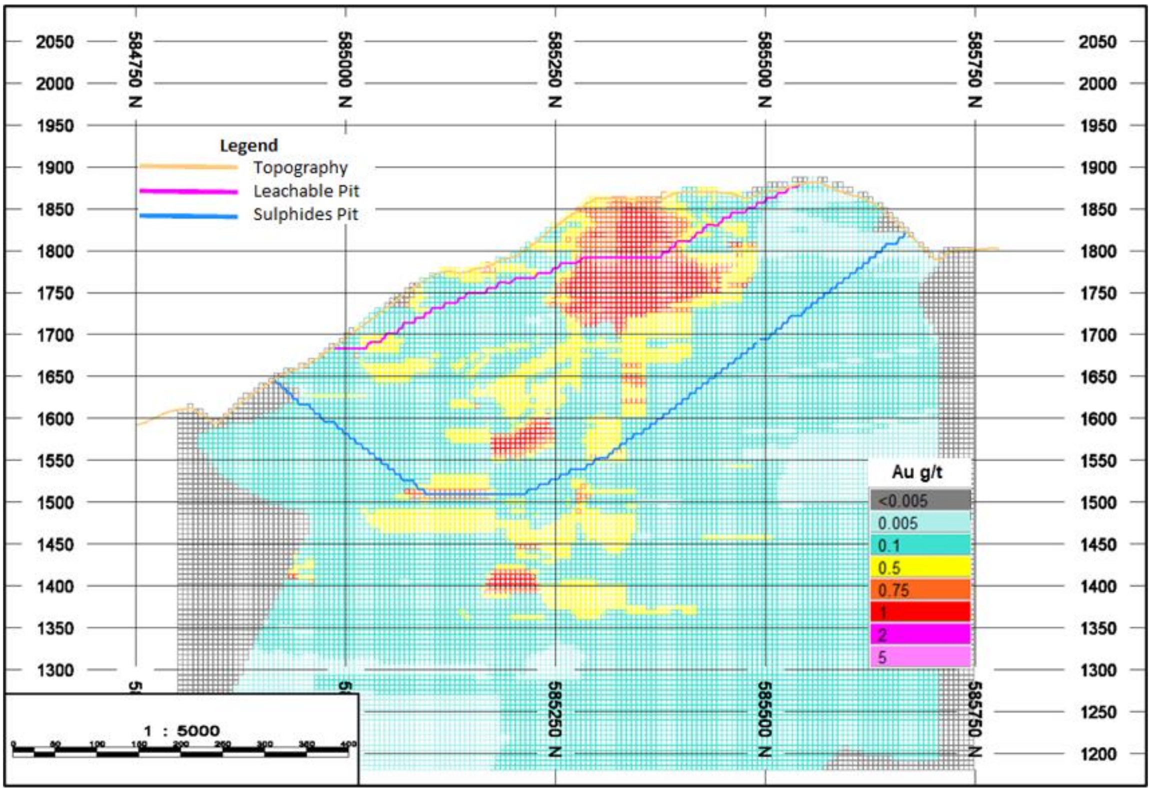


Table 16-31 shows the proposed annual mine production schedule. Table 16-32 and Figure 16-18 shows the proposed processing schedule.

Table 16-31:
Life of Mine Plan

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year: 8	Year: 9	Year 10	Year 11	Year 12	Year 13	Year 14	LoM
Ore Leach kt	3,844	5,400	5,400	5,400	1,293	571	193	232	492	834	550	0	0	0	24,209
Ore Mill kt	65	94	223	34	3,563	6,800	7,534	8,883	10,800	10,800	10,800	10,800	10,800	1,187	82,384
Waste kt	1,605	1,281	1,190	770	2,392	1,948	3,007	6,482	11,952	7,777	7,129	4,163	2,485	9	52,191
Leach Au g/t	0.61	0.81	0.77	0.70	0.52	0.60	0.50	0.42	0.39	0.33	0.30	0.00	0.00	0.00	0.68
Leach Ag g/t	1.46	1.55	1.73	1.72	1.45	1.45	1.94	1.34	1.56	1.23	0.92	0.00	0.00	0.00	1.58
Mill Au g/t	0.52	0.66	0.69	0.74	0.78	0.84	0.65	0.63	0.48	0.44	0.45	0.42	0.42	0.42	0.53
Mill Ag g/t	1.42	1.44	1.49	1.70	1.67	1.73	1.82	1.54	1.45	1.73	1.52	1.37	1.51	1.42	1.57

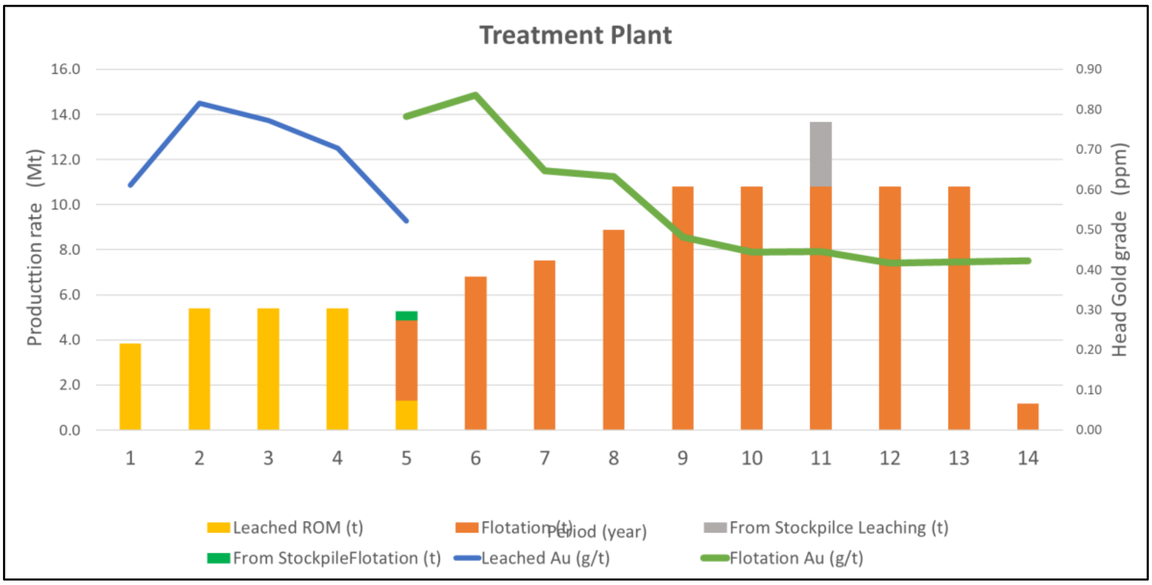
Note: Leach include oxide and transitional ore. Mill is referred as Flotation-Gravimetry plant and include only primary sulfide ore.

Table 16-32:
Processing Schedule

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Heap Leaching														
Processing ore Mt	3.80	5.40	5.40	5.40	1.30						2.9			
Au g/t	0.61	0.81	0.77	0.70	0.52						0.4			
Ag g/t	1.46	1.55	1.73	1.72	1.45						1.33			
Flotation														
Processing ore kt					4.0	6.8	7.5	8.9	10.8	10.8	10.8	10.8	10.8	1.2
Au g/t					0.78	0.84	0.65	0.63	0.48	0.44	0.45	0.42	0.42	0.42
Ag g/t					1.67	1.73	1.82	1.54	1.45	1.73	1.52	1.37	1.51	1.42

Note: Processing ore include leach and sulfide ore. Leach include oxide and transitional ore. Mill is referred as Flotation-Gravimetry plant and include only primary sulfide ore. Treatment plan is a conceptual plan from the strategic mine plan at PEA level prepared by Batero's staff.

Figure 16-18:
Conceptual Treatment Plant by Process



16.10 OTHER RELEVANT FACTOR

Other relevant information about La Cumbre project is referred:

- The proposed ore haulage at the La Cumbre project is based on a conveyor belt system in addition to 35-tonne capacity trucks. Minera Quinchia considers that possibility of using only trucks as transportation method would increase social risks due to increased road traffic.
- Consequently, the ore will be transported by trucks to a temporary zone within the project area, and from there, it is being planned to use conveyor belts to transport it to the interim stockpiles in the La Perla or Matecaña sector. This re-handling is planned to be supported by LHD equipment to the area assigned for final heap leaching.
- The residual material from the leaching, called tailings, is also planned to be hauled to the leached material deposit by an overland system. Waste material could benefit from the belt conveyor system in the case of waste dump in La Perla and Matecaña sector. These systems allow to reduce hauling cost and social risk using public road.
- An additional benefit of using a belt conveyor system is the potential generation of electrical energy from the operations in the favorable -549 vertical distance over 2250 in length. The overland belts have a descending path that allows conceptualizing their regenerative nature, resulting in a possible economic advantage estimated at -0.09 $/ton compared to truck system transport. These possibilities are still under study and have not been defined in this PEA.

16.11 MINE FACILITIES

Research works by Anddes Asociados S.A.C. (March 2020) and SRK (March 2021), provides for major components at the La Cumbre project such as a dynamic leach pad for oxides, an agglomeration plant, cyanide solution handling ponds, temporary or transitional stockpiles, as well as several waste rocks dumps, ore tailings ponds, and a leach tailings pond. For both leach and sulfide processing plants, the concepts are only in conceptual stages.

Waste rock dumps are in conceptual design stages to balance the pit constraints. Geometric and stability definitions for these components are not fully justified and further investigation is required in the upcoming stages. Santa Inés as a component of the TSF is one of the primary structures yet to be preliminarily studied. Table 16-33 and Figure 16-19 show the conceptualization of these structures.

Table 16-33:
Component Overview

Zone	Studies by	Component Description	Component ID	Capacity (Mt)
Paramillo	RDC	DMO	DMO 1	1.1
Matecaña	SRK	Temporal Stockpile	SP 1	2.9
Matecaña	SRK	Waste Dump	WSF1	12
Los Castros	AMEC	Waste Dump (B3)	WSF2	19
Los Castros	AMEC	Waste Dump (B3.1)	DMO2	2.9
Palo Grande	RPA	Waste Storage Facility 3 (B4)	WSF3	40.6
La Perla	SRK	Leached Material (tailings)	LM1	21.3
La Perla	In house	Leached Material	LM2	3.5
La Perla	SRK	Temporal stockpile and blending	SP2	0.26
Santa Ines	In house	Tailing Storage Facility	TSF	82.5

16.11.1 Waste Dump Storage Facilities

At the La Cumbre project, several waste dumps have been proposed. On the north side of the pit at least 4 waste dumps have been identified, and all hauling is only based by truck system. In the southern part of the pit, the waste dumps have limited capacity overlapping the oxide/transition material stockpiles (case SP1 and WSF1). In addition, all residual material from the leaching process, called leachate tailings, is expected to be stored in the LM1/LM2 landfill. While the transition material will be stored in the temporary stockpiles (SP2, SP1).

The waste storage facilities located in the southern part of the pit are planned to be transported either by trucks or by a mixed conveyor belt-truck system, which has not yet been tested, as in the case of the dumps and stockpiles located in the Matecaña and La Perla areas. Figure 16-19 shows the location of these components for the La Cumbre project.

16.11.2 Tailings Storage Facility

Conceptually, the Santa Inés area was considered a tailings storage area for material generated by the flotation plant, requiring a pipeline system to convey the tailings.

16.11.3 La Perla Ore Stockpile and Temporal Stockpile

Oxide dumps will be for leachable materials in transit or temporary dumps in the proximity of the leaching plant. The case for Stockpile SP1 y SP2.

16.11.4 Dynamic Head leaching pad and Cyanide Pond

Located in the area known as La Perla, it consists of a leaching platform and two (2) cyanide management ponds. One pond is called PLS to manage pregnant solutions and the other one is called PME and is planned as a contingency pond in case of extreme precipitation events of up to 100 years return period.

The material will be leached in stages of disposal, leaching and removal periods, thus called dynamic pad. At the end of the leaching of the heaps in the pad, they will be removed with a compact bucket wheel and sent by a conveyor belt system to the leached material deposit for final disposal.

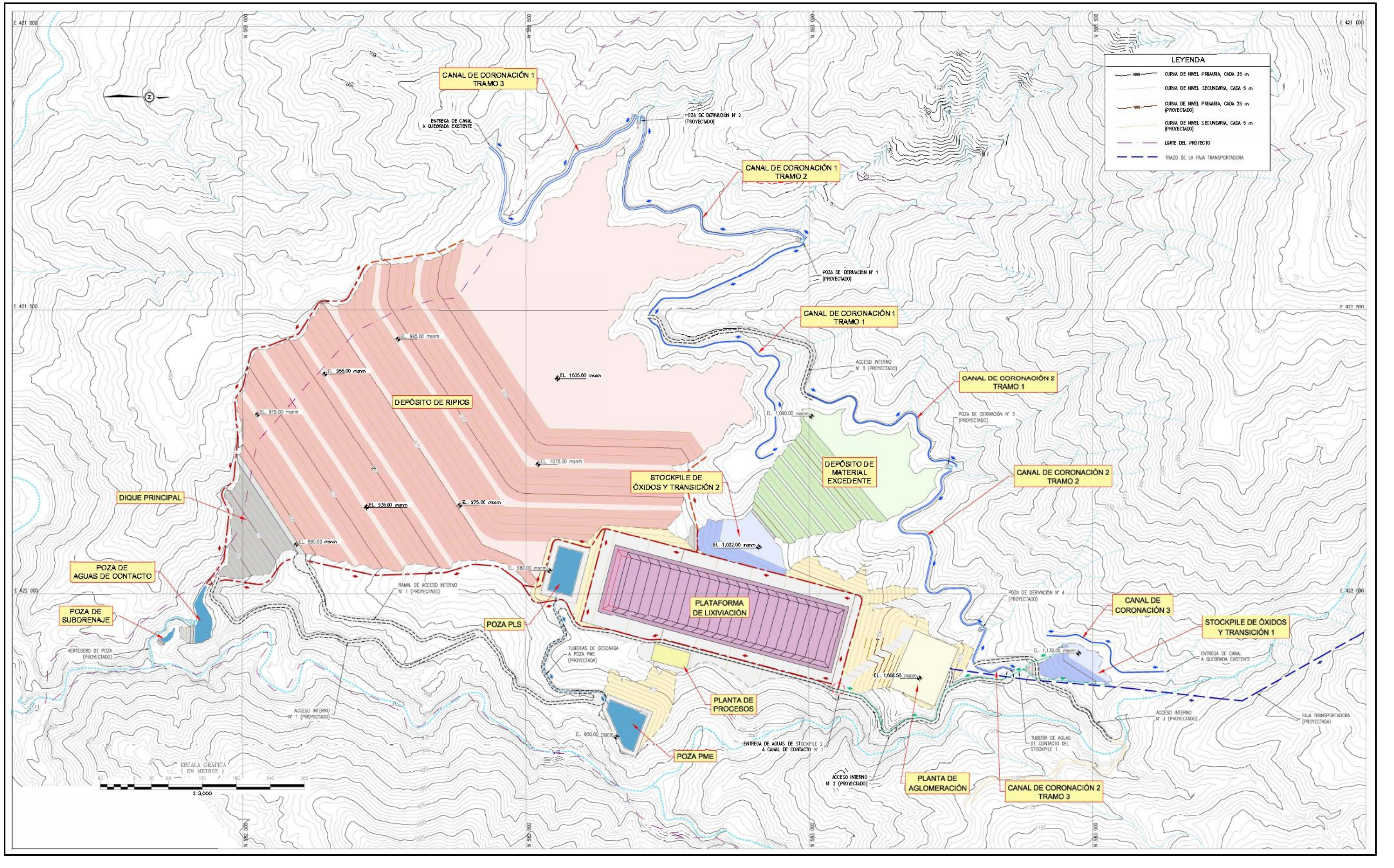
The leaching platform comprises a total area of 8.0 ha, of which 4.4 ha correspond to the effective leaching area and the rest to areas for the installation and operation of equipment for the transport, stacking and extraction of minerals and leached material.

The effective leaching area has a total width of 107 m and length of 413 m. It consists of 22 cells, of which 21 are considered as operating cells and one in standby. Surrounding the effective leaching area is a 6.8 m wide perimeter berm and a 24 m strip for the operation of ore transport and stacking equipment.

16.11.5 ADR Plant and Agglomeration Plant

The ADR plant and agglomeration plant are part of the areas adjacent to the La Perla dynamic leach dumps. It includes the design of an Absorption-Desorption-Reactivation (ADR) plant and an area for the agglomeration plant. In this case, agglomeration is the previous step to the leaching process for all the materials called oxides and transitional. Feasibility studies are underway to consider the feasibility of a leaching-only operation. Components associated with Heap leaching process and ADR plant are showed at Figure 16-19.

Figure 16-19:
Major Components for Leaching Materials (Source: SRK, 2021)



16.12 MINE OPERATIONS UNITS

16.12.1 Drilling and Blasting

Because rock strength test and compression test work have not been done for La Cumbre deposit, LINAMEC estimated powder factor values of approximately 0.28 kg/t and 0.24 kg/t for mineralized material and for waste material, respectively. The average drill penetration rate for mineralized material and waste was estimated to be 38 m/h. Typical pattern designs assumed Atlas Copco PV271 drill equipment with a 251 mm of drill diameter size.

For the pre-strip mining two drills are required, and for the peak years of production, four Atlas Copco PV271 drills are required. Although there is no evidence that water table impaction in blasting, a water-resistant explosive based in emulsion was assumed (ANFO/emulsion type 65%/35%) and it could be demanded for rainy season too.

16.12.2 Loading

Mining at La Cumbre is planned to be carried out with loading equipment based on 4.6 m^3 capacity excavators, type 374F CAT models, which have been considered to fit the bench height considered in the pit design.

16.12.3 Hauling

For the present scenario, the use of a conveyor belt combined with trucks has been contemplated in case of transporting the materials to the leaching plant, located towards the south of the pit in the direction of the area known as La Perla and Matecaña. The southern areas of the pit, where there are waste rock deposits that could benefit from the mixed truck-conveyor belt transport system. In the case of transportation to the waste deposits located in the northern part of the pit, only trucks are planned to be.

The material allocation and the gross hauling distance based on truck system only are between 0.40 and 3.5 kilometer. The distance of transportation by conveyor belt is not part of these estimation. Table 16-34 show a preliminary estimation of the hauling distance from the pit to different destinations only for the case of pit.

Table 16-34:
Hauling Distance by Destinations

Destination	Material	Haulage by trucks (km)
DMO 1	Ash	1.68
SP1	Leached material	0.70
WSF1	Waste	1.20
WSF2	Waste	3.38
DMO 2	Ash	0.75
WSF3	Waste	1.68
SP2	Temporal Stockpile and Blending	0.40
TSF	Tails	1.68

Note: Leached material and TSF are transported by conveyor belt or pipeline system. It is north part of these information.

16.12.4 Ancillary

The estimated mine support includes CAT D8 type tractors for dump construction and road maintenance, CAT 12K type motor graders for road maintenance and 10,000 l water trucks for dust suppression, road maintenance, among others.

16.12.5 Pit Dewatering

In case of developing the limited leachable pit phase, it has been contemplated that surface water management will be based on a set of hydraulic structures consisting of crown channels and ditches within the pit that by gravity mainly carry the water to the south outlet of the pit to a system of sand traps. All structures are concrete in the case of the main structures and as a minimum recommendation shotcrete for erosion control purposes. In the case of the sulfide pit expansion, the water outlet is not possible by gravity. The use of pumps and dewatering wells must be considered in the case of groundwater.

16.12.6 Manpower

The mine plan assume that La Cumbre will operate seven days a week, twenty-four hours per day with four crews rotating to fill the mine roster of 12 hours per shift. Owner manpower rises to 409 mine employees in Year 2. This amount includes technical staff, operators, and pit maintenance.

16.12.7 Main Consumables

Main consumables for mine operations include diesel fuel, ANFO, emulsion and tires.

16.12.8 Grade Control

The Batero's technical staff will conduct in-pit grade control. This effort requires the ability to accurately predict the contact between the different material types (mineralized material and waste) with the aim of controlling dilution. The grade control group will be responsible for:

- Sampling, and geological mapping of blast holes
- Merging assay data with blast hole coordinates
- Generating short range planning block models
- Performing mill feed and waste delineation
- Mine-to-plant reconciliations and quality control.

16.13 MINE ENGINEERING

16.13.1 Pit Geotechnical Parameters

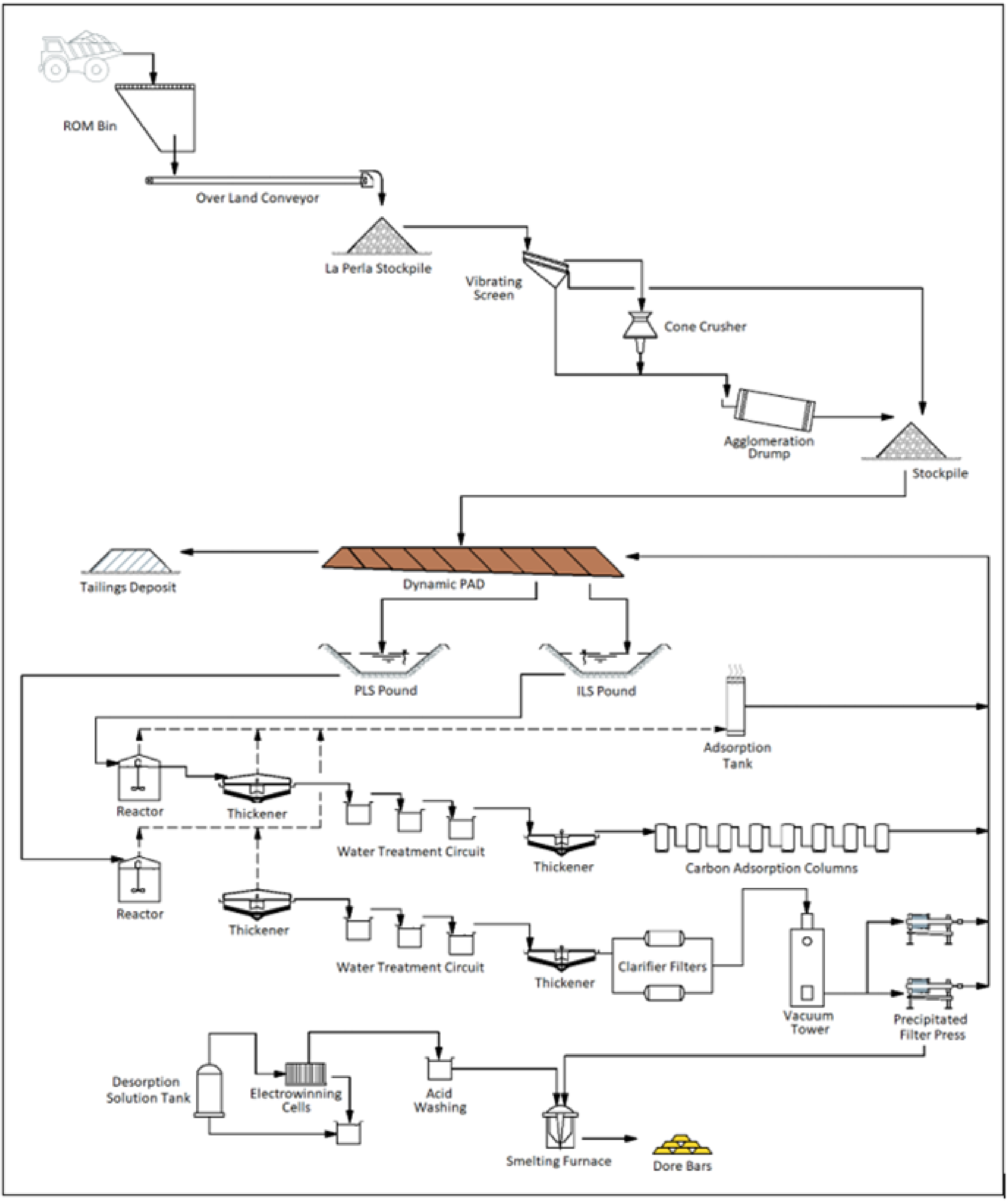
Initial geotechnical studies are available for leaching pit and. IRA of 38° are prevalent in the possible condition of stable, which is for factor of safety (FS) above 1.0 for seismic conditions. Water conditions is considered in physical stability analysis slope. No geotechnical studies for the potential sulfides pit.

17.0 RECOVERY METHODS

17.1 PROCESS DESCRIPTION AND FLOWSHEET OF THE HEAP LEACH PROCESS

Figure 17-1 provides a simplified flowsheet of the heap leach process.

Figure 17-1: Flowsheet of Heap Leach Process



Source: Minera Quinchia, 2021

The processing scheme for the treatment of Au and Ag ores is composed of the following processing areas:

1. Ore Handling at La Cumbre
2. Ore haulage and stockpiling at Matecaña.
3. Crushing at La Perla
4. Agglomeration
5. Ore haulage to the leaching heap.
6. Leaching
7. SART-AVR
8. Merrill Crowe
9. ADR Plant - Adsorption
10. Smelter
11. Contact water detoxification
12. Solution detoxification plant for La Perla zone solutions
13. Preparation and dosing of reagents

The following describes the heap leach operation in more detail.

