NI 43-101 TECHNICAL REPORT ON THE CERRO MARICUNGA PROJECT

PRE-FEASIBILITY STUDY

ATACAMA REGION, CHILE

Prepared For:
Atacama Pacific Gold Corporation

Submitted By:
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Effective Date:
August 19, 2014

Report Date:
October 06, 2014

Revision Number 1

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I, Leticia Conca do hereby certify that:

1. I am General Manager of Alquimia Conceptos S.A., a minerals processing consultant and engineering company, of Santiago, Chile. My address is Alonso de Córdova 5710, Office 404, Las Condes, Santiago, Chile.

2. This certificate applies to the Technical Report titled " NI 43-101 Technical Report on the Cerro Maricunga Project, Atacama Region, Chile", Date effective August 19, 2014", prepared for Atacama Pacific Gold Corporation. (the "Technical Report").

3. I graduated with a Bachelor of Science degree in Civil Mining Engineering from the Universidad de Chile, Santiago, Chile, in 1976 and a Civil Mining Engineer degree in 1993.

4. I am a practicing Mining Engineer and a Registered Member of the Chilean Mining Commission (N° 0058).

5. I have worked as a mining engineer for a total of 38 years. My relevant experience for the purposes of the Technical Report is:

   a. Review and report as a consultant on numerous mining operation and projects around the world for due diligence and regulatory requirements

   b. Pre-feasibility and Feasibility Study work on several copper, gold and silver projects.

6. I have read the definition of “qualified person” set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and confirm that by reason of my education,

affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.

7. I am responsible for the preparation of Items 2, 3, 4, 5, 6, 13, 17, 18, 19, 20, 22, 23 and 27, and portions of Items 1, 21, 24, 25 and 26 of the “NI 43-101 Technical Report on the Cerro Maricunga Project, Atacama Region, Chile", Date effective August 19, 2014. I am responsible for the compilation of information and preparation of the overall Report. Significant contributions were also received from NCL and Eduardo Magri.

8. I am independent of the issuer as set out in Section 1.5 of the Canadian National Instrument 43-101 “Standards of Disclosure for Mineral Projects”.

9. I, or any affiliated entity of mine, has not earned the majority of our income during the preceding three years from Atacama Pacific Gold Corporation, or any associated or affiliated companies.

10. On April 25, 2014, I completed a personal inspection of the Cerro Maricunga Project.

11. I have read National Instrument 43-101 Form 43-101F1 and certify that this Technical Report has been prepared in compliance with the foregoing Instrument and Format.

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

Dated this October 06, 2014

Leticia Conca

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Leticia Conca

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I, Carlos Guzmán do hereby certify that:

1. I am Principal and Project Director with the firm NCL Ingeniería y Construcción SpA., Santiago, Chile. My address is General del Canto 235, Providencia, Santiago Chile.

2. This certificate applies to the Technical Report titled "NI 43-101 Technical Report on the Cerro Maricunga Project, III Region, Chile", Date effective August 19, 2014, prepared for Atacama Pacific Gold Corporation. (the "Technical Report").

3. I am a Graduate of the Universidad de Chile and hold a Mining Engineer title (1995).

4. I am a practicing Mining Engineer, a Fellow Member of the Australasian Institute of Mining and Metallurgy (AusIMM, N° 229036); and a Registered Member of the Chilean Mining Commission (N° 0119).

5. I have worked as a mining engineer for a total of 19 years. My relevant experience for the purpose of the Technical Report is:

    a. Review and report as a consultant on numerous exploration, mining operation and projects around the world for due diligence and regulatory requirements

    b. I have extensive experience in mining engineering. I have worked on mining engineering assignments.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and confirm that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7. I am responsible for the preparation of Items 15 and 16 and portions of items 1, 18, 21, 24, 25 and 26 that are directly relevant to those items of the “NI 43-101 Technical Report on the Cerro Maricunga Project, Atacama Region, Chile", Date effective August 19, 2014.

8. I am independent of the issuer as set out in Section 1.5 of the Canadian National Instrument 43-101 “Standards of Disclosure for Mineral Projects”.

9. I have had prior involvement with the Cerro Maricunga Project. I have been responsible for part of Technical Report (NI 43-101) of the Preliminary Economic Assessment Study for the Cerro Maricunga Project, Region III, Chile, for Atacama Pacific Gold, effective March 15, 2013.

10. On February 10, 2012, I completed a personal inspection of The Cerro Maricunga Project.

11. I have read National Instrument 43-101 Form 43-101F1 and certify that this Technical Report has been prepared in compliance with the foregoing Instrument and Format.

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

Dated this October 06, 2014

Carlos Guzmán

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Carlos Guzmán

Mining Engineer, FAusIMM (229036)

RM Chilean Mining Commission (0119)

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I, Eduardo Magri, do hereby certify that:

1. I am a consulting mining engineer to the mining and mineral exploration industry with an office at Don Carlos 2939, Office 613, Las Condes, Santiago, Chile.
2. This certificate applies to the Technical Report titled "NI 43-101 Technical Report on the Cerro Maricunga Project, Atacama Region, Chile", Date effective August 19, 2014, prepared for Atacama Pacific Gold Corporation. (the "Technical Report").
3. I obtained the following university degrees:
   a. Mining Engineer from the University of Chile, Santiago in 1970
   b. MSc in Mining Engineering from Colorado School of Mines in 1972
   c. Bachelor Honours in Operations Research form the University of South Africa in 1976
   d. PhD in Mining Engineering from the University of the Witwatersrand in 1983
   e. Citation in Applied Geostatistics from the University of Alberta, Canada in 2003.
4. I am a registered and active Fellow of the South African Institute of Mining and Metallurgy since 2004.
5. I have been continuously practicing my profession as a Mining Engineer and consultant since 1972.
6. I have read the definition of “qualified person” set out in National Instrument 43 101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and confirm that by reason of my education, affiliation with a professional association (as defined in NI 43 101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43 101.

7. I am responsible for the preparation of Items 7, 8, 9, 10, 11, 12 and 14, and portions of Items 1, 25 and 26 of the "NI 43-101 Technical Report on the Cerro Maricunga Project, Atacama Region, Chile", Date effective August 19, 2014.

8. I am independent of the issuer as set out in Section 1.5 of the Canadian National Instrument 43-101 "Standards of Disclosure for Mineral Projects".

9. I, or any affiliated entity of mine, has not earned the majority of our income during the preceding three years from Atacama Pacific Gold Corporation, or any associated or affiliated companies.

10. I have had prior involvement with the Cerro Maricunga Project. I have been responsible for part of Technical Report (NI 43-101) of the Preliminary Economic Assessment Study for the Cerro Maricunga Project, Region III, Chile, for Atacama Pacific Gold, effective March 15, 2013.

11. On November 27 and 28, 2012, I completed a personal inspection of the Cerro Maricunga Project.

12. I have read National Instrument 43-101 Form 43-101F1 and certify that this Technical Report has been prepared in compliance with the foregoing Instrument and Format.

13. As at the effective date of this report and certificate to the best of my knowledge, information and belief, the technical report contains all of the scientific and technical information that is required to be disclosed to make the technical report not misleading. I certify that I have actively participated in the following activities: the design and implementation of the sample preparation protocol and QA/QC system; analyses of QA/QC and twin-hole data; geostatistical analyses and geological resource estimation and categorization. Not being a professional geologist, I have relied entirely on other experts in all matters other than the ones mentioned above.

Dated this October 06, 2014

Eduardo Magri

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Eduardo Magri

PhD Mining Engineering

South African Institute of Mining and Metallurgy

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1 EXECUTIVE SUMMARY

1.1 Introduction

This technical report, detailing the results of the Cerro Maricunga Oxide Gold Project ("Cerro Maricunga" or "the Project") Pre-Feasibility Study, has been prepared and compiled by Alquimia Conceptos S.A. ("Alquimia") at the request of Atacama Pacific Gold Corporation ("Atacama"), a public-listed company trading on the Toronto Venture Exchange under the trading symbol ATM. The report was prepared according to the guidelines set out under Canadian Securities Administrators "Form 43-101F1 Technical Report" of National Instrument Standards of Disclosure for Mineral Projects (NI 43-101").

Cerro Maricunga Project is located in the Copiapo Province, III region, Chile, specifically in the Maricunga gold belt, approximately 140 kilometers northeast by road of Copiapo.

The technical report considers the development of a mining operation at Cerro Maricunga. The report takes into account results presented in NI 43-101 reports on Cerro Maricunga dated March 14th, 2013, October 7th, 2011 and August 20th, 2010.

1.2 Property and Location

Cerro Maricunga gold deposit is located in the Maricunga Mineral Belt, the well-known district that contains over 70 million ounces of gold and hosts the La Coipa and Maricunga (Refugio) mines, as well as the Volcan, Caspiche, Marte Lobo and Cerro Casale deposits.

There are no significant population centres or infrastructure in the immediate vicinity of the Cerro Maricunga project. There are a small number of indigenous families communities who raise crops and livestock in areas of the valleys that drain the region but none in the immediate vicinity of Cerro Maricunga.

Chile is an advanced country in terms of mining technology and infrastructure. Copiapo, the nearest major city to the Cerro Maricunga deposit is located approximately 140 km southwest by road. Copiapo has an approximate population of 150,000 people. Experienced mine and plant personnel can be sourced from Copiapo, or elsewhere in Chile where a generally well trained and experienced workforce exists. Furthermore, Copiapo is a well-established support and logistics centre for mining activities in the region.

The project location is shown in Figure 1-1.

Figure 1-1 Location Map of the Cerro Maricunga Gold Project

Atacama is a precious metals exploration and development company with a portfolio of Chilean exploration projects including the flagship 100%-owned Cerro Maricunga gold project. Atacama owns and controls the Cerro Maricunga property through its Chilean subsidiary, Minera Atacama Pacific Gold Chile Limitada.

1.3 Geology and Mineralization

Surface mapping, trenching and drilling indicate that gold mineralization at Cerro Maricunga is confined to a NW-SE trending corridor consisting of a porphyry and breccia complex bounded by fault structures. The strike extension of mineralization was recognized along 2,300 m in NW-SE direction, up to 700 m in NE-SW direction and to depths of 550 m (up to 4,400 masl). The mineralization remains open at depth.

Three mineralized zones have been defined, based on gold distribution, in trenches, outcrops and drill holes: Lynx zone (NW sector), Phoenix zone (Central sector) and Crux zone (SE sector).

Gold mineralization at Cerro Maricunga is hosted in a porphyry and breccia complex, usually associated to black banded quartz veinlets (BBV). The banding in BBV's is due to variable concentrations of tiny gas inclusions, very fine magnetite and rare sulphides aligned parallel to the veinlet margins. Magnetite is variably replaced by hematite due to incipient martitization. Sulfides are relatively rare in the deposit, typically accounting for < 0.1 wt% and are primarily pyrite. A few grains of chalcopyrite, bornite, chalcocite and covellite have been observed.

Gold deportment studies indicate that gold occurs in two forms: native gold and submicroscopic gold in iron oxides ("FeOx"). Native gold is observed mainly as either free grains or attached to FeOx, to rock particles or both. Gold grains are also observed enclosed in rock and/or FeOx.

Free particles of gold in Cerro Maricunga ore are fine, with 75-95% of the grains measuring less than 10-µm. Gold composition is of high purity (on average over 99% Au).

Submicroscopic gold is carried mainly by goethite followed by hematite.

1.4 Exploration

The Cerro Maricunga deposit and surrounding property was explored during the years 2008-2013. This work has consisted in trenching, mapping, geophysics and drilling over four stages.

A total of 26,979 m of trenches were carried out. Trenches were sampled in 5 m lags and assayed for gold.

320 drill holes were drilled into the Cerro Maricunga deposit, totaling 104,948 m. Diamond core comprised 28% of drilling with 29,534 m completed and reverse circulation drilling totaled 75,414 m.

The results of the exploration and drill programs have been reported in a number of 43-101 compliant technical reports filed by Atacama Pacific.

1.5 Mineral Resource

The block size dimensions chosen for the model was 10 x 10 x 10 m. Resources were estimated using Ordinary Kriging (OK).

Measured, indicated, measured and indicated and inferred resources for the Crux, Phoenix and Lynx zones are shown in Table 1-1.

Table 1-1 Mineral Resource

Zone	Measured		Indicated		Measured and Indicated			Inferred		
	Tonnes (Millions)	Grade (g/t Au)	Tonnes (Millions)	Grade (g/t Au)	Tonnes (Millions)	Grade (g/t Au)	Gold Ounces (000's)	Tonnes (Millions)	Grade (g/t Au)	Gold Ounces (000's)
Lynx	20.1	0.46	82.8	0.4	102.9	0.41	1,344	7	0.37	84
Crux	92	0.35	119.1	0.32	211.1	0.33	2,227	28.1	0.3	266
Phoenix	40.7	0.46	79.1	0.42	119.8	0.44	1,678	22.8	0.34	253
Totals	152.8	0.39	281	0.37	433.8	0.38	5,249	57.9	0.32	603

Comments pertinent to results shown in Mineral Resource Table are as follow:

- Mineral resources are reported as global unconstrained resources at a 0.15 g/t Au cut-off grade
- Mineral resources are not confined within a pit using mining parameters
- Rounding may result in apparent summation differences between tonnes, grade and contained gold ounces

- Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

1.6 Mineral Reserve

In order to evaluate the economic potential of the resources, a pit optimization was generated using the Measured and Indicated resources only. Base case pit parameters include a range from 40° to 48° degree slope angle, ore and waste mining average cost of $1.45/t and $3.09/t for process and G&A cost (based on 80,000 tpd throughput rate). The gold price used was $1,300/oz. These parameters were taken from the Preliminary Economic Assessment and/or updated results.

Table 1-2 reports the mineral reserve for the Cerro Maricunga Project based on the production schedule used for this study.

Mineral Reserves have been defined within an open pit mine plan generated considering diluted Measured and Indicated Mineral Resources.

Mineral Resources were converted to Mineral Reserves recognizing the level of confidence in the Mineral Resource estimate and reflecting any modifying factors. The Proven Mineral Reserve is based on Measured Mineral Resources and Probable Mineral Reserve is based on Indicated Mineral Resources after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Project.

The Mineral Reserve is that part of the Mineral Resource which can be economically mined by open pit mining methods. Dilution of the Mineral Resource model and an allowance for ore loss were included in the Mineral Reserve estimate.

Table 1-2 Mineral Reserves

0.15 g/t Au Cut-off Grade	Tonnes (millions)	Grade (g/t Au)	Gold Ounces (000's)
Proven	126.9	0.39	1,603
Probable	167.6	0.40	2,140
Total Proven and Probable	294.4	0.40	3,743

Comments pertinent to results shown in Mineral Reserve Table are as follow:

- Mineral Reserves are reported as constrained within Measured and Indicated pit design, and supported by a mine plan featuring a constant throughput rate. The pit design and mine plan were optimized using the parameters shown above as well as a recovery to dore assumption of 79.5% for gold; $10.0/oz of Au refining charge; 97% mining recovery and 3% of mining dilution
- Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
- Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

1.7 Mining Studies

1.7.1 Pit and Mining Phases Designs

A mine plan was developed for Cerro Maricunga Oxide Gold Project to process 80,000 tpd of resource material. The total annual movement starts at 50 Mt (Year 1) and peaks up to 94 Mt (Year 5).

The final pit design was based on the economic shell generated at $1,300 per ounce. This shell was smoothed and narrow bottoms were eliminated; adding ramps and safety berms where deemed necessary, to obtain an operative final pit with an overall slope angle of 40 to 48 degrees.

Figure 1-2 shows the final pit design with two exits on the west of the pit which give accesses to the primary crusher and to the waste storage areas.

The final pit is 2,500 m long in the SE-NW direction and up to 1,000 m wide in the NE-SW direction. Four pit bottoms can be identified, from north to south at 4,450 mRL, 4,460 mRL, 4,550 mRL and 4,490 mRL. The total area disturbed by the pit is about 196 hectares.

Additional 30 m wide safety berms were included in the design when the slope height exceeds 120 m, accordingly to the geotechnical recommendations.

A set of seven mining phases or pushbacks were designed by analysing the Whittle Four-X series of nested shells. Pit bottoms were selected to project them to surface, applying recommended slopes.

One waste rock storage area at the west and south of the pit was designed for the life of the project. The final configuration is shown in Figure 1-2.

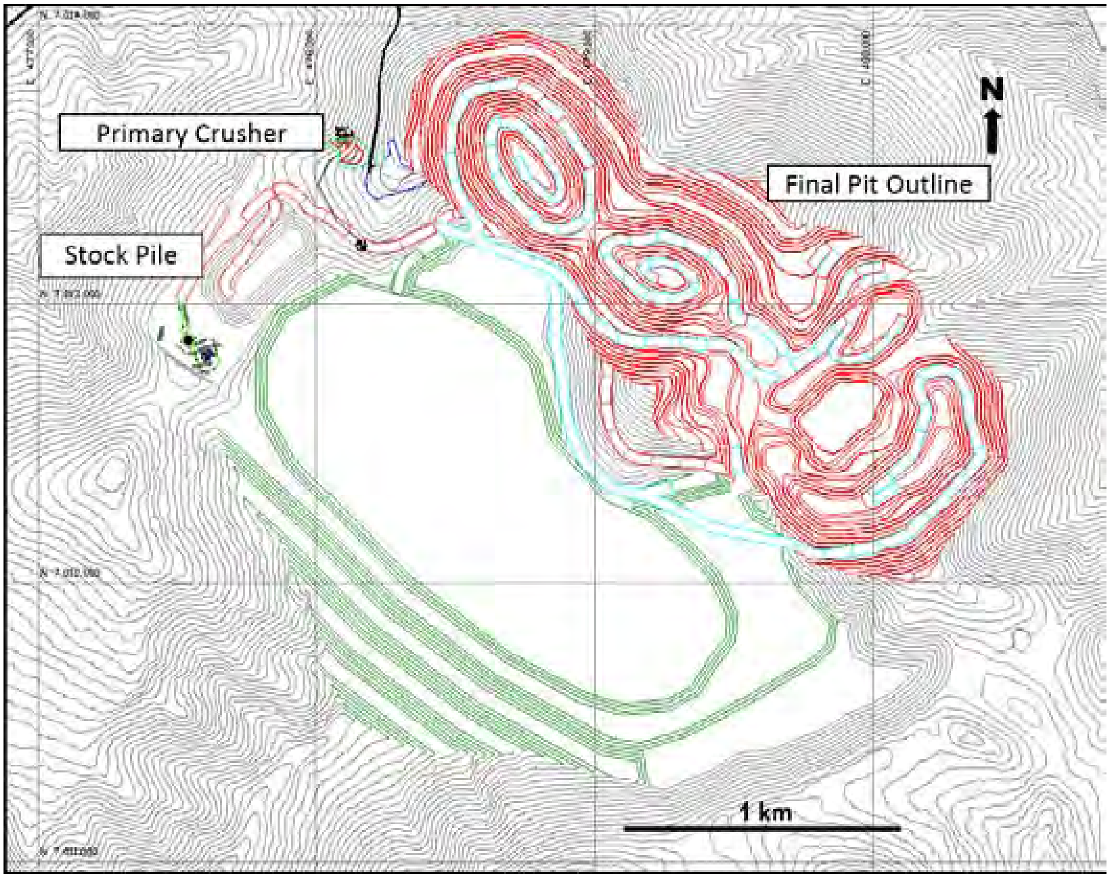


Figure 1-2 Final Pit Design

1.7.2 *Mine Production Schedule*

Table 1-3 shows mine production for each mining year. The schedule is based on 80,000 tpd to match the process plant throughput (29.2 million tonnes per year). The table also shows the total material movement from the mine by year which peaks at 94 million tonnes during Year 5.

Table 1-3 Mine Production Schedule

Year	Ore ('000 tonnes)	Waste ('000 tonnes)	Strip Ratio (waste:ore)	Grade (g/t Au)	Production ('000 oz Au)
-1	5,652	5,348	0.95	0.44	-
1	23,548	26,131	1.11	0.46	285
2	29,200	47,163	1.62	0.42	314
3	29,200	54,038	1.85	0.41	303
4	29,200	54,475	1.87	0.37	275
5	29,200	64,949	2.22	0.38	278
6	29,200	60,669	2.08	0.36	270
7	29,200	55,310	1.89	0.35	257
8	29,200	46,328	1.59	0.36	263
9	20,025	45,579	2.28	0.35	220
10	15,480	31,820	2.06	0.42	165
11	12,778	18,614	1.46	0.47	155
12	8,374	5,626	0.67	0.55	119
13	4,174	1,370	0.33	0.54	58
Totals	294,431	517,419	1.76	0.40	2,963

Comments pertinent to results shown in the Mine Production Schedule Table are as follow:

- Rounding may result in apparent summation errors
- Pre-production ore (5,652 kt) feed process plant in Year 1.

The main highlights of the pit design are the following:

- Material mined: 812 Mt
- Ore processed: 294 Mt @ 0.4 g/t Au (average)
- In-situ gold content: 3.74 Moz
- Average stripping ratio: 1.76:1
- Mine life: 13 years.

1.7.3 *Mine Equipment Requirement*

The mine fleet required is based on annual mine production rate, mine work schedule and hourly equipment production estimates is shown in Table 1-4.

Table 1-4 Major Mine Equipment Requirement

Equipment	Capacity	Pre-Production Requirement	Peak Requirement
Front End Loader	20 m^3	1	1
Hydraulic Shovel	42 m^3	1	4
Haul Truck	290 t	5	17
Diesel Drill	-	2	5

1.8 Metallurgical Testing Summary

Several metallurgical testing campaigns have been carried out with Cerro Maricunga ore since 2010, in order to evaluate its metallurgical behavior in a conventional heap leach process. In addition, some mineralogical studies, in the form of gold deportment, have been conducted to explain the metallurgical results. The test work was undertaken by Kappes, Cassiday and Associates (KCA) and AMTEL Laboratories.

The cyanide leach tests were carried out through bottle roll and column percolation tests under different conditions of residence time and crush/grind size. Although numerous composite samples have been tested through these leach tests since 2010, only twelve composite samples have been considered in this report. The results from the other composite samples, reported by Atacama and summarized in previous 43-101 technical reports, are in line with the 12 composites considered in this report.

Gold deportment studies indicate that free and/or attached gold grains range from 74% to 81% in crushed ore and 78% to 90% in milled ore. Refractory ore in Cerro Maricunga consists of very fine grained gold contained within microcrystalline quartz.

Column tests were performed on eleven of the twelve composite samples (composite 3 was not considered for these tests). The tests were conducted at different crush size, from P_{80} of 100 mm to P_{80} 9.5 mm, and at different residence time from 57 to 113 days and at different sample sizes from approximately 40 kg to 580 kg. All other main conditions like cyanide concentration and pH values were maintained equal for all of the tests.

Conclusions based on the results obtained are as follow:

- Gold extractions of 80% can be achieved, for material with 0.40 g/t Au and at P_{80} 19 mm crush size
- The following model can predict the gold extraction at a P_{80} of 19 mm and ore with gold grades ranging between 0.22 and 1.13 g/t:
    - Gold extraction (%) = 9.1653*(Gold head grade in g Au/t) + 76.534.
- Crushed ore contains three forms of gold; (1): exposed CN-able gold and hence easily recoverable, (2): enclosed CN-able gold and not particularly sensitive to crush size, and (3): refractory gold, which accounts for less than 10% of total gold
- Mineralogical characterization findings were contrasted with the metallurgical tests results, confirming that crush size does not have an important impact in gold extraction and that most of the extractable

gold is recoverable quite fast. These findings allow considering new possibilities for crushing sizes and residence times.

1.9 Mineral Processing and Recovery Methods

Run of mine ore will be crushed in a primary, secondary and tertiary crushing circuit and then treated by a heap cyanidation process, in order to recover gold.

The processing plant has been designed for an annual working rate of 360 days for the crushing plant, and 365 days for the leaching and ADR plant, therefore the crushing plant considers a nominal throughput of 81,150 tpd, while the leaching and ADR plant 80,000 tpd. The average head grade considered for plant design is of 0.5 g/t of Au and 0.25 g/t of Ag. The mineral reserves available for the project are estimated to be 294 Mt, establishing the project life at 13 years.

The project includes the following unit operations or facilities:

- Crushing (primary, secondary and tertiary)
- Heap Leaching.
    - Solution handling.
- ADR plant (Adsorption, desorption and recovery).
    - Adsorption
    - Acid wash and Desorption
    - Recovery (Electrowinning and Smelting)
    - Carbon Reactivation.

A general diagram for the process route is shown in Figure 1-3. Design considers a gold recovery of 80%. The overall calculated recovery for the process is 79.2%.

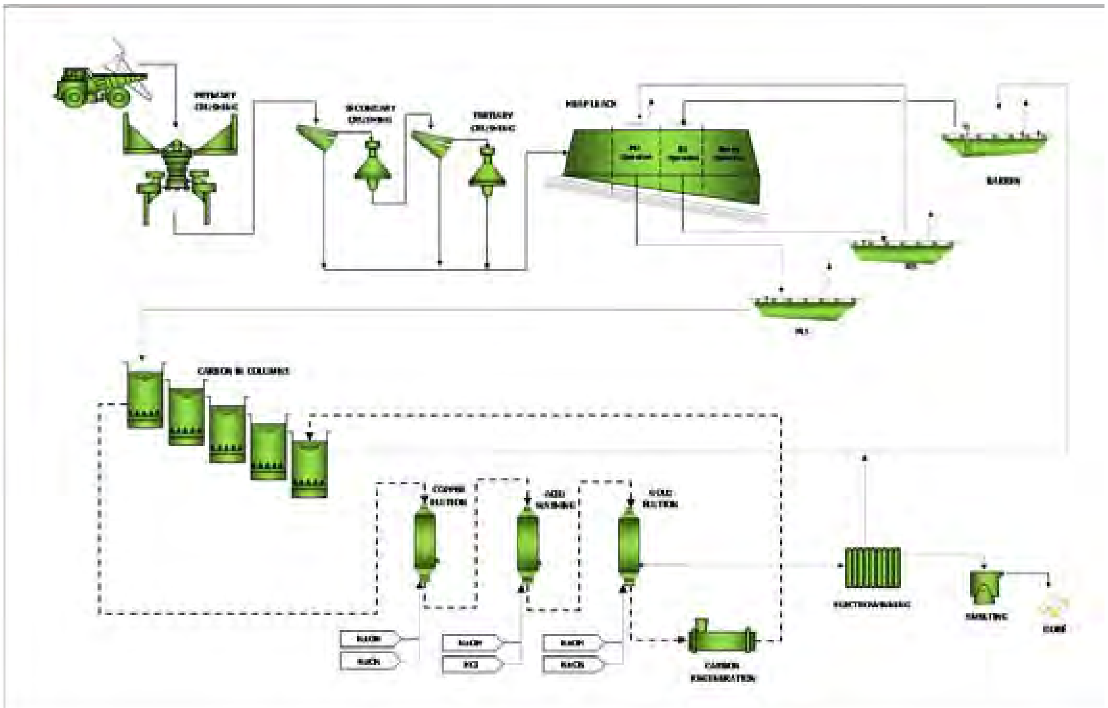


Figure 1-3 Conceptual process plant flowsheet

Table 1-5 shows the main equipment selected for the process.