17.1.1 Ore Handling at La Cumbre

For the mine ore extraction process, the Mine to Mill (M2M) concept will be applied in order to produce an optimal granulometry (not only finer) and to have in balance the transfer granulometry to the crushing-agglomeration circuit and thus improve the overall performance of the circuit.

Ore and wastes are deposited separately in a Grass Hopper that feeds the conveyor belt system. All crushed material will be transported by an Overland Conveyor (OVD) system, the ore will be sent to the La Perla processing plant, the waste material will be sent for storage to the Matecaña area.

The material coming from the mine (ore and waste, ROM) will be transported by conveyor belts to the crushing plant, where it will be temporarily stockpiled in stockpiles according to the destination of the material, as listed below:

- Oxides to process at La Perla
- Oxides to stockpiles at La Perla
- Transitional to process at La Perla
- Transitional to heaps at La Perla
- Sulfides to heaps at Matecaña
- Waste material at Matecaña

Only the material from primary zone in the stockpiles will be crushed separately in a semi-mobile crushing plant. The material will be transported from the stockpiles to the transfer hopper of the semi-mobile crushing plant by a 20 m^3 front-end loader. Both the semi-mobile crusher and the location of the stockpiles will not have a fixed location, as they will be moved as the excavation fronts advance. The location of the semi-mobile plant and the ore stockpiles will always remain within the excavation reach.

The ore will be transported by a front loader from the storage area and then discharged into the transfer hopper, which will have a screen in order to separate particles larger than 12", while the ore passing through the screen will be sent to a crushing circuit. Particles larger than 12" will be fragmented using mobile rock breakers. The ore from the hopper will be extracted from the bottom by a vibrating screen, from where the coarse ore will be discharged to a jaw crusher, the product of the crusher will join the fines from the screen on a conveyor belt.

The final crushed product is deposited in a Grass Hopper that feeds the conveyor belt system. All crushed material will be transported by an Overland Conveyor (OVC) system, the ore will be sent to the La Perla processing plant, the waste material will be sent for storage to the Matecaña area.

17.1.2 Ore Hauling and Stacking in Matecaña

Ore transport and stockpiling at Matecaña will be done using both conveyor belts and haul trucks. The process has a regenerative belt that initiates the transport from La Cumbre and will have a transfer tower at Matecaña, which will allow unloading the waste material and sulfides to their respective deposits. The ore destined for processing and for the oxide and transition piles will continue its transportation to La Perla in the overland.

Sulfide storage will be temporary, since the design of the haulage and stacking system at Matecaña and the regenerative conveyor will allow the sulfides to be moved from one place to another, feeding them at the loading point in the transfer tower.

17.1.3 Crushing at La Perla

Ore from La Cumbre will be transported to the area known as La Perla by means of a conveyor belt system of approximately 2,257 m in length.

At La Perla the ore destined for the process is stored in the reception yard, while the low-grade ore is stored in the oxide and transitional deposits, as appropriate. The crushing and agglomeration plant will be located in this area, where there is an ore reception yard that will store the ore until it enters the system.

From the reception yard the ore will be extracted by two plate feeders that will discharge onto a conveyor belt that will in turn transfer the ore to an 8' x 20' DD (Double Deck) vibrating screen. The vibrating screen will generate coarse (+17 mm), intermediate (-17 to +12.7 mm) and fines (-1/2"). The coarse product will be sent to the conical crushers, the intermediates will be sent to the belt feeding the Grass Hoppers and the fines to the agglomerating drum.

17.1.4 Agglomeration

The fines from the screens (-½") will be agglomerated by dosing cement, 3 to 7 kg/t ore, lime and barren solution in a 1,000 tph capacity agglomerating drum, which will discharge the glomers onto a conveyor belt, where it will join the non-agglomerated ore. This belt feeds the Grass Hopper belt system, which in turn will transfer the agglomerated material to a conveyor belt that will stack the glomers on the leach pad. Curing of the agglomerated ore will take place on the platform in the time it takes to load and assemble the irrigation system.

When the agglomerated material reaches its maximum strength after the curing stage, it is extracted from Stock Pile 2 (SP-2) by means of feeder belts, which will discharge the ore onto a main belt that will feed a continuous conveying and stacking system. On this collecting belt also discharge the belts that extract the intermediate sized stockpile material (> ½"), which allows mixing the agglomerated material with the non-agglomerated material in order to obtain a better permeability.

The collecting belt discharges onto a system of Grass-Hoper type conveyor belts that will transport the material to the Dynamic Pad. This belt system will be modified according to the location of the cell being stacked. The last Grass-Hopper belt will discharge onto a rotating stacker belt that finally places the ore on the Dynamic Pad, moving continuously according to the stacking progress.

17.1.5 Heap Leaching

Prior to the start of ore stacking, the heap leach pad will be waterproofed with the use of polyethylene geomembrane layer and equipped with corrugated and perforated pipes in main and secondary lines for the collection of Au and Ag enriched solutions. This system will be covered by overliner material, made of selected mineral that must have high permeability and meet certain grain size specifications, to prevent the geomembrane from suffering any deterioration at the time of unloading the mineral in the heap, each loading cell will have its own drainage pipe to the desanding ponds.

The heap leach pad system consists of an infeed conveyor with a complete tripper with perpendicular belt to transfer the ore to a mobile stacking system composed of 5 grasshoppers and 2 in-line stackers, where one of them is used to adjust the continuous stacking, when a grasshopper has to be removed or incorporated.

The heap leach pad will have an irrigation cycle of 17.5 days, with a NaCN gold leach solution. The concentration of the leach solution to irrigate the ore will be 1,000 ppm NaCN. The irrigation cycle of the cells is 17.5 effective days of irrigation, the first 5 days will be humectation at a rate of 5 $l/h/m^2$, to condition the permeability of the heap ore preparing it to receive a higher flow in the production stage. The production irrigation cycle takes 12.5 days, this irrigation will have a rate of 10 $l/h/m^2$. Once the irrigation cycle of a cell is completed, the leached material is moved by conveyor belts to the tailings deposit.

The cells will have a surface area of 90m x 18m (1,620 m^2), each cell will have flow lines and drop irrigation systems, starting with the distribution of the main-folds manufactured with SCH 40 iron pipes. The irrigation to the cells will be done with flat hoses that will have a self-compensating system, pressure gauges, flow meters and purge valves.

For the extraction of the precious metals (Au and Ag) from the ore, the heap leaching method will be used, which is a hydrometallurgical process of solid-liquid extraction by dissolution. This method consists of passing a diluted leach solution through the ore heap, so that the cyanide dissolves the Au and Ag particles contained in the mineralogical species, in order to obtain an enriched solution that will be stored in a Pregnant Leach Solution (PLS) pond.

The adsorption desorption and recovery (ADR) plant will have a treatment capacity of 15,000 t/day of ore, the operating variables of the process are:

- Nominal leaching flow 288 m^3/h,
- NaCN concentration, 1,000 ppm,
- Solution pH 10.5 to 11,
- Irrigation rate 10 $l/h/m^2$, and

- Leaching cycle 17.5 days

To ensure good percolation of the leach solution, the solution flow and the percolation that will occur through the ore bed due to gravity will be carefully controlled. The behavior of this descent of the leaching solution will be affected by the characteristics of the solution such as viscosity, density and the characteristics of the mineral such as void space percentage, size distribution, percentage of fines, affinity for the solution and trapped air. The moment of oversaturation of the ore due to the effect of irrigation, occurs the drainage of the heap with discharge of gold and silver loaded solutions that will be conducted to the PLS pond.

The rich solution is classified according to its Au concentration, the solution that percolates during the first 5 days of irrigation will have the highest gold concentrations and will be stored in the PLS pond. The solution percolating for the remainder of the 12.5 days contains lower gold content and will be stored in the Intermediate Leach Solution (ILS) or ILS Pond.

The NaCN-containing leach solution is applied to the surface of the heap, from where it percolates through the mineral bed, dissolving the gold minerals, to produce a gold-enriched solution or PLS, which is collected on an inclined, impermeable surface below the heap, to be transported by piping to the rich solution pond.

The rich solution that passes through the ore pile and is collected by a system of perforated pipes installed under the Dynamic Pile, these pipes are protected by 2½" to 3" diameter rock fragments called Drainage Layer, the solution flows into two main pipes that lead the PLS and ILS solutions to their respective settling ponds. These ponds are designed to precipitate by gravity the solids that were carried away by the percolation of the dynamic pile liquids, since suspended solids are detrimental to the Merrill Crowe process.

From the PLS and ILS desanding ponds, the solution is overflowed through HDPE pipes to the PLS and ILS storage ponds and pumped to the respective process. The PLS solution will be treated with the Merrill Crowe process and the ILS solution with activated carbon columns.

17.1.5.1 Pumps Used in the Process

The poor leach solutions (barren solution), left over from the ADR and Merrill Crowe processes, are conducted to the barren tank, where it is recomposed with caustic soda to control pH (10.5 to 11), NaCN strength at 1,000 ppm, and antifouling at 4 ppm, this recomposed solution will be recirculated to the heap leach pads using a vertical turbine pump.

The inlet lines to the carbon columns and the Hopper tank of the Merrill Crowe circuit are dosed according to the carbonate and sulfate content (before and after dosing). The motors of these pumps will work with variable speed drives in order to give flexibility to the operation.

To maintain the flow of the process solution in equilibrium, it will be necessary to compensate with industrial water or with barren solution from the well of major events, according to the process requirements, for this purpose it has been considered the installation of a submersible pump in operation with its respective flow line.

17.1.6 SART Process

SART (sulfurization, acidification, recycling and thickening) is a process that helps to efficiently manage the recycling of cyanide from operations while complying with environmental regulations related to the destruction of cyanide.

The rich solution, prior to entering its respective process, is treated in the SART circuit. The SART circuit is intended to recover copper from the rich solution in the form of copper sulfide and convert the weak acid dissociable cyanide (CN wad) to free cyanide for subsequent recovery and recirculation in the AVR circuit. The process consists of precipitation, thickening and filtration of copper sulfides as well as neutralization of the rich solution, precipitation and thickening of gypsum. The PLS and ILS solutions have their respective SART and AVR circuits.

The rich solution with high copper content is pumped to an agitated precipitation tank, where sodium hydrosulfide (NaHS) and sulfuric acid (H_2SO_4) are added to react with the dissolved copper cyanide complexes to form copper sulfide (Cu_2S) precipitates and dissolved hydrocyanic gas. The resulting precipitate slurry is thickened in the sulfide thickener. Most of the precipitates in the thickener discharge slurry are pumped to the filter press holding tank, resulting in filtered copper sulfide.

The overflow from the sulfide thickener, containing acidified rich solution, is pumped to the neutralization tanks, passing it through the AVR circuit. Neutralization of the acidified rich solution takes place in a series of three agitated neutralization tanks with the addition of lime. The neutralized rich solution, containing gypsum precipitates ($CaSO_4$) from the neutralization reaction, is precipitated in the gypsum thickener. The gypsum precipitates, discharged from the thickener, are recirculated to the leach pad.

All process tanks containing acidified rich solution or slurry are covered and vented to a scrubber to prevent the escape of hydrogen cyanide gas. The vented air is scrubbed with an alkaline caustic soda solution before being released to the atmosphere. Any NaCN recovered by the caustic soda solution is recirculated back to the leaching process.

17.1.7 AVR Circuit

The Acidification, Volatilization and Reneutralization (AVR) circuit is intended to recover dissolved hydrogen cyanide gas from the acidified rich solution and convert it to sodium cyanide (NaCN) for reuse in the leaching process. The AVR circuit is used to maintain a cyanide balance between the Merrill Crowe, Column Carbon and heap leach processes when there is an excessive amount of cyanide in the rich solution. This condition has its origin in the treatment of transition ores with high copper content that require a high concentration of cyanide in the leaching solution.

Dissolved hydrocyanide gas is released into the gas phase by blowing a large amount of process air into the acidified rich solution in a packed bed desorption column. The resulting gas, a mixture of air and hydrocyanic gas, is then passed through a packed bed absorption column, where an alkaline caustic soda solution converts the hydrocyanic gas to NaCN in solution. Process air is circulated in a closed loop between the desorption column and the absorption column, with a small bleed of clean process air used to maintain vacuum conditions. The AVR process takes place in closed vessels and operates under vacuum conditions to prevent the escape of hydrocyanic gas.

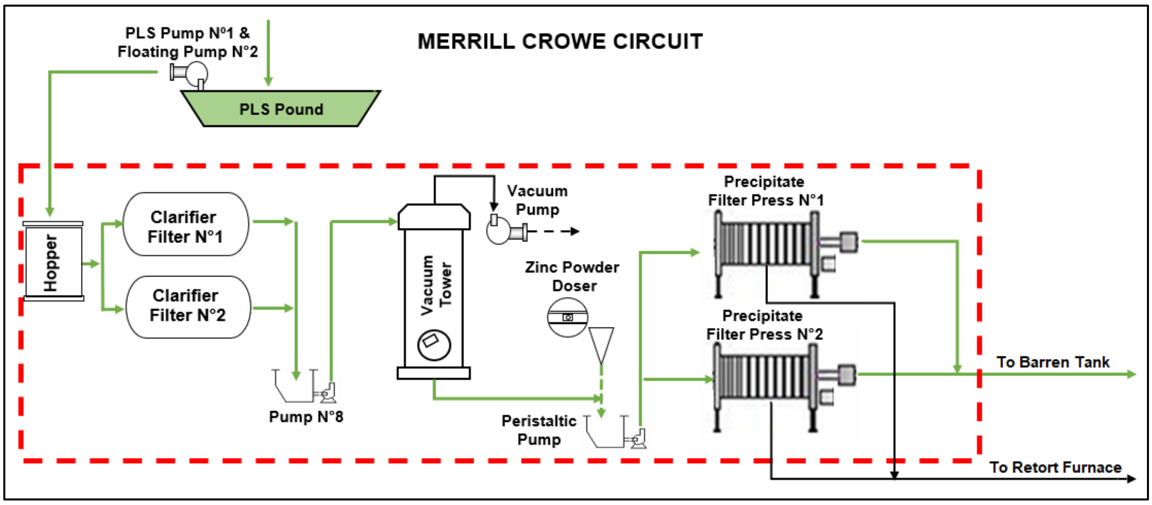
The acidified rich solution from the desorption column is pumped to the neutralization tanks of the SART process for neutralization and subsequent shipment to the Merrill Crowe or carbon-in-columns (CIC) plant as appropriate.

The NaCN solution recovered from the absorption column is recirculated to the dilution water tank for reuse in the leaching process.

17.1.8 Merrill Crowe Circuit

The Merrill-Crowe process is a cementation process involving classical redox reactions. It is typically applied to solutions generated from a solid-liquid separation step after a milling and leaching operation, or from solutions from heap leaching. It has also been used in coal mining effluents and intensive cyanidation solutions (Walton, 2005). The Figure 17-2 shows the Merrill Crowe diagram.

Figure 17-2:
Flowsheet of Merrill Crowe Circuit



Source: Minera Quinchia, 2021

The rich solution, stored in the PLS pond, is pumped to the SART process and then to the Merrill Crowe plant for the Au and Ag extraction process. The solution first enters a Hopper tank and from there passes to the clarifying filters to filter out suspended particles that have been carried over from the heap leach pad to reduce turbidity to less than one nephelometric turbidity unit (NTU). Clarification, in the Merrill Crowe process, is done using diatomite, two preparation tanks are required for the diatomite, one for the preparation for the pre-coat, called precoat mix tank, and the other one for the body feed tank, the feed flow to the Merrill Crowe circuit will be 64 m^3/h.

Once the rich solution is filtered it is passed to the deoxygenation tower, this device is the one that removes the dissolved oxygen from the solution mechanically creating a high vacuum, this process is very important because in the presence of oxygen the precipitation of Au, Ag and Cu cannot be carried out efficiently, at this stage the oxygen concentration is reduced to a value of less than 0.5 ppm. The deoxygenated solution is fed through steel pipes to the precipitation filter presses. Using a dosing device, zinc powder and lead nitrate is added to the pipe carrying the cyanide solution by means of a peristaltic pump.

The gold precipitation reaction occurs instantaneously and the gold is drawn into the filter presses and retained in the filter chambers. When the pressure of the filter presses shows that they are already saturating, the precipitates are harvested.

The filtered solution called barren solution, which contains Au and Ag concentrations of less than 0.02 ppm, is stored in a steel tank, from where it is recharged with cyanide and then pumped to the dynamic stack to irrigate the cells and repeat its Au extraction cycle.

The efficiency of the Merrill Crowe circuit is 98%.

17.1.9 Adsorption, Desorption and Electrowinning Plant

The adsorption system is the process in which the phenomenon of surface adhesion of gold in the internal pores of the activated carbon occurs when the solution loaded in values passes through the activated carbon bed.

There are three ways for adsorption of gold in solution with activated carbon:

1. Carbon in Column (CIC).
2. Carbon in Pulp (CIP)
3. Carbon in Leach (CIL)

17.1.9.1 Adsorption in Columns

The ILS solution passes through the adsorption circuits (CIC) after precipitating the copper in the SART circuit, considering a cascade configuration, so that the Au contents are adsorbed. According to the design criteria it has been considered that the carbon will be loaded until reaching values of 2.4 kg Au per ton of carbon before moving to the next process.

Two CIC banks will work in parallel. An electromagnetic flowmeter will be installed in the inlet lines of each adsorption circuit to record the flow rate, with its control and safety accessories.

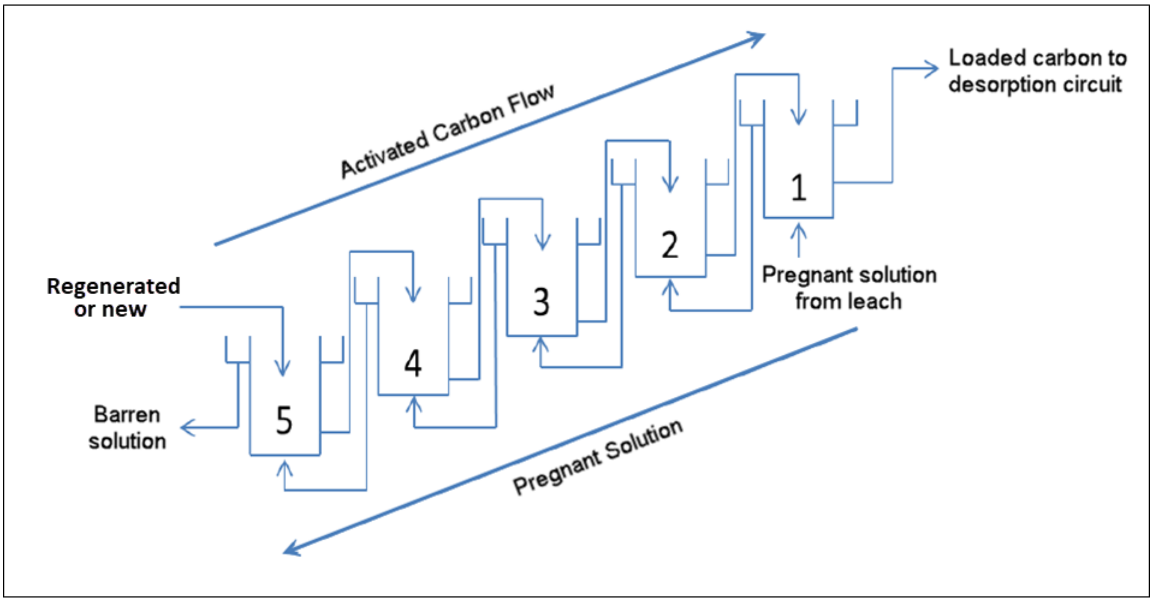
The poor solution coming out of each adsorption circuit (barren solution) will be conducted through two stationary curved screens type DSM (Dutch State Mines), for the separation of carbon particles that could be dragged from the adsorption columns, after which the solution will pass to the Barren tanks.

In the pipeline, antiscalant and 25% leach solution are added at the outlet of each tank to readjust the strength of the solution, and then pumped to the heap leach pads, thus producing a permanently closed circuit.

Once the loading of the first column with activated carbon is finished, it is unloaded and sent to desorption. Subsequently, the carbon from column No. 2 will pass to column No. 1, from column No. 3 to column No. 2, and so on in the opposite direction of the solution flow. Column No. 5 will be loaded with new carbon, see Figure 17-3.

The carbon loaded with precious metal content will pass through a sieve in order to separate the coarse carbon from the fine particles existing in the barren solution.

Figure 17-3:
Flowchart of a Carbon Adsorption Plant



17.1.9.2 Desorption and Electrowinning

The carbon stored in the ponds is transported to the desorption reactor. Carbon desorption is a batch process that receives the rich carbon from the carbon columns. The desorption process takes place in a desorption reactor, which receives the hot stripping solution, at temperatures between 130°C and 140°C. This hot solution comes from the stripping tank, which is heated by means of a heat exchanger, through which hot oil from a heater circulates.

The solution flow out of the desorption reactor is diverted to the duplex filter for fine carbon retention, while the filtered solution passes through a cooler, where the temperature is reduced to 70°C. This flow is diverted to the desorption cells. This flow is diverted to the electrolytic cells where Au is recovered in the form of electrolytic precipitates at the cathode of the cells. The efficiency of the EW is 98% recovery of the Au and Ag dissolved in the rich solution.

The electrolytic precipitate is taken to an acid washing process in reactors, the first one with sulfuric acid and the second one with nitric acid. This procedure is carried out under a fume hood with a neutralization tower, from which air is released into the environment. Finally, the product obtained from the acid washing process is considered the final product of the process, which must pass to a smelting stage.

17.1.9.3 Carbon reactivation

Carbon reactivation starts with chemical washing, which is done with 3% diluted hydrochloric acid at 90 °C temperature in a reactor, which helps to remove carbonates and sulfates from the carbon. Precipitates of carbonates and calcium sulfates trapped in the carbon pores negatively affect the load-carrying capacity of the carbon.

The second stage of carbon reactivation is thermal reactivation in a furnace, which consists of heating the activated carbon gradually and indirectly to a temperature of 700 °C with a short holding time at this temperature. Finally, the reactivated carbon passes through a size classification to discard the fines and be reused in the carbon columns.

17.1.10 Smelting

The precipitates with Au and Ag contents are distributed in trays and taken to the retort furnaces, in order to dry and extract the Hg, for which the furnace is brought to a temperature of approximately 600°C, with a retort cycle time of 24 hours under a vacuum condition of 180 mm Hg. The mercury in vapor form is collected by a system of water-cooled condensers and stored in a collector which is discharged at the end of the cycle to special containers for storage.

In order to remove any remaining gaseous mercury that may be released into the environment, the vacuum stream leaving the retort collector passes through a water cooler immediately downstream. This flow then passes through an activated carbon column and a water separator before going to the vacuum pump and being discharged to the environment. The activated carbon recovers more than 99.5% of the mercury, which ensures that mercury emissions are very low.

The dry and cold precipitate is mixed with the fluxes (sodium borate, silicon dioxide, sodium nitrate, sodium carbonate and calcium fluoride) and charged to the induction furnaces to be melted at a temperature of around 1,300°C to achieve slagging. To obtain the doré bars, the cascade casting method is used. Finally, the bars are cleaned, coded and stored in a vault until they are shipped.

The slag produced is processed to recover any valuable material it may contain. To do this, the solid slag will pass through a closed crushing circuit, which consists of a jaw crusher and a screen with a 0.8 mm mesh opening (Tyler Series 20 mesh). The fine product from the screen is sent to a gravimetric concentrator (Gravimetric Table) in order to concentrate the valuable product. The remaining slag and tailings from the gravimetric concentrator are sent to the tailings deposit, while the concentrate obtained is smelted again with the next batch.

17.1.11 Detoxification of Contact Waters

Contact water from the La Cumbre mine and Matecaña deposits will be analyzed daily to verify the presence of dissolved metals and acidity; if acidity exceeds permitted limits, it will be treated by the same process. The generators associated with acid drainage are sulfates, along with three important metal cations Fe^{+3}, Cu^{+2} and Pb^{+2}.

Drainage from the ROM ore piles at La Perla will be sent to the plant process.

The contact water will be channeled to a collecting pool, from where it will be pumped to an agitator tank. In this tank, lime will be added until it reaches a pH between 9 and 10, thus neutralizing the acidity and the formation of heavy metal hydroxides. The flow then enters an aerator tank where atmospheric oxygen is incorporated into the water by means of the agitation turbines and the iron and sulfates are oxidized. It is then passed to a second tank where a reducing agent is added to precipitate other metals such as Cu and Pb as sulfides and finally in the third tank coagulant is added.