Table 1-5 Process Main Equipment

Equipment	Qty	Characteristics
Primary crushing		
Giratory crusher	1	62 inch x 75 inch , 450 kW
Secondary crushing		
Secondary screen	2	Conventional, 10 ft x 24 ft
Cone crusher	2	nominal opening 35 mm , 746 kW
Tertiary crushing		
Tertiary screen	3	Conventional, 10 ft x 20 ft
Cone crusher	3	nominal opening 18 mm , 597 kW
Heap Leaching		
Barren pond	1	23,200 m^3
ILS pond	1	21,700 m^3
PLS pond	1	21,700 m^3
Emergency pond	1	103,800 m^3
ADR Plant / Smelting		
Adorption column	5	3 train, Capacity= 4,057 m^3/h
Copper elution column	1	28.6 m^3
Acid wash column	1	28.6 m^3
Elution column	1	28.6 m^3
Electrowinning cells	4	Capacity= 3.54 m^3
Smelting	1	Induction furnace for 0.05 t/d

1.10 Mine Geotechnical

The overall open pit wall angle ranges between 40º and 48º, depending on the position of ramp accesses.

The conceptual estimate for the Project's geometry is based on the following mining efficiency, geology and on-site geotechnical information:

- Perimeter and height of the proposed pit
- Bench height: 10 m high benches or double (20 m)
- Geotechnical data for 84 diamond drill holes (RQD, FF attributes, and core photographs)
- Tele-viewer data for 17 drill holes (strike and dip discontinuity)
- Lithological zoning of major bodies: breccia, dacite-andesite porphyry, and volcanic dacite
- Structures: Major Faults: Saturn, Neptune and Pluto. All structures are vertical to sub-vertical with predominant NW and N-NW orientation.

1.11 Plant Site Geotechnical

The leaching pad site sits on a terrace (Quaternary deposits), mainly composed of granular material such as gravel and silty sand of medium to dense compactness. This material sits on volcanic bedrock.

The projected average slope for leaching pad is of 1:2.5 (equivalent to 22°). The leaching pad is static and seismically stable under higher basal gradient conditions (6.6%).

Site soil material may be used as base material for the construction of the geomembrane support material (silty sand) and the draining layer (cover). The results, in terms of particle size and properties, indicate that the most convenient alternative is to screen material under 5 mm mesh, which would allow obtaining the two required aggregates.

1.12 Site Infrastructure

1.12.1 Water Supply

Fresh water make up requirement (85%) is provided by the water treatment plant Aguas Chañar. Atacama has signed an agreement to ensure industrial water, at a flow of 80 litres per second.

Water will be transported from Aguas Chañar plant (Copiapo) to the project site by a pump system, which consists of a 149 km long pipeline with four intermediate pump stations located on specific points in the route.

1.12.2 Power Supply

The energy for Cerro Maricunga project will be supplied by 110 kV transmission line connected to main Chilean electrical transmission grid or “Sistema Interconectado Central [SIC]”, at “Carrera Pinto” electrical substation.

The 110.5 km long high voltage line supplies energy for the process plant, water pump stations and project facilities. Power system is designed to transmit 40 MVA.

1.12.3 Site Access Roads

The location of the Cerro Maricunga deposit provides major advantages for the project’s construction and future exploitation. The existing Ch 31 International road (Copiapo to Argentinean border) is in the vicinity of the project. The access from Ch 31 to the project requires constructing a 24 km road.

1.12.4 Mine Infrastructure

Vendors will provide storage facilities for fuel, lubricants and explosives as part of their supply contract.

Mining trucks maintenance and repairs will be carried out in truck shops, which will be located close to the primary crusher. A 4-bay truck shop with 10 m width manually operated gates is considered.

1.13 Project Implementation

Atacama needs to complete an environmental impact study and detailed engineering in order to meet the required technical studies for the start-up of the Cerro Maricunga Project.

Construction program and pre-stripping activities can commence once the project is approved by authorities. The construction of the project will be executed considering EPC, EP and construction contracts. Atacama will be leading the construction.

The construction phase of the Cerro Maricunga Project is estimated to last 18 months, beginning in August 2016. The climatic conditions and altitude are of consideration, with major foundation work to be completed during the early summer months.

Key milestones are as follow:

- Feasibility Study: November 2014 – June 2015
- Environmental Impact Study: November 2014 – June 2016
- Detailing Engineering and Procurement: July 2015 – August 2016
- Bidding and Evaluation of Construction Contracts: June 2016 – September 2016
- Construction: August 2016 – January 2018
- Start-up and Commissioning: January 2018 – May 2018

1.14 Capital and Operating Cost Estimates

Capital and operating costs have been estimated to an accuracy of -5% to +20% with a 90% probability of occurrence.

Key assumptions used in the economic analysis are presented in Table 1-6.

Table 1-6 Main Economical Parameters

Item	Unit	Value
Exchange Rate	CLP/$	600
Fuel Price	$/l	0.90
Energy	$/MWh	100
Lime	$/kg	178
NaCN	$/kg	2,650
Water	$/m^3	0.75

1.14.1 Capital Costs

Capital costs are estimated to define the total cost of the project and verify its economic viability. A summary of the capital costs are shown in Table 1-7. A detail for each of these costs is presented in section 21.

Table 1-7 Capital Costs Summary

Item	Initial ($ Millions)	Sustaining ($ Millions)
Mine		
Pre-stripping	15.3	0.0
1st Fleet Lease Payment	7.5	0.0
Mining Support	19.6	6.3
Process Plant		
Crushing & Stockpiles	143.4	0.0
Leach Pads	83.7	152.5
ADR & EW/Smelting	23.3	0.0
First Fill	2.0	0.0
Infrastructure		
Support Facilities	34.9	0.8
Roads	10.1	0.0
Owner	12.7	
Capital Costs *(without contingencies)*	352.5	159.6
Contingencies	46.5	23.0
Closure Cost	0	5.0
Total Capital	398.9	187.6

Comments pertinent to results shown in Capital Costs Summary Table are as follow:

- The PFS has been completed to a level of accuracy of +20% to -5%
- Contingencies are 15% of capital costs excluding "Mine" costs
- Rounding may result in apparent summation errors.

1.14.2 Operating Costs

All unitary operating costs are expressed in processed tonnes. These costs have been estimated for the operating areas of Mining, Process Plant, and G&A.

A summary of the operating costs are shown in Table 1-8. Detail is presented in section 21.

Table 1-8 Operating Costs Summary

Area	Total Cost ($ Millions)	Unit Cost ($/t)
Mining	1,125	3.82
Process Plant	741	2.52
G&A	159	0.54
Total	2,024	6.88

1.14.3 Leasing

The project execution considers leasing contracts for mine equipment, truck shop, water and power supply and infrastructure. Table 1-9 shows the total leasing costs by contract.

Table 1-9 Leasing Costs Summary

Item	Total Cost ($ Millions)	Unit Cost
Mine Fleet	227.9	$0.28 per tonne mined
Truck Maintenance Shop	27.4	$0.03 per tonne mined
Power Supply	65.0	$0.22 per tonne processed
Water Supply	202.4	$0.69 per tonne processed
Total	522.7	---

Rounding may result in apparent summation errors.

1.14.4 Cash Cost

Total cash costs include mine site operating costs (mining, processing, G&A, royalties and production taxes).

Total site cash cost is the sum of total cash costs, sustaining capital expenditures, capitalized and expensed exploration that is sustaining in nature and environmental reclamation/closure costs.

Table 1-10 Cash Cost Indicators

Item	($ Millions)	($/oz)	($/t)
Cash Cost	2,024	683.3	6.88
Transport + Insurance	14	4.6	0.05
Infrastructure Leasing	267	90.3	0.91
Mine Fleet Leasing	255	86.2	0.87
Total Cash Cost	2,561	864.3	8.70
Sustaining Capex	188	63.3	0.64
Total Site Cash Cost	2,748	927.6	9.33

1.15 Economic Analysis

Economic analysis is based on the estimated CAPEX and OPEX, production plan, and revenue calculated thereof. Project's NPV is calculated to the annual period prior to initial mining capital expenditure.

The economic analysis is presented in a pre and post-tax format. Sensitivities were performed, for changes in market gold price, metal recovery, operating and capital expenditures and discount rate.

Considering that the analysis is based on a cash flow estimate, it should be expected that actual economic results might vary from these results. The PFS has been completed to a level of accuracy of -5% and +20% with a 90% probability of occurrence.

Economic parameters used for the evaluation are shown in Table 1-11.

Table 1-11 Main Economic Parameters

Item	Unit	Value
Economics		
Au Price	US$/oz	1,350
Discount Rate	%	5.0
Taxes & Royalties		
State Royalty	%	Variable
Income Tax Rate	%	20
Bullion Terms		
Payable Au	%	100.00
Transport & Insurances	US$/t	0.046

Total payable gold production and revenues associated to the product are summarized in Table 1-12.

Table 1-12 Revenues Summary

Item	Unit	Value
Mined Mineral	Mt	811
Au	Moz	2.96
Revenues	US$ M	4,000

Based on the projections resulting from the financial model, before and after tax NPV, IRR and payback periods, are shown in Table 1-13.

Table 1-13 Summary of Economic Evaluation Results

Indicator	Unit	Pre-tax	After-tax
NPV @ 5%	US$ M	521	409
IRR	%	28,6	25.0
Payback Period	years	2.76	3.00

A summary for the after-tax NPV sensitivity analysis is shown in Figure 1-4.

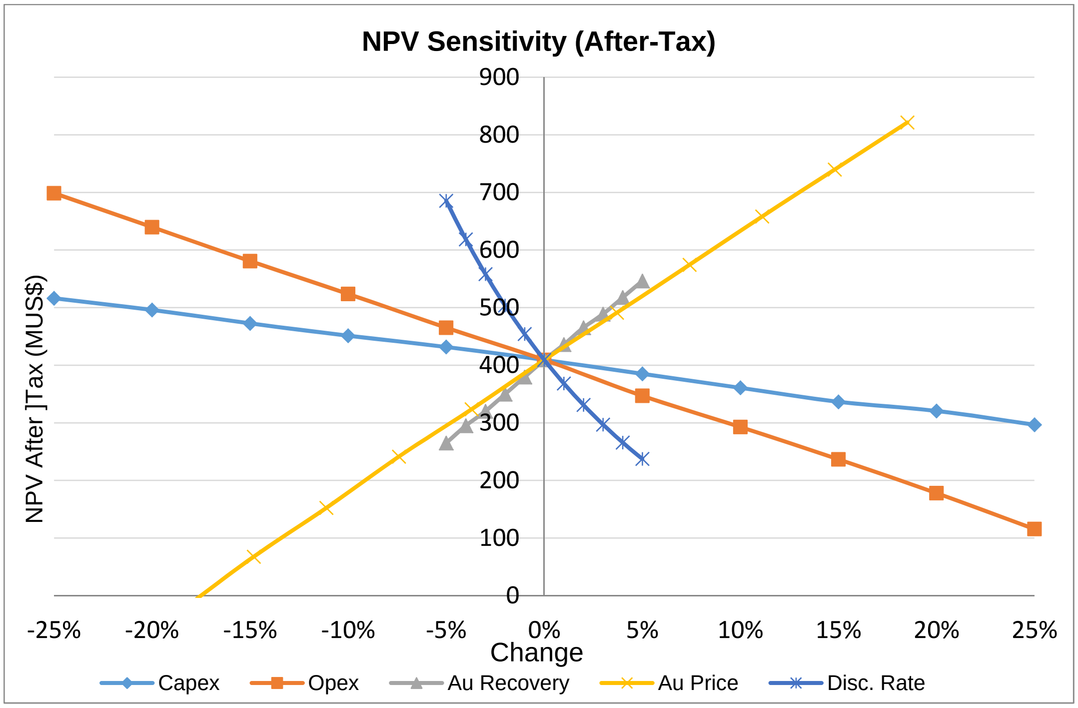


Figure 1-4 After-tax NPV Sensitivity

1.16 Conclusions

The Cerro Maricunga project is located in the Atacama Region (III Region, Chile) within the well-known Maricunga Mineral Belt. The region boasts a well-trained mining workforce, support from experienced and well established mining equipment suppliers, and first class professional and technical consultants. Chile has a long-standing mining culture.

No fatal flaws have been identified during the course of the pre-feasibility study for the Cerro Maricunga Project. Water availability is the most common constrain for mining projects in the area and Cerro Maricunga Project has secured a supply, with the acquisition of the majority of the necessary water resources for the development of Cerro Maricunga, one of the main development risks has been significantly reduced.

On other hand, the increase of 2,000 MW in the Chilean central power grid (SIC) and its connection with the power grid for the northern zone (SING) ensures energy availability for project. It should be noted that the power requirement for the Cerro Maricunga are comparatively low.

The pre-feasibility study for this oxide gold deposit results indicate that, at a gold price of 1,350 $/oz, Cerro Maricunga project is an attractive and robust project, which warrants continued development to full feasibility level.

The main conclusions from the prefeasibility study are as follows:

- Proven and Probable Mineral Reserves total 294.4 Mt grading 0.40 g/t Au
- Mineral Reserve estimated could be most affected by changes in metallurgical recoveries and operating costs
- Gold price, even though the most important factor for revenue calculation, has a lower impact on the Mineral Reserve estimation because the selected Lerchs-Grossman shell used as the guide for practical mine design was obtained using the discounted method and the mine plan considers operational cut-offs higher than the internal cut-off
- Six-month pre-production period and 11 Mt of total material needs to be removed in order to expose sufficient ore to start commercial production in year 1
- A gold extraction of at least 80% is expected for ore with Au grade of 0.40 g/t and at P_{80} 19 mm crush size. The overall recovery of gold in the processing plant, including electro-winning and refining, is 79.2%
- The Cerro Maricunga mining project is feasible from an environmental sustainability viewpoint. All key environmental sustainability variables identified and analysed (potential environmental impacts) can be fully addressed and there are measures in place to effectively manage them
- The financial analysis indicated that the proposed development reflected net positive cash flow and internal rate of return which could support the progression to the next stage of feasibility study
- Chilean government is addressing a new tax regulation and its impact on the project economics needs to be addressed and evaluated.

1.17 Recommendations

In order to continue to the next phase of developing a feasibility study, Alquimia foresees the following activities.

- Develop environmental baselines studies in order to begin the environmental impact study presentation process
- Complete current geotechnical studies in mine, plant and heap leach areas
- Mine fleet optimization studies and mine scheduling may be further developed in order to improve equipment matching and plant scheduling
- Investigate the use of contractors for the open pit mining developments
- Improve the geo-metallurgical model with current and new metallurgical testwork data
- Further metallurgical testwork to confirm likelihood of crushing at 50 mm with minimal loss in recovery
- Process optimization study based on new metallurgical testwork.

2 INTRODUCTION

2.1 Purpose of the Technical Report

This Technical Report, detailing the results of the Cerro Maricunga Pre-feasibility Study, was prepared and compiled by Alquimia Conceptos S.A. ("Alquimia") at the request of Atacama Pacific Gold Corporation. The report was prepared according to the guidelines set out under Canadian Securities Administrators "Form 43-101F1 Technical Report" of National Instrument Standards of Disclosure for Mineral Projects.

The principal consultants used in the preparation of this document are:

- Alquimia Conceptos (Alquimia): Process plant engineering, infrastructure, costing and cash flow
- Magri Consultores (Magri): Geology, database, resource estimate and associated topics
- NCL Ingeniería y Construcción (NCL): Mineral reserves calculation, mine design and production planning
- Derk Ingeniería Geología y servicios (Derk): Mine geotechnical review
- Gabriel Cabezas (Secoia): Leach pad geotechnical review
- Hidromás: Hydrological Study
- Propipe: Water supply pipeline design
- DST Ingenieros Asociados: Power supply line design
- Minería y Medio Ambiente (MYMA): Environmental sustainability study.

2.2 Qualified Persons

The following qualified persons have compiled this technical report:

- Leticia Conca, Principal / Project Director of Alquimia, is responsible for the compilation of the information and preparation of the overall PFS and is responsible for the information provided for the metallurgy and process plant design
- Eduardo Magri, Principal of Magri Consultores is responsible for the mineral resource estimate
- Carlos Guzman, Principal / Project Director of NCL, is responsible for the mining related sections of the PFS including the mineral reserve estimate.

2.3 Effective Dates

The Effective Date of this report is taken to be the date of the completion of the financial model for the Project on August 19th 2014. The dates for critical information used in this report are:

- The Mineral Resource Estimate and block model were completed on January 30th 2014
- The Mineral Reserves Calculation were completed on August 19th 2014
- The final mine plans were issued on July 4th 2014
- Process engineering were completed on July 4th 2014
- Capital and operating costs estimate were completed on July 25th 2014

- The current personal inspections on the project site of all Qualified Persons were completed as follows:
    - Leticia Conca: April 25, 2014
    - Carlos Guzmán: February 10, 2012
    - Eduardo Magri: November 27, 2012.

There were no material changes to the scientific and technical information of the Project between the Effective Date and signature date of the Report.

2.4 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

All units in this report are based on the International System of Units ("SI"), except industry standard units, such as troy ounces for the mass of precious metals. All currency values are in United States Dollars ("$") unless otherwise stated.

This report uses abbreviations and acronyms common within the minerals industry.

Table 2-1 and Table 2-2 respectively identify the terms, abbreviations and units of measurement used in this Report.

Table 2-1 Technical Terms and Abbreviations

Abbreviation	Acronym
AAS	Atomic Absorption Spectrometry
ADR	Adsorption, Desorption, Recovery
Ag	Silver
As	Arsenic
ATM	Atacama Pacific Trading Symbol
Au	Gold
CAPEX	Capital Expenditure
CM	Construction Management
Cu	Copper
DDH	Diamond Drill Hole
DIA	"Declaración de Impacto Ambiental"
ENAMI	"Empresa Nacional de Minería"
EP	Engineering and Procurement
EPC	Engineering, Procurement and Construction
EW	Electrowinning
FEL	Front End Loaders
FeOx	Iron Oxides (Collectively)
HCl	Hydrochloric Acid
Hg	Mercury
ICMC	International Cyanide Management Code
IRR	Internal Rate of Return
K/Ar	Potassium / Argon Geochronology
LG	Lerchs-Grossman
LOM	Life of Mine
MASL	Metres Above Sea Level
MMA	Environment Ministry
NaCl	Sodium Cloride
NPV	Net Present Value
OK	Ordinary Kriging
OPEX	Operational Expenditure
Pb	Lead
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
PLS	Pregnant Leach Solution
QA	Quality Assurance
QC	Quality Control
RC	Reverse-Circulation Drilling Method
ROM	Run of Mine
RQD	Rock-Quality Designation
Sb	Antimony
SEIA	"Sistema de Evaluación de Impacto Ambiental"
SIC	Chilean Power Grid for the Central Zone
SING	Chilean Power Grid for the Northern Zone
Zn	Zinc

Table 2-2 Units of Measurement

Unit	Abbreviation
Bed volume	BV
Cubic metre	m^3
Cubic yard	yd^3
Days per year	d
Degrees relative to true north	°T
Effective productivity per hour	t/op h
Foot / feet	ft
Gram	g
Hectare	ha
Horsepower	HP
Hours	hrs
Hours per period	h/p
Inch	"
Instant productivity per hour	t/hr ef
Kilogram	kg
Kilometre	km
Kilopascal	kPa
kilotonnes	kt
kilotonnes per day	ktpd
kilotonnes per period	kt/p
Kilovolts	kV
Liters	l
Megawatt hour	MWh
Metre	m
Metres Relative Level	mRL
Metric Tonne	t
Micro	µ
Micro Siemens	µS
Millimetre	mm
Million of U.S. dollars	$ Millions
Million ounces	Moz
Million tonnes	Mt
Million volt-amperes	MVA
Minute	min
Normal cubic meter	Nm^3
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percentage	%

Unit	Abbreviation
Pound	lb
Second	s
Square metre	m^2
Thousand	000's
Thousands of ounces	koz
Thousands of U.S. Dollars	k$
Tonnes per day	tpd
United States Dollar	$
Weight percentage	% wt
Year	y

3 RELIANCE ON OTHER EXPERTS

Preparation of this Technical Report has depended on documentation generated by Atacama and Alquimia. It also includes a number of documents with public and private information provided by Atacama and information provided in various previous Technical Reports listed in Section 27 of this Report.

The authors believe that the information provided and relied upon for preparation of this Technical Report is accurate at the time of the Technical Report and that the interpretations and opinions expressed in them are reasonable, based on current understanding of mining and processing techniques and costs, economics, mineralization processes and the host geological setting. The authors have made reasonable efforts to verify the accuracy of the data relied upon in this Technical Report.

The results and opinions expressed in this Technical Report are conditional upon the aforementioned information being current, accurate, and complete as of the date of this Technical Report, and the understanding that no information has been withheld that would affect the conclusions made herein. The authors reserve the right, but will not be obliged, to revise this Technical Report and conclusions if additional information becomes known to the authors subsequent to the date of this Technical Report.

The authors of this Technical Report are not qualified to provide extensive comment on legal issues associated with the Property. For portions of Section 4 dealing with the types and numbers of mineral tenures and licenses, the nature and extent of Atacama's title and interest in the Property, the terms of any royalties, back-in rights, payments or other agreements and encumbrances to which the Project is subject, Alquimia relied on the legal opinion of the law firm Baker Mackenzie Ltd, Santiago.

Other expert persons relied upon for the preparation of this Technical Report were:

- Natasha Tschischow (Principal NTK) was involved in the mineral resource estimation published on January 29th, 2014, specifically in Geology, QA/QC and Modeling
- Juana Galáz (Principal MYMA) was in charge of the environmental information utilized in Section 20 of this Technical Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

Cerro Maricunga gold deposit is located in Chile's III Region (Copiapo), in the Maricunga Mineral Belt, a well-known mining district that boasts over 70 million ounces of gold and hosts La Coipa and Maricunga (Ex Refugio) mines, as well as Volcan, Caspiche, Marte Lobo and Cerro Casale deposits. The region also boasts a trained mining workforce, support from experienced mining equipment suppliers, and excellent professional and technical consultants.

Cerro Maricunga site is surrounded by key mining infrastructure – including ports at Antofagasta and Coquimbo – all connected by the Pan-American Highway and the provincial road network. Chile's central power grid passes within 110 kilometers of the property.

The property is approximately 117 km (straight-line) north east of Copiapo City (III Region Capital), approximately 50 km west of the border with Argentina, along the western flanks of the Chilean Andes at a mean elevation of approximately 4,200 m. The project location is shown in Figure 4-1.

Copiapo is surrounded by the Atacama Desert and receives little rain (12 mm per year). The population of Copiapo was no more than 10,000 in 1903, 11,600 in 1907 and, as of 2013, there are over 150,000 inhabitants. Copiapo has a diversified economy, but mining is the largest economic activity.

Cerro Maricunga is centred at 7,013,000 N and 479,000 E; approximately 20 km south of Kinross Gold's La Coipa Au-Ag mine (currently on standby), 60 km north of Kinross's Maricunga (previously named Refugio) Gold Mine and 40 km north of Hochschild's Volcan Gold Project.

The Cerro Maricunga Project encompasses an area of 15,893 ha.

Figure 4-1 Location Map of the Cerro Maricunga Gold Project

4.2 Land Tenure

Under the mining laws of Chile, mining concessions can be held in perpetuity provided that the appropriate annual payments have been made. There is no requirement that the property be put into production within a specified time frame and there is no requirement to reduce concession sizes as the exploration process advances.

Payments to maintain concessions are made annually in March. The property payments, as made to date, will maintain the Cerro Maricunga property in good standing until October 2014. The total cost to maintain the mining and exploration concessions is estimated to be approximately Ch$ 28.347.681. Table 4-1 lists the mining concessions.

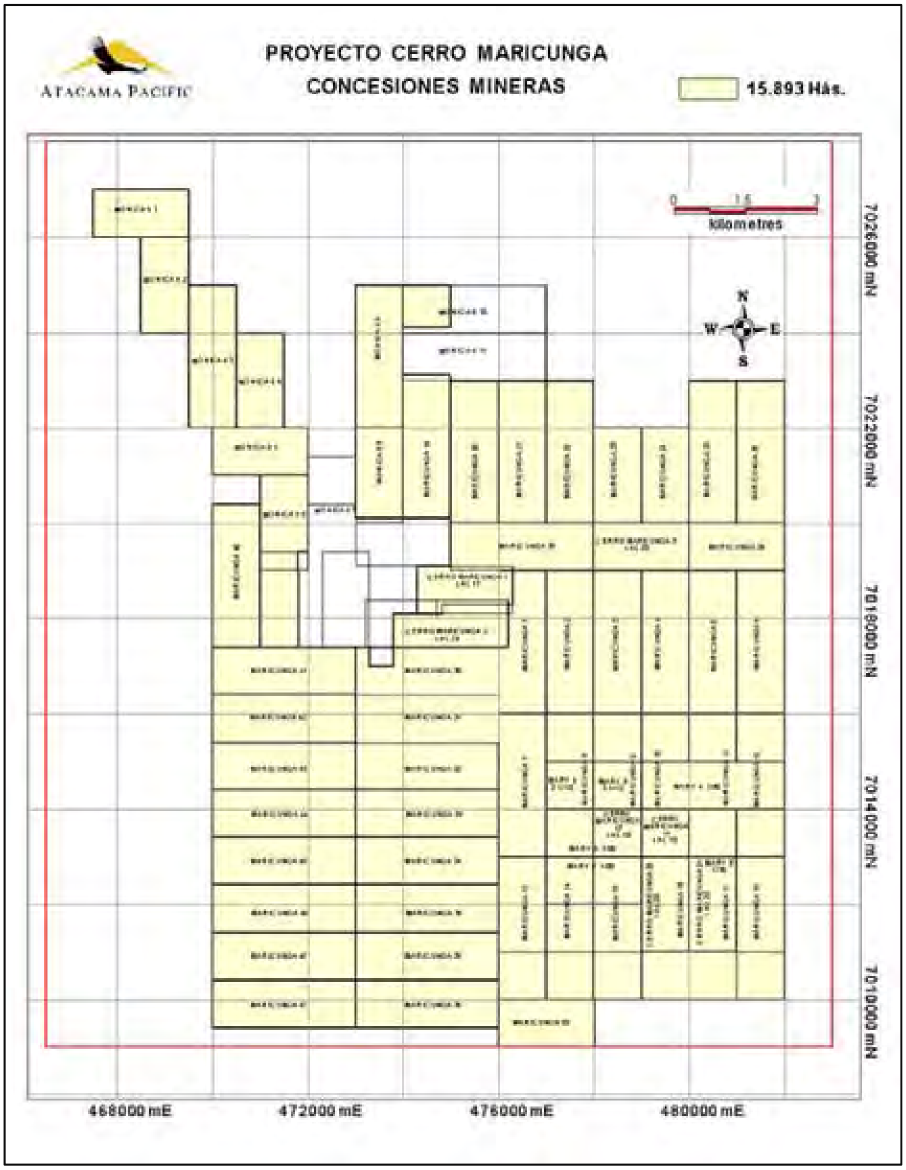


Figure 4-2 Cerro Maricunga Concession Map (source SBX)

Table 4-1 Maricunga Mining Concessions

Concession	Ha	Concession	Ha	Concession	Ha
Maricunga 1	300	Maricunga 29	300	Mónica II 5	200
Maricunga 2	300	Maricunga 30	300	Mónica II 6	200
Maricunga 3	300	Maricunga 31	300	Mónica II 7	100
Maricunga 4	300	Maricunga 32	300	Mónica II 8	200
Maricunga 5	300	Maricunga 33	300	Mónica II 9	300
Maricunga 6	300	Maricunga 34	300	Mónica II 10	300
Maricunga 7	300	Maricunga 35	300	Mónica II 11	300
Maricunga 8	300	Maricunga 36	300		
Maricunga 9	300	Maricunga 37	200	Cerro Maricunga 1 1/17	170
Maricunga 10	300	Maricunga 38	300	Cerro Maricunga 2 1/30	300
Maricunga 11	300	Maricunga 39	300	Cerro Maricunga 3 1/20	200
Maricunga 12	300	Maricunga 40	300	Cerro Maricunga 13 1/30	300
Maricunga 13	300	Maricunga 41	300	Cerro Maricunga 14 1/30	300
Maricunga 14	300	Maricunga 42	300	Cerro Maricunga 20 1/30	300
Maricunga 15	300	Maricunga 43	300	Cerro Maricunga 21 1/30	300
Maricunga 16	300	Maricunga 44	300		
Maricunga 17	300	Maricunga 45	300	Mary Segunda 2 1/10	100
Maricunga 18	300	Maricunga 46	300	Mary Segunda 3 1/10	100
Maricunga 19	300	Maricunga 47	300	Mary 4 1/30	300
Maricunga 20	300	Maricunga 48	300	Mary 5 1/20	200
Maricunga 21	300	Maricunga 49	300	Mary 7 1/20	200
Maricunga 22	300	Maricunga 50	300	Mary 8 1/30	300
Maricunga 23	200	Maricunga 51	300		
Maricunga 24	200			Monica 1 1/40	200
Maricunga 25	300	Mónica II 1	200		
Maricunga 26	300	Mónica II 2	200	Mary 6 1/30	300
Maricunga 27	300	Mónica II 3	300	Mary 9 1/40	200
Maricunga 28	300	Mónica II 4	200	Mary 10 1/30	300
Total	21,570				

Notes to Accompany Maricunga Mining Concessions:

- Total area presented does not consider overlapping concessions.