In the overflow channel of the aeration tank, flocculant is added, after which the flow is conveyed to a clarifier where the precipitates settle out. The overflow from the thickener passes to a sedimentation pond, while the sludge from the clarifier is sent to a filter press. A small dose of sulfuric acid is added at the outlet of the filters to counteract alkalinity if necessary. The produced muds are sent to the tailings deposit for final disposal.

17.1.12 Solution detoxification plant at La Perla

La Perla will have water drainage from the tailings tank. This flow will be collected and sent to the barren tank to be incorporated into the process in order to maintain a water balance in the process plant. On the other hand, there could be an excess of water in the system, so these liquids will be treated at the La Perla solution detoxification plant.

The detoxification process (Detox) will treat the solution without valuable metal content (barren solution). This process starts by sending the barren solution to the detoxification plant, where the flow will be directed to a flow pool, which in turn will supply the barren solution to the reactor tanks. There will be three reactor tanks arranged in series, which will discharge by overflow from one to the other. Hydrogen peroxide, copper sulfate and caustic soda (NaOH) are added to the first tank at a concentration of 10%. The same reagents are added to the second reactor tank as in tank No. 1.

The overflow from this tank enters the second agitator tank where the precipitation of weak leaching complexes is carried out and finally to tank No. 3. In this tank is where the neutralization of the charges is carried out, adding a coagulant that neutralizes colloidal charges and an anionic flocculant to bind the suspended solids and sediment them. The effluent from this tank enters a clarifier where the solid-liquid separation between the sediments and the detoxified water takes place. The overflowing water passes through a carbon column system to ensure that any ion that has not been detoxified can be trapped at this stage. This water, after this treatment, complies with the maximum permissible limits for discharge into the environment at total cyanide and other concentrations in accordance with current regulations. The mud produced in this process will be sent to the tailings deposit.

The reagents used in this stage will be:

- Hydrogen Peroxide (H_2O_2) at 50%.
- Copper Sulfate Pentahydrate ($CuSO_4.5H_2O$)
- Lime (CaO)
- Coagulant
- Flocculant

17.2 PROCESS DESCRIPTION AND FLOWSHEET OF TREATMENT OF PRIMARY SULFIDES

17.2.1 Description of the Metallurgical Process and Flowsheet

The metallurgical treatment of the primary sulfides from the La Cumbre deposit consists of the following processes:

- Crushing at La Cumbre
- Ore haulage and stockpiling at Matecaña.
- Crushing at La Perla
- Milling
- Gravimetry
- Flotation
- Tank Leaching (CIL)
- SART-AVR
- Merrill Crowe

- ADR Plant - Adsorption
- Smelting
- Contact water detoxification
- Solution detoxification plant La Perla
- Preparation and dosing of reagents

In the primary sulfide treatment stage, the agglomeration, heap loading and leaching sections, which were part of the oxide and transition ore processing in the first stage, are replaced by the grinding, gravimetric, flotation and carbon tank leaching sections.

The other sections that were implemented in the first stage of the project will continue to operate under the same procedure as described above, see Figure 17-4 for the schematic flowsheet of metallurgical process for primary sulfides.

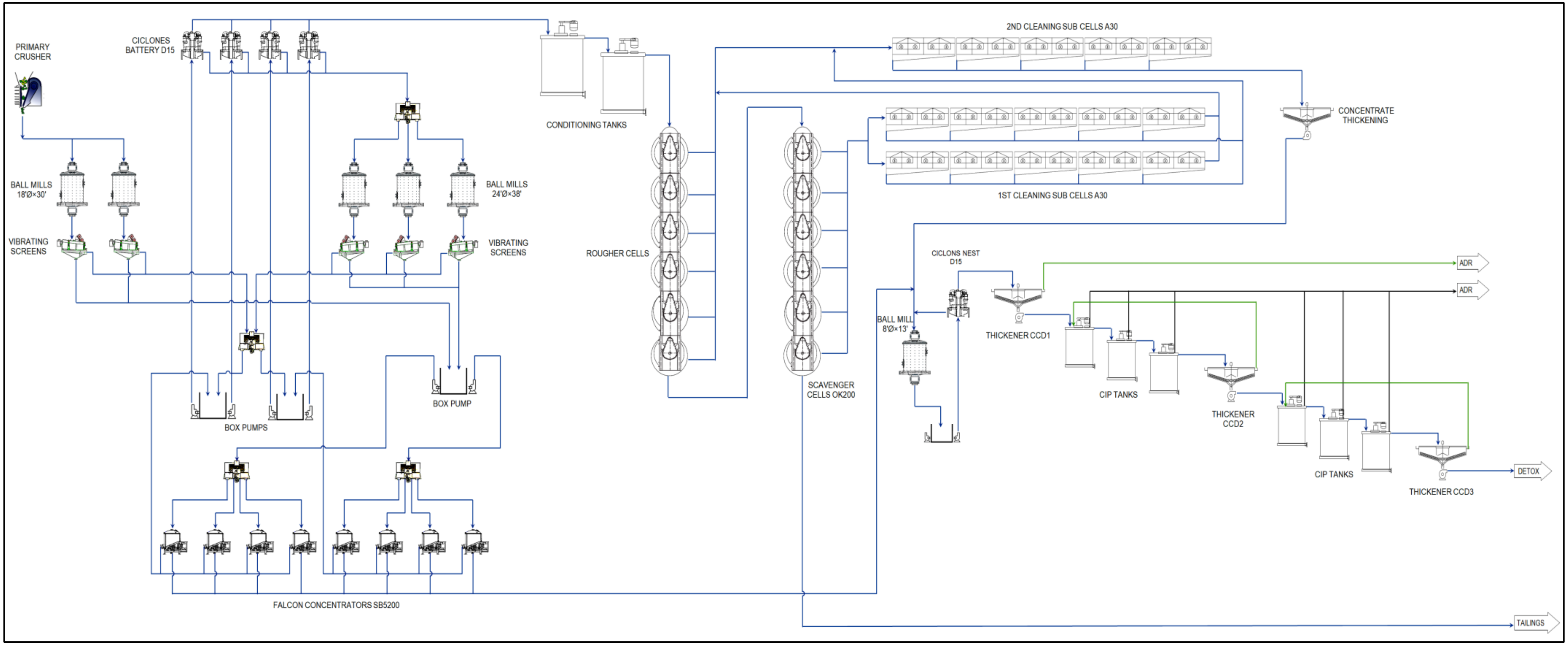
17.2.2 Milling

The crushed ore, product of the La Perla crushing plant, is stored in a fine ore hopper, from where is extracted by means of feeder belts and introduced to the two 15'Ø×25' primary mills. The discharge from each primary mill falls by gravity onto a high-frequency vibrating screen, where it is classified by size. The fraction retained on the screen enters by gravity into a pump box, which drives the ore to the cyclone nest. These cyclones classify the ore by size, the fine ore is sent to the flotation section and the coarse ore fraction is fed to the three 18'Ø×26' secondary mills. Like the primary mills, each secondary mill has a high frequency vibrating screen to discharge the ground ore. The retained fraction from each screen is diverted to the pump box of the cyclone nest. The fine fraction from each vibrating screen, both from the primary and secondary mills, enters the gravimetric section. By means of distributors, the pulp is divided to feed with equal flows to the 8 centrifugal gravimetric concentrators. The gravimetric section delivers the first concentrate, which is sent to the tank leaching section. The gravimetric tailings are sent to the pump box of the cyclone nest.

17.2.3 Gravimetry

High frequency vibrating screens, located at the discharge of each mill, are responsible for classifying the ore to the optimum size to enter the centrifugal gravity concentrators. These concentrators do not work continuously, as they must stop every so often to harvest the gravimetric concentrate. This gravimetric concentrate is stored in a bin until it can be transferred to the leaching section. Tailings from the gravity concentrators are recirculated to the mill through the pump box of the cyclone nest.

Figure 17-4:
Flowsheet of Metallurgical Process for Primary Sulfides



Source: Minera Quinchia, 2021

17.2.4 Flotation

The milled ore is gravity fed to the flotation section. The pulp enters two in-series conditioners, where the flotation reagents are added to initiate the reaction on the surface of the mineral particles. The residence time in this conditioning stage is 14 minutes.

The pulp, with added reagents, enters the first flotation stage, **rougher stage**, which consists of 6 flotation cells (tanks). The discharge or tailing from this first stage enters the second stage of flotation, **scavenger stage**, which consists of 6 flotation cells (tanks). The residence time of the pulps in these two flotation stages is 50 minutes. The discharge from the flotation scavenger is the final flotation tailing and is sent to the tailings dam.

The concentrate from the rougher stage is diverted to cleaning stage 2, which consists of 5 banks of 4 cells each. The concentrate from the scavenger stage is diverted to cleaning stage 1, which consists of 10 banks of 4 cells each.

Tailings from cleaning stage 2 recirculates to cleaning stage 1, tailings from cleaning stage 1 recirculates to the flotation head. The concentrate from cleaning stage 2 is considered the final flotation concentrate and is sent to the concentrate thickener. The thickened concentrate is sent to the tank leach section.

17.2.5 Leaching

The gravimetric and flotation concentrates are brought together and enter the leaching section. The first stage of the leaching consists of a concentrate regrinding in cyanide medium. The concentrate enters an 8'Ø×13' ball mill, where cyanide solution is applied. The mill discharge enters a pump box from where it is propelled to a nest of cyclones. The cyclones classify the ore by size, the coarse fraction recirculates to the ball mill, while the fine fraction enters thickener 1 for solid-liquid separation. The rich solution from the thickened slurry is pumped to the Merrill Crowe plant, while the thickened slurry is diverted to the activated carbon tank leaching section.

The first leaching and adsorption stage is carried out in 3 agitator tanks in series, each with a residence time of 12 hours. The discharge from this first bank of leaching tanks enters thickener 2, where a solid-liquid separation is performed. The liquid overflowing from thickener 2, with low gold content, is recirculated to the first leach tank. The thickened pulp enters the second leaching and adsorption stage; this second stage also consists of 3 agitator tanks with 12 hours of retention each. The discharge from the second bank of leaching tanks enters thickener 3 for solid-liquid separation. The solution overflowing from the thickener is recirculated to the fourth leach tank, while the thickened slurry is diverted to the detox plant.

Carbon harvesting is done whenever the gold content in the liquid streams overflowing from the thickeners increases to 0.05 ppm. The loaded carbon is transferred to the carbon desorption and reactivation plant.

17.2.6 Pulp Tailings

The slurry tailings dam will receive the following flows:

- Flotation Tailings
- Leaching Tailings

The flotation tailings are sent directly to the tailings pond, while the leach tailings must be detoxified before being introduced to the tailings pond. The solids that enter the tailings pond in the form of slurry settle to form a beach and a mirror of clarified water. This clarified water is pumped to the process water storage tank and then recirculated to the process.

17.2.7 Metallurgical Balance

The metallurgical balance of the second stage of the project is shown in the Table 17-1.

Table 17-1:
Metallurgical Balance of Primary Sulfides Treatment

Product	Weight [TMSD]	Au [g/t]	Dist. Au [%]
Gravimetry			
Head	30000	0.99	100.0
Gravimetric Concentrate	55.5	189.5	35.6
Gravimetric Tailings	29944.5	0.64	64.4
Flotation			
Head Flotation	29944.5	0.64	64.4
Flotation Concentrate	277.2	52.68	49.4
Final Tailings	29667.3	0.15	15.1
Leaching			
Head	332.7	75.51	84.9
Tailings	332.7	3.78	4.2
ADR		Au [oz/d]	
Dissolved Gold		767.21	80.7
Recovered Gold		766.44	80.6
Concentrate Recovery			84.9
Leaching Extraction			95.0
ADR Recovery			99.9
Global Recovery			80.6

18.0 PROJECT INFRASTRUCTURE

18.1 INTRODUCTION

The La Cumbre Project infrastructure and services are designed to support the (First Stage) operation of a 15,000 t/d mine and processing leaching plant, operating on a 24 hour per day, 7-day per week basis. It is designed for the local conditions and rugged topography.

The main infrastructure proposed for the project consists of the following facilities:

- A 10 km access road between the existing property entrance and the Quinchia Municipality, originating at the paved road leading to Anselma-Quinchia;
- Gold and silver leaching pad with security, administration, and personnel facilities;
- Mine support facilities including mobile equipment maintenance;
- Camp and Accommodation;
- Explosives, magazine;
- Utility infrastructure for the site: water, sewer, fire protection and communications;
- A 2,257 m long overland conveyor belt system from the open pit to the La Perla sector
- 13.2 kV grid power supply for the pre-production phase;
- Surface water handling infrastructure to manage local streams and runoff from the facilities;

The overall site layout is shown in Figure 18-1.

18.2 ACCESS AND LOGISTICS

The project site is accessible through the road network that connects the city of Medellin with Pereira-Anserma-Quinchia, after turning south towards the Municipality of Quinchia on a main paved road. From the Municipality of Quinchia, access to the La Cumbre Project is via second or third roads in the direction of Paramillo up to the La Cumbre vereda for approximately 10 km.

Minera Quinchia believes that the roads connecting the La Cumbre Project with the main Colombian ports, such as Santa Marta, Barranquilla, Cartagena and Buenaventura, could be used for transporting machinery and equipment during the construction and mounting stage.

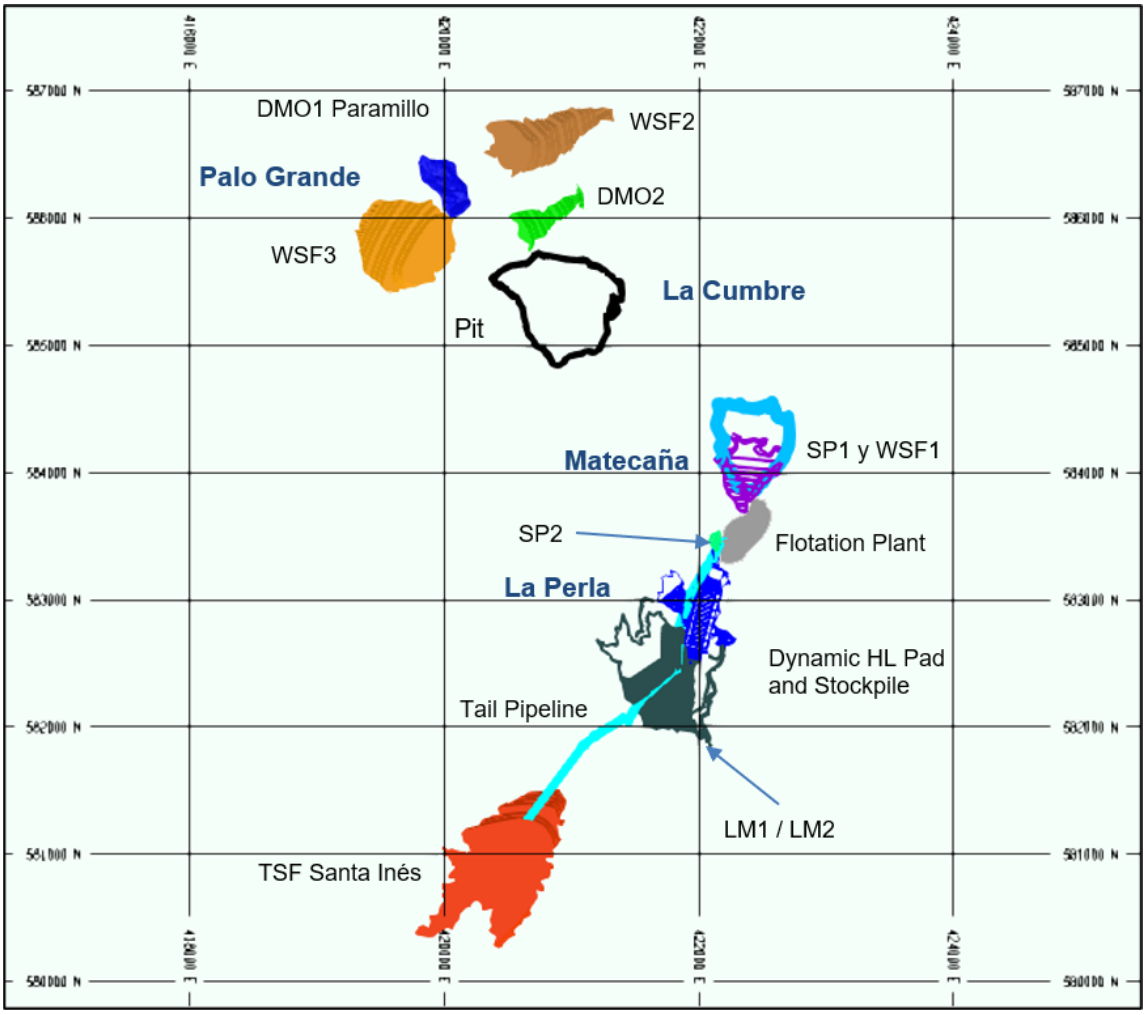
18.3 WASTE STORAGE FACILITIES

The proposed waste rock storage facilities are discussed in Chapter 16.0.

18.4 TAILING STORAGE FACILITIES

During the development of the PEA, the Santa Inés area was considered a tailings storage area for material generated by the flotation plant. Initial development focused on location for a TSF with a storage capacity of 82.5 Mt. Minera Quinchia requires studies to optimize the usefulness of this future tailings' impoundment.

Figure 18-1:
Main Components Plan View at the La Cumbre Project



(Source: Minera Quinchia, 2021)

18.5 WATER MANAGEMENT

Amec Foster Wheeler (November 2017) conducted an analysis of the hydrology of the La Cumbre Project, with the objective of establishing effects on the main constituents including Leaching Pad, Waste Dump No. 1, Waste Dump No. 2, Waste Dump No. 3 and Waste Dump No. 4 due to the action of water inflow, resulting from surface runoff and managing the water that collects on the surface of these constituents.

In 2022, Minera Quinchia states that measures will be taken to:

- Avoid the entry of surface runoff to the disturbed areas, not allowing the contact of these waters with the land affected by the operation, especially the excavation areas.
- Prevent the filtration of contact water into the natural soil by impermeabilizing the waste and sulfide storage areas by using textured geomembranes on both sides, fixed to the ground by anchor trenches and placed on a non-woven geotextile sheet, as protection against the puncturing of soil particles.
- Collect the contact water in contact ponds and send it to the detoxification or neutralization plant, as the case requires.

18.5.1 Non-Contact Water Management

The design of a system of crown canals located over the highest elevation of the main components in order to capture surface runoff water from the upper basins and discharge it to the neighboring natural streams.

18.5.2 Contact Water Management

The construction of a system of contact channels adjacent to the planned components that will capture surface runoff from the leach pad, waste rock dump, and oxide and mixed-ore deposits, and then convey it to the contact water pond located at the bottom of the waste rock dump.

Also, measures to avoid possible seepage of contact water into the natural soil. The waste, oxide, sulfide and mixed-ore deposits will have an impermeable coating composed of a 2 mm thick LLDPE geomembrane and a basal drainage system (or seepage system) that will capture any seepage and divert it to the contact water pond.

18.5.3 Seepage Systems

Simultaneously, the impermeabilization of the storage area includes the installment of a seepage collection system at the base of the leached material storage area (on top of the impermeable geomembrane), that will allow the collection of possible seepage of contact water and discharge it to the contact water pond, located at the bottom of the leached material storage area.

18.5.4 Subdrainage Systems

Below the storage areas and in natural terrain, a subdrainage system will be installed to capture any groundwater (non-contact) through a network of main, secondary and tertiary subdrains to the subdrainage well located downstream of the planned contact water pond.

18.5.5 Crown Channels

The design and construction of a crown channel located above the maximum storage level of each storage reservoir to capture and discharge surface runoff water outside the reservoir area to neighboring natural catchments.

18.5.6 Contact Channel

As a complementary measure, the construction of lateral channels has been planned to capture the surface runoff of contact water and convey it to the projected contact water pond.

18.5.7 Contact Water Ponds

In order to temporarily retain the contact water collected by the seepage system and contact channels, the construction of contact water ponds has been foreseen.

18.5.8 Subdrainage Ponds

In order to temporarily retain the possible groundwater conducted by the planned subdrainage system, the construction of storage ponds located at the bottom of the contact water pond has been foreseen. These ponds will have a compacted fill dam and a spillway channel that will convey the water to the natural channel.

18.5.9 Surface Water Management in Paramillo

The design of hydraulic structures for the drainage and management of surface water inside the Paramillo landfill was performed having in consideration the different stages of the filling of the reservoir.

The construction of a system of drainage structures, piping, and road and perimeter gutters, which will complement each other at each stage. This system can be extended or maintained for each new phase and is complemented by different water discharge structures that reduce the flow velocities at the outflow of gutters and pipes.

18.6 SITE INFRASTRUCTURE

Different facilities will be implemented at the La Cumbre Project including:

18.6.1 Operations support facilities.

A central operations area near the pit area, comprising maintenance shops, warehouses, support facilities and offices for the operations workforce.

18.6.2 Community Support

Modular mine administration building.

18.6.3 Maintenance Shops and Warehousing

Maintenance shops for the fleet of equipment for the mine operation including the spare parts warehouse, welding shop, tire shop, wash bay, light equipment maintenance shop and assembly area. In addition, a special washing area for mining equipment and the small equipment maintenance shop will have a special washing area.

Similarly, the contracting of a main workshop located outside the municipality of Quinchia owned by Minera Quinchia for the maintenance of the major mining equipment.

18.6.4 Power Supply

The installation of three Caterpillar 315 355 kW portable generating plants.

18.6.5 Fuel Storage Areas

Installation of two 10,000-gallon fuel tanks to store Diesel B10, i.e., extra diesel with a 10% biodiesel blend, which will be sufficient to meet the operation's needs. This volume accounts for 30% of the monthly consumption, which will ensure a continuous supply of fuel and thus avoid supply difficulties due to public order problems or road failures.

A containment dam will be built around the storage tanks to contain all the stored fuel and withstand the lateral pressure transmitted to the tank walls. The dam will be built in reinforced concrete, depending on the type of soil and the seismic zone of the site.

18.6.6 Laboratory

At the La Perla location there is to be a chemical laboratory to control the cyanidation process in order to continuously determine the content of precious metals in the mineral samples that go to the pile, control the content of precious metals in solution, carry out environmental tests of cyanide degradation, columns to continuously test the variability of the minerals. This same laboratory will also have the necessary equipment and specialized personnel for the elaboration of physicochemical analysis to determine the content of precious metals in different phases of the recovering process.

This laboratory should basically comprise:

- Small jaw crusher
- Small ball mill
- Set of sieves and their Ro-tap
- Cyanidation columns
- Weighing scales
- Analytical balance with six decimal places
- Electric muffles
- Laboratory elements such as tweezers, quarting tongs, scoops, etc.
- Various glassware: graduated cylinders, beakers, porcelain crucibles, etc.
- Clay crucibles
- Payoneras
- A reagent warehouse for both heap cyanidation and laboratory operations.
- Other consumables.

18.6.7 Communications

All PLCs within the plant will be connected to a common Ethernet communications network, where practical. The communications network backbone will be through a repeater, which will allow communications via radiotelephone; cellular service is also available in the project area.

The main hub for all PLC communication will be installed in the plant services switching room. Junction boxes, Ethernet switches, media converters and power supplies will be provided and installed where required.

All I/O signals to the PLCs will be through standard digital and analog modules. PLC equipment installed within each area will operate autonomously, so that a PLC failure in one plant area will not affect the other areas.

18.6.8 Aqueduct

Although the mine site is supplied by local aqueducts, Minera Quinchia has decided to install a water treatment plant to meet the potable water demand of the entire mining complex. In emergency events, this plant will be able to supply drinking water to the inhabitants in the immediate surroundings of the project.

Total projected storage will be 5,000 gallons, including the reserve for the fire-fighting system.

18.6.9 Fire-Fighting System

The fire-fighting system will be designed to comply with the minimum acceptable levels of fire safety in the different camp buildings, through different protection systems, and in accordance with the criteria established in current regulations.

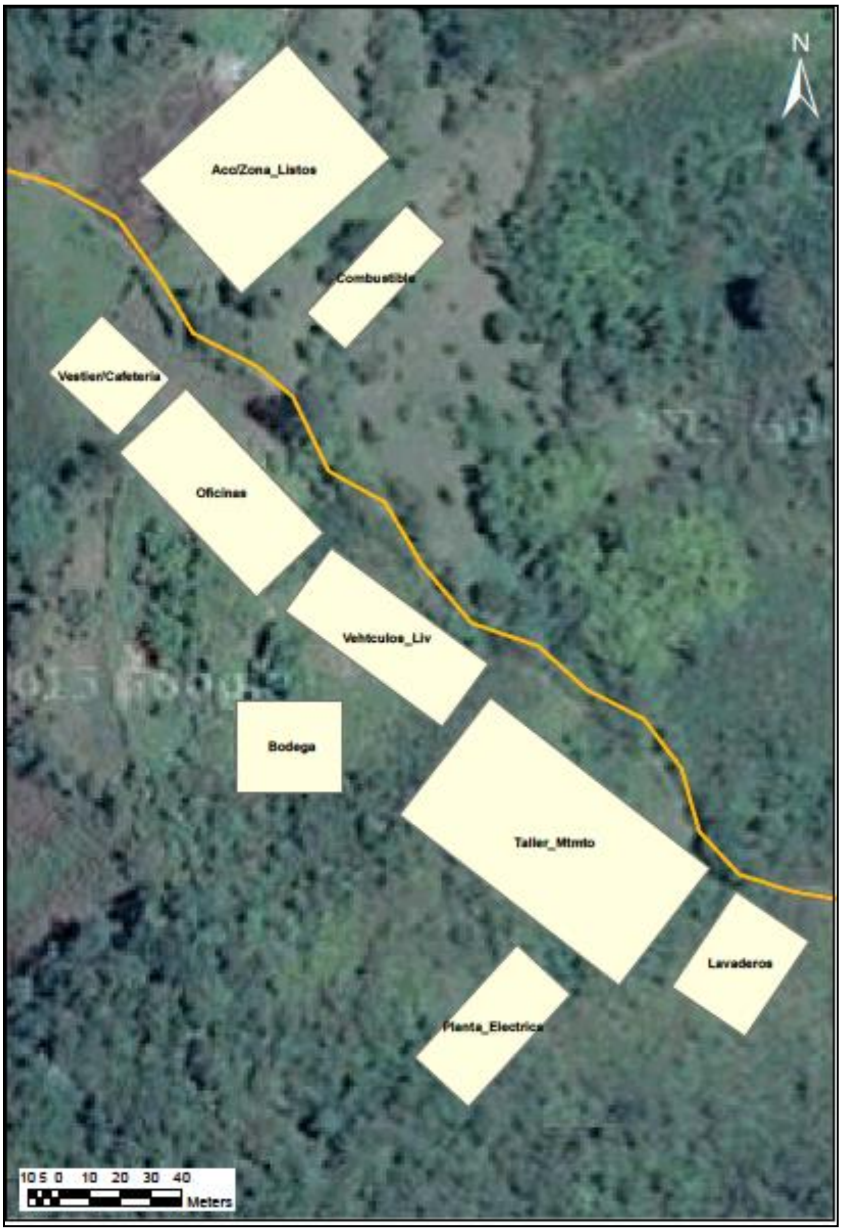
Figure 18-2 and Figure 18-3 shows the proposed main mine installations at La Cumbre Project.

18.7 CAMPS AND ACCOMMODATION

A key priority of Minera Quinchia is to recruit local labor; therefore, operations and maintenance personnel will be housed in their own homes near the project or will travel from the municipal capital in vehicles rented for this activity.

Maintenance technicians and equipment operators plus support personnel will be transported from their homes to the work area (process plant and mine portal) by bus. The aim is to reduce the traffic of private vehicles and motor vehicles inside the process facility.

Figure 18-2:
Schematic Layout of Mining Facilities - Mining Excavation Area



(Source: Coal Support SAS)

Figure 18-3:
Schematic Layout of Mining Facilities – Matecaña Area

(Source: Coal Support SAS)

18.8 POWER AND ELECTRICAL

Minera Quinchia expects that the electrical energy demand at the La Cumbre Project in Stage 1 (exploitation-processing of oxide and transition minerals) will be low compared to similar mining operations, and will be basically met from a main substation connected to the current 13.2 kVA grid, it should be noted that government permits will be required for the corresponding use.

In addition, three Caterpillar 315 portable generating plants of 355 kW each will be installed, which will be used in the operation in the event of any failure in the supply of the national interconnected network.

The estimated energy demand for the mining complex will reach 3 MW. This requirement was calculated taking into account the medium and low voltage electrical installations, lighting and power, grounding and shielding systems against atmospheric discharges, and the supply of electrical energy to the project facilities. The estimate also calculated the maximum demand required by the maintenance shops, the mill, the water and sewage services, the camps and the offices.

It is important to note that the overland conveyor belt system has the capacity to generate energy during the descent of the material, thus reducing energy consumption costs, and is environmentally friendly.

LINAMEC recommends that CMQ update this information in future works incorporating energy consumption for the flotation plant.

18.9 SOLID WASTE MANAGEMENT

This Section is based on the information provided by Minera Quinchia in the Environmental Impact Study (EIS of the La Cumbre Project), where it states that an ordinary, hazardous and special solid waste management system will be managed to prevent and mitigate any potential contamination of the natural resources in the area of influence of the project. The solid wastes that are expected to be generated during the development of the La Cumbre Project in its different stages are shown in Table 18-1.

This system will be implemented based on suitable collection, classification, provisional storage, transportation (internal and external), and final disposal. Waste generation will be differentiated for each stage of the project based on the methodology proposed by the basic environmental Water and Sanitation Technical Regulations (RAS) (RAS - Resolutions 330 of 2017 and 799 of 2021) - Title F or any similar method.

Table 18-1:
Solid Waste Generated in the Project for the Different Stages.

Description	Waste composition distribution	Construction and assembly stage waste production	Waste production during operation stage	Production of waste at the closure stage
Number of inhabitants (inhab)		1000	350	71
PPC (kg/inhab/day)		0.5	0.5	0.5
Total waste (kg/inhab/day)		500	175	35.5
Organic waste (kg/day)	29.5%	147.5	103.25	20.95
Ordinary waste (kg/inhab/day)	30.0%	150.0	105.00	21.30
Recyclable waste (kg/day)	40.0%	200.0	140.00	28.40
Hazardous waste (kg/day)	0.5%	2.5	1.75	0.36
Density of solid waste (kg/m³)		500	500	500
Total waste (m³/day)		1.0	0.7	0.142
Organic waste (m³/day)	29.5%	0.295	0.2065	0.0419
Ordinary waste (m³/day)	30.0%	0.300	0.210	0.0426
Recyclable waste (m³/day)	40.0%	0.400	0.280	0.0567
Hazardous waste (m³/day)	0.5%	0.005	0.0035	0.0007

The estimated total volume of solid waste at the La Cumbre Project will be:

- Construction stage is an estimated 2,555 m^3 (2 years)

- Production stage is an estimated 3,832.5 m^3 (15 years)
- Closure stage is an estimated 10,000 m^3 (5 years)

Minera Quinchia plans the installation of waste disposal sites (ecological points - Resolution 2148 of 2019) based on those areas with the most waste generation such as dining rooms, kitchen, workshops, warehouses, bathrooms, common or recreational areas, industrial process areas, guardhouses, among other targeted areas.

Each type of waste (non-hazardous and hazardous) must be properly characterized, classified, and labeled (hazardous waste, according to NTC 1692) and then disposed of in accordance with Minera Quinchia's environmental protocols.

Non-hazardous solid waste classified according to current environmental regulations will be deposited in appropriately labeled containers of sufficient capacity, specially prepared at the points of generation, for subsequent marketing and use.

Ordinary waste will be disposed of in sanitary landfills that have the proper legal authorizations and have the capacity and viability to dispose of the required volumes. At present, Minera Quinchia has not established a proposal for this type of landfill.

Most of the hazardous waste will consist of lubricants, used grease and oil, oil filters, drums, air filters, containers and rags contaminated with hydrocarbons, solvent and paint containers, chemical packaging, batteries, batteries, soil contaminated with hydrocarbons, etc., mainly from equipment maintenance activities. To a lesser extent, other hazardous waste such as printer cartridges and toner and cleaning product containers will be generated at the camp and offices. These wastes will be stored in places that guarantee minimum health and environmental risks, taking into account their distance from water sources and the possibility of flooding. Furthermore, a minimum possible period will be avoided to ensure that the waste does not decompose or the presence of vectors.

These wastes must be periodically delivered to companies duly established for this purpose, and must have the appropriate environmental license in force for the type of waste to be disposed of in authorized locations beyond the area of influence of the project.

In the case of special handling waste, it will be disposed of in containers or sent for temporary disposal (construction demolition material (debris, wood, tires, etc., etc.), depending on the volume.

Minera Quinchia plans to build a temporary waste collection and storage center, which must be located in a site within the project equidistant from the generation areas; it must be covered to avoid contact or exposure to rainwater and must have a capacity to store waste generation for one week for non-hazardous waste. It is important that the facility have adequate identification of each type of waste stored.

The waste management companies must issue a certificate of proper disposal of the waste. This certificate must contain the name of the generator, the date of collection, the amount disposed of by type of waste, the treatment given to each type of waste, if applicable, and the type of final disposal.

18.10 COMMENTS ON SECTION 18

Infrastructure requirements have been assessed at the PEA level to support open pit mining activities. Next stages in the La Cumbre Project requires not only the updating of technical information of the different components to be built but also those governmental permit requirements.

19.0 MARKET STUDIES AND CONTRACTS

19.1 MARKET STUDIES

No market studies have been completed. The doré that will be produced by the mine is readily marketable.

19.2 COMMODITY PRICE PROJECTIONS

The gold and silver prices provided for Mineral Resource estimation are US$1,750/oz and US$22/oz, respectively. The financial evaluation in the 2022 PEA uses a US$1,750/oz gold price and US$22/oz silver price.

Gold prices for the full-year 2021 fell behind other major asset classes in 2021 retracting 4.0% and an additional 1.4% through January 2022. The reduction in gold prices occurred despite the multi-decade high inflation in the majority of the developed world including the United States where the consumer price index reached 7.0% in December, a level not seen since 1990. The gold price oscillated above the US$1,750 line for 2021 and peaked on February 3rd at US$1,804/oz.

For 2022 and onwards the main themes affecting gold prices are related to investor fear over inflation, the conflict in Ukraine, possibility of new COVID variants, recovery of the jewelry sector but offset by interest rate hikes and higher bond yields.

Market consensus composed by 30 banks expect the gold price to remain between US$1,600 and US$1,962 in 2022 while the long-term price estimates range largely between $1,360/oz and $2,030/oz, see historical gold prices in Figure 19-1 below.

**Figure 19-1:
Historical Gold Prices**

19.3 CONTRACTS

There are currently no material contracts in place for the development and construction of the project or the operation of the mine.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 GENERAL LEGAL GUIDELINES

The Political Constitution of Colombia as the normative nucleus of environmental legislation along with the five main norms in which most of the regulations on environmental matters are developed including:

- Law 23 of 1973, by which extraordinary authority was granted to the President of the Republic to issue the Code of Natural Resources and Environmental Protection and other provisions were dictated. It has 29 of 31 articles related to regulations to prevent, control environmental damage and to look for the improvement, conservation and restoration of renewable natural resources, in compliance with the health and welfare of all the inhabitants of the national territory. The following are considered contaminable resources: air, water and soil. Moreover, necessary measures to coordinate the actions of the governmental entities that directly or indirectly advance programs for the protection of natural resources; by establishing minimum allowable levels of contamination and use for each one of the resources that integrate the environment.
- Decree Law 2811 of 1974, which regulates Part III of Book II of Decree Law 2811 of 1974: "On non-maritime waters and partially Law 23 of 1973. The purpose of this Act is to regulate the rules related to the water resource in all its states. It sets out that water may only be used by virtue of a concession, where the use of water is restricted to the availability of the resource and to the needs imposed by the purpose for which it is allocated.
- Law 9 of 1979: "Whereby water preservation is provided for and other regulations are issued".
- Law 99 of 1993: "Whereby the Ministry of the Environment is created, the Public Division in responsibility of the management and conservation of the environment and renewable natural resources is reorganized, the National Environmental System, SINA, is organized, and other provisions are enacted". The declaration in Article One, General Environmental Principles, where it states that the Colombian Environmental Policy will follow the following general principles: "The Colombian economic and social development process shall be oriented according to the universal and sustainable development principles contained in the Rio de Janeiro Declaration of June 1992 on Environmental and Development."
- Law 1333 of 2009: “Subject matter of the preventive measures. The purpose of deterrent measures is to prevent or avoid the occurrence of an event, the performance of an activity or the existence of a circumstance that threatens the environment, natural resources, landscape or human health". In Fifth Article Five, Infractions, any action or omission that constitutes a violation of the norms contained in the Renewable Natural Resources Code, Decree-Law 2811 of 1974, in Law 99 of 1993, in Law 165 of 1994 and in the other environmental provisions in force that replace or modify them, and in the administrative acts issued by the competent environmental authority, is considered an environmental infraction. The commission of damage to the environment will also constitute an environmental infraction, with the same conditions established in the Civil Code and complementary legislation for the configuration of extra-contractual civil liability, namely: The damage, the generating fact with guilty or dishonest intent and the causal relation between the both. When these elements are configured, they will give rise to an environmental administrative sanction, without prejudice of the liability that the fact may cause for third parties in civil laws.

- Law 685 of 2011 Mining Code, whose objective is the public interest to promote the technical exploration and exploitation of state and privately owned mining resources; to stimulate these activities in order to meet the demands of internal and external market and to ensure that their use is undertaken in accordance with the principles and regulations for the sustainable exploitation of non-renewable natural resources and the environment, within an overall concept of sustainable development and the economic and social growth of the country.

20.2 ENVIRONMENTAL STUDIES AND BACKGROUND INFORMATION

Minera Quinchia holds the La Cumbre Project, located in the municipality of Quinchia in the Department of Risaralda.

Based on the information provided by Minera Quinchia, a summary of the work undertaken in the La Cumbre Project in compliance with legal and environmental regulations of the Colombian Government, including:

2010-2019

- Mining Licenses
- Prospecting-exploration
- Subsoil reconnaissance
- Preliminary Economic Evaluation

2017-2021

- Program of Tasks and Works (PTO)
- Preparation of the Environmental Impact Assessment (EIA)

2022

- Protocolization of Preliminary Consultation
- Filing of the EIA with the National Environmental Licensing Authority (ANLA).

As role of the activities required for the approval of the EIA, Minera Quinchia stated that it has initiated a series of monitoring procedures required by the authority, as summarized in Table 20-1.

Table 20-1:
EIA Environmental Monitoring Matrix

Components	Tasks	Environmental Permits	Monitoring Date
Biotic	Flora	Collection of Wild Species Specimens	Started: 2017
	Fauna	Collection of Wild Species Specimens	Started: 2017
Abiotic	Water Quality	Surface Water Concessions Sewage Permits Channel Occupation Permits	Started: 2017
	Atmosphere	Air and Noise Atmospheric Emission Permits	Started: 2017
	Hydrogeology	Groundwater Concessions (No permit required)	Started 2017 Ended: 2018
	Geotechnics and Geomorphology	Construction materials (Quarries)	Started 2019 Ended: 2020

20.3 MAIN PROJECT AREA ENVIRONMENTAL CHARACTERISTICS

20.3.1 Climate and Meteorology

Annual precipitation in the La Cumbre Project area ranges from 900 mm/year to 3,000 mm/year. There are two rainy periods during the year, between March to May and September to December. Annual average daily temperatures range from 18°C to 24°C. The average relative humidity is 80%.

The data used were gathered at three IDEAM stations near the area of influence of the project, identified as Guerrerito, Bellavista and San Clemente. Table 20-2 provides the general characteristics of the stations.

Table 20-2:
General Characteristics of the Stations Used

Code	Station Name	Category	Municipality	Elevation (masl)	Coordinates UTM (WGS 84 18N	
					Easting	Northing
26170260	Guerrerito	PM	Quinchia	797	426,315	583,771
26145020	Bellavista	CO	Anserma	2,017	411,336	583,223
26140110	San Clemente	PM	Guática	2,173	412,723	587,855

(Source: Servicios Ambientales y Geográficos S.A.)

20.3.2 Hydrology

The hydrological network in the area of influence of the La Cumbre Project consists mainly of the Guanquía and Guerrero stream basins. The former is part of the Quinchia River sub-basin and the latter is part of the Opiramá River sub-basin. Both rivers flow directly into the Cauca River on its western bank.

The Guanquía stream originates at an elevation of approximately 2,075 masl. Its catchment area is 10.45 km^2 and it flows into the Quinchia River on its left bank, at approximately 890 meters above sea level.

The Guerrero Creek starts at an elevation of approximately 2,000 masl. Its catchment area is 9.86 km^2 and it flows into the Opiramá River on its left bank, at approximately 820 masl.

Based on the classification of Howard (1967) and Eagleson (1970), the Guanquía and Guerrero basins may be classified as lotic systems with a dendritic drainage pattern, similar to small strands or threads, characterized by small, short and irregular streams that flow in all directions, cover wide areas and join the main river at any angle.

The estimation of the average flow of each basin at the defined control point using precipitation and real evapotranspiration maps, resulting in reports for each delimited basin with the tool "Zonal Statistics" available in the ArcGIS 10.2 Software (CoalSupport S.A.S., 2022), resulting in an average flow of 379 l/s and 313 l/s for the Guanquía and Guerrero streams, respectively. Table 20-3 shows the Hydric Balance for the Guanquía and Guerrero streams.

Table 20-3:
General Characteristics of Used Stations

Code	Station Name	Guanquia Creek	Guerrero Creek
Precipitation, mm/year		2,194.20	2,085.40

Evaporation, mm/year	Cenicafé[1]	1,049.80	1,065.90
	Turc	1,048.40	1,101.50
Runoff, mm/year	Cenicafé	1,144.40	10,195
	Turc	1,145.80	983.9
Area, km^2		10.45	9.86
Multi-year average flow rates, (l/s)	Cenicafé	379.2	318.8
	Turc	379.7	307.6
	Average	379.5	313.2

Source: Servicios Ambientales y Geográficos S.A.
[1] National Coffee Research Center

20.3.3 Water Quality

As provided by Minera Quinchia (2022), there are four monitoring points distributed in the Cumbres, Piedras, Guerrero and Mandeval streams, respectively. The values obtained for parameters allow classifying the water of the streams as Good Quality, being the monitoring point PM1 (Cumbres stream) the one with the most favorable sanitary quality conditions, while the monitoring point PM3 (Guerrero stream) with less optimal indices but still of Good Quality.

Table 20-4 and Table 20-5: Results of the Calculation of the ICA Water Quality IndexTable 20-5 present the values obtained for the rainy period (October 2017) and dry period (July 2018).

Table 20-4:
Results of the Calculation of the ICA Water Quality Index

Parameter	Weighting factor	Q_i value			
		PM1 Cumbres Creek	PM2 Piedras Creek	PM3 Guerrero Creek	PM4-1 Mandeval1 Creek
Dissolved oxygen (% Saturation)	0.18	75	87	91	86
Fecal coliforms (NMP/100 ml)	0.17	63	39	37	29
pH (Und)	0.12	51	77	92	87
DBO5 (mg/l)	0.12	70	70	70	70
Nitrates (mg/l)	0.11	67	95	96	49
Phosphates (mg/l)	0.11	91	91	91	91
Temperature (°C)	0.11	93	93	93	93
Total Suspended Solids (mg/l)	0.08	82	85	82	84
ICA $_{WQI\ NSF}$	1.00	73	77	79	72
Classification		Good	Good	Good	Good

(Source: Servicios Ambientales y Geográficos S. A.)

Table 20-5:
Results of the Calculation of the ICA Water Quality Index

Parameter	Weighting factor	Q_i value			
		PM1 Cumbres Creek	PM2 Piedras Creek	PM3 Guerrero Creek	PM4-1 Mandeval1 Creek
Dissolved oxygen (% Saturation)	0.18	98	87	99	97
Fecal coliforms (NMP/100 ml)	0.17	99	37	33	22
pH (Und)	0.12	62	81	63	90
DBO5 (mg/l)	0.12	56	56	56	56
Nitrates (mg/l)	0.11	96	96	96	96
Phosphates (mg/l)	0.11	77	91	91	91
Temperature (°C)	0.11	93	93	93	93
Total Suspended Solids (mg/l)	0.08	86	83	83	82
ICA $_{WQI\ NSF}$	1.00	85	76	75	76
Classification		**Good**	**Good**	**Good**	**Good**

(Source: Servicios Ambientales y Geográficos S. A.)

20.3.4 Hydrogeology

Minera Quinchia has postulated a conceptual hydrogeological model, based on a series of hypotheses and assumptions that should reduce the groundwater flow problem and the real domain of the groundwater environment to a simplified version of reality, including aspects including geology, geomorphology, hydro geochemistry, hydrogeology, hydrology and climate.

The outcome of this analysis enables to propose four hydrogeological units (HGU), which are integrated as indicated below:

HGU-I - Basalt, monzonite and porphyritic andesite.
HGU-II - Diorite and contact breccias
HGU-III - Volcanic tuffs
HGU-IV - Residual soils and slope deposits.

These units were defined according to their main lithological, structural, geomorphological and hydraulic characteristics, supporting their definition on the survey of groundwater points and hydro geochemical sampling (Figure 20-1).

Figure 20-1:
Hydrogeological Units at La Cumbre Project

(Source: Servicios Ambientales y Geográficos S. A.)

For the calculation of permeability of the hydrogeological units located in the study area, it is necessary to use secondary information to determine the most approximate values and intervals of this parameter for each unit, since in-situ permeability tests are not available in the area.

Figure 20-1 does not show HGU-IV, which is interpreted as subjacent to the other hydrogeological units.

20.3.4.1 HGU-I: Basalt (Kvb), Monzonite (Kmi) and Porphyrytic Andesite (Tadi)

This unit occupies about 78% of the surface in the study area and from the hydrogeological point of view they can be clustered, since their intrinsic permeabilities are similar. The HGU-I has seven springs found, with average conductivity is 287 µS/cm and dissolved oxygen is 6.11 mg/l, suggesting fast flows that do not have a high interaction time with the subsurface medium. In addition, the upwelling media where the springs outcrop are typically residual soils, soil-rock contact (saprolite) and organic matter (IA).

This unit has only RQD tests in basalts (Kvb), where an alternation between high (> 75%), intermediate (50% - 75%) and low (< 50%) values are obtained, with intermediate values predominating. The percentages lower than 50% are associated with fault zones as a result of the strain present in the study zone, whose thickness ranges between 2.3 m to 2.5 m at the depth of 49.2 m of drill hole DDH-ZO-42 and 94 m of drill hole DDH-ZO-45A.

This unit is considered as an aquitard, except that the first 49 m have a higher secondary permeability (1 x 10-3 to 1 m/d), due to the degree of fracturing evidenced in the RQD, while after 49 m to 120 m this unit is less fractured.

This type of unit has the capacity to store water, but the flow in the medium is very slow, compared to aquifers; although it is important to clarify that the state of the rocks is unknown from 120 m depth, and it is possible that at these depths there is a decrease in the degree of fracturing of the material and therefore has a lower permeability.

20.3.4.2 HGU-II: Diorite (Mdio) & Contact Breccia (Mbx)

This unit is formed by intrusives of dioritic composition (fine and medium grain) and the contact breccias associated with the mineralized body that are: hydrothermal, fault zone, magmatic and polymictic, being rocks that intrude the other units in the area, generate shear zones at the contacts that are important for the conduction of groundwater flows.

From the surface to 100 m depth, it is fractured and weathered, with RQD less than 50% with fracturing associated with the emplacement of this intrusive body in the pre-existing rocks, and those zones of hydrothermal alteration, where a potassic core with stockwork of quartz veinlets and magnetite veinlets is found, with an over imposition of clays in an assemblage of phyllic stockwork and intermediate argillic alteration, which is called telescopic.

The most significant feature of this hydrogeological unit is the presence of a secondary faulting system formed by faults with preferential N-NE and N-NW direction, not knowing its hydraulic characteristics. At depths less than 100 m, zones with low RQD can be observed, associated with fault zones with thicknesses of 0.2 to 11.5 m, but at depths greater than 100 m there is an interchanging RQD index between high (> 75%), intermediate (50% - 75%) and low (< 50%) values, with intermediate values predominating.

On surface, the secondary permeability of this unit is important, with ranges between 1 x 10-5 m/d, up to 1 m/d, making possible the existence of local groundwater flows in the fractured environment, coming from the HGU-IV together with the volcanic ash, towards these zones with greater fracturing having interconnection of fractures. Given the properties of the hydrothermal alteration that occurs in the outer parts of these bodies, it could be assumed that up to 550 m this unit performs as an aquitard, capable of conducting water slowly towards the low gradient zones, although no outcrops of water points were recorded over this unit.

Local and intermediate bedrock flows are predominantly to the NE, but flows may occur to the NW and across subvertical faults.

On the other side, it is possible that the HGU-II after 550 m depth could be considered as an aquifuge, that is, a unit that does not store or transmit water because the degree of fracturing decreases, as observed in the drill holes. Therefore, the secondary permeability would also decrease (1 x 10-9 to 1 x 10-5 m/d), so it is required to confirm this hypothesis, the continuity and hydraulic characteristics of these structures.

20.3.4.3 HGU-III: Volcanic Tuffs (Tmc)

Conformed by volcanic tuffs and agglomerates of the Combia Formation, with low to moderate primary permeability, with surface weathering and highly fractured with fault zones between 0.6 to 2.1 m thick (geotechnical soundings).