4.3 Environmental Liabilities

MYMA prepared and filed the background information required for the "Estudio de Impacto Ambiental" (Environmental Impact Study) on behalf of Atacama. The conclusions reached by MYMA are as summarized following:

- Cerro Maricunga mining project is fully feasible from the environmental sustainability viewpoint. The consultant does not foresee any serious shortfall at this stage of the project development

- All key environmental sustainability variables identified and analyzed (potential environmental impacts) in this report can be fully addressed and there are measures in place to effectively manage them
- In connection with potential socio-economic impacts, it is foreseen that the mining operation will not cause a significant alteration in the lifestyle or the customs of the inhabitants or their dwellings; no cultural or anthropological changes are foreseen in the human groups indicated above, and in turn it is anticipated as a positive effect the increase in the supply of local manpower
- A successful environmental permitting is closely linked to the availability of relevant (project design) and essential information (baseline studies and evaluation of environmental impacts)
- Atacama should continue discussing the Project with the Environmental Authorities and the neighboring communities to reinforce the relationship and to facilitate the communication during the environmental evaluation of the project. Although there are no indigenous communities in the area where the Project will be developed, it is fundamental to maintain good relationships with the neighboring communities to enhance communications and to facilitate the environmental permitting of the project.

4.4 Permits Acquired

Drilling has been permitted through the approved DIA's (Environmental Impact Declaration) submitted by Atacama. No additional permits, in particular an Environmental Impact Assessment, will be required for exploration activities at Cerro Maricunga. Additional permits will be required for the exploitation of the Cerro Maricunga orebody.

4.5 Ownership, Royalties and Other Payments

The Cerro Maricunga property is 100%-owned by Atacama and is subject to no third party royalties, back-in rights or payments.

5 ACCESS, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Climate

Cerro Maricunga ore deposit is located at 4,700 metres above sea level, in the west side of central zone of the Chilean Andes Mountains. Climate is extremely dry and only 200 to 300 millimeters per year precipitations and consisting largely of snow during winter South American months of June through mid-September. Some sporadic, rain storms of short duration occur during the summer and autumn months (January to May). Strong winds may develop during these periods due the high altitude.

The average annual temperature is of the order of 2°Celsius, and ranges between -30°Celsius at night in winter to 20°Celsius during the day in summer.

Evaporation from surface water and soils varies between 1,500 to 2,000 mm/yr (Bartlett, et. al., 2004) resulting in the extremely arid conditions observed in the various areas.

Local wildlife is sparse although vicuñas may be encountered.

5.2 Vegetation and Wildlife occasionally

See section 20.3.2 and 20.3.3 of this report.

5.3 Physiography

The planned mine, heap leach pad and process plant are to be located approximately 40 km west of the Argentine border in the high Andes and at elevations of between 3,800 and 5,000 masl. The principal topographic features are the result of the combination of horst and graben block tectonics in the Cordillera Occidental and the Cenozoic to Recent volcanism that has produced the various strato volcanoes and dome complexes, which host the alteration/mineralization that has been identified to date. The topography within the property is almost entirely volcanic in nature and consists of broad open areas of moderate relief and prominent ridges with limited cliff zones of exposed bedrock.

5.4 Access

Copiapo is the closest city to Cerro Maricunga Project, located 800 km of the capital Santiago, accessible by national road and several commercial flights.

The Cerro Maricunga gold project is accessed from the centre of Copiapo by a combination of paved highway, two-lane asphalted road (just before the La Coipa Mine), and then by maintained single track dirt roads. Access to the property by pickup (standard or 4-wheel drive) takes approximately 2.5 hours (140 km) from the centre of Copiapo. Directions to the property are as follows: from Copiapo, travel southeast approximately 10 km out of the centre of town towards the ENAMI Paipote smelter, and then turning north on the Inca de Oro road for 15 km and then turning off NE along the salt-paved road to Paso de San Francisco for 104 km (Road Ch 31), and to the turn-off for La Coipa Mine. At approximately 800 m south east of the La Coipa Mine turn-off, swing right to the south west and follow the dirt road which follows the Quebrada (drainage) Pelada gulch for 24 km to reach the project site. It is considered the construction of an access road of 24 km for project Cerro Maricunga.

Figure 5-1 Project access from Copiapo (Source: Alquimia)

5.5 Local Resources and Infrastructure

There are no significant population centres or infrastructure in the immediate vicinity of the Cerro Maricunga project. There are a number of indigenous families who raise crops and livestock in specific areas of the valleys that drain the region. Chile is an advanced country in terms of mining technology and infrastructure and supplies high quality mining professionals to other countries. Copiapo, the nearest major city to the Cerro Maricunga site is about 140 km by road. Copiapo has an approximate population of 150,000 people. Experienced mine and plant personnel should be easily sourced from Copiapo, or elsewhere in Chile where a generally well trained and experienced workforce exists. Furthermore, Copiapo is a well-established support and logistics centre for mining activities in the region.

All workforce transport is by private vehicles. Existing mines and exploration projects house their workers in fully serviced camps with workers travelling in and out on a roster system. Atacama will follow this procedure during construction and operation of Cerro Maricunga project.

Electrical energy required for the project will be obtained from the Central Interconnected System (SIC) of Chile by a power line of approximately 110 km. The high voltage transmission line (110 kV) will be the responsibility of Atacama. The construction and operation will be arranged with third parties. The SIC is highly reliable and generally does not require significant backup generators on site with the exception for critical areas and camps.

There is no water supply in the vicinity of the project and water resource is scarce in the area. Cerro Maricunga has acquired a water supply by sourcing industrial water from largest supplier of water in the city of Copiapó.

Cerro Maricunga is a green field site, and thus existing site infrastructure is limited to an exploration camp and roads.

6 HISTORY

6.1 Cerro Maricunga History

Atacama was founded in 2008 as an exploration company focused on the acquisition and subsequent exploration of the Cerro Maricunga Project, which it acquired on January 26, 2010 when it purchased Minera Atacama Pacific Gold Chile Limitada from SBX Asesorias E Inversiones Limitida ("SBX"). The Company is run by a team of exploration professionals with an acknowledged track record of significant gold discoveries and extensive public market expertise. Management has a long history of exploration experience in Chile; a country which combines all the necessary qualities that make it one of the top prospective jurisdictions in the world for the discovery of major mineral deposits and their subsequent timely development.

Preliminary prospecting conducted in the early 1980's identified a "possible high level, high sulfidation system" in the Ojo de Maricunga strato-volcano containing silica (opaline)-clay altered pyritic breccias, tuff and quartz-feldspar porphyry.

In December 2007, SBX constructed access roads and conducted sampling trenches and preliminarily mapping at 1:25.000 scale detecting classical Maricunga style black banded veinlets along 2.5 km along a NW trend in the intrusive complex in the central part of the volcanic system. Trench and outcrops sampling in the area detected anomalous gold grades ranging from 0.2 ppm to 3 ppm.

In early 2008, Minera Newcrest Chile Ltda ("MNCL") conducted a preliminary evaluation of the property during which time MNCL took 325 samples which confirmed the presence of elevated gold mineralization along a NW-SE trending zone.

In 2008, Gold Fields (GFC) entered into an exploration/joint venture/option agreement with SBX and conducted trenching, mapping and channel sampling. During the following exploration season (2008 – 2009), GFC completed an induced potential/resistivity and magnetic survey. The work performed by GFC confirmed that Maricunga was a potential gold target, and that the property warranted additional exploration including additional mapping, trenching/sampling and drilling.

6.2 Recent Exploration at Cerro Maricunga

Extensive surface trenching and sampling along with geophysical surveys, metallurgical testing and an eight hole Phase I drill program (2,142 m) were funded privately from 2007 through to early 2010 with the goal of de-risking the project prior to obtaining a public listing. During years 2008 - 2010 the Maricunga deposit and surroundings areas were extensively mapped and sampled (rock chips and trenches). Local mapping, at a scale of 1:2,500, centered in the deposit, was carried out by A. Hodgkin. Surroundings, totaling 163 km^2, were mapped at scales of 1:10,000 and 1:25,000 by Dr. A. Dietrich. Petrographical studies have been performed by Dr. P. Cornejo on approximately 50 samples taken from the field and drill core.

In October 2010, Atacama commenced a Phase II drill program with two drill rigs to follow up on the positive drill results from the private Phase I program. As the Phase II campaign progressed and the results continued to provide support for a possible large oxide-associated gold deposit, two additional drill rigs were mobilized onto the property and by the end of April 2011; 31,438 metres of drilling had been completed in 82 holes.

Exploration work including detailed 1:500 mapping of 27 km of roads cuts and trenches drilling and trenching were carried out during 2011 - 2012 seasons.

During 2011, initial metallurgical testing results indicated that Cerro Maricunga deposit was amenable to heap leach processing. Eleven column tests and 36 bottle roll tests confirmed earlier results with gold recoveries in the range of 80% at 19 to 25 mm (3/4 to 1 inch) crush. A column test completed on coarse crush size material returned positive results with gold recovery of 78% at a 50 mm (2 inch) crush.

During the Phase III program (2011-2012 exploration season), Atacama completed 45,942 m of drilling on the Cerro Maricunga deposit largely defining the extents of the mineralization. Surface trenching was completed and metallurgical testing continuing.

A Phase IV drilling program (2012 - 2013 season) was completed in May 2013 with 25,427 m of completed. The majority of the drill program was dedicated to infill drilling. Metallurgical testing continued.

In January 2013, Company issued a detailed preliminary economic assessment (PEA) establishing the engineering parameters and demonstrating the potential economic viability of the large Cerro Maricunga gold deposit.

Since 2011, Atacama has prepared and published a number of resource updates which have been appropriately disclosed.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting

Basement rocks in the Maricunga Belt comprise a series of volcanic-plutonic-sedimentary arcs of Mesozoic-Cenozoic age which are associated with the subduction of the Pacific Plate beneath the South American Plate.

High and low angle reverse faulting occurred as a result of compression induced by subduction zone flattening. Major alignments in the Maricunga Belt have northwest strikes. This structural pattern is observed in faults as well as alteration and mineralization components.

A large volcanic caldera complex developed over basement rock of Paleozoic-Triassic and Mesozoic-Early Tertiary age. Initial stages of the caldera complex were associated with the development of large andesitic (dacitic) strato-volcanoes of Oligocene-Miocene age (23-14 Ma - based on K/Ar dating) located mainly west of Salar de Maricunga. The Miocene volcanic rock and associated alteration and mineralization consists of two partly overlapping sub-belts; Western sub-belt of early Miocene (24-20 Ma) age and the Eastern sub-belt of Miocene age (14-13 Ma).

Several hydrothermal systems developed during period resulting in the formation of various currently known deposits, such as Marte-Lobo, La Pepa, La Coipa, El Volcan, Maricunga (previously Refugio), which are grouped in the so called Maricunga Belt, a world class metallogenic district with resources in the order of 70 million ounces gold (Figure 7-1).

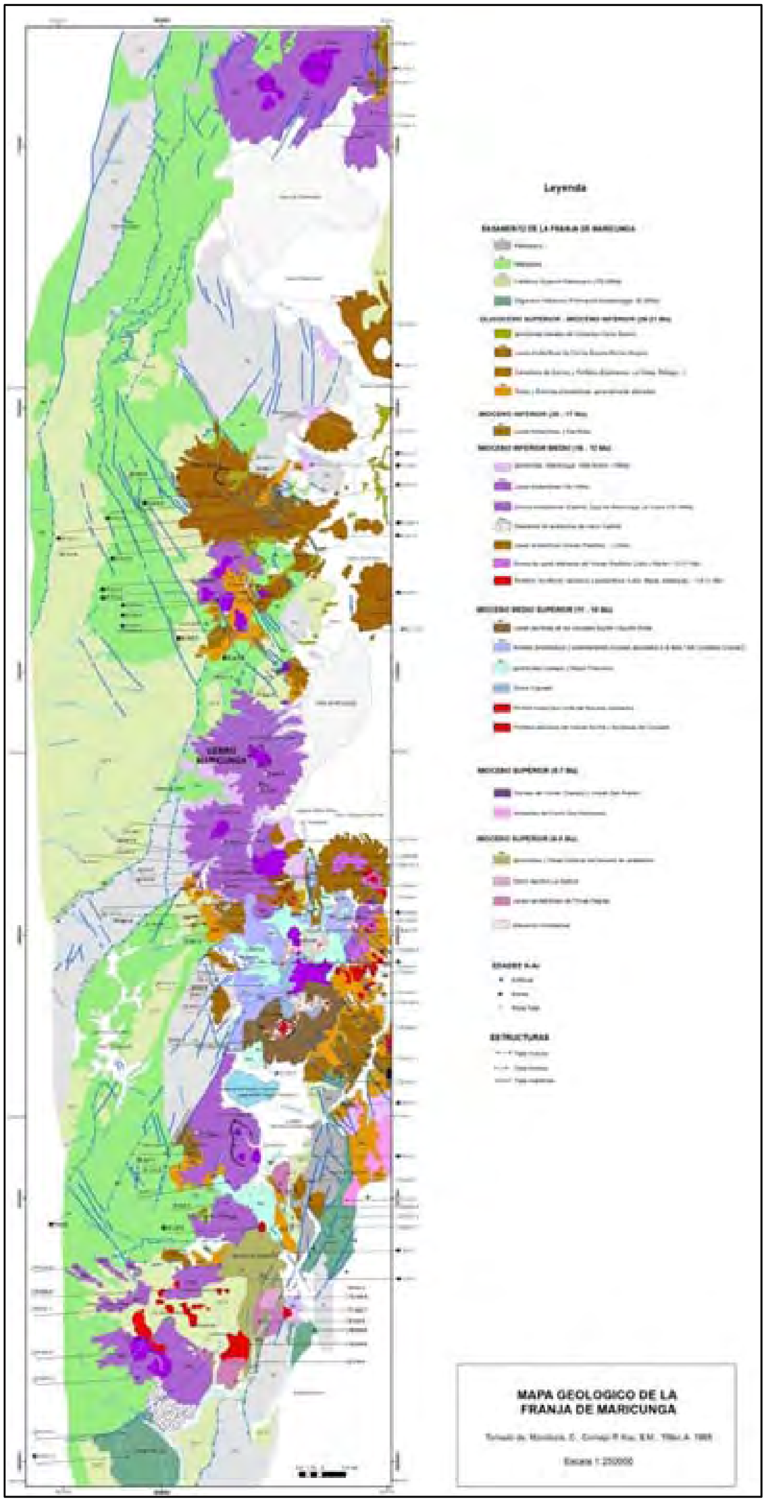


Figure 7-1 Regional Geology and Deposits of the Maricunga (Mpodozis et al 1995)

Hydrothermal and solfataric activity in the Maricunga Belt resulted in the generation of a large number of hydrothermal alteration and precious metals mineralization zones grading from very high epitermal systems characterized by extreme argillized and silicified rocks with important amounts of alunite and sulphur, such as Ojo de Agua sector in the El Volcán deposit, to deeper expressions of epithermal systems such as in the Marte Lobo System. The latter present clear characteristics of Au-Cu porphyry-type related to deeper seated (often telescoped) K-silicate alteration, which are preserved at Maricunga (Refugio), and Aldebarán (Cerro Casale). The deposits are typically overprinted and obliterated by sericite-clay-chlorite assemblages of intermediate argillic type.

Subsequent explosive extrusion of large volumes of ignimbrites with the resultant development of craters/calderas type structures covered and truncated hydrothermally altered rocks in many areas and resulted in the formation of a volcanic plateau to the west. These ignimbrites are considered to be coeval with the San Andres ignimbrites and extend to as far as a La Puerta and Puquios to the north, where over 100 metres of weakly consolidated volcanic ash are preserved overlying Miocene terrace gravels. Resurgence of sub-volcanic intrusive magmas (ca. 5-4 Ma) may have raised the volcanic edifice(s) culminating in the formation of the 6,052 metres high Volcan Copiapo.

7.2 Local Geology

Mineralization at Cerro Maricunga is associated with shallow (sub-volcanic) dacitic and andesitic intrusive domes and breccia (phreatic, phreatomagmatic and magmatic) complexes located in the nuclei of an eroded strato-volcano, which is composed of very extensively developed mid-Miocene (15-17 Ma) pyroclastic volcanic rock represented by volcanic breccia, pyroclastic flows, lapilli and crystal tuff, and dacitic-andesite lava flows, and very locally by tuffaceous arenite, volcano-clastic conglomerate and laharic deposits.

The Tertiary volcanic sequence is developed un-conformably on the volcano-clastic sediment and conglomerate, shale and limestone of Upper Triassic-Lower Jurassic age of the El Mono Formation.

7.2.1 Structures

The Cerro Maricunga deposit is located along a major NW structural trend, consisting of faults with pre and post mineral activity.

Mineralization in the deposit is controlled by NW structures mainly. Three main systems have been defined as depicted in Figure 7-2.

<u>NW Fault System:</u> Three main sub-parallel SE-NW striking faults cross cut the northern portion of Phoenix and Lynx zones. Dips were interpreted as being vertical to sub-vertical. Movements along strike partially control the location intrusions and mineralization.

<u>NNE Fault System:</u> NNE-striking faults cross cut the southern end of the Crux zone. Dips and movements along strike are similar to those observed in the NW fault system.

EW Structures: Intrusions, as well as mineralization, show a consistent EW / vertical oriented structural pattern within the dacitic intrusive complex (magenta bodies shown in Figure 7-2). Mineralization within these structures is found within multi-directional black banded veinlet sets with widths ranging from 0.2 mm to 10 cm.

North-south contacts have been observed occasionally at surface, which are probably associated original regional structures.

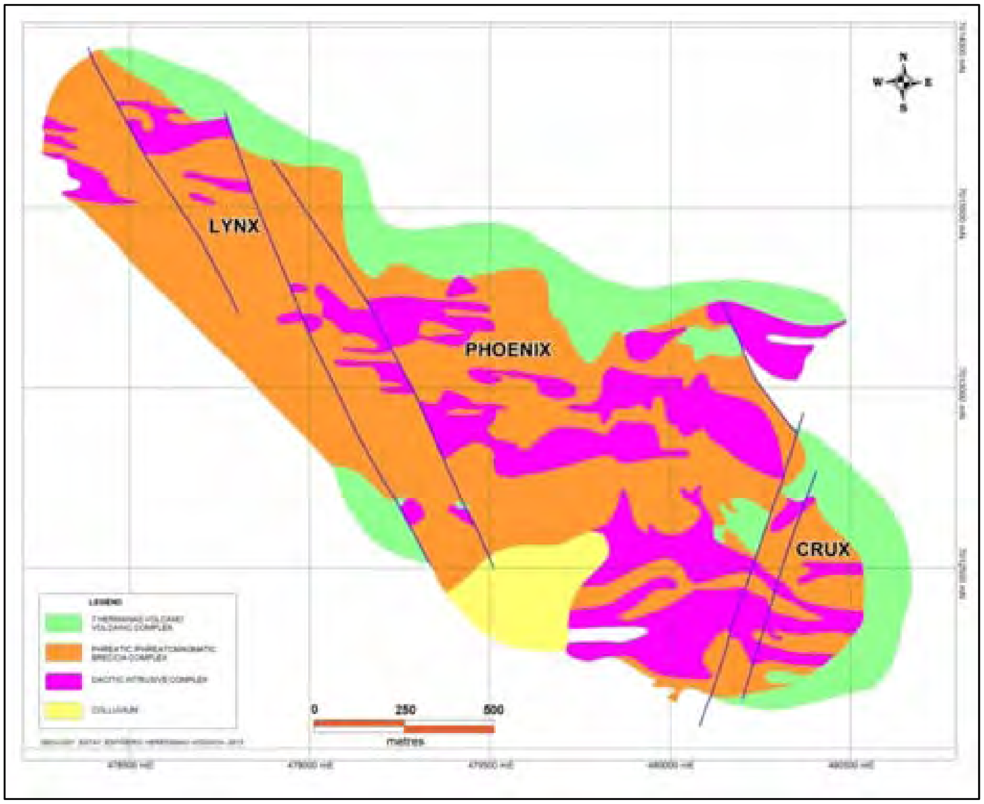


Figure 7-2 Summarized Structural Geology of the Central Part of Cerro Maricunga Deposit (Cepeda, Estay 2014)

7.2.2 Lithology

Gold mineralization at Cerro Maricunga is hosted in a porphyry and breccia complex as shown in Figure 7-3.

Porphyry Units

Daciandesite Porphyry (DAP): DAP consists of feldspar (20-30%) and hornblende (10-15%) phenocryst with variable and minor amounts of biotite (up to 10%) and quartz (up to 5%). Petrographic studies carried out by Cornejo indicate that hornblende is often re-absorbed and transformed to micro granular aggregates of clinopyroxene with magnetite over the remnant crystals altered to hydro-biotite and smectite. The groundmass has a felting fabric, with fresh plagioclase and clinopyroxene microcrystals associated to abundant magnetite. Small degassing cavities contain quartz (0.2-0.5mm) and carbonates (ankerite).

Dacite Porphyry (PDA): This unit consists mainly of coarse plagioclase phenocrysts (up to 15 mm) with minor hornblende, and occasional rounded quartz eyes, and is transitional to rhyodacite quartz-eye porphyry. Microscopy shows a weakly fluidal groundmass, plagioclase (0.1-0.3mm), abundant re-absorbed amphibole, biotite and smectite, and interstitial fine quartz, feldspar, apatite, smectite and sulfides (0.05-0.3mm) or magnetite.

Rhyodacite Quartz-eye Porphyry (QEP): QEP consists medium-size feldspar (25%) and fine to medium-size hornblende>biotite (10-15%) and quartz (10%). Quartz-eyes are corroded, medium to coarse-size, with diameters that may reach 10mm. These are often “smoky” and stained with iron oxide along micro fractures.

Breccia Units

Magmatic Breccia and Auto-Breccia (BMA): This breccia unit consists of crystalline matrix with mingling textures of two magmas; pale-leucocratic and dark grey-melanocratic magma phases. This breccia type is formed during cooling and fracturing of a magma body. Clasts are hazy. The matrix is slightly finer-grained than clasts.

Phreatomagmatic Breccia: Phreatomagmatic breccia is the result of the interaction between magma and an external water source. They are characterized by the presence of juvenile clasts (up to 30%), which are irregularly shaped, often displaying concave embayment towards the outside, which are unlikely the result of mechanical transport and abrasion (Dietrich, 2010). Their composition ranges from andesite to dacite. The matrix is composed of rock flour and sometimes scarce crystalline components. Phreatomagmatic breccia occurs as irregular bodies or dikes.

Phreatic Breccia: Phreatic breccia are monomictic or polymictic, matrix supported clastic rocks. The matrix consists of rock flour matrix, minor tuffisite, and silica or gypsum cement. The polymictic variety contains sub-rounded to angular clasts of a wide spectrum of lithology types; dacite, dacite-andesite, andesite, early breccia, clasts with banded veinlets, etc. Monomictic breccia is related to the contacts with the country rocks, and their clasts are generally angular shaped. Dietrich (2010) noted that the phreatic breccia overlaps with the characteristics of the effusive-pyroclastic dacitic country rock sequence.

Hydrothermal Breccia (BHT): This unit comprises breccia with hydrothermal matrix-cement and angular clasts of host-rock composition (“crackle breccia”). The matrix consists of hydrothermal components such as magnetite, silica, chlorite, gypsum and clay.

Tuffisite Breccia Dikes: These often sub-vertical fluidal structures cross-cut the majority of lithological units observed in Cerro Maricunga and; therefore, are interpreted as a late-stage phase. Tuffisite breccia dikes are matrix-supported and have rounded to sub-rounded clasts and often present stratification and cross banded textures .Widths normally range from 1to 20 cm. but metric widths are not uncommon.

Occasionally, tuffisite dikes are rimmed by banded veinlets, minor events of veining, and/or crosscut by minor veining events.

Andesite Dikes Sills and Plugs (AND): Andesite sub-vertical dikes and occasional plugs are dark grey to green and fine-grained. Widths range between 1 to 30 m. Andesitic units generally cross-cut the breccia and porphyry unitsintruding through contact surfaces and faults, suggesting a fairly late origin.

Occasionally, trace pyrite mineralization is observed in andesite dikes.

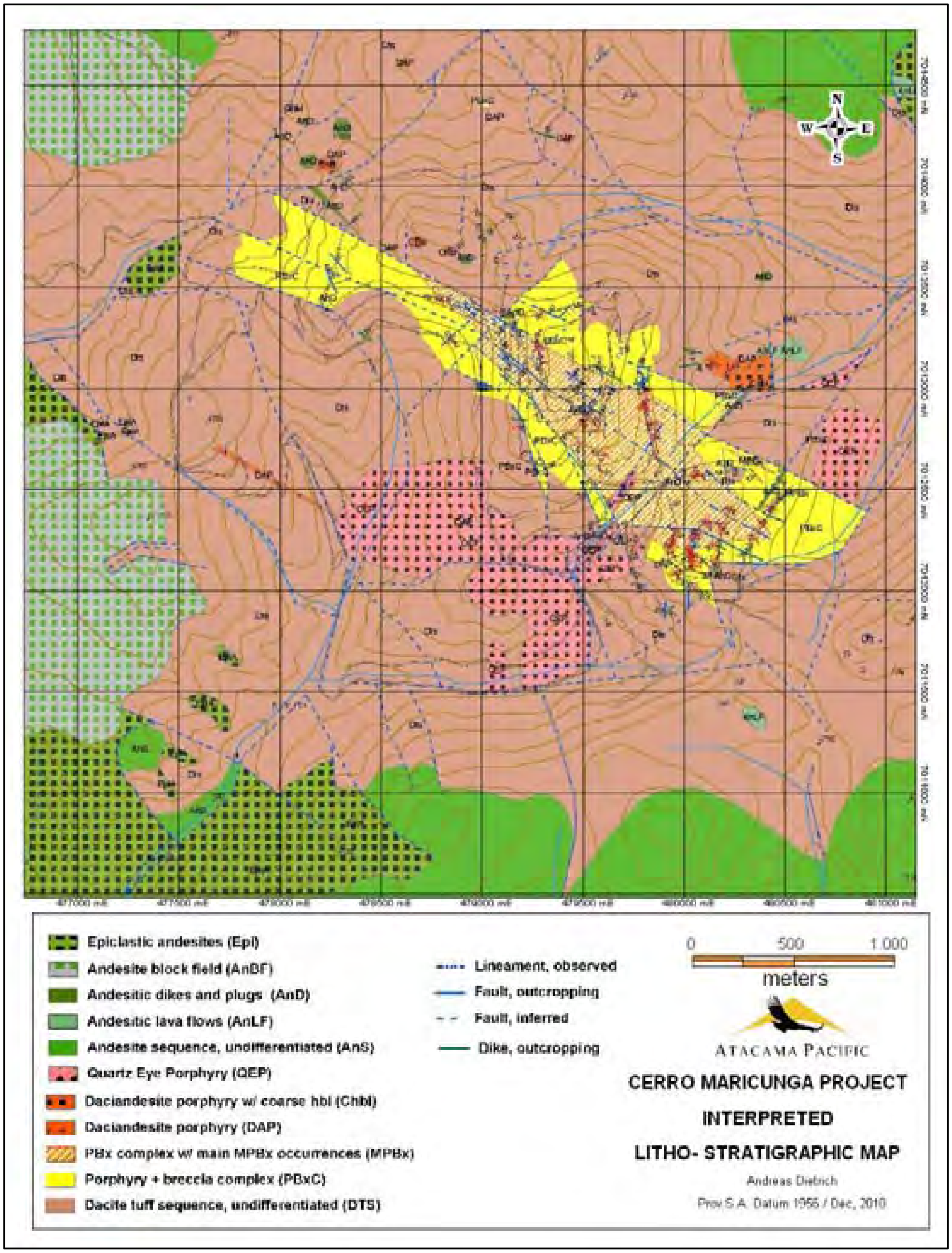


Figure 7-3 Geology of Cerro Maricunga (Dietrich, 2010)

7.3 Alteration and Mineralization

7.3.1 Alteration

Early alteration at Cerro Maricunga is characterized by a change from high temperature hydrous (amphibole, biotite) to anhydrous products (pyroxene, plagioclase, Fe-Ti oxides) under decompression. The later alteration (intermediate argillic alteration with smectite/illite) is the result of typical acidification with falling temperature due to the breaking of bonds in the complexes of chlorides, carbonates and sulfates (minor). At Cerro Maricunga, this type of alteration is quite weak, as is shown by the preservation of pyroxenes near the center of the area; pyroxene being notoriously unstable under hydrous acidic conditions.

Figure 7-4 is a microphotograph of a porphyry sample with pervasive argillic and illite-smectite-hydrobiotite alteration and a silicified mass (located in the left portion of the picture). At least 25% of the groundmass consists of devitrified glass and 20% of secondary clinopyroxene of the diopside - hedenbergite series.

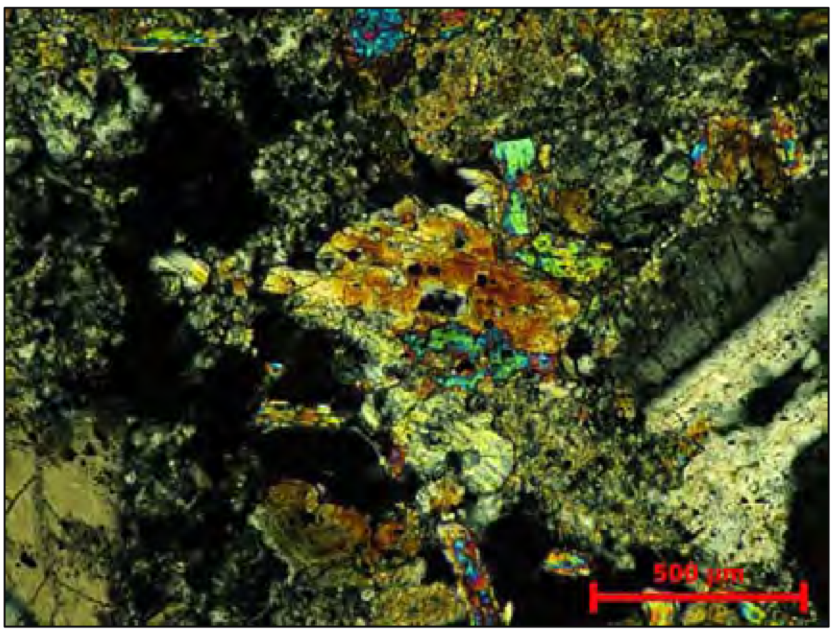


Figure 7-4 Microphotograph – Altered Porphyry

7.3.2 Mineralization

Microscopy studies indicate that gold mineralization at Cerro Maricunga primarily occurs within black and grey banded veinlets (BBV and GBV) in porphyry and breccia, and secondarily within early chlorite-magnetite-quartz veinlets. A schematic cross section of the deposit depicting location of gold occurrences is shown Figure 7-5.

As can be seen gold mineralization may be encountered in phreatomagmatic breccia, surrounding hydrothermal breccia, in dacite porphyry and also surrounding andesitic dikes and plugs.

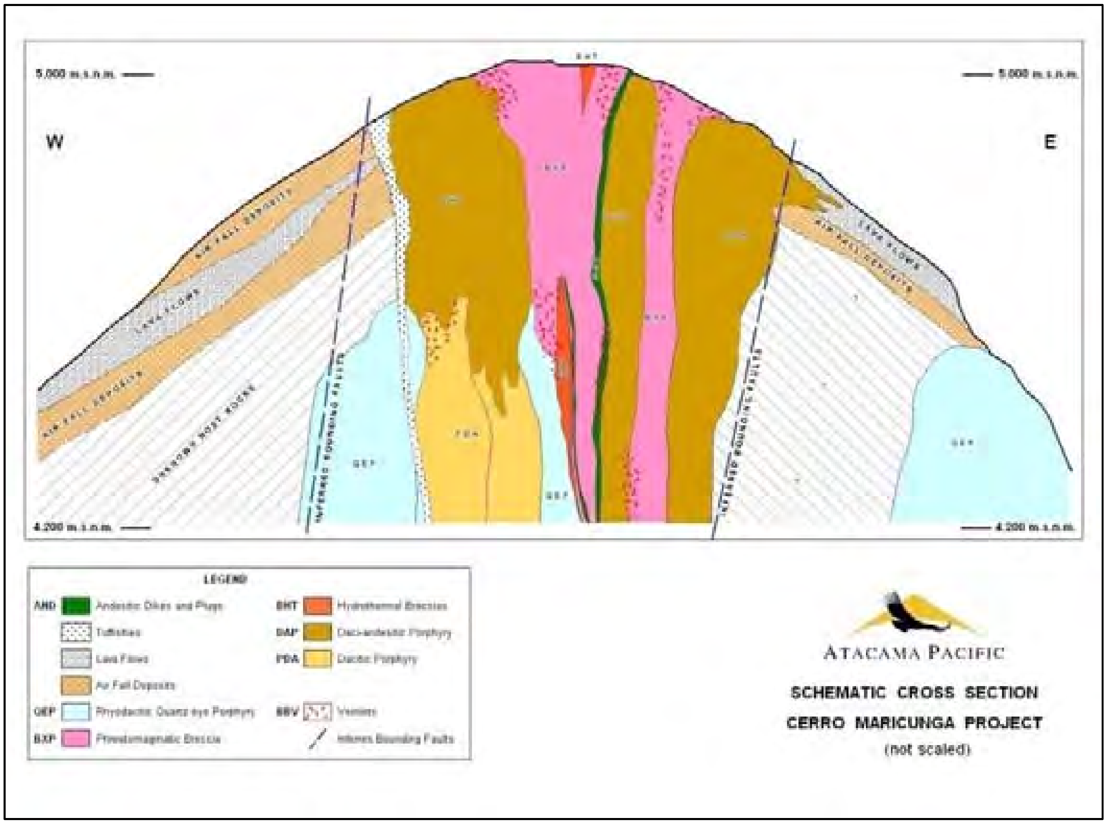


Figure 7-5 Schematic Cross Section - Cerro Maricunga Project (Villegas and Díaz, 2011)

Black veinlets consist of tiny gas inclusions, very fine magnetite and trace sulphide Images of black veinlets are shown in Figure 7-6 and Figure 7-7. Magnetite is variably replaced by hematite due to incipient martitization. Sulfides are relatively rare throughout the deposit, typically accounting for < 0.1 wt% and are primarily pyrite. Few grains of chalcopyrite, bornite, chalcocite and covellite have been observed.

Figure 7-6 Black Banded Quartz Veinlet

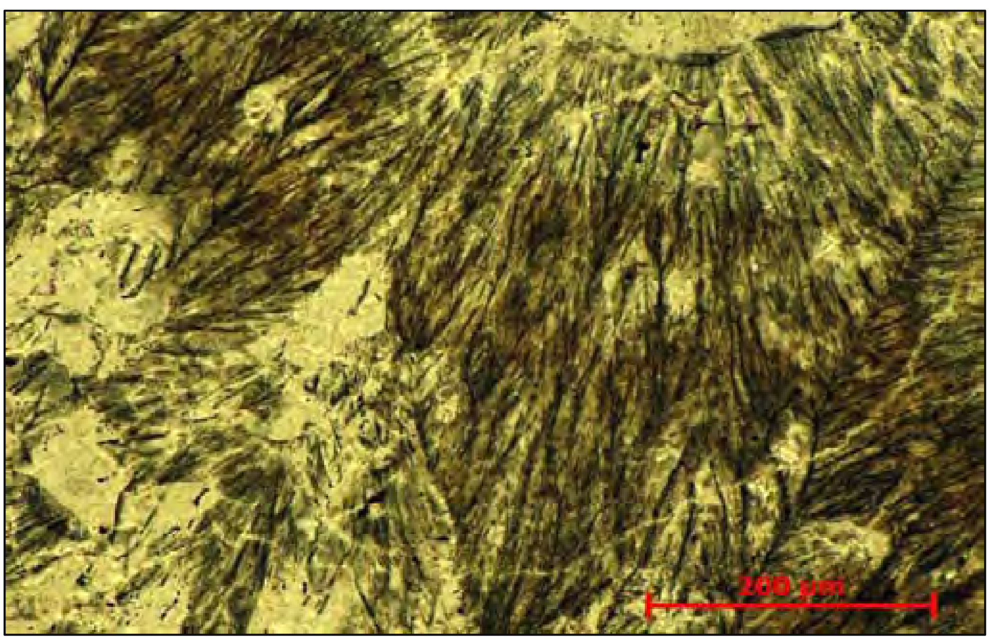


Figure 7-7 Feathery boiling textures in a banded quartz veinlet. The dark color is due to an enormous amount of vapor fluid inclusions and tiny magnetite crystals.(Lohmeier,Lehman 2012)

7.3.3 Gold Form and Carriers

Mineralogical data presented herein was prepared by AMTEL[1].

Gold deportment studies were carried out in three composites of the Phoenix zone with results indicating that gold occurs in two forms: native gold and submicroscopic gold in FeOx.

- Native gold is observed as free particles; attached to FeOx, to rock or both; and enclosed (in rock and/or FeOx) as shown in Figure 7-8
- Free particles of gold within the composites are very small with 75-95% smaller than 10 µm. Gold composition is of high purity (on average >99% Au)
- Submicroscopic gold (noted as “refractory”) is carried mainly by goethite followed by hematite (referred collectively as FeOx).

Quantitative data obtained from deportment studies in composites are summarized in Table 7-1.

Table 7-1 Gold Carriers in Cerro Maricunga Composites

Composite-ID	Head-Au-g/t	GOLD GRAINS (%)		
		Exposed Attached	Enclosed CN-Able	Refractory
4	0.287	81	14	5
5	0.466	79	17	4
6	0.505	74	13	13

Native Gold

- Carriers of native gold; free gold – attached (collectively called “exposed”) – enclosed
- The composition of gold grains have high purity (on average >99% Au)
- Gold grains are very small with 75-95% smaller than 10 µm.

Submicroscopic Gold

- Submicroscopic gold is carried mainly by goethite followed by hematite (referred collectively as FeOx).

[1] Deportment of Gold in Maricunga Low & Medium Grade Ore Composites for Atacama Pacific Gold Corporation, January 10th, 2012.

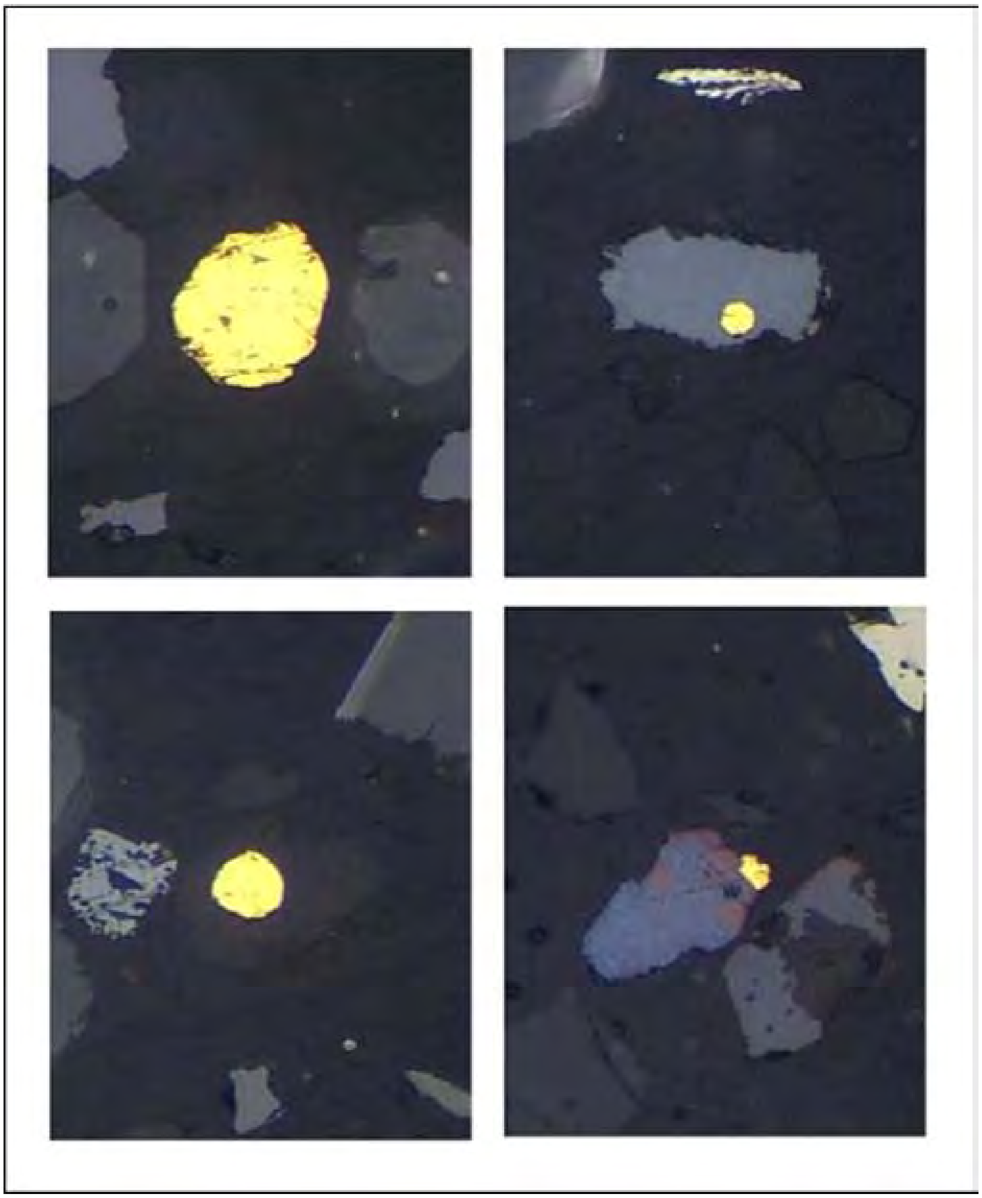


Figure 7-8 Top Left-Free gold; Top Right: Gold grain enclosed in hematite; Bottom-Left: Gold grain enclosed in rock; Bottom-Right: Gold grain attached Cu sulfide (digenite and bornite)

8 DEPOSIT TYPES

Cerro Maricunga has characteristics similar to other known deposits which occur within the Maricunga Mineral Belt of Chile. The deposit type being explored for is a porphyry gold deposit developed in, and associated with, Miocene domal intrusives. These characteristics can include mineralization/alteration types which appear to be intimately associated with, or occur below, high level, high sulfidation epithermal mineralizing systems developed in variably eroded and collapsed Oligocene-Upper Miocene stratovolcanoes and within recurrent intrusive dacitic domes. Hydrothermal and phreatic breccias are frequently developed flanking and transecting (and below the steam heated zones) the domal intrusives and most commonly at fault intersections and/or zones of dilation.

9 EXPLORATION

During years 2008 - 2011 the Cerro Maricunga deposit and surroundings were extensively mapped and sampled (rock chips and trenches).

Initial detailed exploration mapping was carried out by Atacama during the 2007 - 2008 exploration seasons. Approximately 30 km^2 of the Cerro Maricunga property, including the deposit area, was mapped at a 1:25,000 scale by A. Cepeda. In 2011, detailed mapping at a scale of 1:2,500, centered on the deposit area, was carried out by A. Hodgkin and 163 km^2 around the deposit was mapped at a scale of 1:25,000 by Dr. A. Dietrich.

Petrographical studies have been performed by Dr. P. Cornejo on approximately 50 samples taken from the field and drill core.

Geophysical surveys, both ground magnetic and induced polarization, were completed by the company over the Cerro Maricunga property. The survey results are summarized on the September 29th, 2010 and October 7th, 2011 technical reports appropriately filed by Atacama.

Four phases of drilling have been completed on the property. The results from the first three phases of the drill program have been summarized in appropriately filed Cerro Maricunga technical reports dated September 29th, 2010, October 7th, 2011, November 9th, 2012 and March 15th, 2013. The data from the fourth phase of drilling, largely infill, has been incorporated in the resource estimate which formed the basis for the current reserve estimate provided in this report.

In addition to the drilling program, 26,979 m of surface trenching and associated rock chip sampling at 5 m intervals (5,398 samples) were completed during the first three phase of exploration, the results of which have been disclosed in the aforementioned technical reports.

10 DRILLING

The first drilling campaign at Cerro Maricunga, which consisted of 8 exploratory drill holes, was carried out in 2010 (Phase I). Thereafter, three additional campaigns were performed in seasonal basis, totalling 320 drill holes. The number and type of drill holes bored and metres, by drill phase is shown in Table 10-1.

A total of 104,948.2 m were drilled combining reverse circulation and diamond drilling techniques. Diamond drilling amounted to 28% of total metres drilled (29,534 metres), while reverse circulation drilling totaled 75,414 m.

Drilling was carried out along 50 metres spaced NE sections, approximately perpendicular to the NW mineralization trend of three major ore bodies: Crux, Phoenix and Lynx. An additional minor orebody was discovered in the south west portion of Phoenix (Pollux). Drill hole locations as well as mineralized zones are shown in Figure 10-1.

Table 10-1 Number of Holes and metres Drilled per Phase

Phase	I	II	III	IV	Total
Year	2010	2011	2012	2013	2010 - 2013
N° Drillholes	8	82	130	100	320
Reverse Circulation (m)	1,422.0	24,564.0	31,584.0	17,844.0	75,414.0
Diamond Drilling (m)	720.4	6,873.7	14,357.5	7,582.6	29,534.2

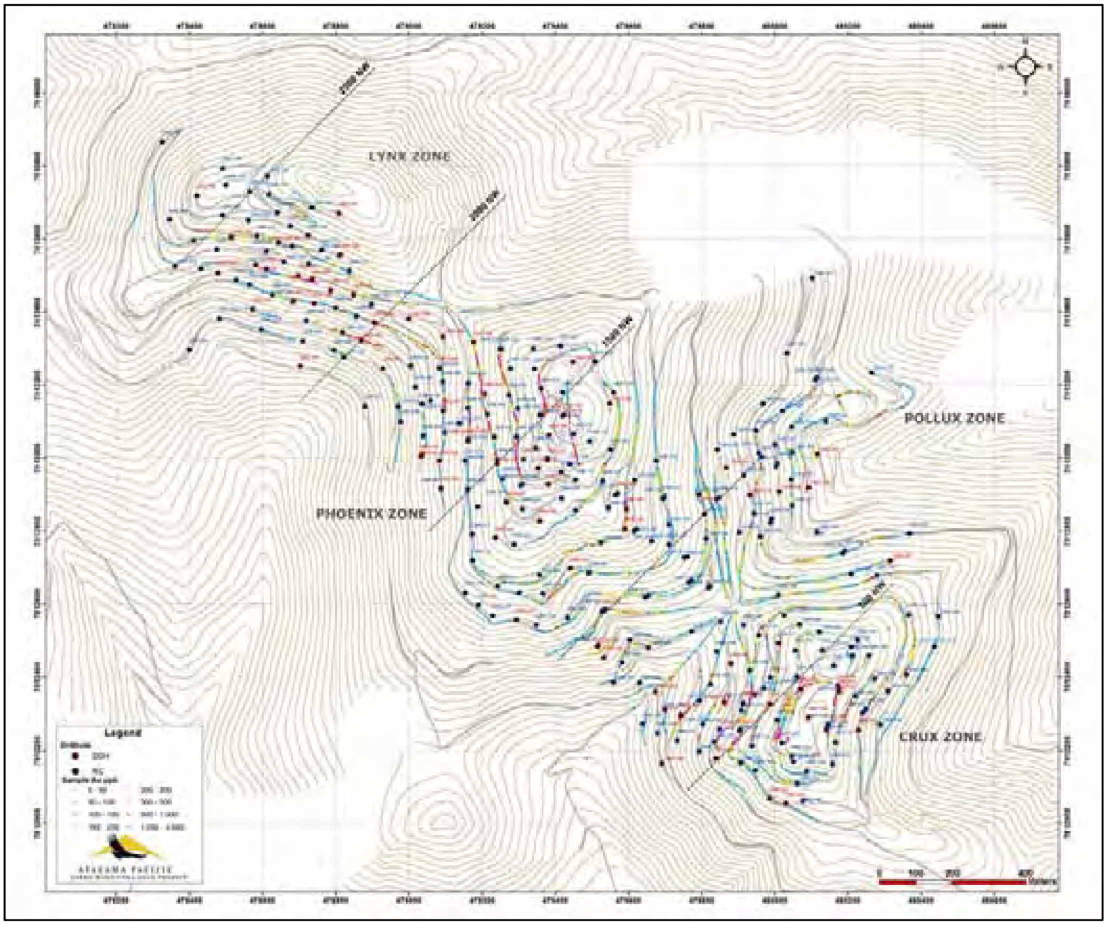


Figure 10-1 Drill Hole Location and Mineralized Zones

A total of 42,542.8 metres drilled fall within the mineralized (modeled) ore bodies; that is 41% of total metres drilled along the deposit. The percentage of total metres within Crux, Phoenix (including Pollux) and Lynx are 26%, 55% and 19% respectively. Further details are shown in Table 10-2.

Table 10-2 Cerro Maricunga Drilling Phases – Metres Drilled & Metres Assayed

Zone	Phase	DDH HOLES			RC HOLES			DDH + RC HOLES		
		Drilled (m)	Assayed (m)	Not Assayed (m)	Drilled (m)	Assayed (m)	Not Assayed (m)	Drilled (m)	Assayed (m)	Not Assayed (m)
Crux	I	202.0	202.0	0.0	542.0	542.0	0.0	744.0	744.0	0.0
	II	1,068.0	1,068.0	0.0	452.0	452.0	0.0	1,520.0	1,520.0	0.0
	III	1,482.0	1,482.0	0.0	4,700.0	4,700.0	0.0	6,182.0	6,182.0	0.0
	IV	362.0	362.0	0.0	2,086.0	2,084.0	2.0	2,448.0	2,446.0	2.0
	Sub-Total	3,114.0	3,114.0	0.0	7,780.0	7,778.0	2.0	10,894.0	10,892.0	2.0
Phoenix	I	263.1	263.1	0.0	564.0	564.0	0.0	827.1	827.1	0.0
	II	1,069.4	1,069.4	0.0	7,966.0	7,964.0	2.0	9,035.4	9,033.4	2.0
	III	3,369.3	3,369.3	0.0	3,894.0	3,890.0	4.0	7,263.3	7,259.3	4.0
	IV	2,440.0	2,440.0	0.0	3,880.0	3,874.0	6.0	6,320.0	6,314.0	6.0
	Sub-Total	7,141.8	7,141.8	0.0	16,304.0	16,292.0	12.0	23,445.8	23,433.8	12.0
Lynx	I	161.9	161.9	0.0	0.0	0.0	0.0	161.9	161.9	0.0
	II	813.1	813.1	0.0	2,200.0	2,200.0	0.0	3,013.1	3,013.1	0.0
	III	1,280.0	1,280.0	0.0	1,916.0	1,916.0	0.0	3,196.0	3,196.0	0.0
	IV	860.0	860.0	0.0	972.0	972.0	0.0	1,832.0	1,832.0	0.0
	Sub-Total	3,115.0	3,115.0	0.0	5,088.0	5,088.0	0.0	8,203.0	8,203.0	0.0
Out	I	93.4	93.4	0.0	316.0	316.0	0.0	409.4	409.4	0.0
	II	3,923.2	3,923.2	0.0	13,946.0	13,932.0	14.0	17,869.2	17,855.2	14.0
	III	8,226.2	8,226.2	0.0	21,074.0	21,040.0	34.0	29,300.2	29,266.2	34.0
	IV	3,920.6	3,920.6	0.0	10,906.0	10,886.0	20.0	14,826.6	14,806.6	20.0
	Sub-Total	16,163.4	16,163.4	0.0	46,242.0	46,174.0	68.0	62,405.4	62,337.4	68.0
All zones	I	720.4	720.4	0.0	1,422.0	1,422.0	0.0	2,142.4	2,142.4	0.0
	II	6,873.7	6,873.7	0.0	24,564.0	24,548.0	16.0	31,437.7	31,421.7	16.0
	III	14,357.5	14,357.5	0.0	31,584.0	31,546.0	38.0	45,941.5	45,903.5	38.0
	IV	7,582.6	7,582.6	0.0	17,844.0	17,816.0	28.0	25,426.6	25,398.6	28.0
	Total	29,534.2	29,534.2	0.0	75,414.0	75,332.0	82.0	104,948.2	104,866.2	82.0

Down hole survey measurements were carried out routinely. Eight (8) out of 320 holes were not surveyed due to operational obstacles. Two (2) out of 8 were not surveyed due to extreme ground collapse in the upper 30 m. The remaining 6 holes were not surveyed due to deeper ground collapse. Details are shown in Table 10-3.Non surveyed holes (in metres) correspond to 2.2% and 0.9% for RC and DDH holes, respectively drilled.