The average thickness of this unit is 378 m, considering that for the first 42 m of this hydrogeological unit there is saprolite material, with a RQD less than 50% considered very important for groundwater conduction and storage, classifying this section towards HGU-IV.

Of the springs identified in the entire study area only three outcrops on rocks of this unit with electrical conductivities between 39 and 89.7 µS/cm with dissolved oxygen values averaging 6.5 mg/l, possibly with the presence of fast flows that do not come into contact with the medium, which creates the hypothesis that this material at depth has a considerable degree of lithification.

From 42 m depth, this unit is considered as an aquitard (described as HGU-IV), since at surface it has a higher secondary permeability (8.6 x 10-2 to 8.6 x 10-1 m/d).

This HGU has the capacity to retain water, but the flow in the medium is very slow, compared to aquifers. These formations can be of great importance when studying regional water motion, since over relatively large areas they can transmit large amounts of water despite the low flow velocity.

20.3.4.4 HGU-IV: Residual Soils

It includes the residual soil unit (IIA-IC) of basalts (Kvb), diorites (Mdio) and volcanic tuffs (Tmc). The granulometry of the IIA-IC horizon ranges from silty to silty-clayey, which have a high potential to infiltrate and transmit water to the deeper zones.

The thicknesses of this hydrogeological unit, according to the drillings, are from 16.9 to 34.6 m for the volcanic tuffs (Tmc), from 27 to 70 m for the Diorites (Mdio) and from 34.6 m for the Basalts (Kvb).

A total of 53 springs emerges in this unit, which have neutral pH, and mean dissolved oxygen of 5.6 mg/l and electrical conductivities ranging from 90.9 to 300 µS/cm with an average of 244.6 µS/cm, with the presence of 9 springs with electrical conductivities > 400 µS/cm considered as anomalous values. In this HGU-IV, it is expected that the water has low to intermediate residence times in the host medium, and that when it infiltrates it moves through sub-surface flows, and then quickly rises to the surface. The flow rates identified in the springs of this unit are low and are between 0.01 and 0.43 l/s and the primary permeability for silty-clay and silty materials is between 1 x 10-4 m/d and 2 m/d, between low and moderate.

This unit is considered within the conceptual hydrogeological model as a zone of moderate recharge and groundwater transit, in which sub-surface and local flows occur in the most superficial level of the zone. On the other hand, in the study area there are two transit units that, due to their limited width and extension, were not classified as hydrogeological units, but they play an important role in the hydrogeology, being this the case of volcanic ashes and hillside deposits.

Volcanic ashes are segmented over the upper parts of the Piedras and Guanquía basins, characterized by a silty-clay granulometry, low density, brown color, and variable humidity, sometimes mixed with organic matter. Due to its primary permeability, it is a propitious zone for water infiltration and transport to the underlying hydrogeological units. The thickness of this unit ranges from 1 to 7.6 m and according to geoelectric tests it is usually wet unsaturated.

The hillside deposits are found punctually in several sites in the Piedras and Guanquía basins, the largest being those identified in La Perla and Matecaña, with a fine granulometry, mature and humid, whose width is between 5.9 to 19.8 meters. With a primary permeability, these areas may be suited for infiltration and transport of sub-surface water flow, as it is a transit zone with 29 registered springs, where the conductivity is between 89.9 and 520 µS/cm and the average dissolved oxygen is 6.2 mg/l.

Table 20-6 summarizes the characteristics of the hydrogeological units in the La Cumbre Project.

Table 20-6:
Summary of the Hydrogeological Units (HGU) in the Area

HGU	Rock Type & Unit	Springs	Permeability: m/d	Average Thick. m	Classification	Observation
I	Basalt, monzonite & porphyry andesite	7	1x10-3 to 1x10-5	120	Acuitard	Fractured in surface (49 m)
II	Diorite & contact breccia	0	1x10-5 to 1	550	Acuitard	Very fractured in surface (100 m)
III	Volcanic tuff	3	8.6x10-5 to 8.6x10-5	378	Acuitard	Fractured in surface (42 m)
IV	Residual soil (IC-IIA)	7	1x10-3 to 1x10-5	120	Transit zone-Subsurface flows	

(Source: Servicios Ambientales y Geográficos S. A.)

20.3.5 Vegetation

The study area is classified as a Very Humid Premontane Tropical Forest zone (Soluciones Ambientales AP&A, 2010), characterized by temperatures ranging between 17° to 24°C, with annual rainfall ranging between 2,000 to 4,000 mm, with altitudes ranged from 1,000 to 2,000 masl It was also identified only secondary forests or disturbed areas (Amec Foster Wheeler, 2017).

20.3.6 Fauna

The identification of 133 species distributed in 62 families, 25 species among amphibians and reptiles, 19 species in mammals and 89 species of birds (Soluciones Ambientales AP&A, 2010). Among the most significant avifauna families found in the area are the seed-eaters, tanagers and honeycreepers and Trochilidae (hummingbirds), tanagers (Tarthus, T vassorii and Thraupis episcopus), Picidae (woodpeckers), Cracidae (guans) and the Ramphastidae family (toucans). Among the mammal species families, Phyllostomidae (bats) and Didelphidae (chuchas) were included. Among the most common amphibian and reptile families, Hylidae (frog), Strabomantidae (frog), Colubridae (false coral, black hunter) were recorded.

20.4 PERMITTING

20.4.1 Environmental License

In the PEA (RPA, 2013), it is mentioned that, exploration activities involving construction of accesses and platforms within the framework of an Environmental Management Plan approved by the Risaralda Regional Autonomous Corporation (CARDER), in that sense, Minera Quinchia has an Environmental Management Plan (Soluciones Ambientales AP&A, 2010) whose approval by CARDER could not be confirmed (Amec Foster Wheeler, 2017).

For mining projects, the Colombian Mining Legislation (Decree 1076, 2015) establishes that, the approval of an Environmental Impact Assessment (EIS) by the National Environmental Licensing Authority (ANLA) is required, where a minimum content must be included based on the Terms of Reference for the Preparation of the Environmental Impact Assessment of Mining Exploitation Projects - TdR-13 (Resolution 2206, 2016). Minera Quinchia initiated the elaboration of the Environmental Impact Study in 2017, and it is expected to be completed in the second half of 2022.

Minera Quinchia summarizes chronologically the permits obtained for the La Cumbre Project, as listed below:

- Resolution No. 0730 dated 10-March 2011; Water Concession for domestic, Industrial and dumping use.
- Resolution No. 1123 dated April 18, 2011; Contingency plan for the prevention and control of spills and/or emissions.
- Resolution No. 2995 dated August 19, 2011; Resolves the request for new water concessions.
- Resolution No. 1979 dated July 19, 2012; Authorization for final disposal of excess material from excavation and stripping.
- Resolution No. 0072 dated 19-January 2017; Extension of concessions for use of surface water.
- Renewal Resolution No. 0072; File No. 9603 dated May 20, 2022; Renewal of Water and Dumping Concession.

20.4.2 Other Environmental Permits

It is worth noting that Minera Quinchia obtained on March 7, 2011, the authorization for the use of surface water granted by the Corporación Autónoma Regional de Risaralda (CARDER) for the catchment of water from the Piedras stream (mining industrial use), Reinerio stream (mining industrial use), Palogrande stream (mining industrial use) and Mandeval stream (mining industrial and human use). This authorization was for a renewable period of 5 years, with Minera Quinchia having requested the extension of this term in January 2017.

In the case of domestic water use permit, it was approved by the regional environmental authority in 2013, and was valid until 2017.

The La Cumbre Project area has pre-montane forests and high humidity, which requires a study of the quality of these ecosystems, identifying possible sensitive and/or endemic species to establish sustainability efforts prior to developing subsequent phases (Amec Foster Wheeler, 2017).

20.5 ENVIRONMENTAL MANAGEMENT

Minera Quinchia hired the specialized company Servicios Ambientales y Geográficos S.A. to draft the Environmental Management Plan for the La Cumbre Project, resulting in the document "EIA MQ_CAP10.1_Plan_Manejo_Ambiental" (March 2022) where the different processes for environmental management in the key components that will be affected in the current and future works at the La Cumbre Project are described (see Table 20-7).

Abiotic Environment: which involves Water Management procedures (domestic wastewater, non-domestic wastewater, runoff water and watercourse intervention, groundwater); Slope Stability Management, Deposits and Exploitation Areas; Atmospheric Emissions Management; Noise Management and Control, and so forth.

Biotic Environment: which covers procedures for the Management of Vegetation Cover Removal; Management for Sensitive Flora Species; Management for Epiphytic and Terrestrial Vascular Flora in National Prohibition in the Intervention Area (epiphytic and terrestrial orchids and bromeliads); Management of Terrestrial Fauna (birds, mammals, reptiles and amphibians); Management of the Hydrobiological Resource.

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Table 20-7:
Environmental Management (EMP) and Follow-up and Monitoring Programs (FMP)

SETTING	EMP	FMP	DETAILS[1]
Abiotic	PMA-AB-01 Integrated Management of Domestic Wastewater ARD	PSM-AB-01 Monitoring and Follow-up of Integral Management of Domestic Wastewater ARD PSM-AB-16 Surface Water Monitoring and Follow-Up	MQ-AB-06 Soil Quality Alteration MQ-AB-07 Alteration to the quality of groundwater resources MQ-AB-09 Hydrogeomorphological alteration of fluvial dynamics and/or sedimentological regime MQ-AB-10 Alteration to the quality of surface water resources MQ-AB-11 Alteration in the supply and availability of surface water resources MQ-BT-01 Alteration to the hydrobiota including aquatic fauna MQ-SE-06 Generation and/or alteration of social conflicts
	PMA-AB-02 Integral Management of Non-Domestic ARnD Wastewater (industrial and contact or acidic)	PSM-AB-02 Monitoring and Follow-up of Integrated Management of Non-Domestic ARnD Wastewater (industrial and contact or acidic) PSM-AB-16 Surface Water Monitoring and Follow-up PSM-AB-16 Surface Water Monitoring and Follow-up	MQ-AB-06 Soil Quality Alteration MQ-AB-07 Alteration to the quality of groundwater resources MQ-AB-09 Hydrogeomorphological alteration of fluvial dynamics and/or sedimentological regime MQ-AB-10 Alteration to the quality of surface water resources MQ-AB-11 Alteration in the supply and availability of surface water resources MQ-BT-01 Alteration to the hydrobiota including aquatic fauna MQ-SE-06 Generation and/or alteration of social conflicts
	PMA-AB-03 Runoff Water Management and Stream Intervention	PSM-AB-03 Monitoring of Surface Water Environment PSM-AB-16 Surface Water Monitoring and Follow-up	MQ-AB-03 Alteration of Geologic Conditions MQ-AB-04 Alteration of the geoform of the terrain MQ-AB-05 Alteration of Geotechnical Conditions MQ-AB-06 Alteration of soil quality MQ-AB-07 Alteration to the quality of groundwater resources MQ-AB-08 Alteration to the supply and/or availability of groundwater resources MQ-AB-09 Hydrogeomorphological alteration of fluvial dynamics and/or sedimentological regimes MQ-AB-10 Alteration in the quality of surface water resources MQ-AB-11 Alteration in the supply and availability of surface water resources MQ-BT-01 Alteration to hydrobiota including aquatic fauna

1 Standardized Category to be Managed

SETTING	EMP	FMP	DETAILS1
	PMA-AB-04 Water Supply Management	PSM-AB-04 Water Supply Management Monitoring and Follow-up PSM-AB-16 Surface Water Monitoring and Follow-up	MQ-AB-08 Alteration in the supply and/or availability of groundwater resources MQ-AB-09 Hydrogeomorphological alteration of fluvial dynamics and/or groundwater regime MQ-AB-11 Alteration in the supply and/or availability of surface water resources MQ-BT-01 Alteration to hydrobiota including aquatic fauna MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-08 Modification of physical and social infrastructure, public and social services
	PMA-AB-05 Integrated Groundwater Management: Availability and Quality	PSM-AB-05 Monitoring and Follow-up of Integrated Groundwater Management: Availability and Quality	MQ-AB-06 Alteration to Soil Quality MQ-AB-07 Alteration to the quality of groundwater resources MQ-AB-08 Alteration to the supply and/or availability of groundwater resources
	PMA-AB-06 Slope Stability Management of Slopes, Deposits and Exploitation Areas	PSM-AB-06 Monitoring and follow-up of slope stability management of deposits and mining areas	MQ-AB-03 Alteration of geologic conditions MQ-AB-04 Alteration of the geoform of the terrain MQ-AB-05 Alteration of the geotechnical conditions MQ-AB-06 Alteration of soil quality MQ-AB-07 Alteration to the quality of groundwater resources MQ-AB-09 Alteration to hydrogeomorphological alteration of fluvial dynamics and/or sedimentological regime MQ-SE-02 Alteration in the visual perception of the landscape MQ-SE-07 Modification of accessibility, mobility and local connectivity
	PMA-AB-07 Management of atmospheric emissions	PSM-AB-07 Monitoring and follow-up of the Management of atmospheric emissions	MQ-AB-01 Air Quality Disturbance MQ-SE-06 Generation and/or alteration of social conflicts
	PMA-AB-08 Noise management and control	PSM-AB-08 Noise Management and Control Monitoring and Follow-Up	MQ-AB-02 Alteration in sound pressure levels MQ-SE-06 Generation and/or disruption of social conflicts MQ-SE-07 Modification of local accessibility, mobility and connectivity
	PMA-AB-09 Integral Management of Ordinary, Hazardous, and Special Solid Waste	PSM-AB-09 Monitoring and follow-up of integrated management of ordinary, hazardous, and special solid waste	MQ-AB-06 Alteration to the quality of the soil MQ-AB-07 Alteration to the quality of groundwater resources MQ-AB-10 Alteration to the quality of groundwater resources MQ-BT-01 Alteration to hydrobiota including aquatic fauna MQ-SE-08 Alteration to physical and social infrastructure, public and social services
	PMA-AB-10 Management of Cyanide and Chemical Substances	PSM-AB-10 Cyanide and Chemicals Management Monitoring and Follow-up	MQ-AB-06 Alteration to soil quality MQ-AB-07 Alteration to groundwater quality MQ-AB-10 Alteration to surface water quality MQ-BT-01 Alteration to hydrobiota including aquatic fauna MQ-SE-07 Modification of accessibility, mobility, and local connectivity

SETTING	EMP	FMP	DETAILS1
	PMA-AB-11 Explosives and blasting management	PSM-AB-11 Monitoring and Follow-up of Blasting and Explosives Handling	MQ-AB-01 Air Quality Disturbance MQ-AB-02 Alteration to Sound Pressure Levels MQ-AB-03 Alteration of Geologic Conditions MQ-AB-05 Alteration of geotechnical conditions MQ-AB-06 Alteration to soil quality MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-07 Modification of accessibility, mobility, and local connectivity
	PMA-AB-12 Soil Management and Reclamation	PSM-AB-12 Soil Management and Reclamation Monitoring and Follow-up	MQ-AB-03 Alteration of geologic conditions MQ-AB-04 Alteration of the geoform of the land MQ-AB-05 Alteration of Geotechnical Conditions MQ-AB-06 Alteration of soil quality MQ-AB-07 Alteration to the quality of groundwater resources MQ-AB-08 Alteration to the supply and/or availability of groundwater resources MQ-AB-09 Hydrogeomorphological alteration of fluvial dynamics and/or sedimentological regime MQ-AB-10 Alteration in the quality of surface water resources MQ-AB-11 Alteration in the supply and availability of surface water resources MQ-BT-01 Alteration to hydrobiota including aquatic fauna MQ-BT-01 Alteration to hydrobiota including aquatic fauna MQ-SE-02 Alteration in the visual perception of the landscape MQ-SE-03 Change in land use MQ-SE-06 Generation and/or alteration of social conflict MQ-SE-07 Modification of accessibility, mobility, and local connectivity
	PMA-AB-13 Management of Construction Materials and Excavation Waste	PSM-AB-13 Monitoring and follow-up of construction materials management and excavation overburden	MQ-AB-03 Alteration of geologic conditions MQ-AB-04 Alteration of the geoform of land MQ-AB-05 Alteration of geotechnical conditions MQ-AB-06 Alteration of soil quality MQ-AB-07 Alteration to groundwater quality MQ-SE-03 Change in land use MQ-SE-08 Modification of physical and social infrastructure, public services and social services
	PMA-AB-14 Landscape Management	PSM-AB-14 Landscape Management Monitoring and Follow-Up	MQ-AB-04 Alteration of the geoform of the terrain MQ-SE-02 Alteration in the visual perception of the landscape MQ-SE-06 Generation and/or Alteration of Social Conflict
	PMA-AB-15: Management of Artificial Lighting Systems	PMS-AB-15: Monitoring and Tracking of Light Intensity	MQ-SE-02 Alteration in the visual perception of the landscape MQ-SE-06: Generation and/or alteration of social conflicts

SETTING	EMP	FMP	DETAILS1
Biotic	PMA-BT-01 Vegetation Cover Removal Management Program	PSM-BT-01 Monitoring and follow-up of the vegetation cover removal management program	MQ-AB-04 Alteration of the geoform of the land MQ-AB-05 Alteration of geotechnical conditions MQ-AB-06 Alteration to Soil Quality MQ-AB-07 Alteration to the quality of groundwater resources MQ-AB-08 Alteration to the supply and/or availability of groundwater resources MQ-AB-08 Alteration to the supply and/or availability of groundwater resources MQ-AB-09 Hydrogeomorphological alteration of fluvial dynamics and/or sedimentological regime MQ-AB-10 Alteration in the quality of surface water resources MQ-AB-11 Alteration in the supply and availability of surface water resources MQ-BT-02 Alteration to vegetation cover MQ-BT-03 Alteration to flora community MQ-BT-04 Alteration to the terrestrial fauna community MQ-SE-02 Alteration to the visual perception of the landscape MQ-SE-03 Change in land use MQ-SE-06 Generation and/or alteration of social conflict MQ-SE-08 Modification of physical and social infrastructure, public services, and social services
	PMA-BT-02 Management Program for Sensitive Plant Species (national, regional, gradual, threatened, and sensitive plant species)	PSM-BT-02 Monitoring and follow-up of the management program for sensitive plant species (national, regional, gradual, threatened, and sensitive species)	MQ-BT-03 Disturbance to the flora community
	PMA-BT-03 Management Program for epiphytic and terrestrial vascular flora in national closure in the Intervention Area (epiphytic and terrestrial orchids and bromeliads)	PSM-BT-03 Monitoring and follow-up of the Management Program for epiphytic and terrestrial vascular flora in national closure in the Intervention Area (epiphytic and terrestrial orchids and bromeliads)	MQ-BT-03 Disturbance to the flora community
	PMA-BT-04 Management program for non-vascular organisms in national closure in the Project Intervention Area (epiphytic and terrestrial lichens and mosses)	PSM-BT-04 Monitoring and follow-up of the management program for non-vascular organisms in national closure in the project intervention area (epiphytic and terrestrial lichens and mosses)	MQ-BT-03 Disturbance to the flora community

SETTING	EMP	FMP	DETAILS1
	PMA-BT-05 Terrestrial Fauna Management (birds, mammals, reptiles, and amphibians)	PSM-BT-05 Monitoring and follow-up of terrestrial fauna management (birds, mammals, reptiles and amphibians)	MQ-BT-04 Disturbance to terrestrial fauna community MQ-SE-06 Generation and/or alteration of social conflicts
	PMA-BT-06 Management of the hydrobiological resource	PSM-BT-06 Monitoring and follow-up of the management of hydrobiological resources. PSM-BT-07 Monitoring and follow-up of the quality of the environment of the hydrobiological resources.	MQ-BT-01 Alteration to hydrobiota including aquatic fauna
Socioeconomic	PMA-SE-01 Information and Community Participation Program	PSM-SE-01 Monitoring and Follow-up of the Information and Community Involvement Program	MQ-AB-01 Air Quality Disturbance MQ-AB-02 Alteration in Sound Pressure Levels MQ-AB-07 Alteration to the supply and availability of groundwater resources MQ-AB-10 Alteration in the quality of surface water resources MQ-AB-11 Alteration in the supply and/or availability of surface water resources MQ-BT-01 Alteration to hydrobiota including aquatic fauna MQ-BT-02 Alteration to vegetation coverage MQ-BT-03 Alteration to flora community MQ-BT-04 Alteration to terrestrial fauna community MQ-SE-01 Change in demographic variables MQ-SE-03 Change in land use MQ-SE-04 Alteration of cultural dynamics MQ-SE-05 Alteration to archaeological heritage MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-07 Modification of accessibility, mobility, and local connectivity MQ-SE-08 Modification of physical and social infrastructure, public and social services MQ-SE-09 Modification of economic activities in the area MQ-SE-10 Population to be resettled MQ-SE-11 Citizen participation

SETTING	EMP	FMP	DETAILS1
	PMA-SE-02 Training program for personnel involved in project	PSM-SE-02 Monitoring and follow-up of training program for project personnel	MQ-AB-01 Air Quality Disturbance MQ-AB-02 Alteration to sound pressure levels MQ-AB-06 Alteration to soil quality MQ-AB-06 Alteration to soil quality MQ-AB-10 Alteration to surface water quality MQ-AB-11 Alteration in the supply and availability of surface water resources MQ-BT-01 Alteration to hydrobiota including aquatic fauna MQ-BT-02 Alteration to vegetation cover MQ-BT-03 Alteration to flora community MQ-BT-04 Alteration to terrestrial fauna community MQ-SE-01 Change in demographic variables MQ-SE-02 Alteration in the visual perception of the landscape MQ-SE-03 Change in land use MQ-SE-04 Alteration in cultural dynamics MQ-SE-05 Alteration to archaeological heritage MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-07 Modification of accessibility, mobility, and local connectivity MQ-SE-08 Modification of physical and social infrastructure, public and social services MQ-SE-09 Modification of economic activities in the area MQ-SE-11 Citizen participation
	PMA-SE-03 Infrastructure Restitution and Payment of Damages Program (private and community)	PSM-SE-03 Monitoring and follow-up of the infrastructure restitution and damage payment program (private and community)	MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-07 Modification of accessibility, mobility, and local connectivity MQ-SE-08 Modification of physical and social infrastructure, public and social services MQ-SE-09 Modification of economic activities in the area MQ-SE-10 Population to be resettled MQ-SE-11 Citizen participation
	PMA-SE-04 Archaeological Heritage Management	PSM-SE-04 Archaeological Heritage Management Monitoring and Follow-up	MQ-SE-05 Alteration to archaeological heritage

SETTING	EMP	FMP	DETAILS1
	PMA-SE-05 Labor, Assets, and Services Contracting Program	PSM-SE-05 Monitoring and follow-up of the labor, assets, and services contracting program	MQ-SE-01 Change in demographic variables MQ-SE-03 Change in land use MQ-SE-04 Change in cultural manifestations MQ-SE-05 Alteration of archaeological heritage MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-07 Modification of accessibility, mobility, and local connectivity MQ-SE-08 Modification of physical and social infrastructure, public and social services MQ-SE-09 Modification of economic activities in the area MQ-SE-10 Population to be resettled MQ-SE-11 Citizen participation
	PMA-SE-06 Land and Easement Acquisition Program	PSM-SE-06 Monitoring and Follow-up of the Land Acquisition and Easements Program	MQ-SE-01 Change in Demographic Variables MQ-SE-03 Change in land use MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-09 Modification of economic activities in the area MQ-SE-10 Population to be Resettled MQ-SE-11 Citizen participation
	PMA-SE-07 Road safety management	PSM-SE-07 Road Safety Management Monitoring and Follow-up	MQ-AB-01 Air Quality Disturbance MQ-AB-02 Alteration to sound pressure levels MQ-BT-04 Disturbance to the terrestrial fauna community MQ-SE-04 Alteration of cultural dynamics MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-07 Modification of accessibility, mobility, and local connectivity MQ-SE-08 Modification of physical and social infrastructure, public and social services
	PMA-SE-08 Cultural and Territorial Enhancement Program	PSM-SE-08 Monitoring and follow-up of the cultural and territorial harmonization program	MQ-SE-04 Alteration of Cultural Dynamics
	PMA-SE-09 Resettlement Program	PSM-SE-09 Resettlement Program Monitoring and Follow-Up	MQ-SE-10 Population to be Resettled MQ-SE-06 Generation and/or alteration of social conflict
	PMA-SE-10 Integral reinforcement program for the ethnic community.	PSM-SE-08 Monitoring and follow-up of the Program for the compliance with protocolized agreements CP	MQ-SE-04 Alteration of cultural dynamics MQ-SE-05 Alteration to archaeological heritage MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-07 Modification of accessibility, mobility and local connectivity MQ-SE-08 Modification of physical and social infrastructure, public and social services MQ-SE-09 Modification of the area's economic activities MQ-SE-11 Citizen participation

SETTING	EMP	FMP	DETAILS1
	PMA-SE-11 Compensation Plan for the sustainability of the water supply at the rural level (V. Piedras, Veracruz y Miraflores)	PSM-SE-09 Monitoring and follow-up of the Compensation Plan for the sustainability of water supply at the rural level (V. Piedras, Veracruz and Miraflores).	MQ-AB-08 Alteration in the supply and/or availability of groundwater resources. MQ-AB-11 Alteration in the supply and availability of surface water resources MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-08 Modification of physical and social infrastructure, public services and social services
	PMA-SE-12 Environmental education program for the community	PSM-SE-12 Monitoring and follow-up of the community environmental education program	MQ-AB-01 Air quality disturbance MQ-AB-02 Alteration in sound pressure levels MQ-AB-07 Alteration in the supply and availability of surface water resources MQ-AB-09 Alteration in the supply and/or availability of groundwater resources MQ-BT-01 Alteration to hydrobiota including aquatic fauna MQ-BT-02 Disturbance to vegetation cover MQ-BT-03 Alteration to flora community MQ-BT-04 Disturbance to terrestrial fauna community MQ-SE-02 Alteration to visual perception of landscape MQ-SE-03 Change in land use MQ-SE-04 Alteration in cultural dynamics MQ-SE-05 Alteration to archaeological heritage MQ-SE-06 Generation and/or alteration of social conflicts MQ-SE-07Modification of accessibility, mobility and local connectivity MQ-SE-08 Modification of physical and social infrastructure, public and social services MQ-SE-08 Modification of physical and social infrastructure, public and social services MQ-SE-09 Modification of the area's economic activities MQ-SE-11 Citizen participation

Note: [1] Standardized Category to be managed
(Source: Minera Quinchia S.A.S.)