Table 10-3 Non Surveyed Drill Holes due to Ground Collapse

Zone	Non Surveyed Drillholes	Non Surveyed (m)	Section	Drillhole Type
Crux	CMR-148	214	350 NW	RC
Phoenix	CMR-018	444	1400 NW	
Non-Surveyed RC	658			
Non-Surveyed (%)	2.2			
Phoenix	CMD-010	165.35	1400 NW	DDH
Phoenix	CMD-152	134.80	1400 NW	
Phoenix	CMD-091	25.95	1600 NW	
Phoenix	CMD-021	143.15	1600 NW	
Lynx	CMD-122	173.50	2150 NW	
Lynx	CMD-036	22.95	2300 NW	
Non-Surveyed DDH	665.70			
Non-Surveyed (%)	0.9			

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

Diamond drill core and reverse circulation cuttings are handled by Atacama personnel and/or SBX sub-contracted personnel from the moment that the core/cuttings exit the drill. Core and cuttings are transported weekly to Atacama's Paipote core logging and core-cutting /storage facility.

11.1.1 Reverse Circulation Drill Holes-RC

- RC 2 m cuttings, weighing approximately 80 kg, were split at the drilling site in a standard splitter down ¼ (approximately 20 kg).Two 20 kg samples were bagged and put into pre-labelled plastic bags under the supervision and control of Atacama personnel. Also, a geological technician collected representative samples (dust and cuttings) every 2 m intervals in properly marked and identified plastic "chip" trays, which were used for logging purposes
- Field duplicate samples are inserted at a rate of approximately 1 per 20 samples. Once the holes are sampled, the samples are transported to the core shed located in Paipote
- At Paipote, 7 kg bagged blank reference material are inserted after each duplicate and then sent to the sample preparation facility run by Geoanalitica, located approximately 1 km from Atacama's core shed. The sample preparation stream, as well as the QA/QC protocol is shown in Figure 11-1.

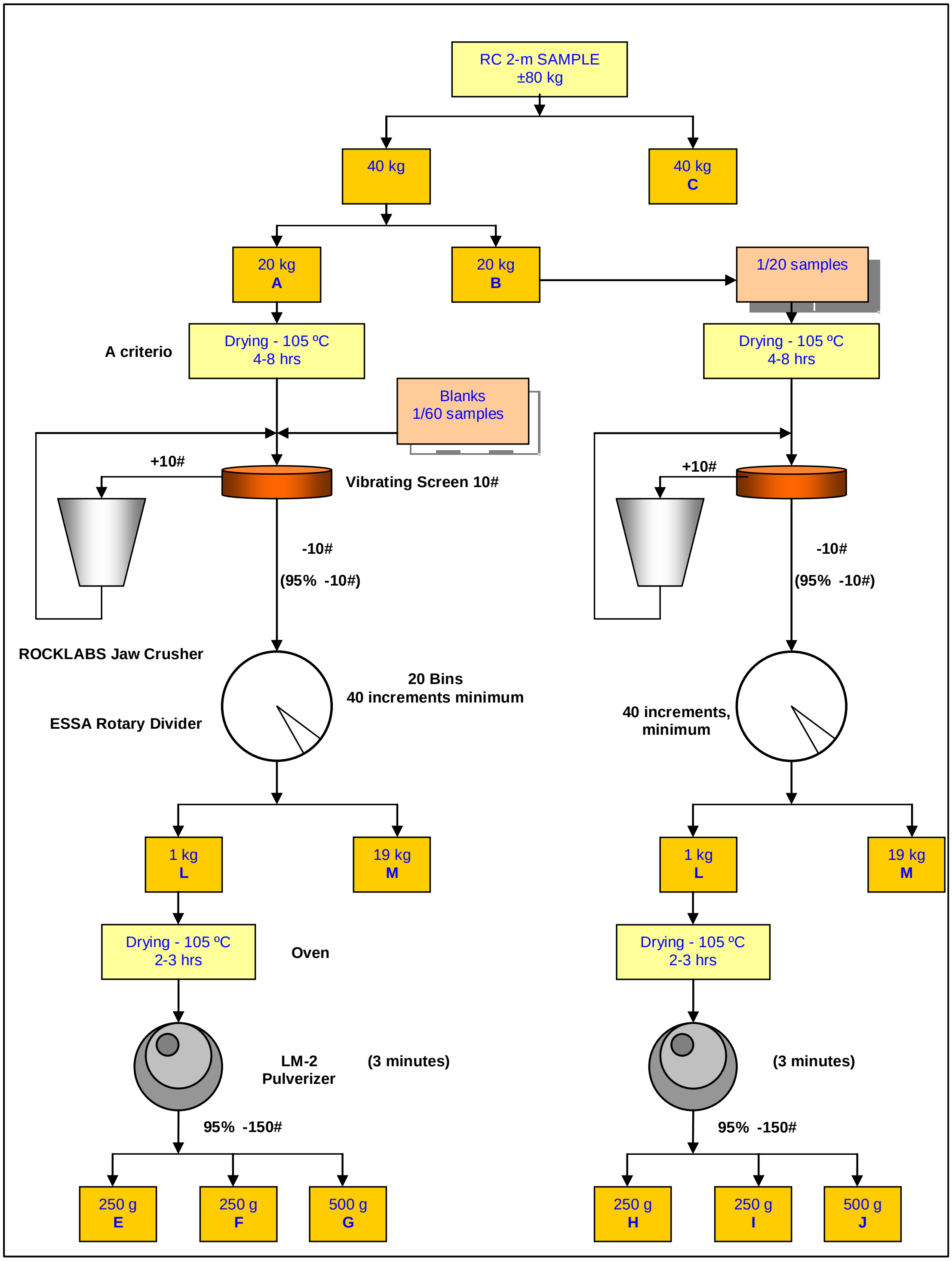


Figure 11-1 Sample Preparation Protocol-RC and QA-QC

11.1.2 Diamond Drill Holes – DDH

- Diamond drill core is boxed in aluminum trays at the drill site, where it is properly taken from the core barrel. The recovery, RQD, and fracture frequency are measured by a geological technician
- A senior geologist marks the axis along the drill hole as well as the starting and ending points for 2 m samples. Each sample is given a specific number.
- Core is then pre-logged ("quick-log") by a senior geologist at site in order and the geologist selects the 2 m samples that will be duplicated in the sample preparation facility (approximately one every 20 m). The identification of samples selected for duplicates are recorded. Samples selected as duplicates should "ideally" contain gold mineralization
- The core boxes are properly sealed such that there will be no movement or separation of the core, and are then transported to the camp
- Diamond saw splitting is carried out in the Atacama core shed located in Paipote
- One half of the core is returned to the core box for final logging and storage; the other half is properly bagged and labeled, 7 kg in-house blank sample are inserted, and delivered to Geoanalítica for preparation together with the list of samples selected as duplicates. The sample preparation stream for DDH is shown in Figure 11-2.

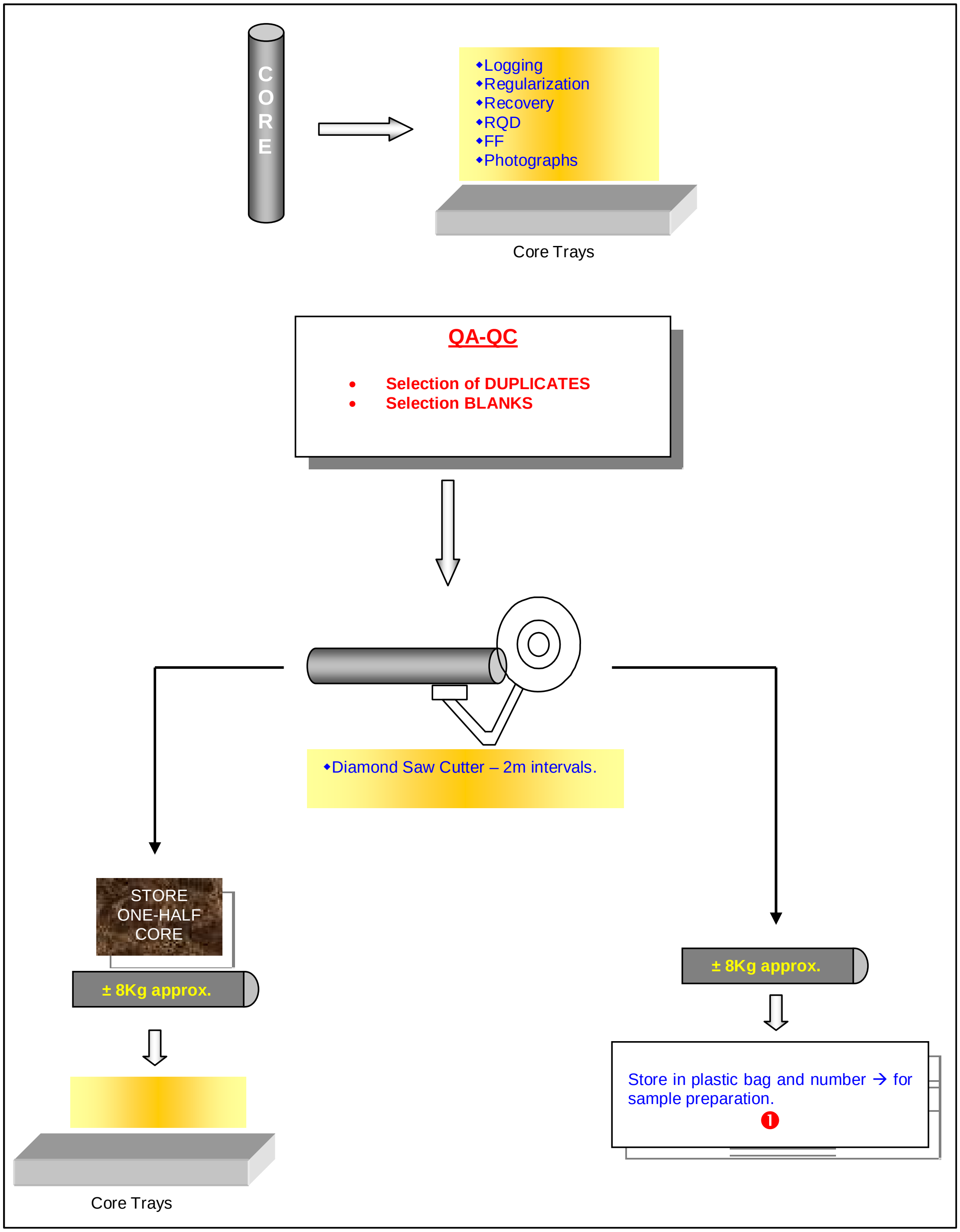


Figure 11-2 Sample Preparation Protocol – DDH and QA-QC

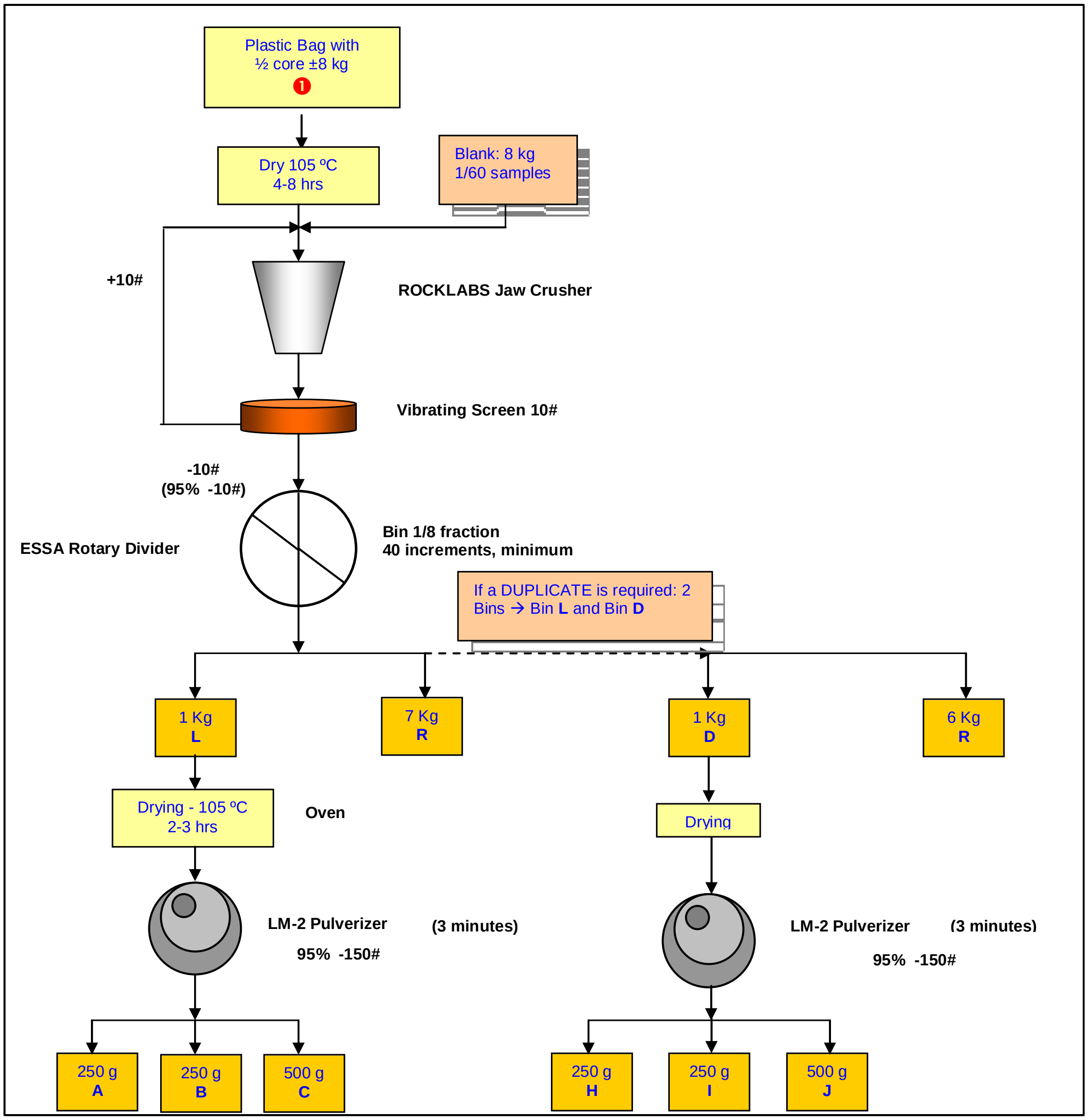


Figure 11-3 Sample Preparation Protocol

11.2 Analyses

Once Geoanalitica returned the prepared samples, Atacama personnel inserted 250 g standards (#150) every 20 m and re-numbered the samples with bar codes.

Finally, each hole contained the following quality control material:

RC Holes:

1. Field duplicate (every 20 m) – Envelope H in Figure 11-1.
2. Pulp duplicate (every 20 m) – Envelope I in Figure 11-1.

3. Standard (every 20 m).

4. Blank (every 60 m).

DD Holes:

1. Coarse (-10#) duplicates (every 20 m) - Envelope H in Figure 11-2.

2. Pulp duplicates (every 20 m) – Envelope I in Figure 11-2.

3. Standards (every 20 m).

4. Blanks (every 60 m).

Samples were analyzed in Activation Laboratories in Coquimbo, Chile.

11.3 Security

Core trays, cutting boxes, 150# pulps and coarse rejects are orderly and safely stored in Atacama's logging and storing facility in Paipote.

Coarse rejects were stored in plastic bottles containing approximately 2.5 kg each.

11.4 Conclusion

The overall conclusion is that sampling, sample preparation, analyses and security protocols used by Atacama during the 2011 – 2013 drilling campaigns meet acceptability criteria and therefore data collected may be used with confidence for resource modelling and estimation.

12 DATA VERIFICATION

12.1 Data Management

During the Phase I through Phase IV drilling campaigns, sample quality assurance and quality control measures included the insertion of duplicates and standards, as well as in-house and commercial blanks. This section of the report presents statistical analyses of data collected during Phases: I (2010), II (2010/2011), III (2011/2012), and IV (2012/2013). Details are shown in Table 12-1.

Table 12-1 Cerro Maricunga Database Quality Assessment and Quality Control

Phase	I	II	III	IV	Total
Years	2010	2011	2012	2013	2010-2013
N° Drillholes	8	82	130	100	320
Meters Assayed	2,142.40	31,421.70	45,903.50	25,398.60	104,866.20
N° Samples Assayed	1,072	15,729	22,993	12,728	52,522
QA-QC Assays					
N° Standards	48	534	894	481	1,957
N° Blanks-In-House	18	238	388	0	644
N° Commercial Blanks	0	0	263	179	442
N° RC Field Duplicates	22	417	417	335	1,191
N° DDH 10# Duplicates	17	117	238	133	505
N° Pulp Duplicates	39	534	655	468	1,696
Total QA-QC Samples	144	1,840	2,855	1,596	6,435
QA-QC Data (%) Database	13.4	11.7	12.4	12.5	12.3

As will be seen in the following sections, results indicated that sample preparation and analyses were acceptably precise and exact during the 2010-2013 drilling campaigns.

The following action was taken in preparing the QA-QC data for statistical analyses:

- Values for Au reported as "<0.005" were replaced by "0.0025" (this corresponds to values below the 5 ppb detection limit for gold).

12.2 Analysis of Duplicate Samples

Table 12-2 summarizes the QA-QC results for all RC field duplicates, DDH coarse duplicates (10#) and pulp duplicates for RC and DDH samples.

Table 12-2 Summary of QA-QC-RC Field Duplicates-DDH 10# Duplicates and Pulp Duplicates

Results	RC - Au (ppm)		DDH - Au (ppm)		Pulps - Au (ppm)	
	Original	Duplicate	Original	Duplicate	Original	Duplicate
Number of Samples	1,191	1,191	505	505	1,696	1,696
Minimum	0.00	0.00	0.01	0.01	0.00	0.00
Maximun	2.79	2.99	4.50	3.39	3.39	3.33
Mean	0.24	0.24	0.36	0.35	0.27	0.27
Standard Deviation	0.30	0.30	0.46	0.44	0.35	0.35
T Test	-0.64		0.60		-0.76	
Mean Relative Error	13.93		7.52		10.71	
Bias (%)	-0.34		0.60		-0.23	
Correlation	0.99		0.99		1.00	
Intercept	0.00		0.02		0.00	
Slope	0.99		0.95		1.00	
Hyperbola (% rejected)	2.35		1.98		6.19	

In all cases the original and duplicate data show good agreement:

- Results for the T Tests (all values are within [-1.96, 1.96]) show that the original and duplicate means were not significantly different, based on 95% confidence intervals
- Mean relative errors were close to 14% for the RC field duplicates and around 7.5% for DDH coarse duplicates. However, the mean relative error for pulp duplicates was 10.71%, which was considerably higher than that for DDH coarse duplicates. The reason for this increase was due to the fact that there were many low grade values in the pulp duplicates, which inflated the relative errors
- In all three cases, correlation values were high (very close to 1), intercepts were low and slopes were close to 1, indicating a high degree of correspondence between the original and duplicate samples
- The Min-Max analysis was applied. The accepted criterion is that less than 10% of pairs should be rejected, that is above the hyperbola. In this case the percentage rejected ranged from 1.98 to 6.19%, which was acceptable.

The effect of eliminating low grade samples on the mean relative error for RC field duplicates, DDH coarse duplicates (10#) and DDH plus RC pulp duplicates were verified by repeating the statistical analyses presented in Table 12-2 after eliminating pairs with an average Au value lower than 0.1 ppm. Results of this reanalysis are presented in Table 12-3. A threshold of 0.1 ppm was selected because samples with grades lower than this are not likely to be of interest for modeling the resources for open pit planning, and they contributed large amounts of relative error as many of them were close to the gold detection limit.

Table 12-3 Summary of QA-QC results for RC Field Duplicates, DDH 10# and Pulp Duplicates – Au ≥ 0.1 ppm

Results	RC - Au (ppm) > 0.1		DDH - Au (ppm) > 0.1		Pulps - Au (ppm) > 0.1	
	Original	Duplicate	Original	Duplicate	Original	Duplicate
Number of Samples	730	730	358	358	1087	1087
Minimum	0.09	0.01	0.10	0.10	0.04	0.06
Maximun	2.79	2.99	4.50	3.39	3.39	3.33
Mean	0.36	0.36	0.48	0.48	0.40	0.40
Standard Deviation	0.32	0.32	0.49	0.47	0.38	0.38
T Test	-0.82		0.59		-0.77	
Mean Relative Error	10.91		5.41		6.57	
Bias (%)	-0.47		0.62		-0.24	
Correlation	0.98		0.98		0.99	
Intercept	0.01		0.03		0.00	
Slope	0.99		0.94		1.00	
Hyperbola (% rejected)	3.56		1.96		5.52	

As can be seen, 461 RC duplicates, 147 DDH duplicates and 609 pulp duplicates were eliminated, indicating that a considerable amount of the data was below 0.1 ppm Au.

Eliminating low grade duplicates had the following effects:

- The mean Au grades increased from 0.24 to 0.36-ppm for RC, from 0.36 to 0.48-ppm for DDH and from 0.267 to 0.40-ppm for pulps
- The mean relative errors decreased. This was significant for pulp duplicates, where the mean relative error decreased from 10.71 to 6.57%
- The elimination of low grade samples also affected the percentage of data meeting the absolute relative difference criteria, as summarized in Table 12-4
- The percentage of rejected samples by the hyperbola test was lower than 10% in all cases; therefore, results are acceptable according to this criterion.

Table 12-4 QA-QC Criteria and Results for Au duplicates.

Duplicate Type	Criteria	ALL Au Data %	Au Data > 0.1 ppm %
RC Field Duplicates	90% Data have |Rel Diff| < 20%	91.1	95.2
DDH 10# Duplicates	90% Data have |Rel Diff| < 15%	93.3	97.0
Pulp Duplicates	90% Data haver|Rel Diff| < 10%	86.0	93.5

Acceptability criteria were not met for pulp duplicates when all the data were analyzed, most likely due to the large relative difference of low grade samples. When low grade samples (<0.1 ppm Au) were excluded, all three types of samples met the acceptability criteria. Figure 12-1 to Figure 12-6 show detailed results for the RC field duplicates, DDH coarse duplicates and for the pulp duplicates, respectively.