20.5.1 Surface Water Management

Amec Foster Wheeler (November 2017) conducted an analysis of the hydrology of the La Cumbre Project, with the objective of establishing effects on the main constituents including Leaching Pad, Waste Dump No. 1, Waste Dump No. 2, Waste Dump No. 3 and Waste Dump No. 4 due to the action of water inflow, resulting from surface runoff and managing the water that collects on the surface of these constituents.

In 2022, Minera Quinchia states that measures will be taken to:

Avoid the entry of surface runoff to the disturbed areas, not allowing the contact of these waters with the land affected by the operation, especially the excavation areas.
Prevent the filtration of contact water into the natural soil by impermeabilizing the waste and sulfide storage areas by using textured geomembranes on both sides, fixed to the ground by anchor trenches and placed on a non-woven geotextile sheet, as protection against the puncturing of soil particles.
Collect the contact water in contact ponds and send it to the detoxification or neutralization plant, as the case requires.

20.5.1.1 Non-Contact Water Management

La construcción de un sistema de canales de coronación ubicados por encima de la cota más elevada de los principales componentes con el fin de captar las aguas de escorrentía superficial de las cuencas superiores y derivarlas hacia los cauces naturales vecinos.

20.5.1.2 Contact Water Management

The construction of a system of contact channels adjacent to the planned components that will capture surface runoff from the leach pad, waste rock dump, and oxide and mixed-ore deposits, and then convey it to the contact water pond located at the bottom of the waste rock dump.

Also, measures to avoid possible seepage of contact water into the natural soil. The waste, oxide, sulfide and mixed-ore deposits will have an impermeable coating composed of a 2 mm thick LLDPE geomembrane and a basal drainage system (or seepage system) that will capture any seepage and divert it to the contact water pond.

20.5.1.3 Seepage Systems

Simultaneously, the impermeabilization of the storage area includes the installment of a seepage collection system at the base of the leached material storage area (on top of the impermeable geomembrane), that will allow the collection of possible seepage of contact water and discharge it to the contact water pond, located at the bottom of the leached material storage area.

20.5.1.4 Subdrainage Systems

Below the storage areas and in natural terrain, a subdrainage system will be installed to capture any groundwater (non-contact) through a network of main, secondary and tertiary subdrains to the subdrainage well located downstream of the planned contact water pond.

20.5.1.5 Crown Channels

The design and construction of a crown channel located above the maximum storage level of each storage reservoir to capture and discharge surface runoff water outside the reservoir area to neighboring natural catchments.

20.5.1.6 Contact Channel

As a complementary measure, the construction of lateral channels has been planned to capture the surface runoff of contact water and convey it to the projected contact water pond.

20.5.1.7 Contact Water Ponds

In order to temporarily retain the contact water collected by the seepage system and contact channels, the construction of contact water ponds has been foreseen.

20.5.1.8 Subdrainage Ponds

In order to temporarily retain the possible groundwater conducted by the planned subdrainage system, the construction of storage ponds located at the bottom of the contact water pond has been foreseen. These ponds will have a compacted fill dam and a spillway channel that will convey the water to the natural channel.

20.5.2 Surface Water Management in Paramillo

The design of hydraulic structures for the drainage and management of surface water inside the Paramillo landfill was performed having in consideration the different stages of the filling of the reservoir.

The construction of a system of drainage structures, piping, and road and perimeter gutters, which will complement each other at each stage. This system can be extended or maintained for each new phase and is complemented by different water discharge structures that reduce the flow velocities at the outflow of gutters and pipes.

20.5.3 Environmental Geochemistry

Several studies were undertaken by Minera Quinchia or by specialized consultants in an attempt to complete the geochemical evaluation of the oxide, transition and sulfide zones defined within the excavation polygon, as well as the materials coming from the potential quarries identified for the La Cumbre project. These studies allowed to establish that there is a potential generation of acid drainage for the materials coming from the transition and sulfide zones. The main studies are listed as follows:

- Estimation of the Acid Drainage Generation Potential of the "La Cumbre" Open Pit Gold Mining Project - Quinchia - Risaralda; prepared by SGS Colombia S.A.S. (July, 2020).
- ABA, NAG Tests - Final Report; report prepared by SGS del Peru S.A.C. (October, 2020).
- Acid Pit Drainage Results (ZO) Infrastructure 2021; report prepared by Minera Quinchia (March 16, 2021).
- ABA, NAG Tests - Final Report "9 samples"; report prepared by SGS del Peru S.A.C. (February, 2021).

20.5.3.1 ABA Test Result

The results of the NP (Neutralization Potential) and AP (Acidification Potential), which will determine whether the samples are acid drainage generators, are presented below. The values obtained are summarized in Table 20-8.

Table 20-8:
NP and AP Test Results - Head Sample

Element	pH	AP	NP
Unit		kg CaCO³/t	
66289	6.9	11.8	1.1
66291	6.7	35.3	4.8
63932	8.1	110.1	38
66067-66068	6.2	38.7	3.4
66070	7.9	33.3	5.6
67527-67528	7.4	61.4	6.1
20021-20022-20023	7.9	6.3	9.3
66491-66492	7.6	11.2	5.4
20048-20049	7.3	92.8	12.4
66931	4.3	168.2	1.9
66959	7.8	49.3	16.6
*DUP 66070	7.9	33.9	5.8
*DUP 20021-20022-20023	7.9	6.3	9.4

(Source: SGS del Perú SAC, 2021)

Table 20-9 provides the classification of the potential for acid drainage generation according to the two evaluation criteria for the Acid Base Accounting (ABA) test: Net Neutralization Potential (NNP) and the Neutralization Potential / Acidification Potential (NPR) Ratio.

Table 20-9:
ABA Test for Material Classification - Head Sample

Element	NNP	Material Classification by NNP (*)	NP/AP	Material Classification by NPR (*)
Unit	kg $CaCO_3$/t			
66289	-10.70	Uncertain	0.09	Producer
66291	-30.50	Producer	0.14	Producer
63932	-72.10	Producer	0.35	Producer
66067-66068	-35.30	Producer	0.09	Producer
66070	-27.70	Producer	0.17	Producer
67527-67528	-55.30	Producer	0.10	Producer
20021-20022-20023	3.00	Uncertain	1.48	Uncertain
66491-66492	-5.80	Producer	0.48	Producer
20048-20049	-80.40	Producer	0.13	Producer
66931	-166.30	Producer	0.01	Producer
66959	-32.700	Producer	0.34	Producer
*DUP 66070	-28.10	Producer	0.17	Producer
*DUP 20021-20022-20023	3.10	Uncertain	1.49	Uncertain

Table 20-10 shows the classification of the potential for acid water generation according to the two evaluation approaches for the ABA test: Net Neutralization Potential (NNP) and the Neutralization Potential / Acidification Potential (NPR) ratio.

Table 20-10:
ABA Classification - Moisture Cell Tailings

Element Unit	pH	AP kg $CaCO_3$/t	NP kg $CaCO_3$/t	NNP kg $CaCO_3$/t	Material Classification by NNP (*)	NP/AP	Material Classification by NPR (*)
66289	6.7	14.1	4.4	-9.70	Uncertain	0.31	Acid Producer
66291	6.8	36.2	8.9	-27.30	Acid Producer	0.25	Acid Producer
63932	8.2	93.8	38	-55.80	Acid Producer	0.41	Acid Producer
66067-66068	5.8	30.3	5.6	-24.70	Acid Producer	0.18	Acid Producer
66070	7.7	28.7	8.6	-20.10	Acid Producer	0.30	Acid Producer
67527-67528	7.5	46.1	8.4	-37.70	Acid Producer	0.18	Acid Producer
20021-20022-20023	7.8	6.0	10.4	4.40	Uncertain	1.73	Uncertain
66491-66492	7.3	7.5	7.1	-0.40	Uncertain	0.95	Acid Producer
20048-20049	7.3	76.5	13.4	-63.10	Acid Producer	0.18	Acid Producer
66931	4.2	130.3	2.1	-128.20	Acid Producer	0.02	Acid Producer
66959	7.6	51.6	15.6	-36.00	Acid Producer	0.30	Acid Producer
*DUP 66959	7.6	51.6	15.6	-36.00	Acid Producer	0.30	Acid Producer

(Source: SGS del Perú SAC, 2021)

In the study "Estimation of the potential for acid drainage generation" (SGS Colombia SAS, 2020), samples were selected from the drill holes that were within the pit polygon, concluding "...*The statistical analysis for all the drill holes within the perimeter of the pit, allowed establishing that the deposit has potential for acid drainage generation, especially in the geochemical units of transition and sulfides*".

Table 20-11 provides details of the results obtained in the previously mentioned study.

Table 20-11:
Neutralization Capacity and Acid Potential

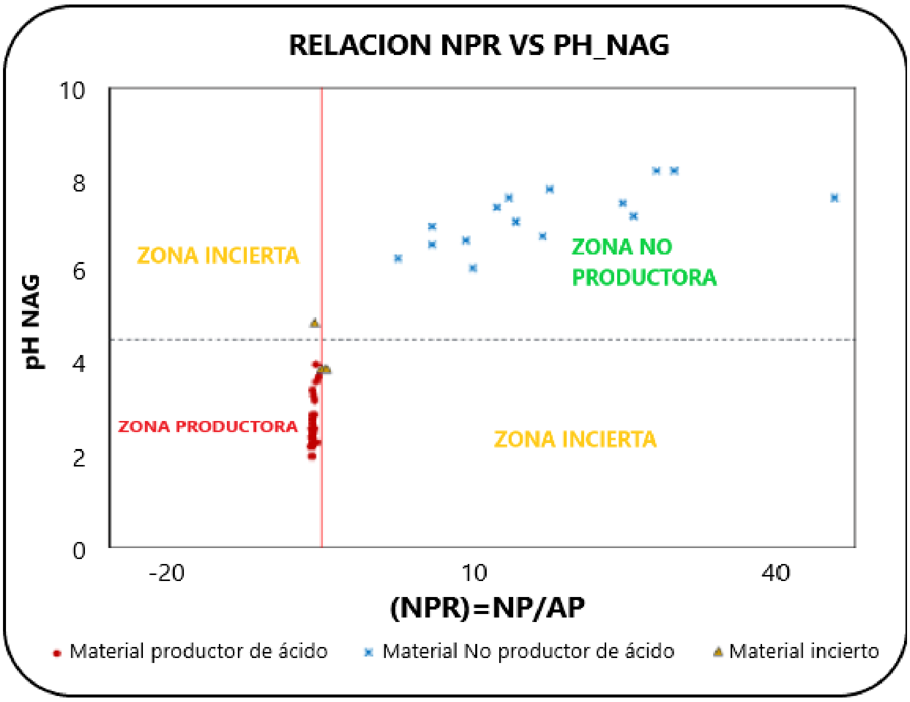
Geochemistry Unit	Samples with neutralization capacity	Samples with Acid Potential	Totals
Oxide Zone	20	16	36
Transition Zone	3	20	23
Primary Zone	0	25	25
Total	23	61	84

(Source: SGS Colombia SAS, 2020)

Additionally, this study involved the evaluation of the surface bodies located in the area of interest, concluding that "The area evidences the generation of acid rock drainage as evidenced by the surface runoff sample taken in La Amarilla creek, both for its historical records and for the measurement taken in June 2020".

At the same time, in the study "Pruebas ABA, NAG - Informe Final" (SGS del Perú S.A.C., 2020) 56 samples were studied and static tests were performed (Acid Base Count (ABA) and Net Acid Generation (NAG)). This study determined that of the 56 samples studied, 38 samples (68%) showed potential for acidity generation, 15 samples (27%) had no potential for acidity generation and 03 samples (5%) had uncertain behavior. Figure 20-2 displays the relationship between pH NAG (NAG assay) with the NP/AP ratio (ABA assay).

Figure 20-2:
NPR vs. pH Ratio - Oxides, Transition and Sulfide Zones



(Source: SGS del Perú SAC, 2020)

20.6 SOCIAL OR COMMUNITY IMPACT

20.6.1 General Context

The La Cumbre Project is located in the Municipality of Quinchia, also known as "Valle de los Cerros", Department of Risaralda; having a territorial extension of 149.8 km². According to censuses conducted by the National Administrative Department of Statistics (DANE), in 2005, there was a population register of 31 996 inhabitants; while in 2016 the population is estimated at 33 816 inhabitants (51.2% men and 48.8% women). The distribution by age groups allows identifying that about 30% of the population is between 0 and 14 years of age, while 32% between 30 and 59 years of age; having that 76% of the population is located in rural areas, while 24% of the population is located in urban areas. The illiteracy index is 11.4%, with a net secondary education coverage of 39.5%.

Between 2017 and 2021, Minera Quinchia gathered information in the Area of Influence of the La Cumbre Project, regarding the different socio-economic components, such as: demographic, spatial, political, organizational, and economic.

Demographic component: a population of 3366 inhabitants was reported, corresponding to a total of 1548 males (46%) and 1818 (54%) females distributed in 1114 dwellings.

Spatial component: there were 18 community huts, 5 educational centers, 5 sugar mills, 5 religious centers, 17 sports venues, and 1 cemetery.

Political and Organizational Component: 2 indigenous groups, 17 Emberra-Chami groups, 22 Karamba groups, community organizations with 20 Community Action Boards.

Economic Component: mainly associated with agricultural activities with coffee, sugar cane and banana plantations; mining activities with mining projects such as Chuscal Alto, Miraflores, La Palma Llanadas and La Cumbre.

20.6.2 Social Management System

Minera Quinchia, through its Social Area, reports on activities to inform about the inherent aspects of the impacts of the mining project, addressing non-ethnic populations or communities in the Area of Influence of the La Cumbre Project, with the development of the community information and participation program (PIPC), which is executed in three stages during the preparation of the EIA, in these activities the project is socialized to the communities and stakeholders of the IA, spaces are provided for the identification of impacts and the formulation of management measures with the participation and contributions of the community.

First Moment: started on August 19, 2020, and ended on August 13, 2021, where activities were carried out such as: beginning of Socialization for the EIA, application of the Veredal File, identification of Community Facilities, identification of Infrastructure in Easement, and the beginning of the Prior Consultation Process.

Second Moment: started on September 14, 2021, and ended on January 27, 2022, reporting: the presentation of EIA Progress together with activities of the La Cumbre Project, validation and identification of impacts inherent to the project, economic valuation surveys and eco-systemic services, evaluation of Landscape Perception.

Third Moment: started on February 21, 2022, and ended on May 25, 2022, where the return of the EIA results is reported.

In general terms, Minera Quinchia aligns its socio-economic management in the Area of Influence of the La Cumbre Project, in issues related to: Education, Social Infrastructure Development, Strengthening Productive Projects, Health Systems, Culture and Heritage Development. The analysis matrix includes the implementation of projects, quantification of beneficiaries, and establishing favorable impacts of these actions. Table 20-12 summarizes this Social Management Plan.

Table 20-12:
Social Management Plan - La Cumbre Project

Lines of Action	Tasks	Number of Beneficiaries	Effect
Education	Special education for adults	48 adults	The beneficiaries were able to complete their elementary school studies and 60% of them have continued their studies in high school.
	Educational project “Escuela & Panela”	400 students	Reinforcement of the generational replacement in the sugarcane sector of the local municipality
	Delivery of School Kits and Reading and Writing Libraries	1000 IA students	Facilitate academic access to education by increasing enrollment coverage in the 15 educational centers of IA
Social Infrastructure Development	Improvement of community shelters	2,464 IA inhabitants	Strengthening of community activities in the 12 beneficiary settlements of the project, providing better facilities for their meetings
	Improvement of Educational Facilities	1,300 IA Students and Teachers	Improvement of the infrastructure of the 15 educational facilities of the IA, positively impacting the teaching-learning spaces and preventing school dropout
	Maintenance and improvement of roads	3,366 IA inhabitants	Maintaining favorable conditions of access roads to the project; favoring the population living in the 20 villages of the IA, and improving opportunities to market their agricultural products, improving their welfare and quality of life
	Improvement and construction of community aqueducts	1,760 IA families	Aqueduct construction and improvement, allowing the inhabitants of the IA to benefit from potable water
Strengthening Productive Development Programs	International Cooperation Projects (IACD/MQ/IOM) Strengthening and promotion of the sugarcane sector in the municipality	212 producer families in the municipality	Increased income by 40%; consequently, higher production in the honey plant and obtaining INVIMA registration, improving the commercialization of panela at different scales. Identification of four innovative business projects (RETO QUINCHIA), two of them currently underway. This results in the improvement of the quality of life and business development in the economic sector of the municipality.
	Strengthening and support to AID farmers. Partnership with the Local Government, SENA, National Federation of Coffee Farmers	3,354 coffee growers in the municipality	Increased crop renewal, higher productivity and lower production costs over the last 4 years
	Partnership with the Local Government - SENA	24 IA families	Implementation of pilot production projects, transforming production systems with increased income for beneficiary families, and is useful as a reference model, with a multiplier effect for other IA farmers
Health Care	Healthcare Unit In alliance with the Nazareth de Quinchia Hospital, support is provided for the provision of medical and dental care in the camp's nursing station	550 users assisted per year	Facilitating health services access for the inhabitants of the IA, supporting health promotion and disease prevention strategies
	La Cumbre Rural School Band Project Musical Training	60 children from IA	Opportunities for AI infants to participate in musical training projects, allowing the incorporation of better habits and lifestyles from childhood

Culture & Heritage Development	Support for rural sports	1,560 children and adults	Encourage and promote sports events, supporting sports training processes in the rural areas of the municipality and promoting the appropriate use of leisure time among the communities of the IA

20.7 CLOSURE

This study of the PEA La Cumbre Project outlines closure actions, which are divided into the following phases:

Initial Closure Plan: summarizes in general terms the use and morphology that will be given to the land, as well as the quality of the biophysical and social environment components.

Progressive closure plan: includes the different closure-related activities that can be implemented during the project operation stage.

Final closure plan: includes geomorphological designs, final revegetation programs and final dismantling of structures that will not provide any services after closure.

Temporary closure plan: sets out the minimum necessary activities to be carried out if adverse conditions arise that require the temporary suspension of project activities for technical, environmental, economic, political or financial reasons.

Post-closure activities: include, among others, physical, chemical, fauna and flora monitoring and maintenance activities associated with the closure measures adopted.

Table 20-13 shows the relationship between the mining activities of the different stages of the project with the activities of the Environmental Management Plan (EMP) that are related to the mine closure plan in any of its phases.

**Table 20-13:
Mining versus Environmental Activities**

<table>
<tr><th>Stage</th><th>Mining Activity</th><th>Environmental Management Plan activity related to the Closure Plan</th></tr>
<tr><td rowspan="10">Construction and Installation</td><td>Explanations, earth movements</td><td>Construction and/or conformation of temporary soil stockpiles</td></tr>
<tr><td>Opening of roads and accesses</td><td>Erosion and soil loss control</td></tr>
<tr><td>Construction of infrastructure and facilities</td><td>Revegetation of affected areas with exposed soils</td></tr>
<tr><td>Land preparation and construction of drainage works</td><td>Construction of water supply and hydraulic control works</td></tr>
<tr><td rowspan="6">Protection structures (dikes, drains, etc.).</td><td>Construction of mine drainage settling and dilution works.</td></tr>
<tr><td>Construction of domestic sewage treatment systems (PTARD).</td></tr>
<tr><td>Construction of hydraulic drainage systems for runoff and rainwater, such as canals, ditches, dispersers, among others.</td></tr>
<tr><td>Construction of industrial sewage (leachate) handling and treatment systems.</td></tr>
<tr><td>Construction of industrial water management and treatment systems (PTARI); effluents from processing plants, workshops, fuel and lubricant supply stations, laboratories, and tailings dams and reservoirs.</td></tr>
<tr><td>Construction of handling and treatment systems for contact water from the Matecaña and La Perla reservoirs.</td></tr>
<tr><td>Mine</td><td></td><td>Management, monitoring and follow-up of surface water management (rain and runoff) and natural water network.</td></tr>
</table>

Stage	Mining Activity	Environmental Management Plan activity related to the Closure Plan
	Operation: start-up, loading, ore transport and ore processing	Management, monitoring and follow-up of domestic and industrial sewage management.
		Management, monitoring, and follow-up of mining drainage from the mining slope (pumping).
		Management, monitoring and follow-up of erosion processes and slope stability (exploitation, roads, mining constructions and assemblies, deposits, and leaching platform).
		Management, monitoring and follow-up of geotechnical stability in the area of direct influence of the project.
		Management, monitoring and follow-up of deposits located in the Paramillo, Matecaña and La Perla areas and their discharges.
		Management, monitoring, and follow-up on the treatment and final disposal of waste from the surface water drainage system, both rainwater and runoff.
		Separation, storage, and final disposal of solid and hazardous waste generated.
		Geomorphological, landscape and forest restoration in stages.
		Monitoring and follow-up of socio-environmental management projects, royalty investment projects, and inter-institutional support programs with company personnel and communities in the project's area of influence.
	Contingency plan	Monitoring and follow-up on the handling and temporary storage of chemical inputs, hazardous substances, fuels, oils and grease, and hazardous waste.
		Monitoring and follow-up on geotechnical stability of surface works.
Closure and abandonment	Dismantling of facilities, demolition of constructions, etc.	Morphological and soil restoration in abandoned areas
		Reforestation and final revegetation of restored areas
		Physical and chemical stabilization of sewage.
		Control, mitigation and compensation in areas affected by erosion and mass removal phenomena.
		Final disposal of sediments from water treatment plants and sedimentation tanks.
		Geomorphological, landscape and forest restoration of the disturbed area, implementation of the final planned land use.
	Closure	Morphological restoration, revegetation, enclosure and signage of access areas to the exploitation works, such as the mining area, the deposits, and the beneficiation area and processing plant
		Removal of chemical inputs.
		Final disposal of hazardous waste.
		Abandonment of facilities and removal of machinery and equipment.
		Information and intervention with affected social groups (workers, contractors and their families).
		Signaling and definitive closure of mining works, DMI/DMO and riprap deposit areas.
Post-Closure		Maintenance and monitoring of the progress of revegetation and reforestation processes of recovered areas.
		Maintenance and monitoring of geotechnical stability of the abandoned and restored mining area, in addition to the mining support works.
		Social monitoring.
		Monitoring of the physical-chemical stabilization system of surface and groundwater.
		Monitoring of terrestrial and aquatic fauna and habitats.

Stage	Mining Activity	Environmental Management Plan activity related to the Closure Plan
		Monitoring and follow-up of final closure plan programs (land use implemented).

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

21.1.1 Initial Capital Costs

The initial capital cost for Phase I totals US$169.5 million, including contingencies and is expected to complete construction in one year. Table 21-1 Initial Capital Cost Summary – Phase I and summarizes the initial capital cost expenditure for the initial Phase I oxide processing plant. Before the oxide ore is depleted, the Phase II sulfide plant processing expansion commences with the view to complete construction in two years with a distribution of 60% in the first year and 40% in year two as shown in Table 21-2 Initial Capital Cost Summary – Phase II.