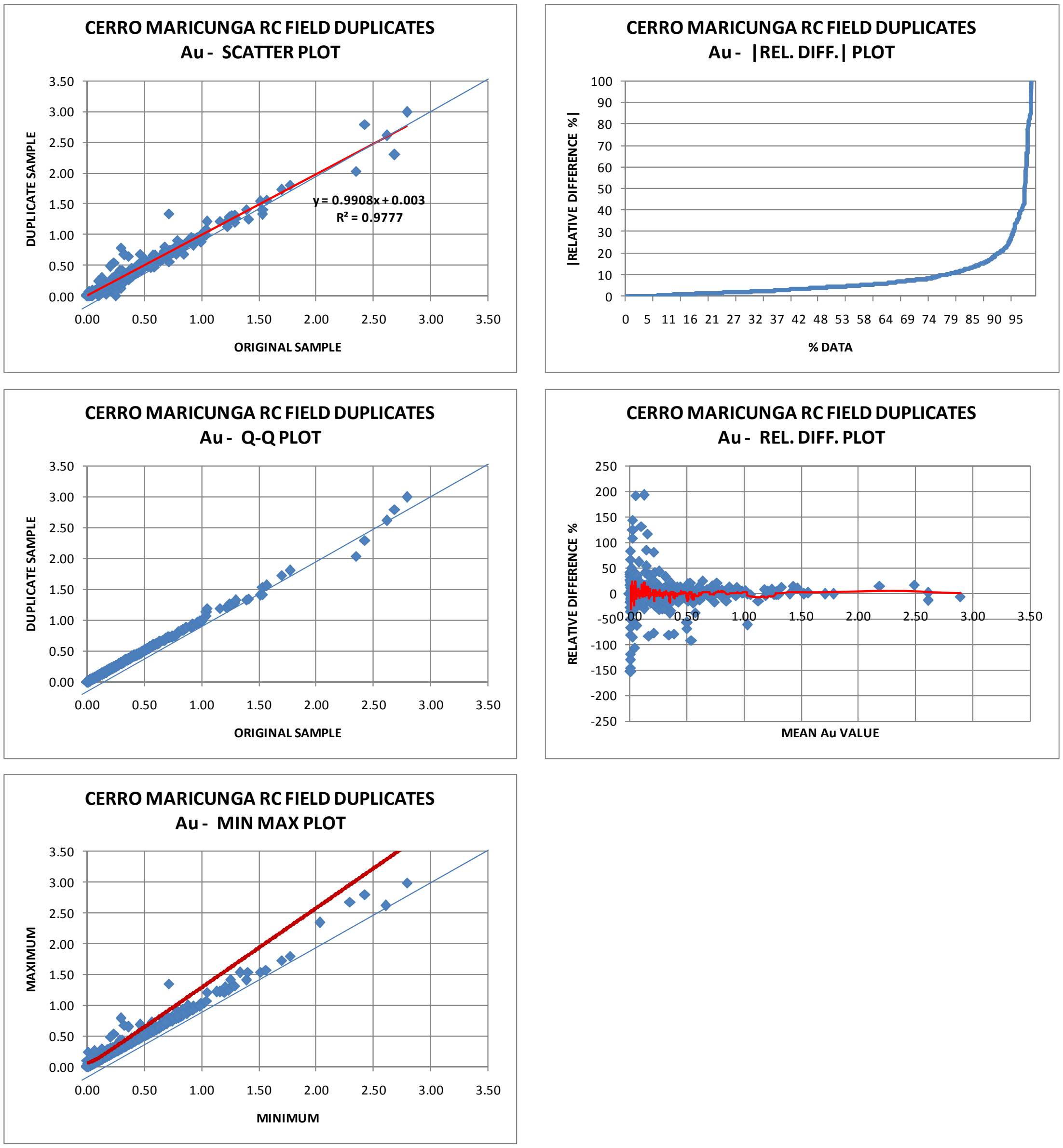


Figure 12-1 Results for all RC Field Duplicates – Au

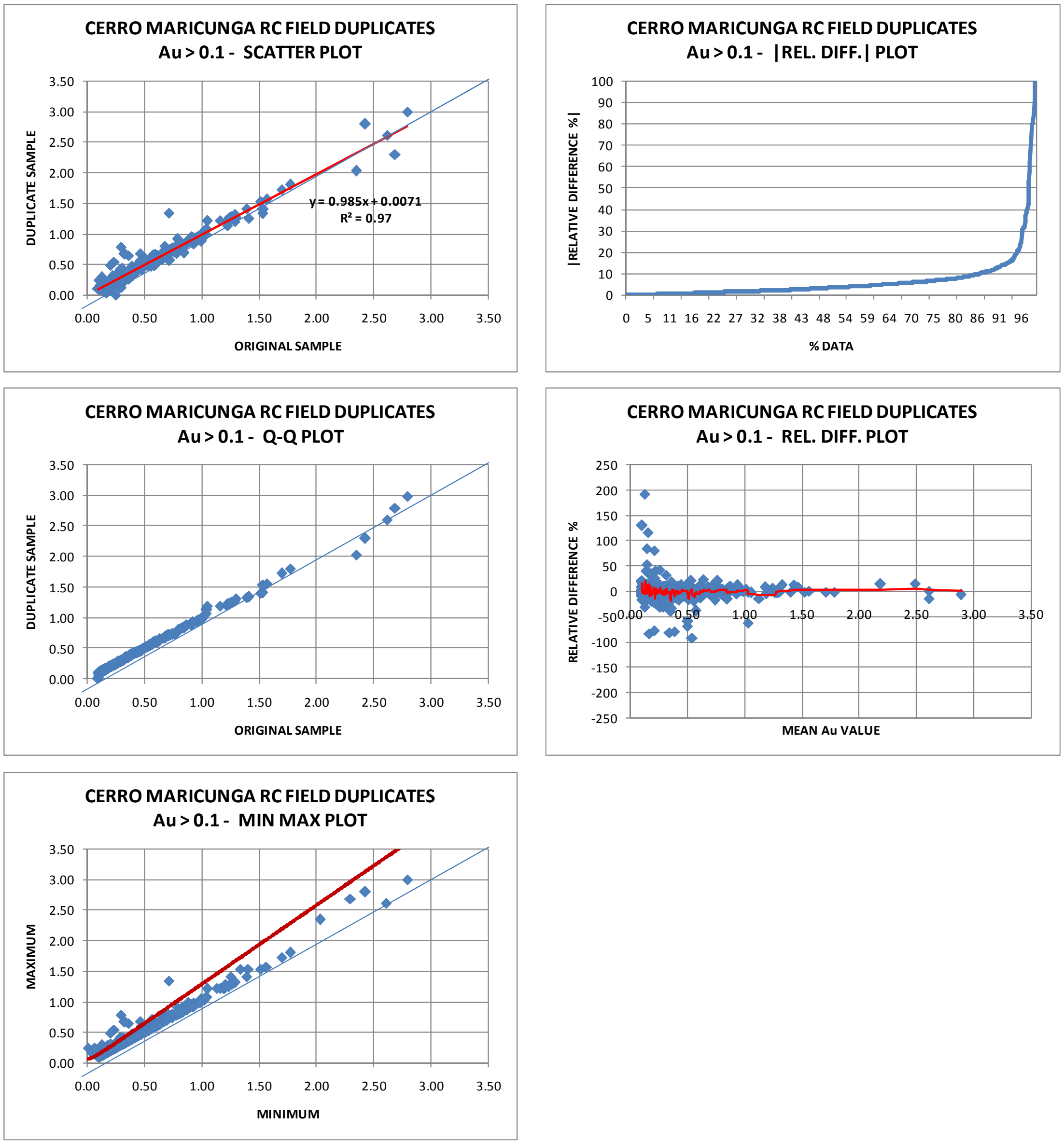


Figure 12-2 Results for the RC Field Duplicates - Au ≥ 0.1 ppm

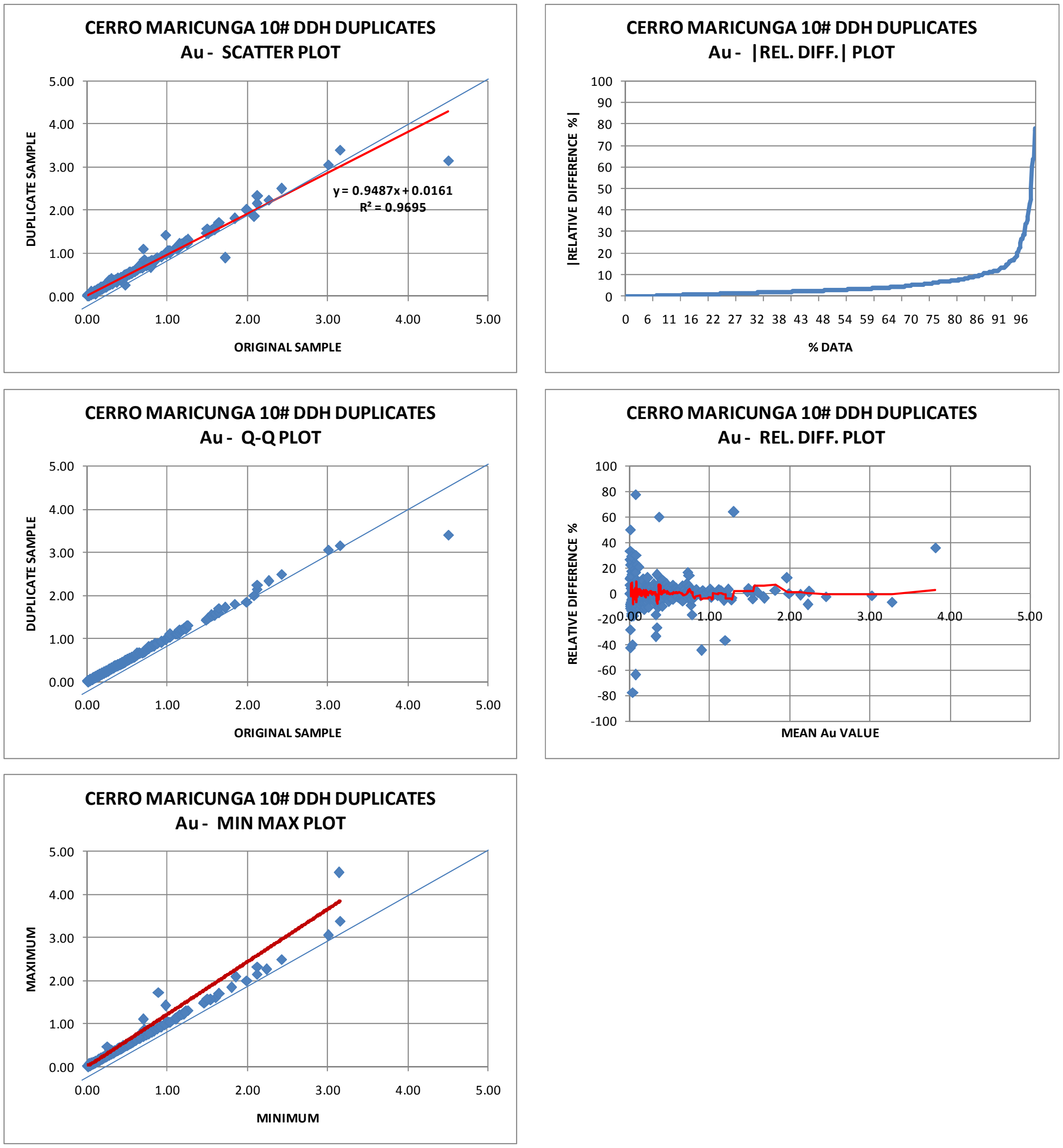


Figure 12-3 Results for DDH Coarse Duplicates - Au

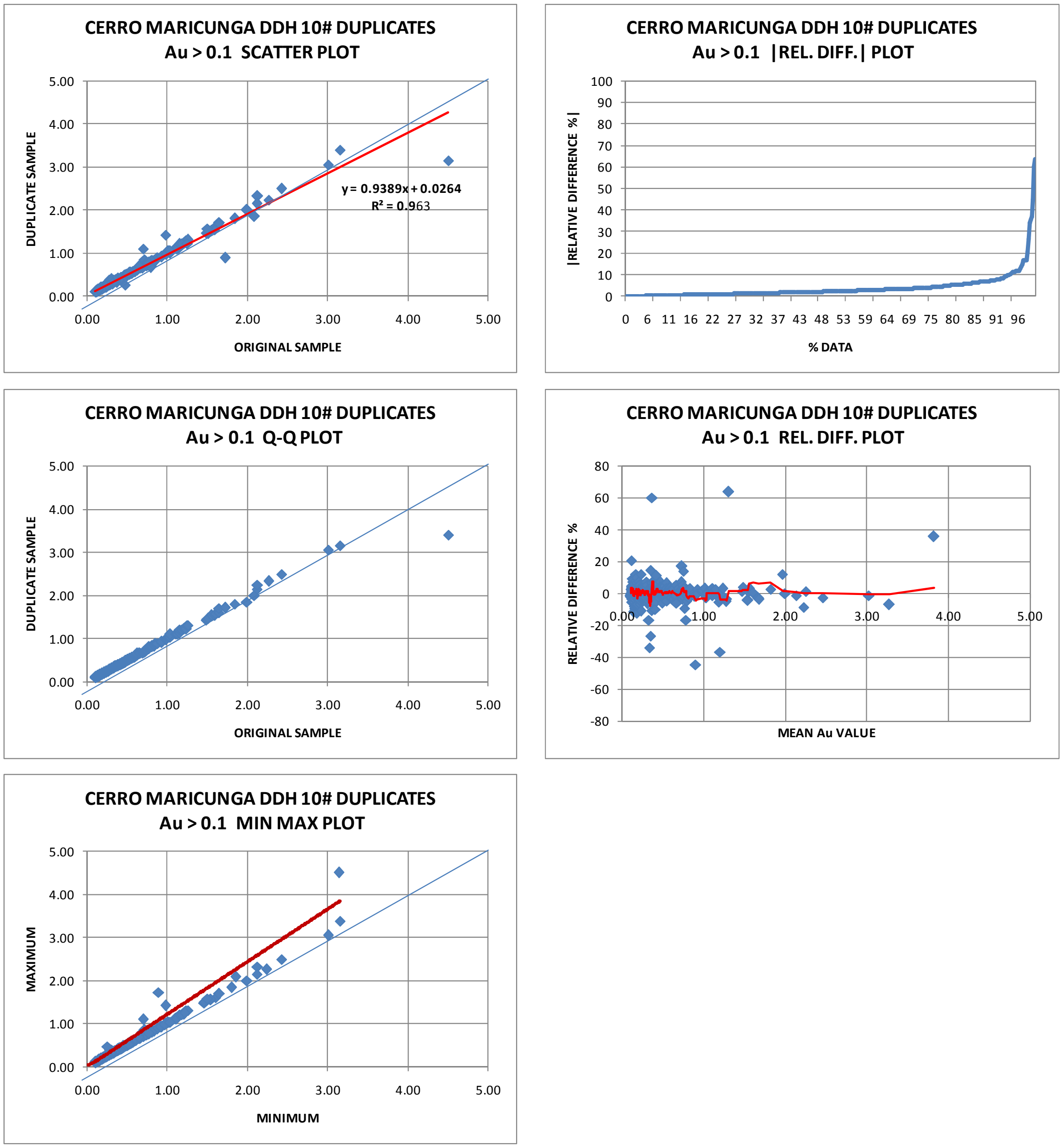


Figure 12-4 Results for the DDH Coarse Duplicates - Au ≥ 0.1 ppm

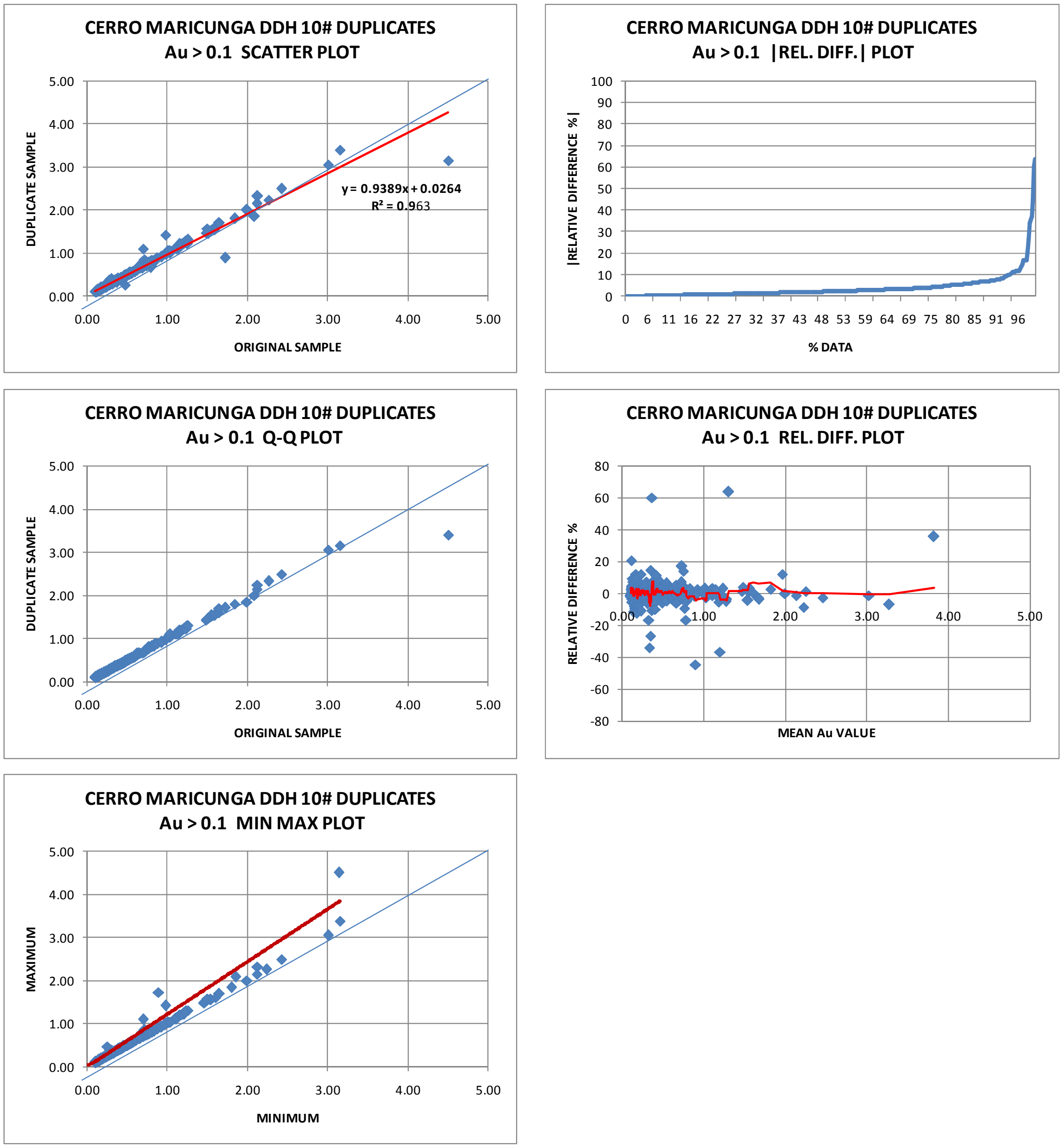


Figure 12-5 Results for all Pulp Duplicates - Au

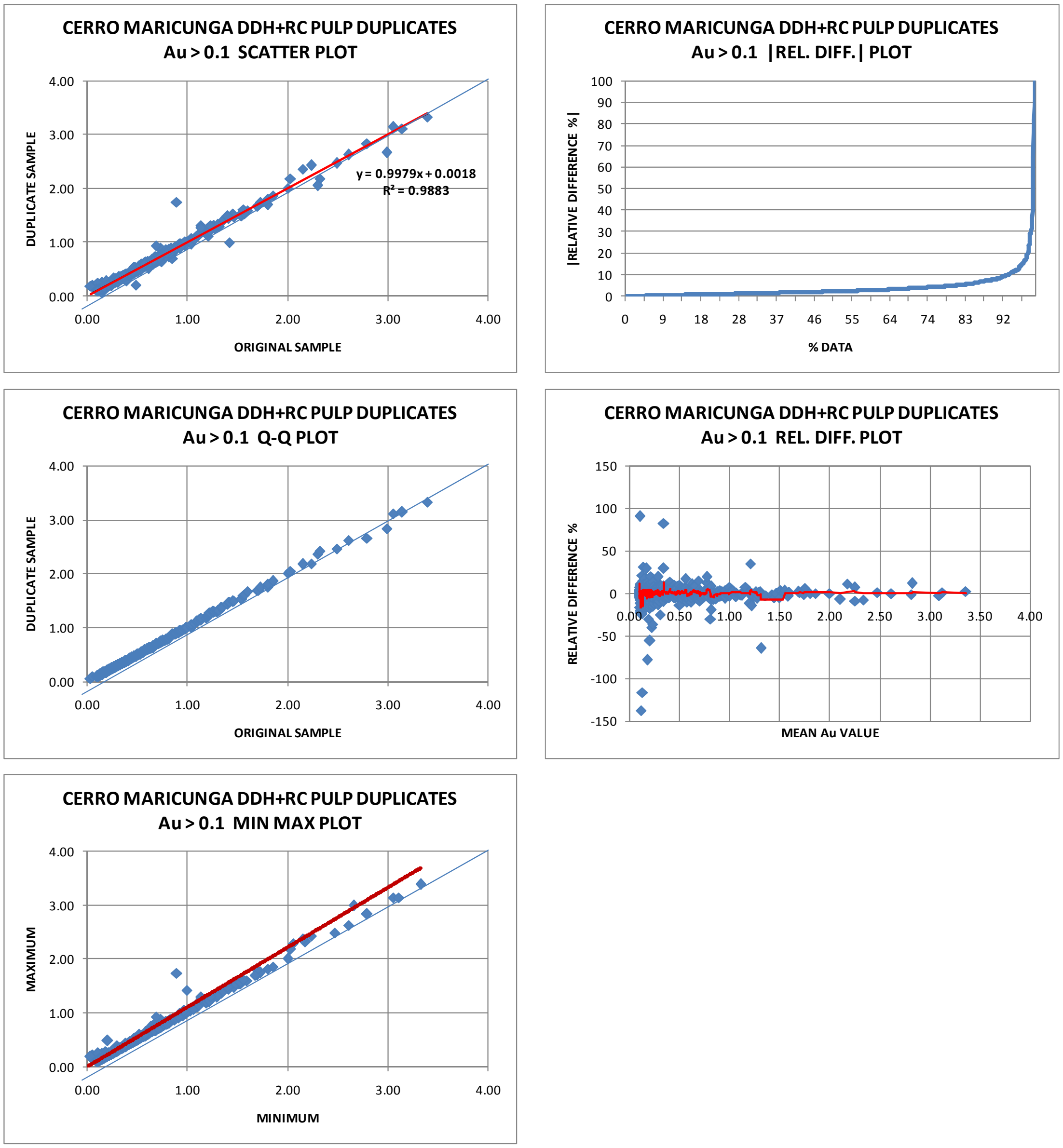


Figure 12-6 Results for the Pulp Duplicates - Au ≥ 0.1 ppm

In general, statistical analyses of all Au duplicate data examined (reverse circulation field duplicates, diamond drill hole 10# duplicates and duplicate assays), especially those above 0.1 ppm Au showed good precision, indicating that the protocols used for sample preparation and assaying were adequate.

12.3 Analysis of Standard Samples

Atacama acquired six standards at Geostats Pty during the 2010-2013 drilling period. Standards G301-1 and G301-3 were discontinued in 2011. These were replaced by G907-2 and G907-7. In addition, it was decided to acquire another standard with a grade similar to the mean grade of the deposit; G909-7 (0.495). A total of 1,957 standards were assayed for Au. Details are shown in Table 12-5.

Table 12-5 Summary of GEOSTATS PTY Standards Used in Cerro Maricunga Project

Standard ID	BEST Au Value (ppm)	σ	Number of Tested Standards - per Drilling Campaign			
			Phase I	Phase II	Phase III	Phase IV
G 301 - 1	0.847	0.092	18	177	0	0
G 301 - 3	1.958	0.162	14	172	0	0
G 303 - 8	0.261	0.063	16	185	227	121
G 909 - 7	0.495	0.031	0	0	226	125
G 907 - 2	0.89	0.056	0	0	221	120
G 907 - 7	1.541	0.065	0	0	220	115

Table 12-6 shows a summary of results for each standard tested during Atacama's QA-QC program.

Table 12-6 Summary Statistical Results for Standards

Standard ID	Best Au-Value	N	Mean	STD	Bias	N out 95% Interval	% Out 95% Interval
G 301 - 1	0.847	195	0.857	0.117	1.181	3	1.5
G 301 - 3	1.958	186	1.952	0.094	-0.306	1	0.5
G 303 - 8	0.261	549	0.269	0.016	3.065	1	0.2
G 909 - 7	0.495	351	0.490	0.021	-1.010	11	3.1
G 907 - 2	0.890	341	0.898	0.021	0.899	2	0.6
G 907 - 7	1.541	335	1.501	0.020	-2.596	1	0.3

Bias (%) was calculated as: (Observed mean – Nominal value) / Nominal value x 100.

The observed bias for the lowest grade standard (G303-8) was slightly high (3.065%). Standards G909-7 and G907-2, which represent a relevant portion of the resources behaved very well. The high grade standard (G907-7) showed a consistent negative bias (-2.596%), however it affected less than 1% of the samples. The overall bias amounted to -0.432% which was acceptable. It is worth noting that the high grade standard G907-7 had a similar behavior of consistent negative bias in the 2 last campaigns. Previously reported biases for this standard were -2.40% (2012) and -3.05% (2013). This could be indicative of a problem with this particular standard.

Results for each standard sample are shown in Figure 12-7.

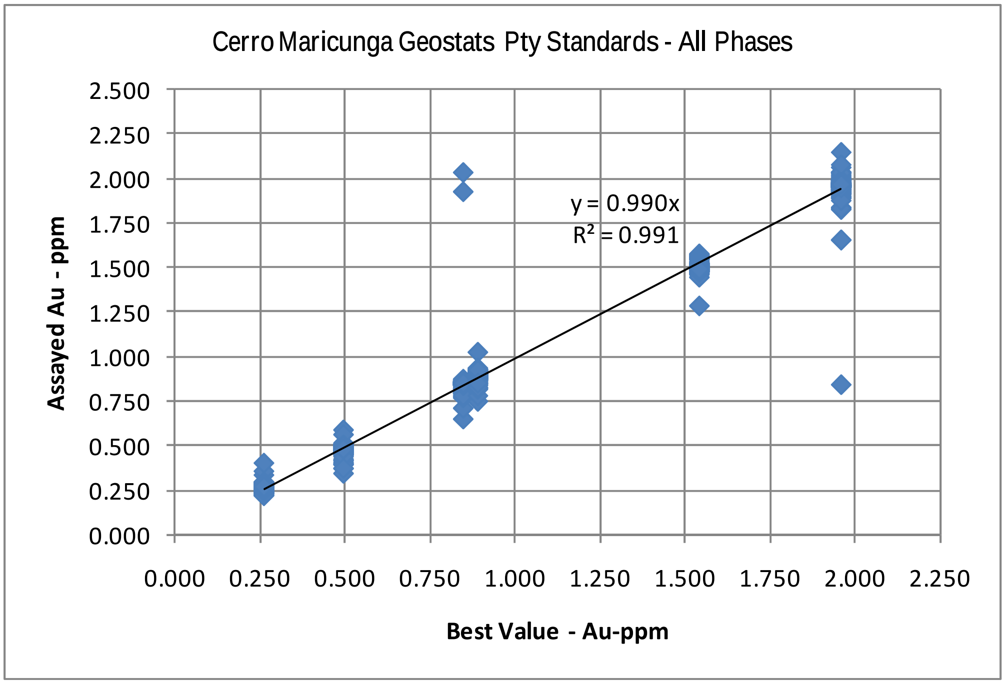


Figure 12-7 Results for all Standards

The slope of the regression line (with an intercept fixed to zero) should ideally be equal to 1.000. In this case, the observed slope was 0.990 (including five clear outliers) which was 1.96% lower than the desired value, which was considered acceptable. The correlation coefficient was very high (0.995), indicating that the deviations from the regression line were low. Additionally, dispersions of the assay values for all three standards were low, indicating good assay accuracy.

Control charts for standards G301-1, G301-3, G303-8, G909-7, G907-2 and G909-7 are shown in Figure 12-8 to Figure 12-13.

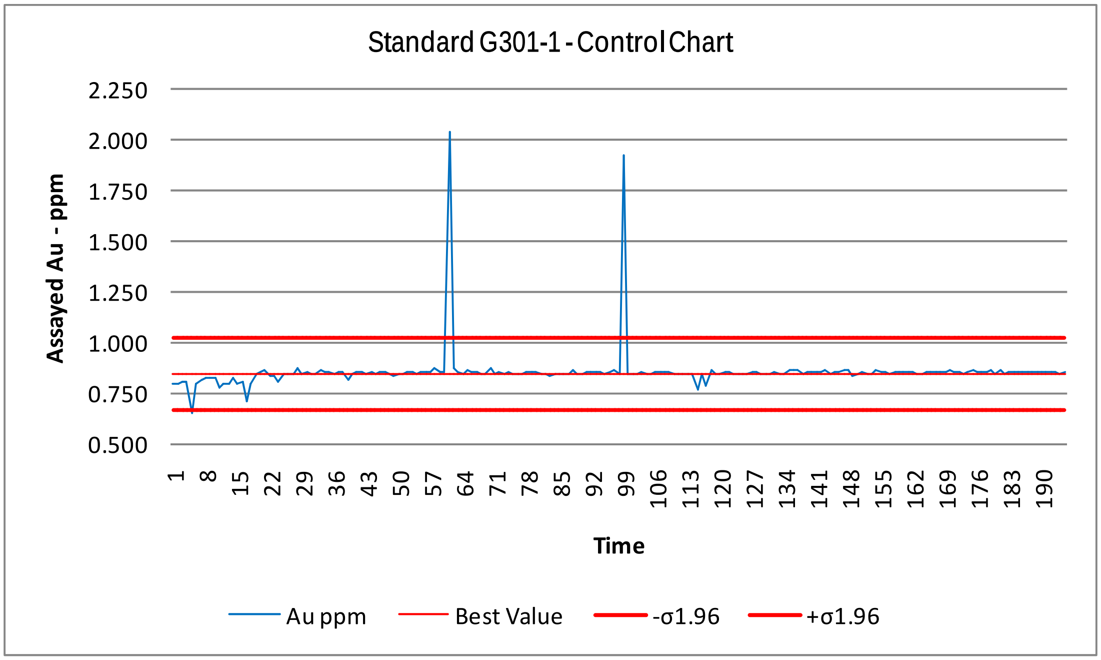


Figure 12-8 Control Chart for Standard G301-1

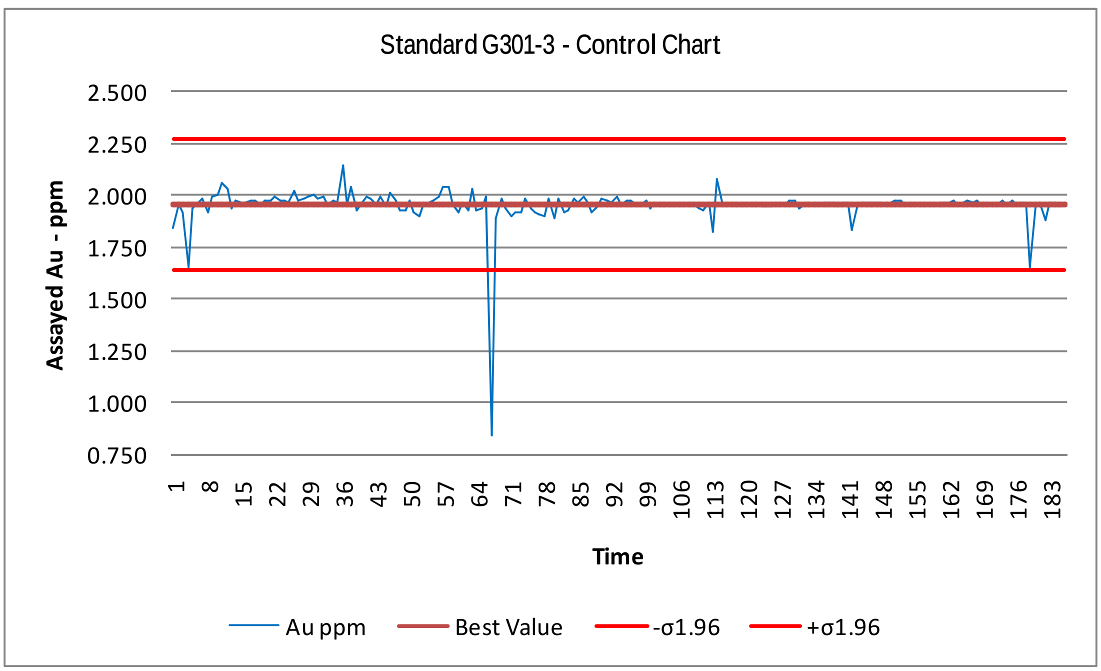


Figure 12-9 Control Chart for Standard G301-3

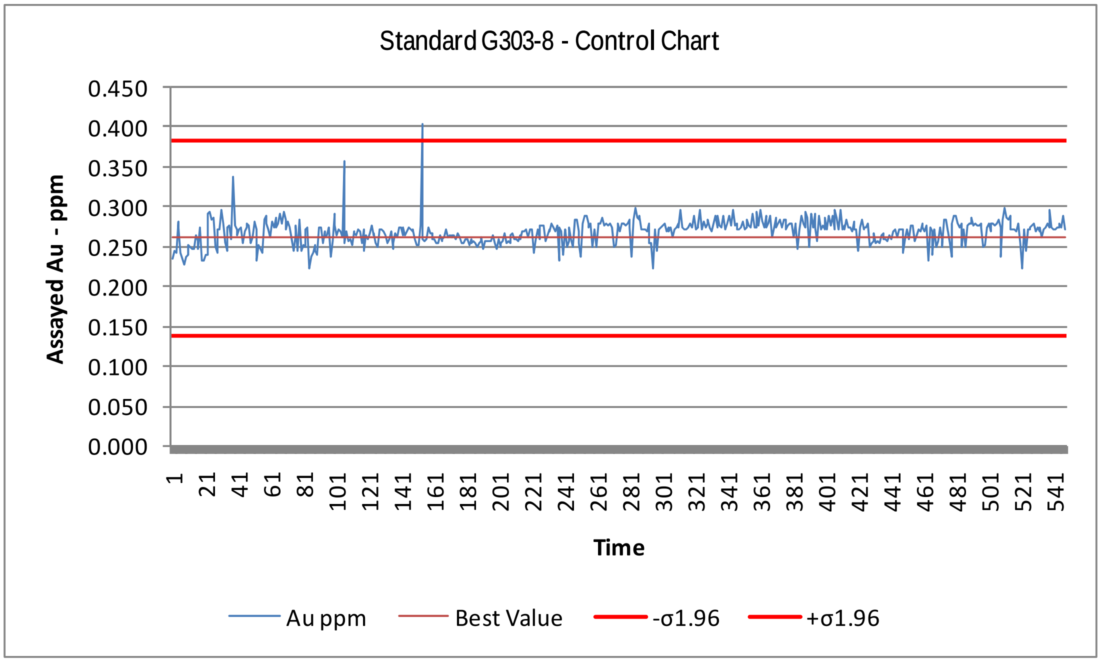


Figure 12-10 Control Chart for Standard G303-8

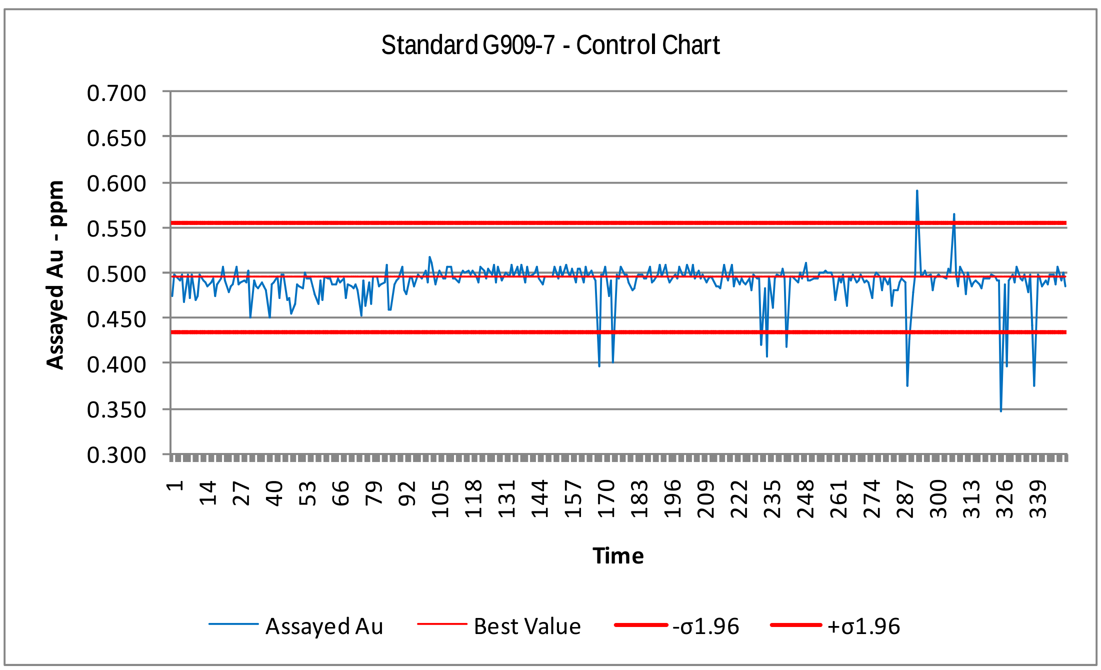


Figure 12-11 Control Chart for Standard G909-7

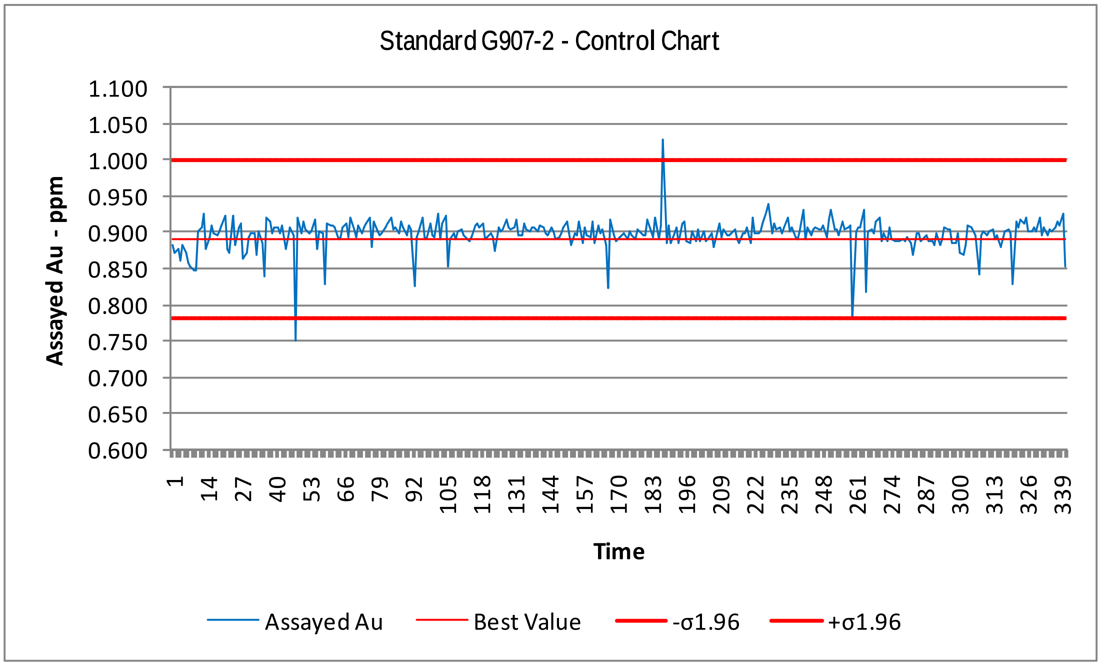


Figure 12-12 Control Chart for Standard G907-2

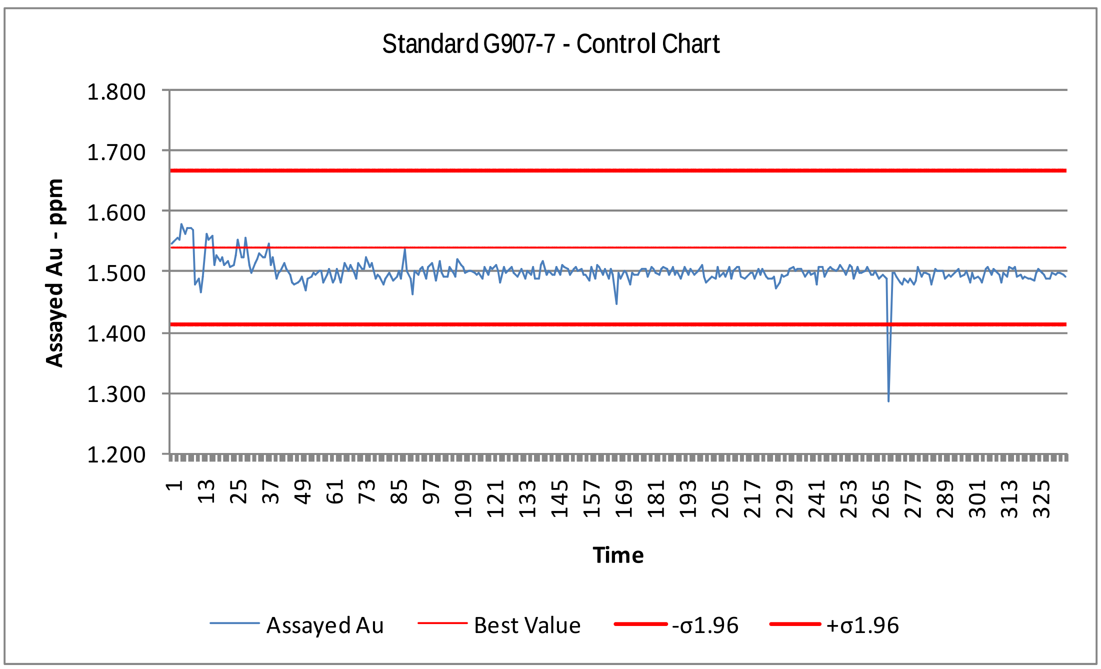


Figure 12-13 Control Chart for Standard G907-7

Control charts and Table 12-6 show that some samples lied beyond the 2 standard deviation upper and lower limits. The "out of bounds" percentage should be at most 5%. This condition was met by all standards.

In conclusion, the analyses of standards used in the Phase I through Phase IV exploration campaigns showed acceptable accuracy and precision and therefore drilling results could be used with confidence for resource modeling and estimation.

12.4 Analysis of In-House Blank Samples

Blank samples were inserted into the sample preparation facility processing order to assess if there was any cross-contamination between samples.

Figure 12-16 shows a sequential Au assay plot for blank samples inserted during the 2010 – 2013 drilling campaigns.

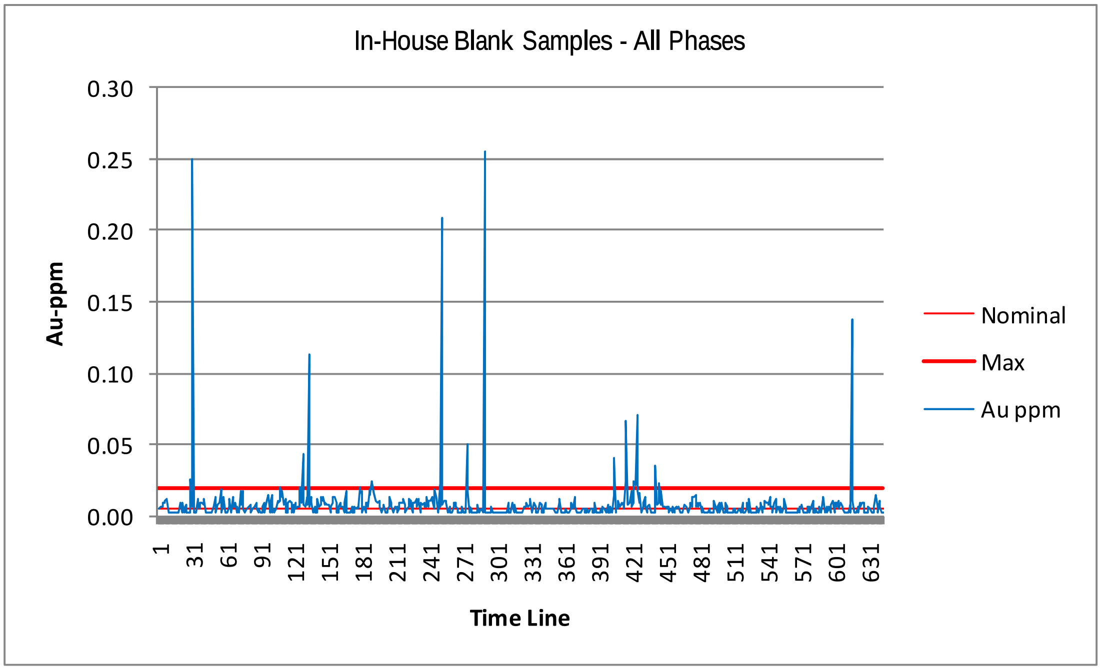


Figure 12-14 Time Sequenced Au Values – In House Blanks

Eighteen (18) samples supersede the maximum acceptable gold grade (> 0.20 g/t) which corresponds to 2.8% of total assayed blanks. Results are shown in Table 12-7 and Figure 12-15 .

Table 12-7 Frequency Table – In House Blanks

Au-ppm Range	Samples (%)	N Samples
≤ 0.005	49.8	321
0.006 - 0.010	34.2	220
0.011 - 0.015	10.2	66
0.016 - 0.020	3.0	19
0.021 - 0.030	1.1	7
0.031 - 0.040	0.2	1
≥ 0.040	1.6	10

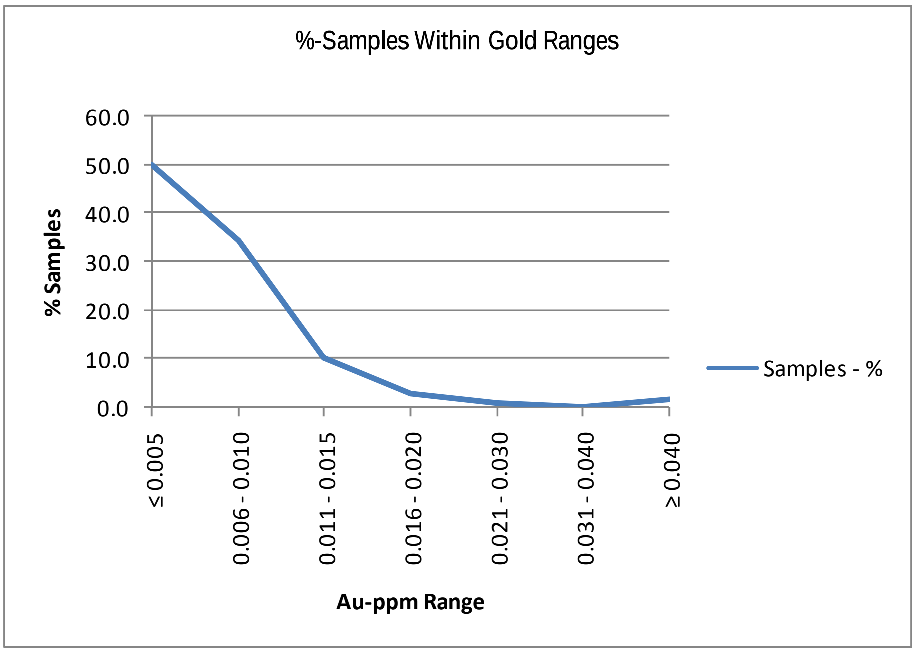


Figure 12-15 Frequency Plot - % Samples within Gold Ranges

Results indicated the following:

- The average grade of all blanks was 0.008 ppm
- Percentage of blanks above 0.02 ppm was 2.8%
- The percent of blanks above 0.03 ppm was 1.8%
- The highest gold value was 0.256 ppm
- Processing orders were reviewed in order to detect probable contamination between samples in cases where blank material had anomalously high gold values. Results showed that there was little correspondence between high grade blank samples and the grade of the samples immediately preceding them in the laboratory's processing order. This suggested that there were no contamination problems between samples, and that these values probably arosed from mislabelled samples (ore samples were mistaken for blanks).

12.5 Analysis of Commercial Blank Samples

During Phases III and IV (2012 and 2013 drilling campaigns) a set of 442 500g (-150#) sachets of blank certified material, acquired at Geostats Pty, were inserted to control possible contamination in the analytical laboratory.

A brief analysis is shown in Figure 12-16 (assayed gold values) and Figure 12-17 (frequency plot). The percentages of samples within different Au-ranges are shown in Table 12-8.

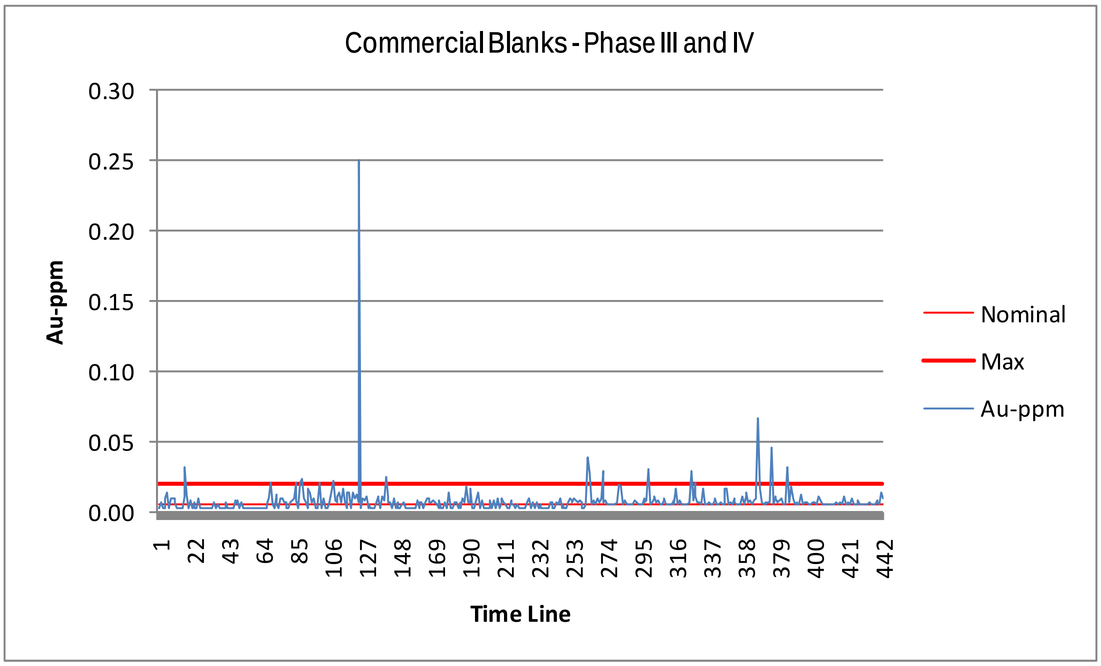


Figure 12-16 Geostats Blank Certified Material - Gold Values

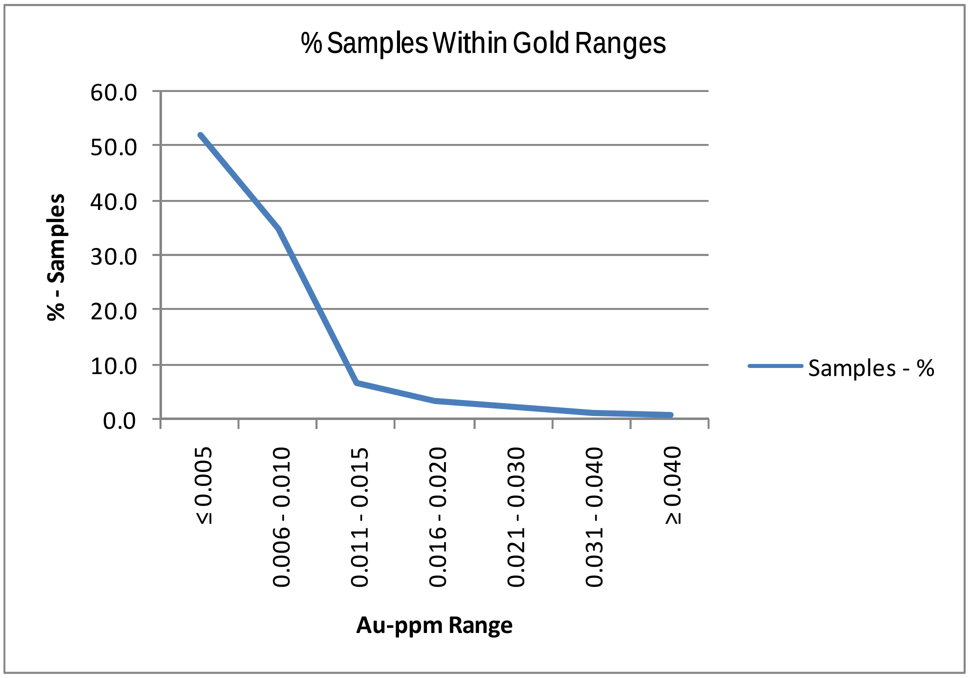


Figure 12-17 Geostats Blank Certified Material - Frequency Plot

Table 12-8 Geostats Blank Certified Material - Frequency Table

Au-ppm Range	Samples (%)	N Samples
≤ 0.005	52.0	230
0.006 - 0.010	34.6	153
0.011 - 0.015	6.6	29
0.016 - 0.020	3.2	14
0.021 - 0.030	2.0	9
0.031 - 0.040	0.9	4
≥ 0.040	0.7	3

Results were as follow:

- The average grade of all blanks was 0.007 ppm
- The percent of blanks above 0.02 ppm was 3.6%
- The percent of blanks above 0.03 ppm was 1.6%
- The highest blank assayed 0.250 ppm, which probably corresponded to Standard G303-8.

Figures showed that no serious cross contamination between samples occurred.

12.6 Twin Hole Analyses

Eleven sets of twin holes were been performed in the Cerro Maricunga Project; two (2) in Lynx, six (6) in Phoenix and three (3) in Crux. Identification and length of each hole, as well a section locations are listed in Table 12-9.

Table 12-9 List of Twinned Holes – Cerro Maricunga Project

Twin hole Set	DDH - Hole-ID	DDH-Length	RC – Hole-ID	RC-Length	Section	Zone
1	CMD004	181.85	CMR209	450	2300	Lynx
2	CMD198	80.35	CMR089	350	2200	Lynx
3	CMD010	165.35	CMR018	444	1400	Phoenix
4	CMD092	589.60	CMR002	342	1600	Phoenix
5	CMD093	531.00	CMR041	348	1400	Phoenix
6	CMD096	351.85	CMR030	374	1500	Phoenix
7	CMD099	700.00	CMR067	450	1550	Phoenix
8	CMD178	107.85	CMR045	312	1400	Phoenix
9	CMD192	320.00	CMR097	200	400	Crux
10	CMD193	180.00	CMR129	400	550	Crux
11	CMD196	250.02	CMR098	250	350	Crux

Gold assay results were compared for each twin hole set. Comparisons were carried with pairs of samples that complied with the following conditions:

- Pairs lied within the mineralized bodies, according to the 2012 geological resource model
- Distances between sample pairs were 10.00 m maximum. Distances between pairs were calculated using the central coordinates of each 2 m sample. The formula used was:

Distance between Pairs = $\sqrt{\Delta x^2 + \Delta y^2 + \Delta z^2}$

Where:

Δx = Difference in North Coordinate between pairs of samples.

Δy = Difference in East Coordinate between pairs of samples.

Δz = Difference in Elevation between pairs of samples.

Analyses of Individual Sets

Statistics and a set of plots were prepared for each pair of twinned holes. Graphs for each pair of twinned holes consisted of:

- Trend plots for sample pairs located at a maximum distance of 10.0 m
- Trend, scatter, quantile-quantile (Q-Q), and Au-relative difference plots for sample pairs within the 0.15 g/t contoured solids located at a maximum distance of 10.0 m.

Drill hole intervals and number of sample pairs used for these graphs are shown in Table 12-10.

Table 12-10 Sample Pairs with Distances ≤ 10.0 m & Sample Pairs within Solids

Twin Holes	Samples Pairs ≤ 10.0-m Distances			Sample Pairs ≤ 10-m Distances within Solids				
	N° Pairs	Min	Max	From	To	N° Pairs	Min	Max
CMD004 – CMR209	91	0.6	3	34	181.85	74	0.7	3
CMD198 – CMR089	40	2.1	2.9	0	72	36	2.1	2.9
CMD010– CMR018 (1)	83	5	5.5	0	126	63	5	5
CMD-092 – CMR002	171	2	8.7	0	126	63	2	3.8
CMD093 – CMR041	77	3.1	9.9	38	130	46	3.5	7.4
CMD096 – CMR030	91	3.3	9.7	14	136	61	3.4	6
CMD099 – CMR067(2)	57	3.5	9.8	-	-	-	-	-
CMD178 – CMR045	49	3.5	10	0	98	49	3.5	10
CMD192 – CMR097	75	3.3	9.7	0	126	63	3.3	8.1
CMD193 – CMR129	90	3.5	9.7	0	126	63	3.5	6.1
CMD196 – CMR098	125	2	6.5	0	126	63	2	2

(1): Not surveyed (2): Do not intersect solid

It should be noted that twin holes CMR-018 and CMD010 were not surveyed; therefore "real" distances between pairs of samples was unknown. The two holes were included in the comparison since surveys of other paired drill holes returned minimal deviation between holes at a 126 m hole depth – the depth of CMR018 and CMD010 – as shown in Table 12-11.