Table 21-1:
Initial Capital Cost Summary – Phase I

Description	Total (US$k)
Conveyors	6,758
Overland	34,370
Conveyor transport	12,270
Matecaña deposit	7,803
Crushing / agglomeration circuit	9,364
Leaching circuit	19,622
Detox and neutralization treatment circuit	2,637
Conveyors feeding / Unloading of dynamic pad	18,019
Domestic and drinking water treatment circuit	1,356
Dynamic pad, tailings deposit, left over material	36,878
Infrastructures and services	6,023
Contingency	14,388
Total initial capex	**169,489**

Table 21-2:
Initial Capital Cost Summary – Phase II

Description	Total (US$k)
Crushing plant	3,855
Flotation plant 30 ktpd	132,777
Tailing deposits	33,966
Tailing pipeline 2.9 km	642
Waste deposit 60 Mt	36,200
DME	1,662
Land acquisition	2,745
Contingency	36,479
Total initial capex	**248,325**

21.1.2 Sustaining Capital Costs

The total sustaining capital expenditure during the operational periods amounts to US$ 9.5 million for Phase I and additional US$ 11.3 million for Phase II, excluding the closing costs.

21.2 CLOSURE COSTS

Closure costs for Phase I total US$11.9 million and another US$ 15.8 million for Phase II. An overview of the total capital costs and distribution for both Phase I and Phase II of the project as shown in tables Table 21-3 and Table 21-4.

Table 21-3:
Initial and Sustaining Capital Summary – Phase I (US$m)

Description	Total	Y -1	Y 0	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7
Conveyors	6.8	6.8	-	-	-	-	-	-	-	-
Overland	34.4	34.4	-	-	-	-	-	-	-	-
Conveyor transport	12.3	12.3	-	-	-	-	-	-	-	-
Matecaña deposit	7.8	7.8	-	-	-	-	-	-	-	-
Crushing / agglomeration circuit	9.4	9.4	-	-	-	-	-	-	-	-
Leaching circuit	19.6	19.6	-	-	-	-	-	-	-	-
Detox and neutralization circuit	2.6	2.6	-	-	-	-	-	-	-	-
Conveyors feeding	18.0	18.0	-	-	-	-	-	-	-	-
Water treatment circuit	1.4	1.4	-	-	-	-	-	-	-	-
Dynamic pad	36.9	36.9	-	-	-	-	-	-	-	-
Infrastructures and services	6.0	6.0	-	-	-	-	-	-	-	-
Contingency	14.4	14.4	-	-	-	-	-	-	-	-
Total initial capex	169.5	169.5	-	-	-	-	-	-	-	-
Sustaining capex	9.5	-	-	-	-	9.5	-	-	-	-
Closure cost	11.9	-	-	-	-	-	-	-	11.9	-
Total LoM Capex	**190.9**	**169.5**	**-**	**-**	**-**	**9.5**	**-**	**-**	**11.9**	**-**

Table 21-4:
Initial and Sustaining Capital Summary – Phase II (US$m)

Description	Total	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14	Y 15
Flotation plant 30 ktpd	3.9	2.3	1.5	-	-	-	-	-	-	-	-	-	-
Crushing plant	132.8	79.7	53.1	-	-	-	-	-	-	-	-	-	-
Tailing deposits	34.0	20.4	13.6	-	-	-	-	-	-	-	-	-	-
Tailing pipeline 2.9 km	0.6	0.4	0.3	-	-	-	-	-	-	-	-	-	-
Waste deposit 60 Mt	36.2	21.7	14.5	-	-	-	-	-	-	-	-	-	-
DME	1.7	1.0	0.7	-	-	-	-	-	-	-	-	-	-
Land acquisition	2.7	1.6	1.1	-	-	-	-	-	-	-	-	-	-
Contingency	36.5	21.9	14.6	-	-	-	-	-	-	-	-	-	-
Total initial capex	248.3	149.0	99.3	-	-	-	-	-	-	-	-	-	-
Sustaining capex	11.3	-	-	-	-	-		11.3	-	-	-	-	-
Closure cost	15.8	-	-	-	-	-	-	-	-	-	-	-	15.8
Total LoM Capex	**275.4**	**149.0**	**99.3**	**-**	**-**	**-**	**-**	**11.3**	**-**	**-**	**-**	**-**	**15.8**

21.3 WORKING CAPITAL COSTS

The working capital requirement was calculated based on the assumptions of average terms for receivables, accounts payable and inventories. The average terms considered are shown in Table 21-5.

Table 21-5:
Working Capital Assumptions

Working Capital Assumptions	Days
Trades receivable	45
Inventory	45
Trades payable	30
Royalties payable	-
Capex payable	-

The maximum capital requirement for the project's working capital is around US$28.2 million in the second year of operations with US$15.6 million being required for the first year of operations.

21.4 OPERATING COSTS

The operating cost estimate is based on a conveyor plus contractor-operated truck and shovel mining operation, leaching processing facility, flotation processing facility and Tailings Storage Facility. Mine operating cost estimates are provided in Table 21-6. The Table 21-7 shows the unitary operating cost per tonne.

Table 21-6:
Life of Mine Total Operating Costs

Item	LoM costs (US$m)
Mining costs	390.6
Processing costs	743.9
Site G&A	23.6
Treatment, refining, penalties	13.4
By-product credits	(58.3)
C1 cash cost	**1,113.2**
Royalties	92.5
Sustaining capital expenditures	48.5
AISC*	**1,254.2**

*AISC = All-in sustaining cost

Table 21-7:
Unit Operating Costs per Tonne

Item	LoM costs (US$/t milled)
Mining costs (per mined ton)	2.46
Mining costs	3.66
Processing costs	6.98
Site G&A	0.22

Treatment, refining, penalties	0.13
By-product credits	(0.55)
C1 cash cost	**12.90**
Royalties	0.87
Sustaining capital expenditures	0.45
AISC	**14.23**

All-in sustaining cost (AISC) is a metric used by mining companies to reflect the cost of gold mining in a consistent format useful to both investors and mining professionals.

The unitary operating cost per ounce are showing in Table 21-8. The PEA estimates that the C1 operating costs will average US$684/oz of Au.

Table 21-8:
Unit Operating Costs per Ounce

Item	LoM costs (US$/oz)
Mining costs	240.1
Processing costs	457.3
Site G&A	14.5
Treatment, refining, penalties	8.2
By-product credits	(35.9)
C1 cash cost	**684.2**
Royalties	56.9
Sustaining capital expenditures	29.8
AISC	**770.9**

The estimation for the operating cost were completed by Minera Quinchia including the variable and annual fixed costs for both phases as shown in Table 21-9 and Table 21-10.

Table 21-9:
Unit Operating Costs – Phase I

Description	Variable		Fixed	
	Units	Amount	Units	Amount
Mining (Ore)	US$/ mined	$2.29	US$m p.a.	$1.93
Processing	US$/ milled	$7.85	US$m p.a.	$1.48
G&A	US$/ milled		US$m p.a.	$1.52

Table 21-10:
Unit Operating Costs – Phase II

Description	Variable		Fixed	
	Units	Amount	Units	Amount
Mining	US$/ mined	$2.28	US$m p.a.	$2.21
Processing	US$/ milled	$6.44	US$m p.a.	$1.70
G&A	US$/ milled		US$m p.a.	$1.75

22.0 ECONOMIC ANALYSIS

22.1 FORWARD-LOOKING INFORMATION CAUTIONARY STATEMENTS

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

- Mineral resource estimates
- Assumed commodity prices and exchange rates
- The proposed mine production plan
- Projected mining and process recovery rates
- Assumptions as to mining dilution and ability to mine in areas previously exploited using mining methods as envisaged the timing and amount of estimated future production
- Sustaining costs and proposed operating costs
- Assumptions as to closure costs and closure requirements
- Assumptions as to environmental, permitting, and social risks

Additional risks to the forward-looking information include the following:

- Changes to costs of production from what is assumed
- Unrecognized environmental risks
- Unanticipated reclamation expenses
- Unexpected variations in quantity of mineralized material, grade, or recovery rates
- Accidents, labor disputes, and other risks of the mining industry
- Geotechnical or hydrogeological considerations during mining being different from what was assumed
- Failure of mining methods to operate as anticipated
- Failure of plant, equipment, or processes to operate as anticipated
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- Ability to maintain the social license to operate
- Changes to interest rates
- Changes to tax rates.

A financial model was developed to estimate the La Cumbre Project open pit LoM plan comprised of mining the Measured and Indicated Resources within the open pit. The LoM plan covers a period of just over twelve years from first production. Table 22-1 presents a summary of the LoM financial parameters and valuation. All costs are in 2022 US dollar nominal terms and inflation has not been considered in the cash flow analysis.

22.2 FINANCIAL MODEL PARAMETERS

22.2.1 Assumptions

The economic analysis was performed assuming the base case gold price of US$1,750/oz, and silver price of US$22/oz. These metal prices were based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metals price expectation over the life of the project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast. The economic analysis also used the following assumptions:

- Construction period for the initial oxide plan at one year and subsequent expansion to sulfides at two years
- Cost estimates are in constant 2022 US dollars with no inflation or escalation factors considered
- Results are based on 100% ownership with additional royalties or stream besides government royalties
- Capital costs are funded with 100% equity (no financing assumed)
- All cash flows are discounted to the start of the construction period using a mid-period discounting convention.
- All metal products will be sold in the same year they are produced.
- Project revenue will be derived from the sale of gold and silver doré
- Currently, there are no contractual refining arrangements

22.2.2 Taxes

The project has been evaluated on a post-tax basis to provide an approximate value of the potential economics. Calculations are based on the tax regime as of the date of the PEA technical report. At the effective date of this report, the project was assumed to be subject to the following tax regime:

- The Colombian corporate income tax system consists of 30% income tax
- Royalties for gold and silver in Colombia at 4% of the gross metal value (as per Article 16 of Law 141 in 1994). Gross metal value is defined at 80% of the London Metals Exchange spot prices for gold and silver, totalling 3.2% effectively

The economic analysis was performed assuming a 5% discount rate. The pre-tax NPV discounted at 5% is US$730 million; the IRR is 47.5%, and payback period is 1.9 years. On a post-tax basis, the NPV discounted at 5% is $481 million, the IRR is 32.1%, and the payback period is 2.5 years. A summary of project economics is shown in Table 22-2. The analysis was done on an annual cash-flow basis; the cash-flow output is shown in Figure 22-1 and Figure 22-2.

Table 22-1:
LoM Financial Valuation and Parameters

Item	Unit	Open pit
Commodity Prices (Long term)		
Gold Price	US$/oz	$1,750
Silver Price	US$/oz	$22.00
LoM Mine Plan Summary		
Mine Life	ears	14.0
Minable resource	kt	106,594
Gold grade	g/t	0.56
Silver grade	g/t	1.57
Processing Rate	tpd	15,000-30,000

Item	Unit	Open pit
LoM Processing Recovery (Oxide and Trasitional Materials)		
Gold Recovery	%	85.5%
Silver Recovery	%	46.9%
LoM Revenue		
Net Revenue	US$M	$2,905.4
LoM Operating Cost		
Mining	$/t processed	3.66
Processing	$/t processed	6.98
Site G&A	$/t processed	0.22
Treatment, Refining, Freight	$/t processed	0.13
By-product credits	$/t processed	(0.55)
C1 Cash Operating Cost	US$/oz	684.22
AISC Cost	US$/oz	770.89
Operating Costs	US$M	$1,171.5
Royalties	US$M	$92.5
LoM Cash Flow		
EBITDA	US$M	$1,641.4
Net Cash Flow		
Less: Cash taxes	US$M	($352.4)
Less: Change in working capital	US$M	$0.0
Less: Capital expenditures	US$M	($466.3)
Net Cash Flow	US$M	$822.7
Post-Tax NPV 5%	US$M	$480.6
Post-Tax IRR	US$M	32.1%
Payback (1st phase)	Years	2.5

Figure 22-1:
LoM Payable Gold

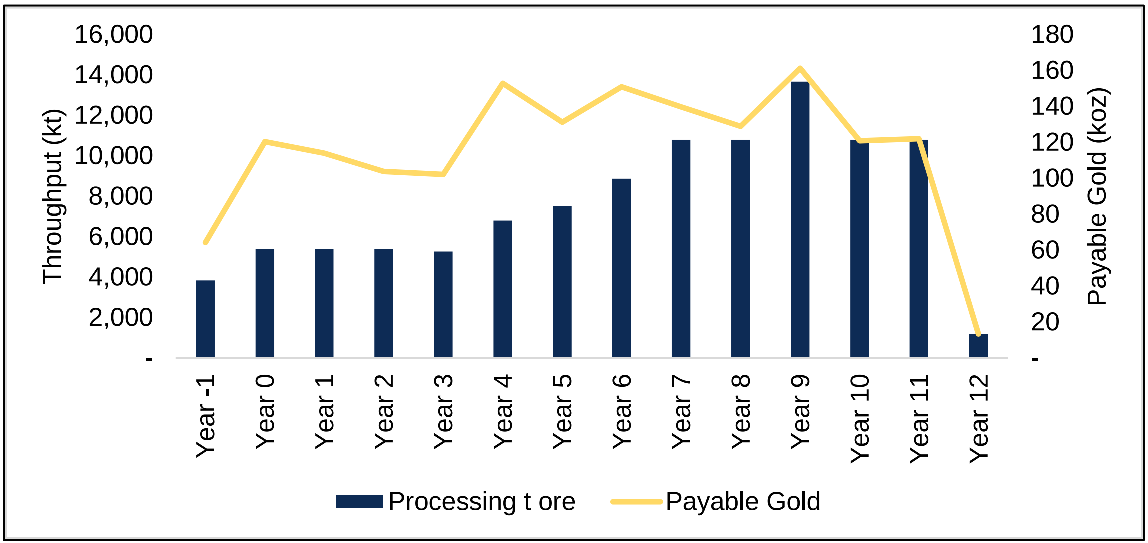


Figure 22-2:
LoM Net Free Cash Flow After Tax

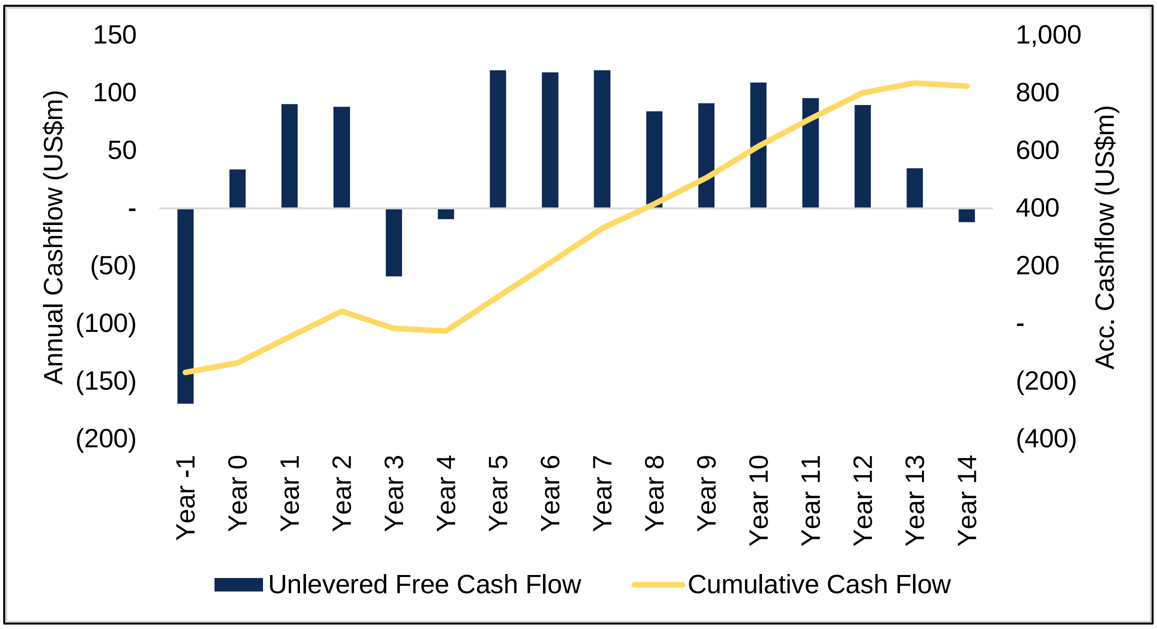


Table 22-2:
Financial Model Summary

Item	Units	Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Total LoM
Gold price	US$/oz	1750	1750	1750	1750	1750	1750	1750	1750	1750	1750	1750	1750	1750	1750	1750	1750	1750
Silver price	US$/oz	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22	22
Mining t ore	kt	-	3,909	5,494	5,623	5,434	4,856	7,371	7,727	9,115	11,292	11,634	11,350	10,800	10,800	1,187	-	106,594
Mining t waste rock	kt	-	1,605	1,281	1,190	770	2,392	1,948	3,007	6,482	11,952	7,777	7,129	4,163	2,485	9	-	52,191
Strip Ratio	W:O	-	0.41x	0.23x	0.21x	0.14x	0.49x	0.26x	0.39x	0.71x	1.06x	0.67x	0.63x	0.39x	0.23x	0.01x	-	0.49x
Processing t ore	kt	-	3,844	5,400	5,400	5,400	5,273	6,800	7,534	8,883	10,800	10,800	13,672	10,800	10,800	1,187	-	106,594
Gold Grade	%	-	0.61	0.81	0.77	0.7	0.72	0.84	0.65	0.63	0.48	0.44	0.44	0.42	0.42	0.42	-	0.56
Silver Grade	%	-	1.46	1.55	1.73	1.72	1.62	1.73	1.82	1.54	1.45	1.73	1.48	1.37	1.51	1.42	-	1.57
Payable Gold	koz	-	64	120	114	104	102	153	131	151	140	129	161	121	122	14	-	1,627
Payable Silver	koz	-	82	123	137	136	135	190	222	222	253	302	321	239	264	27	-	2,651
Gold revenues	US$m	-	113	211	200	182	179	268	230	264	245	226	282	212	214	24	-	2,847
Silver revenues	US$m	-	2	3	3	3	3	4	5	5	6	7	7	5	6	1	-	58
Total Revenue	US$m	-	114.3	213.4	202.7	184.8	182	271.7	234.6	269.1	250.1	232.4	289.5	217	219.4	24.2	-	2,905.40
(-) Downstream costs	US$m	-	-0.5	-0.9	-0.9	-0.8	-0.8	-1.2	-1.1	-1.2	-1.2	-1.1	-1.4	-1	-1.1	-0.1	-	-13.4
Net Revenue	US$m	-	113.8	212.4	201.8	184	181.2	270.5	233.5	267.9	249	231.3	288.1	216	218.3	24.1	-	2,892.00
(-) Mining costs	US$m	-	-13.6	-16.7	-16.8	-15.7	-18.8	-23.5	-26.8	-38	-55.6	-46.7	-44.6	-36.4	-32.5	-4.9	-	-390.6
(-) Processing costs	US$m	-	-31.7	-43.9	-43.9	-43.9	-37.5	-45.5	-50.2	-58.9	-71.3	-71.3	-93.8	-71.3	-71.3	-9.4	-	-743.9
(-) G&A	US$m	-	-1.5	-1.5	-1.5	-1.5	-1.7	-1.7	-1.7	-1.7	-1.7	-1.7	-1.7	-1.7	-1.7	-1.7	-	-23.6
(-) Royalties	US$m	-	-3.6	-6.8	-6.5	-5.9	-5.8	-8.7	-7.5	-8.6	-8	-7.4	-9.2	-6.9	-7	-0.8	-	-92.5
EBITDA	US$m	-	63	144	133	117	117	191	147	161	112	104	139	100	106	7	-	1,641
EBITDA Margin	%	-	55%	67%	66%	63%	64%	70%	63%	60%	45%	45%	48%	46%	48%	30%	-	56%
C1 cash cost	US$m	-	45.5	60.3	60.1	59	55.9	67.8	75	95	124.2	114.2	134.5	105.2	100.8	15.6	-	1,113.20
AISC	US$m	-	49.2	67.1	76.1	64.8	61.7	88.3	82.4	103.6	143.5	121.6	143.7	112.1	107.8	16.3	15.8	1,254.20
C1 cash cost	US$/oz	-	707.7	501.3	527.1	567.3	546.7	443.5	571.1	629	888.8	885.2	833.1	869.6	826.2	1,152.30	-	684.2
AISC	US$/oz	-	764.3	557.8	667.3	624	603.4	577.9	628	685.8	1,026.40	942.6	890.3	926.7	883.5	1,209.40	-	770.9
(-) Cash taxes	US$m	-	-13.6	-40.2	-36.9	-29.5	-27.9	-47.9	-33.8	-35.9	-18.1	-15.7	-21.9	-14.3	-16.1	-0.5	-	-352.4
(-) Change in working capital	US$m	-	-16.1	-12.9	1.3	2.3	0.5	-11.7	4.3	-5.1	1.1	2.6	-7.9	10.2	-0.1	27.9	3.7	-
(-) Capital expenditures	US$m	-169.5	-	-	-9.5	-149	-99.3	-11.9	-	-	-11.3	-	-	-	-	-	-15.8	-466.3
Unlevered Free Cash Flow	US$m	-169.5	33.7	90.4	87.9	-59.2	-9.5	119.6	117.8	119.7	84.1	91.1	108.9	95.6	89.5	34.7	-12.1	822.7
Cumulative Cash Flow	US$m	-169.5	-135.8	-45.4	42.5	-16.7	-26.1	93.5	211.3	331	415.1	506.2	615.1	710.7	800.1	834.8	822.7	

22.3 SENSITIVITY ANALYSIS

Key economic assumptions were examined by running sensitivity analysis on the following to determine their relative importance as value drivers:

- Gold prices
- Silver prices
- Operating costs.
- Initial capital costs
- Sustaining capital costs

The cash flow and NPV5% at various scenarios are summarized in Table 22-3 and Figure 22-3 presents a sensitivity analysis the main drivers for NPV5% and Table 22-4 and Figure 22-4 for IRR.

Table 22-3:
NPV5% at Various Scenarios

Variation	Gold prices (US$m)	Silver prices (US$m)	Opex (US$m)	Initial capex (US$m)	Sustaining capex (US$m)
(20%)	223	476	586	535	486
(10%)	352	478	533	508	483
Base Case	481	481	481	481	481
10%	609	483	428	453	478
20%	737	486	375	426	475

Table 22-4:
IRR at Various Scenarios

Variation	Gold prices (%)	Silver prices (%)	Opex (%)	Initial capex (%)	Sustaining capex (%)
(20%)	18.7%	31.9%	37.0%	41.8%	32.4%
(10%)	25.6%	32.0%	34.6%	36.5%	32.3%
Base Case	32.1%	32.1%	32.1%	32.1%	32.1%
10%	38.4%	32.3%	29.7%	28.5%	32.0%
20%	44.5%	32.4%	27.1%	25.5%	31.9%

Figure 22-3:
NPV5% Sensitivity Analysis

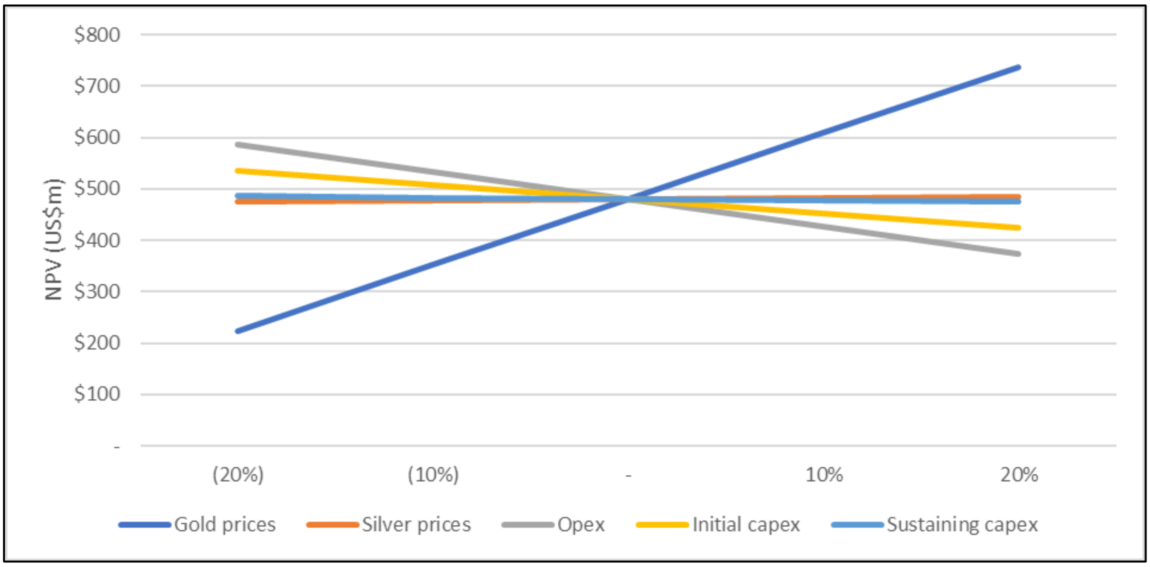
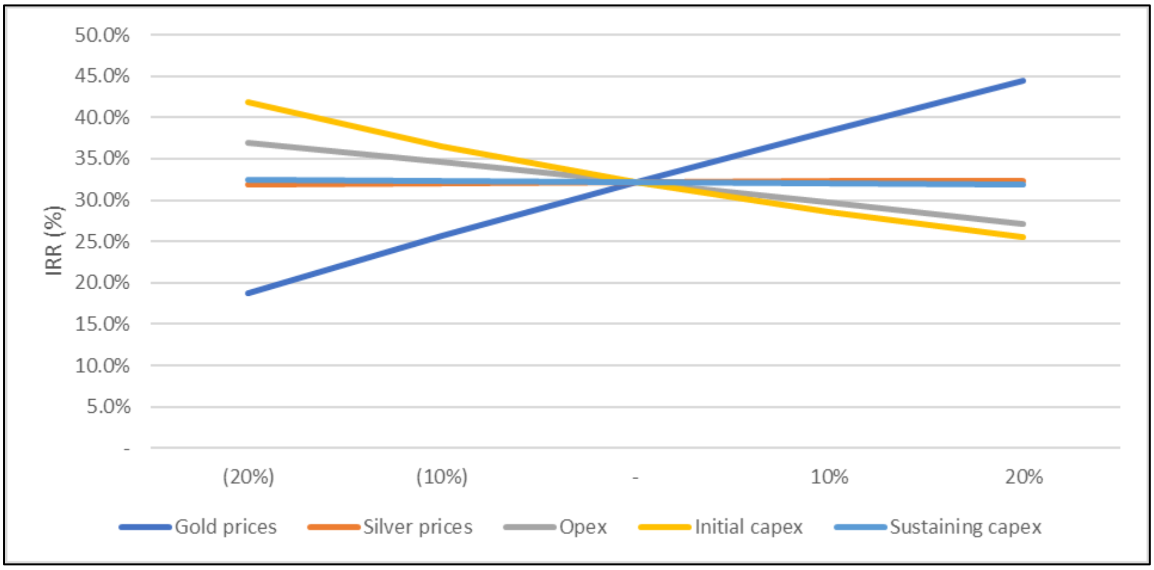


Figure 22-4:
IRR Sensitivity Analysis

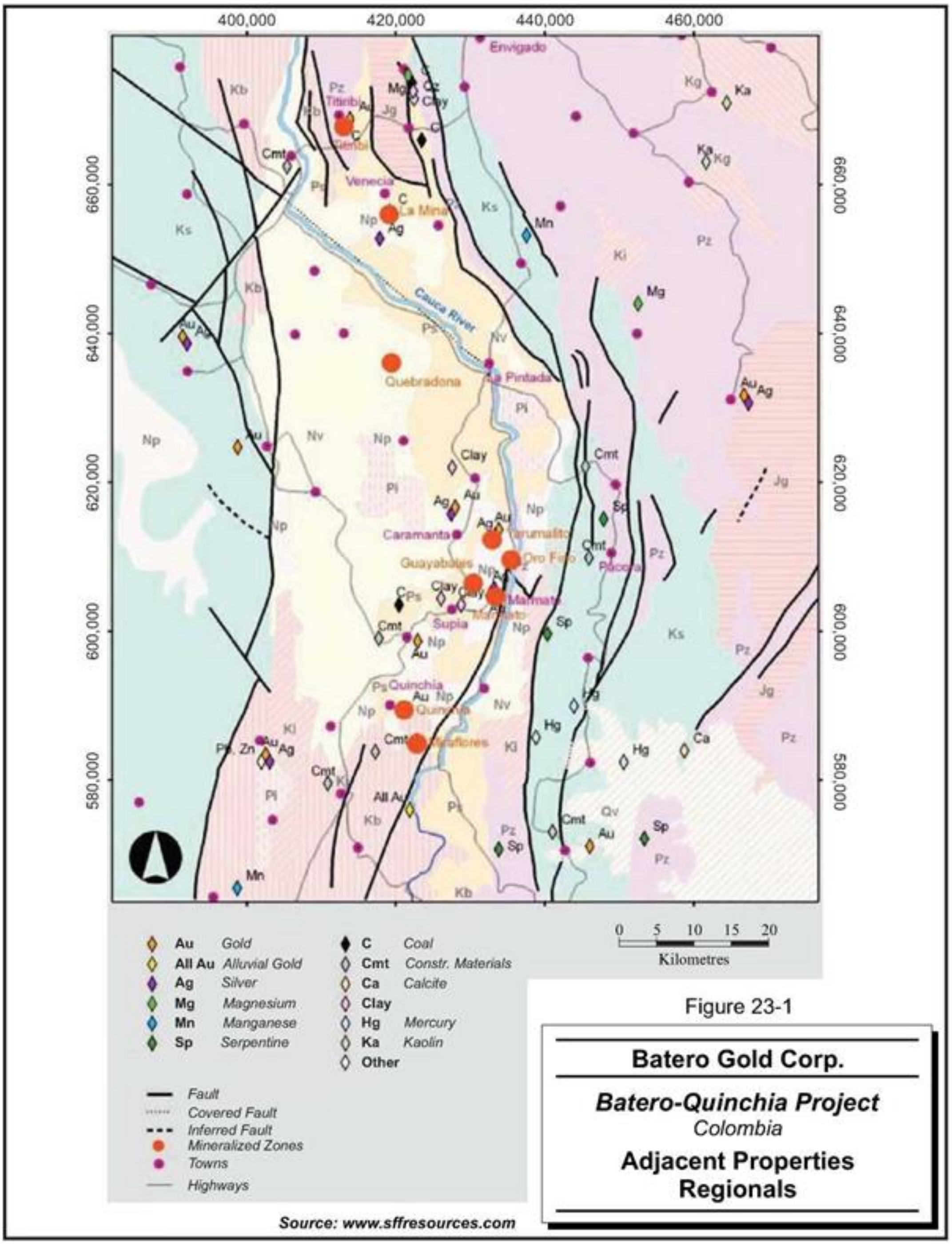


23.0 ADJACENT PROPERTIES

The La Cumbre project is located along the Middle Cauca porphyry belt, which hosts many multi-million-ounce discoveries as defined by Sillitoe (2008). The belt extends about 300 km in a north-south direction from Buriticá in the north to La Colosa in the south.

Figure 23-1 shows the location of the main mineralized occurrences near the La Cumbre Project up to a radius of 50 km in a north-south direction, and comprised in the Middle Cauca Belt.

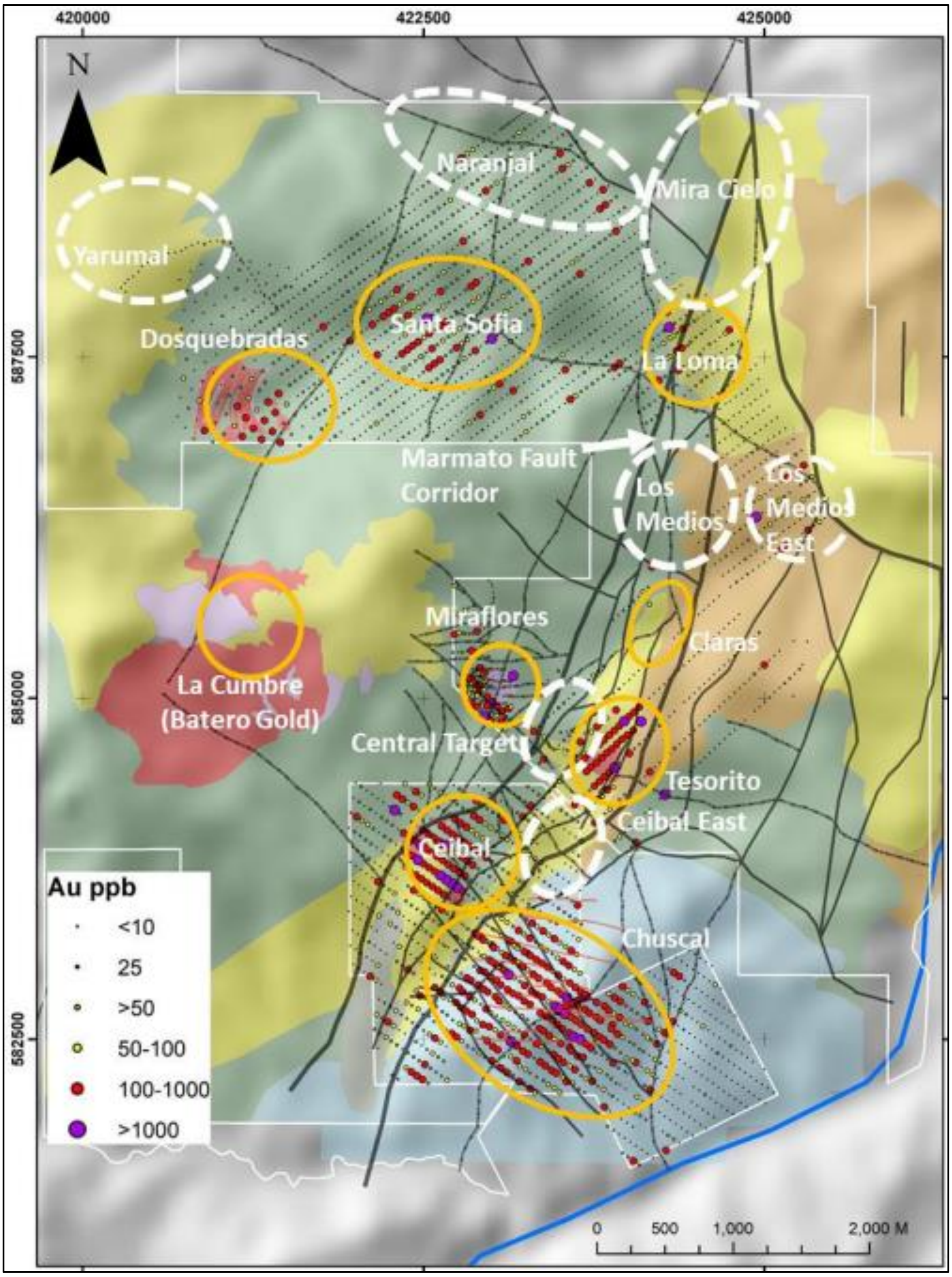
Figure 23-1:
Adjacent Mineralized Properties - Regionals



Similarly, the main mineralized zones near the La Cumbre project are owned by Los Cerros Limited (LCL, a resulting merger between Metminco Ltd. and Andes Resources Limited) that are related to porphyry systems, where we mainly have the Miraflores, Tesorito, Dosquebradas, Chuscal and Ceibal projects.

Figure 23-2 shows the main evidence of gold-silver mineralization around the La Cumbre Project.

Figure 23-2:
Principal Mineralized Zones closed to La Cumbre



Source: Los Cerros Ltd., 2022

23.1 MIRAFLORES

The Miraflores deposit is located immediately east of Batero's License 18567 and consists of a gold-silver rich, magmatic-hydrothermal breccia pipe, settled within a fertile hypabyssal porphyry cluster.

During the period of 2006 to 2013, three diamond drilling programs were carried out totaling 25,884 meters in 73 diamond drill holes and 236 meters of underground channel samples. Based on this information, a total of 13,194 two-meter composites were constructed to for the mineral resource estimation.

Metal Mining Consultants in accordance with the guidelines of JORC Code (2012 edition) prepared the last Mineral Resource Estimation which was published on 14 March 2017.

- Measured Mineral Resources of 2.95Mt @ 2.98g/t Au and 2.5g/t Ag
- Indicated Mineral Resources of 6.31Mt @ 2.74g/t Au and 2.9g/t Ag
- Measured and Indicated Mineral Resources of 9.27Mt @ 2.82g/t Au and 2.77g/t Ag
- Inferred Mineral Resources of 0.49Mt @ 2.36g/t Au and 3.64g/t Ag
- Total M&I Resources contain 840,000ozs Au and 826,000ozs Ag

In October, 2017 the Ore Reserve was estimated using a gold price assumption of US$1,200/oz, a cut-off grade of 1.53g/t Au, constrained by the Mineral Resource Estimate and based entirely on the Measured and Indicated Resources.

- Proved: 1.70Mt @ 2.75g/t Au and 2.20g/t Ag
- Probable: 2.62Mt @ 3.64g/t Au and 3.13g/t Ag
- Proved and Probable: 4.32Mt @ 3.29g/t Au and 2.77g/t Ag

23.2 TESORITO

Tesorito located east of Batero License 18567, near Miraflores, presents evidence of porphyry gold mineralization originating from an intrusive of dioritic composition in contact with breccia units, being discovered in the third quarter of 2020.

Since 2020 Los Cerros Ltd has drilled a total of 22,620 m in 58 diamond drill holes, reporting:

- Very wide porphyry intercepts from near surface. 16 drill holes of 200+m grading ~1g/t Au including 320m @ 1.5g/t Au from surface.
- 50+ diamond holes, targeting MRE calculation H1 2022
- Inferred Resource of 1.3Moz @ 0.81g/t

(https://www.loscerros.com.au/site/projects/quinchia-gold-project)

23.3 DOSQUEBRADAS

Dosquebradas is a copper-poor gold-bearing porphyry system surrounded by basalts, located north of Batero license 22270.

Based on the results of the various drill programs dated since 2009, Cerros Ltd. reports in March 2022, a Mineral Resource Estimate (0.5 g/t Au cut-off grade):

Inferred Resource: 0.46Moz @ 0.71g/t Au, including 511m @ 0.58g/t Au from surface including 200m @ 0.98g/t Au from 122m in QDQ_DH_02

23.4 CHUSCAL AND CEIBAL

The Chuscal and Ceibal projects are located south of Batero License 18567 and are in the Greenfield stage. In the case of the Chuscal Project, two styles of gold mineralization are described, associated to porphyry with low grade disseminated mineralization, with overprinting of epithermal veins with high gold contents, having drilled up to 12 drill holes with a best intercept 350m @ 0.57g/t Au from surface in CHDDH01.

The Ceibal project is a porphyry gold system identified in 2021, where a total of 9 drill holes were drilled with a best intercept of 586m @ 0.51g/t Au from surface in CEDH02.

24.0 OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this technical report understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

1. Hydrothermal magnetite is abundant in the La Cumbre deposit and may be associated with gold mineralization. Magnetite forms part of the potassic alteration.
2. Diorite is the most abundant lithology in the La Cumbre deposit and at least two phases have been identified: a fine-grained diorite and a medium grained diorite with green to gray color.
3. The La Cumbre deposit at Quinchia is a discrete porphyry gold center in which the drilling campaigns have revealed an average gold content reaching economic tenor even though the quartz-veinlet intensity is relatively low (Sillitoe, 2006).
4. The porphyry intrusions that are related genetically to all gold-rich porphyry deposits belong exclusively to I-type, magnetite series suites (e.g., Ishihara, 1981).
5. The abundance of hydrothermal magnetite in gold-rich porphyry deposits, like La Cumbre deposit, suggests that their host intrusions are highly oxidized, sulfur-poor and are representatives of the magnetite series (Sillitoe, 1979).
6. From a review of the drill core, LINAMEC concluded that there are zones without sulfide mineralization but which do have economic gold grades (free gold?) with magnetite.
7. LINAMEC agrees with the deposit type and model previously postulated by Sillitoe (2006) as it seems appropriate for the La Cumbre deposit, which could be classified as a center of a discrete porphyry Au-Ag deposit.
8. In LINAMEC's opinion, the geophysical method of magnetometry yielded the best results in this type of deposit and may be used in future geological prospecting activities including diamond drilling.
9. LINAMEC considers that the current drilling and sampling procedures undertaken at the La Cumbre Deposit are adequate for use in the mineral resource estimation. No major deficiencies or problems were found in the verification and audit procedure.
10. To establish bulk densities for the La Cumbre Project updated mineral resource estimate, 59 samples were collected for bulk density determinations to be used in the tonnage calculation of both oxide and transitional zones. The SG value of 2.520 was used to calculate the tonnage in both mineral zones.
11. The review of CRM's charts concludes that the relative bias for gold and silver is reasonable and that the accuracy and precision of assays from ALS laboratory is considered to be good. The analytical results are suitable for inclusion in mineral resource estimation.
12. Gold grades were validated using RMA plots, that showed a bias below 10% when comparing pairs of primary laboratory grades (ALS) vs. secondary or testing laboratory grades (SGS) and vice versa. LINAMEC concluded that these values could be used for resource estimation for La Cumbre Deposit.
13. The ordinary kriging resource models are adequate based on the current information and show very little bias as compared to the nearest neighbor validation models when analyzed by swath plots.
14. There are no mineral reserves categorized for the La Cumbre project as the status of PEA statement.
15. Gold and silver prices for the economic evaluation of the La Cumbre project were agreed with the corporate office at US$1,750/oz for gold and US$22.00/oz for silver consistent over the life of mine.
16. The final products produced on both phases are gold doré with silver content and are expected to be readily marketable
17. The current and future operations of Batero-Quinchia, including exploration, development and commencement of production activities on the property require permits pertaining to development, mining, production, taxes, labor standards, occupational health, waste disposal, toxic substances, land use, environmental protection, mine safety and other matters, may be required as the project progresses.

18. In 2017, the concession for the use of surface water was renewed extended for five more years, under resolution No. 0072 of January 19, 2017 issued by the Regional Autonomous Corporation of Risaralda (CARDER), which had been granted through the Resolution No. 0730 of March 7, 2011.
19. As of the date of this report, there are no material contracts in place for the construction of the project.
20. The capital expenditures for both phases include direct and indirect cost for the development of the processing plants, open pit mine, associated required infrastructure, offices and general services. A key assumption is the construction of the conveyor to transport ore and waste from the open pit and the equipment required to extract the potential energy and transform it to sellable power to the national grid.
21. The estimate for phase one capital expenditure includes a contingency of 12.8% and totals US$169.5 million of initial capital while phase two includes a contingency of 25% and totals US$248.3 million for the expansion.
22. The operating costs were provided by the corporate office for mining, processing and G&A by phase and broken down by variable and fixed cost. The revenues generated by the power sale related to the conveyor belt act as a discount to the processing cost of US$0.09/t processed. This offset calculation was completed by the corporate team and HLC Ingenieria.
23. The total operating cost including mining, processing, site G&A, treatment and refining adds to a total of US$1,113 million. The C1 cash cost on a by-product basis over the life-of-mine totals US$684/ oz of gold or US$12.90/t milled
24. LINAMEC completed the Economic Analysis for the La Cumbre Project applying the free cash flow valuation methodology. The method attempts to estimate the current value as of the beginning of the project construction based on the future cash-flows the project will generate over the life-of-mine discounted by applying a weighted average cost of capital (WACC). The valuation was completed in 2022 real terms using a 5% per year WACC.
25. The financial model prepared for the report presents an NPV5% post-tax of US$480.6 million, with a Post-Tax, IRR of 32.1% and a 2.5-year payback time. The base case scenario was valued under a gold price of US$1,750/oz, silver price of US$22.00/oz and the exchange at USD: COP 1.00:3963.
26. LINAMEC understands that the La Cumbre Project is economically viable and attractive based on these prices.

26.0 RECOMMENDATIONS

LINAMEC recommends the following be undertaken at the La Cumbre deposit:

1. Determine the gold mineral associations and gold particle size by microscopy of polished sections of mineralized samples.
2. Based on field observations, as well as the geological model proposed for the La Cumbre project, a petrographic study of the lithological units located mainly in the primary zone is required. This would allow correlating gold contents by lithological type and hydrothermal alterations especially within the primary zone.
3. Complete magnetic susceptibility measurements of the cores from the 2016-2017 drilling campaign. These measurements can then be used to determine the mineral zones instead of the sulfur content of the samples. The oxide zone has lower values of magnetic susceptibility and there are zones without, or with very low, quantities of sulfides (py – cpy) but with economic gold contents.
4. Investigate whether there is a correlation between magnetite and Au-Ag mineralization, especially in areas where Cu-Fe sulfides are not present.
5. Downhole surveys must be taken with gyroscopic devices due to the abundance of magnetite in the La Cumbre deposit. The use of magnetic devices such as Reflex EZ-Trac should be avoided.
6. Perform internal audits to the geological databases: lithology, alteration, mineral zones and structures to improve and validate the geological interpretation of mineral zones.
7. Increase the number of density determinations, taking into account the lithology, alteration, mineral zones and structures to get proportional number of samples for density determination of each mineral zone, rock and alteration types.
8. Improve the method of determining the densities, which should include a QAQC program using standards and submit 5% of the samples to a certified laboratory (check samples).
9. Improve the QA/QC program for future drilling campaigns in the La Cumbre Project. This program should cover all activities involved in mineral exploration, geological logging, geotechnical logging, density determination, database inputs, magnetic susceptibility measurements, etc.
10. The QAQC program and all exploration activities must be conducted by a qualified person as defined by international codes JORC and NI 43-101.
11. Undertake X-ray diffractometry (XRD) studies on saprolite samples to determine the type of clay minerals and abundance to prevent problems in heap leaching due to creation of pools by waterproof clay material.
12. Implement cyanide-soluble gold and silver analyses together with total gold and total silver to create geo-metallurgical models to identify zones that are more soluble and to improve the mineral distribution on the leach pads.
13. LINAMEC recommends to evaluate the nominal plant capacity and ramp-up for the sulfide processing plant expansion given that the full capacity is only achieved five years after construction. There is a possibility to decrease the size or implement a progressive expansion to optimized capex and idle capacity.
14. Increase the number of metallurgical tests for the ore from primary zone in a certified laboratory, taking into account the lithology, and alteration types.
15. Update the La Cumbre deposit to the Pre-Feasibility or Feasibility stage to be in accordance with the current work carried out, as recommended by international codes and NI 43-101.

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28.0 CERTIFICATES OF QUALIFIED PERSONS

28.1 QUALIFIED PERSONS

Mr. Walter La Torre, MAusIMM (CP), is a Qualified Person under National Instrument 43-101 Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators. Mr. La Torrre reviewed and approved the scientific or technical disclosure in this release and has verified the data disclosed.

28.2 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

a. I, Walter La Torre, Independent Consultant with LINAMEC S.A.C. Prolongacion Javier Prado, Mz. M Lote 8 Tercera Lima, Lima 03 PERU; do hereby certify that:

b. This certificate applies to the Technical Report entitled “Batero Gold Corp. La Cumbre Gold Project, Department of Risaralda, Colombia NI 43-101 Technical Report on Updated Mineral Resource Estimate and Preliminary Economics Assessment” (the “Technical Report”), dated 01 July 2022.

c. I am a Member of the Australasian Institute of Mining and Metallurgy (AusIMM # 992508) and Chartered Professional (CP) and Professional in Geology of the Engineer College of Peru (Registration No. 57222).

d. I graduated with a Master of Earth Science Degree in Mineral Exploration from International Institute for Aerospace Survey and Earth Sciences (ITC), Delft, Holland in 2000; I have a Diploma in Engineering Geology from National University of Engineering from Lima (Peru) in 1996. I have practiced my profession for 30 years. I have been directly involved in underground and surface operations, mining consulting, resource estimation and mineral exploration in the development of mining projects in Peru and Colombia.

e. I have read the definition of ‘qualified person’ set out in National Instrument 43-101 (“the Instrument”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

f. I last visited the property from December 09 to 12, 2021.

g. I am the co-author of the technical report titled “Batero Gold Corp. La Cumbre Gold Project, Department of Risaralda, Colombia NI 43-101 Technical Report on Updated Mineral Resource Estimate and Preliminary Economics Assessment”. I am co-responsible for this report in its entirety.

h. I am independent of Batero Gold Corp., as independence is described by Section 1.5 of NI 43–101.

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

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

Dated at Lima, Peru, this 31th day of August 2022.

Walter La Torre Chambi, MAusIMM (CP)

CERTIFICATE OF QUALIFIED PERSON

a. I, Fernando Linares Quiroa, Independent Consultant with LINAMEC S.A.C. Calle Antonio Olivera 440 Department 501-B, AA-HH Alto Perú, Chorrillos Lima, Peru; do hereby certify that:

b. This certificate applies to the Technical Report entitled "Batero Gold Corp. La Cumbre Gold Project, Department of Risaralda, Colombia NI 43-101 Technical Report on Updated Mineral Resource Estimate and Preliminary Economics Assessment" (the "Technical Report"), dated 01 July 2022.

c. I am a Member of the Australasian Institute of Mining and Metallurgy (AusIMM # 311886) and Professional in Geology of the Engineer College of Peru (Registration No. 69546).

d. I graduated with a Master of Earth Science Degree in Mining Geology from the National Engineering University of Peru in 1987, I have a Diploma in Engineering Geology from National University of San Agustin from Arequipa (Peru) in 1994. I have practiced my profession for 40 years. I have been directly involved in underground and surface operations, mining consulting, resource estimation and assisting in the development of mining projects in Peru, Argentina, Colombia, Ecuador, and Mexico.

e. I have read the definition of 'qualified person' set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

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Dated at Lima, Peru, this 31th day of August 2022.

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Fernando Linares Quiroa, MAusIMM