Table 12-11 Distances Between Pairs – 126 m Depth

DH Hole-ID	RC Hole-ID	Max Dist
CMD004	CMR209	1.4
CMD092	CMR002	3.8
CMD093	CMR041	7.1
CMD096	CMR030	5.4
CMD192	CMR097	8.1
CMD193	CMR129	5.8
CMD196	CMR098	5.9

Twin holes CMD099 and CMR067 are not included in this analysis, since neither intersected the solid.

Table 12-12 shows mean gold grades for each set of twin holes. Means were calculated for all pairs at a maximum distance of 10.0 m, as well as for pairs within solids (at a maximum distance of 10.0m). Results showed that the highest means occurred in either DDH or RC holes (highest values are highlighted in red in Table 12-12). Additionally, the overall average gold grades in DDH and RC holes were very similar, thus indicating that there was no global bias.

Table 12-12 Comparison - Average Au Grades: Pairs Maximum Distance ≤ 10 m & Pairs Maximum Distance ≤ 10 m within Solids

Hole-ID	(Au g/t) in DDH		Hole-ID	(Au g/t) in RC Holes	
DDH	All Pairs	Pairs in Solid	RC	All Pairs	Pairs in Solid
CMD004	0.382	0.434	CMR209	0.413	0.475
CMD198	0.852	0.893	CMR089	0.6	0.66
CMD010	0.277	0.231	CMR018	0.278	0.265
CMD092	0.861	0.718	CMR002	0.721	0.534
CMD093	0.191	0.247	CMR041	0.282	0.369
CMD096	0.304	0.418	CMR030	0.242	0.315
CMD178	0.331	0.331	CMR045	0.513	0.513
CMD192	0.44	0.425	CMR097	0.472	0.469
CMD193	0.509	0.673	CMR129	0.518	0.613
CMD196	0.57	1.09	CMR098	0.645	1.239
Mean	0.502	0.542		0.493	0.545

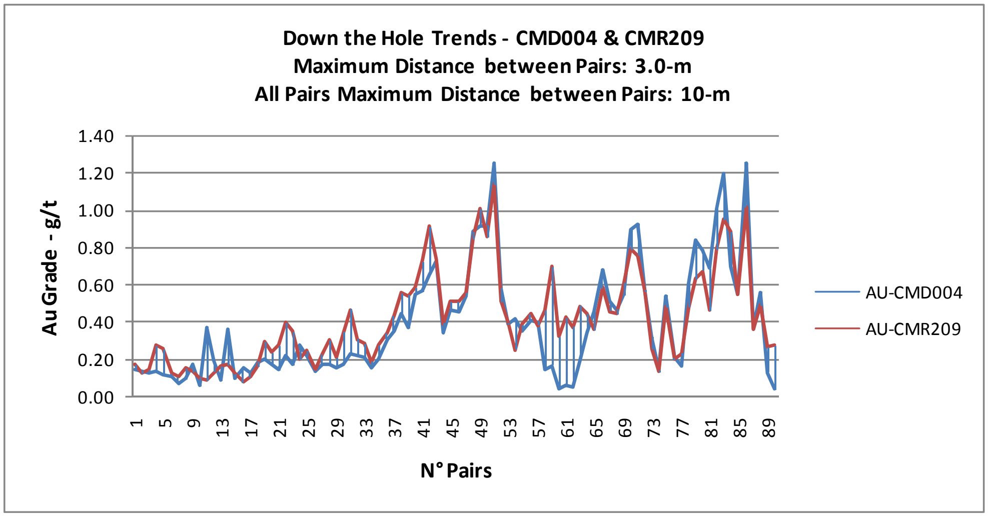
12.6.1 Statistics and Graphs for CMD004-CMR209-Lynx Zone

The following comments were pertinent regarding data shown in Table 12-13, Figure 12-8 and Figure 12-9

- The T statistic (-2.36) showed a significant difference between means, being the DDH mean higher than the RC mean, the global bias being -9.39
- The trend-plots (Figure 12-18) showed consistency between gold values of CMR-209 and CMD-004
- The scatter-plot (Figure 12-19) had a relatively low dispersion as shown by the correlation coefficient (0.875)
- The Q-Q plot (Figure 12-19) showed that RC grades were slightly higher than the DDH grades for gold values under to 0.40 ppm. Thereafter, gold grades followed a similar pattern
- The pair-wise relative difference v/s mean grade plot with a 10-term moving average (red line) showed a similar trend to the Q-Q plot.

Table 12-13 Summary Data – CMD004 – CMR 209 – Lynx Zone

Parameter	Summary of Data	
	CMD004	CMR209
Number	74	74
Minimum	0.044	0.139
Maximum	1.259	1.139
Mean	0.434	0.475
Estándar Deviation	0.301	0.233
T Test	-2.359	
Mean Rel. Err.	35.526	
BIAS (%)	-9.395	
Correlation-r	0.875	



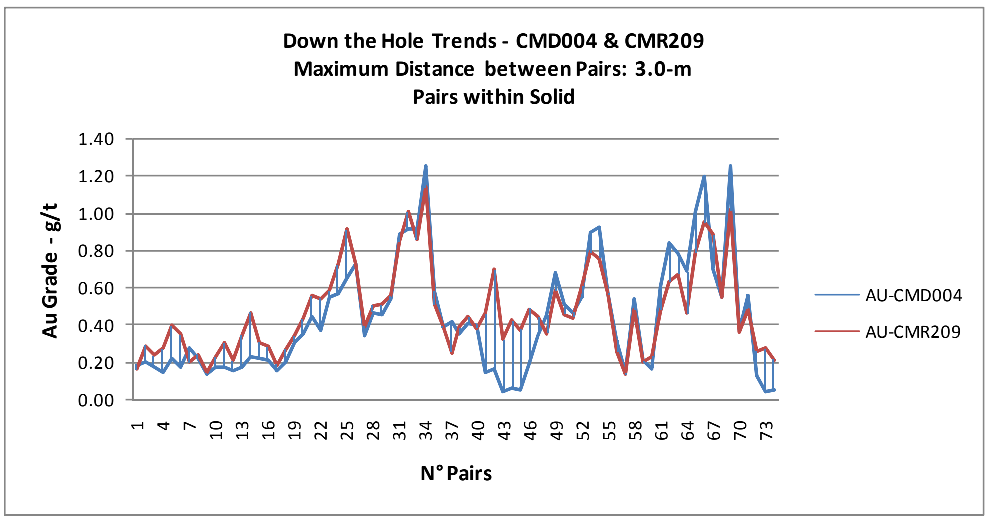


Figure 12-18 Trend Plots – Au Grades

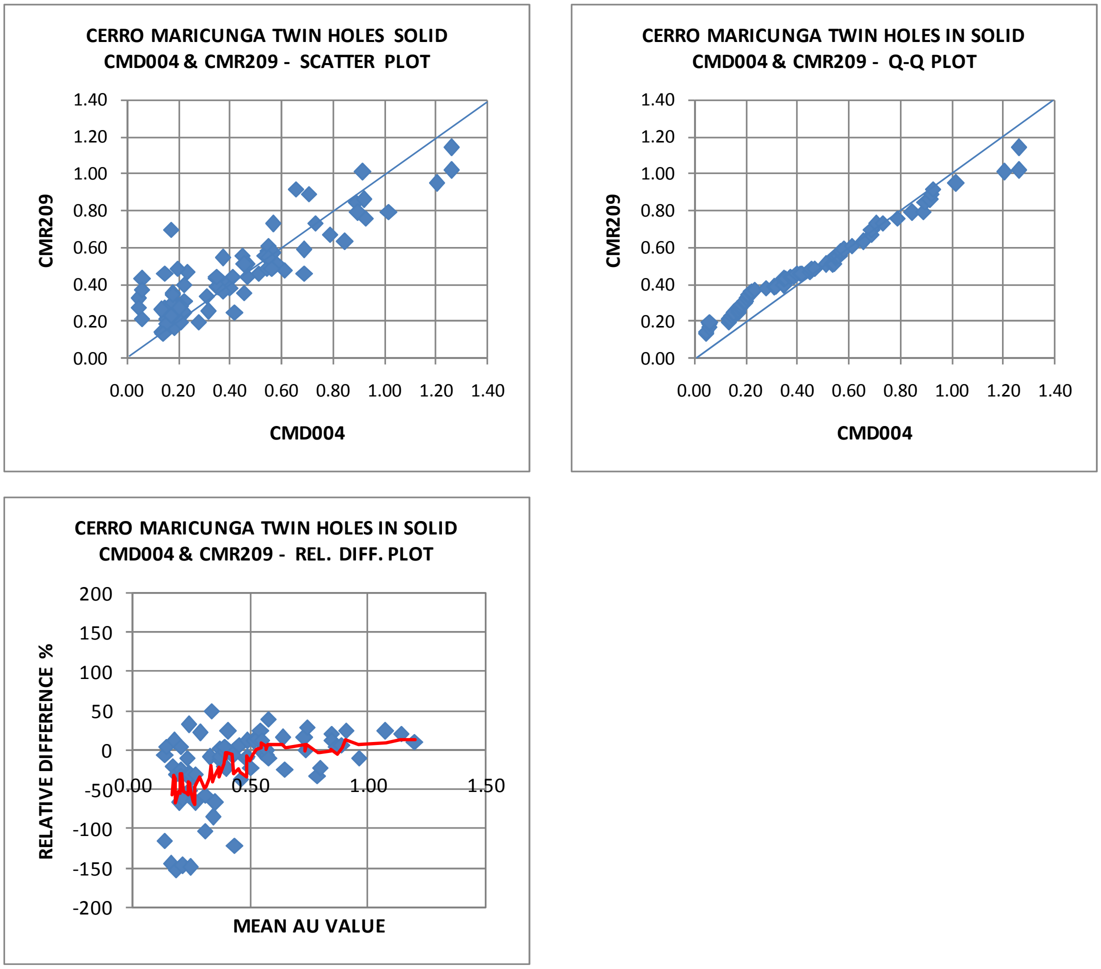


Figure 12-19 Scatter, Q-Q and Relative Difference Plots – Au

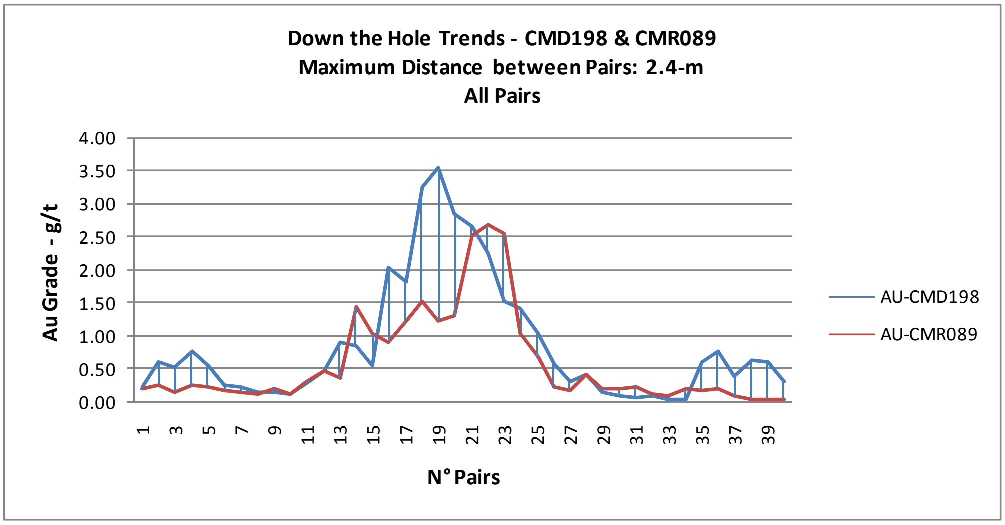
12.6.2 Statistics and Graphs for CMD198-CMR089-Lynx Zone

The following comments were pertinent regarding data shown in Table 12-14, Figure 12-21 and Figure 12-22.

- The T statistic (2.21) showed a significant difference between means, being the DDH mean higher than the RC mean, the global bias being 26.07%
- The trend-plots (Figure 12-21) showed similar overall trends, nevertheless DDH gold values were higher than RC values
- The scatter plot (Figure 12-22) had a relatively low dispersion for grades below 0.3 g/t, thereafter, DDH gold values tended to be higher
- The Q-Q plot (Figure 12-22) showed that DDH grades were higher than RC grades for gold values above 0.3 g/t
- The pair-wise relative difference v/s mean grade plot with a 10-term moving average (red line) showed a similar trend to the Q-Q plot.

Table 12-14 Summary Data – CMD198 – CMR 089 – Lynx Zone

Parameter	Summary of Data	
	CMD198	CMR089
Number	36	36
Minimum	0.042	0.103
Maximum	3.548	2.683
Mean	0.893	0.660
Estándar Deviation	0.970	0.728
T Test	2.213	
Mean Rel. Err.	46.795	
BIAS (%)	26.074	
Correlation-r	0.759	



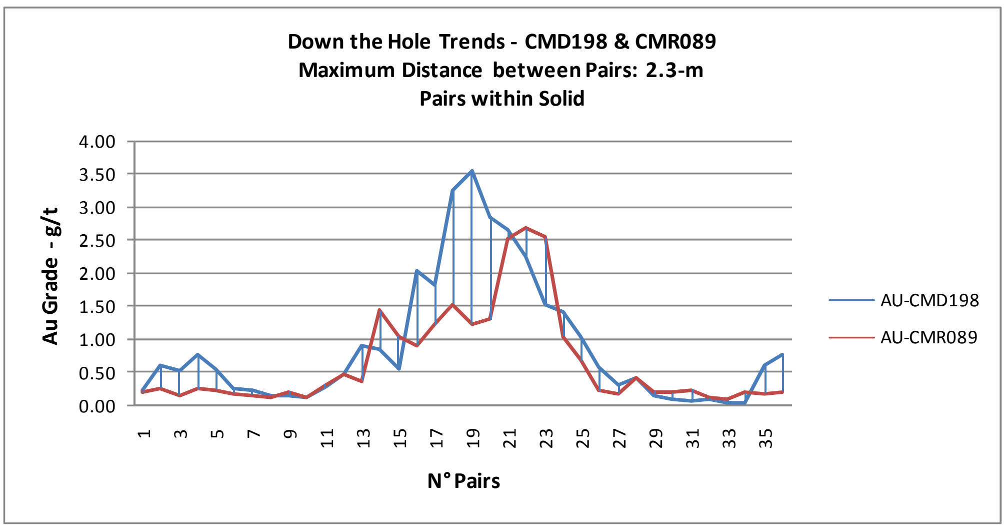


Figure 12-20 Trend Plots – Au Grades

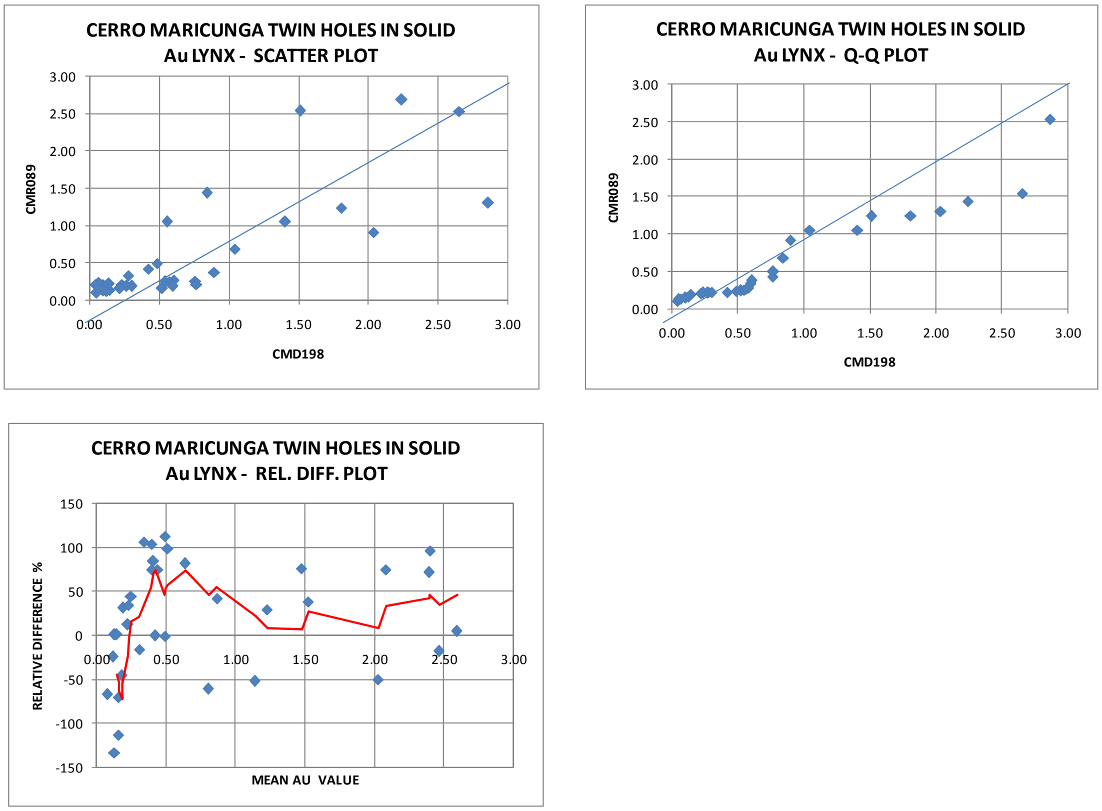


Figure 12-21 Scatter, Q-Q and Relative Difference Plots – Au

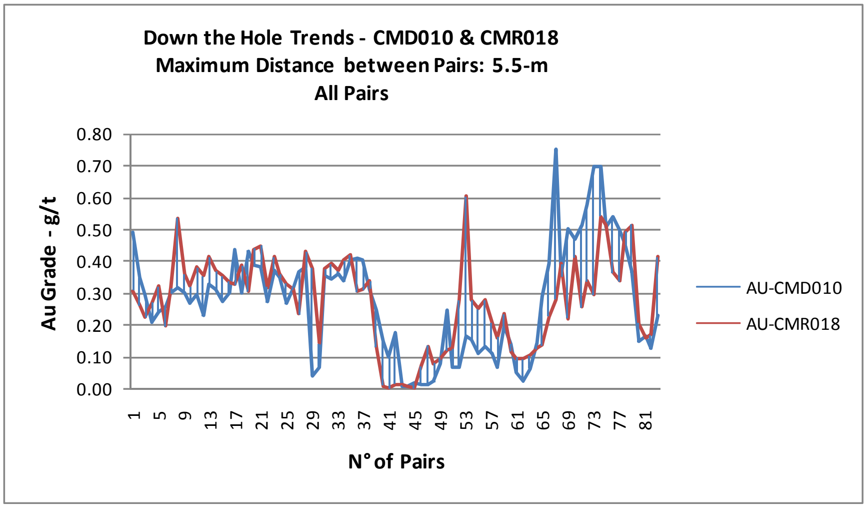
12.6.3 Statistics and Graphs for CMD010-CMR018-Phoenix Zone

The following comments were pertinent regarding data shown in Table 12-15, Figure 12-22 and Figure 12-23.

- The T statistic (-2.47) showed a significant difference between means (global bias equals (-14.68%))
- The trend-plot (Figure 12-22) showed fairly consistent trends between gold values of CMR-018 and CMD-010, except for a few erratic samples
- The scatter-plot (Figure 12-23) had a fairly large dispersion as indicated by the low correlation coefficient (0.695) and the large pair-wise mean relative error (50.29%)
- The Q-Q plot (Figure 12-23) showed a notable bias, being gold values in CMR018 higher than those assayed in CMD010. This observation was also observed in the pair-wise relative difference v/s mean grade plot with a 10-term moving average.

Table 12-15 Summary Data – CMD010 – CMR 018– Phoenix Zone

Parameter	Summary of Data	
	CMD010	CMR018
Number	63	63
Minimum	0.008	0.003
Maximum	0.493	0.607
Mean	0.231	0.265
Estándar Deviation	0.137	0.142
T Test	-2.467	
Mean Rel. Err.	50.295	
BIAS (%)	-14.679	
Correlation-r	0.695	



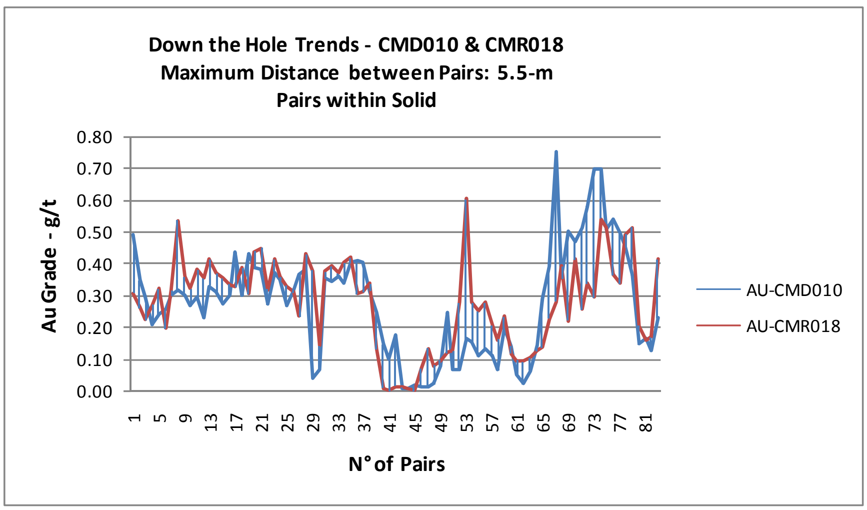


Figure 12-22 Trend Plots – Au Grades

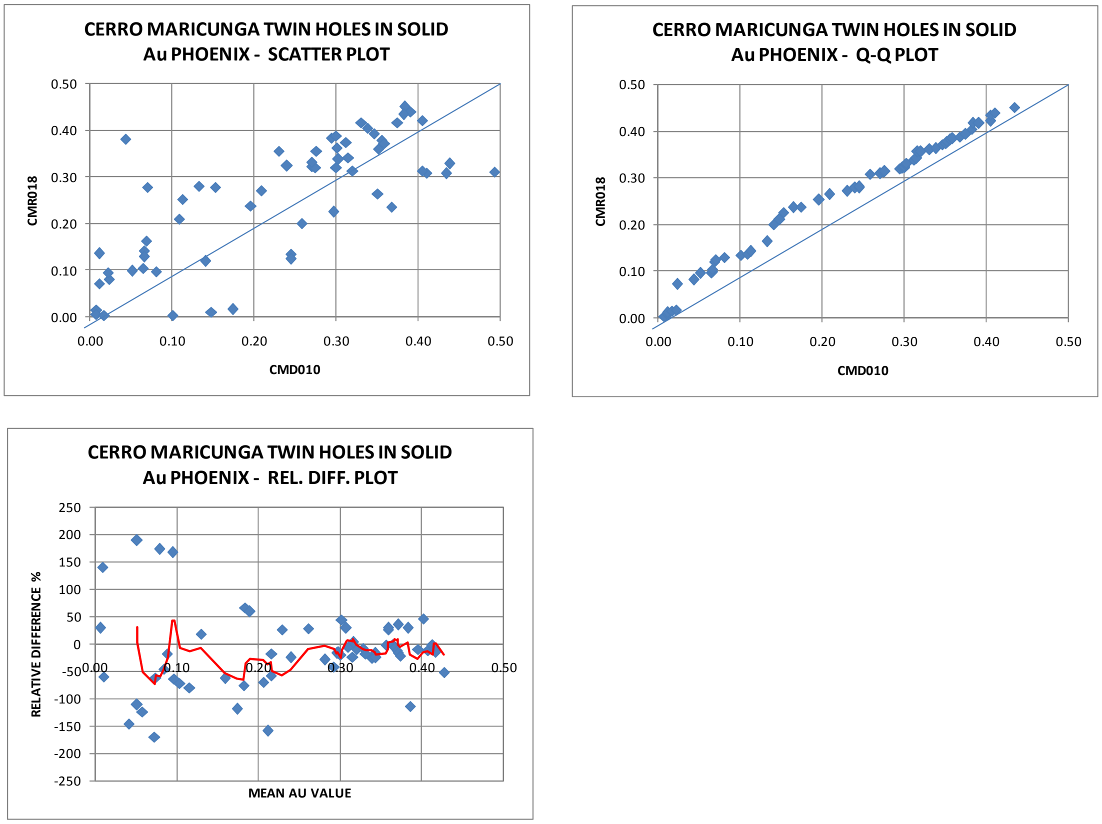


Figure 12-23 Scatter, Q-Q and Relative Difference Plots – Au

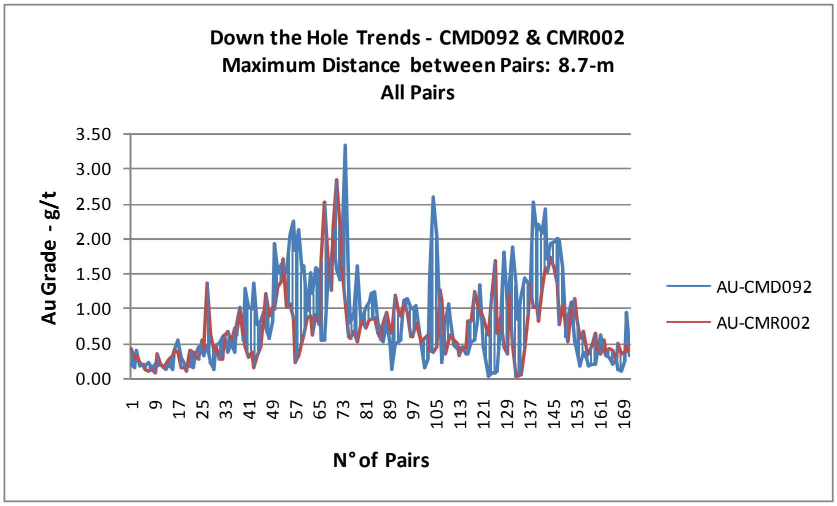
12.6.4 *Statistics and Graphs for CMD092-CMR002-Phoenix Zone*

The following comments were pertinent regarding data shown in Table 12-16, Figure 12-24 and Figure 12-25.

- The T statistic (2.86) showed a significant difference between means, being the DDH mean higher than the RC mean, the global bias being 25.58%
- The trend-plot (Figure 12-24) showed consistency between gold values of CMR-002 and CMD-092 down to approximately 78 m depth
- The scatter-plot (Figure 12-25) had a large dispersion as shown by the low correlation coefficient (0.536) and the large pair-wise mean relative error (45.90%), even though the distance between pairs of samples varied between 2.4 and 3.8 m
- The Q-Q plot (Figure 12-25) showed that RC grades were slightly higher than the DDH grades for gold values under to 0.7 ppm. Thereafter, the gold trend was reversed
- The pair-wise relative difference v/s mean grade plot with a 10-term moving average (red line) showed a similar trend to the Q-Q plot (Figure 12-25).

Table 12-16 Summary Data – CMD092 – CMR002– Phoenix Zone

Parameter	Summary of Data	
	CMD092	CMR002
Number	63	63
Minimum	0.117	0.078
Maximum	2.239	1.699
Mean	0.718	0.534
Estándar Deviation	0.600	0.369
T Test	2.863	
Mean Rel. Err.	45.902	
BIAS (%)	25.578	
Correlation-r	0.536	



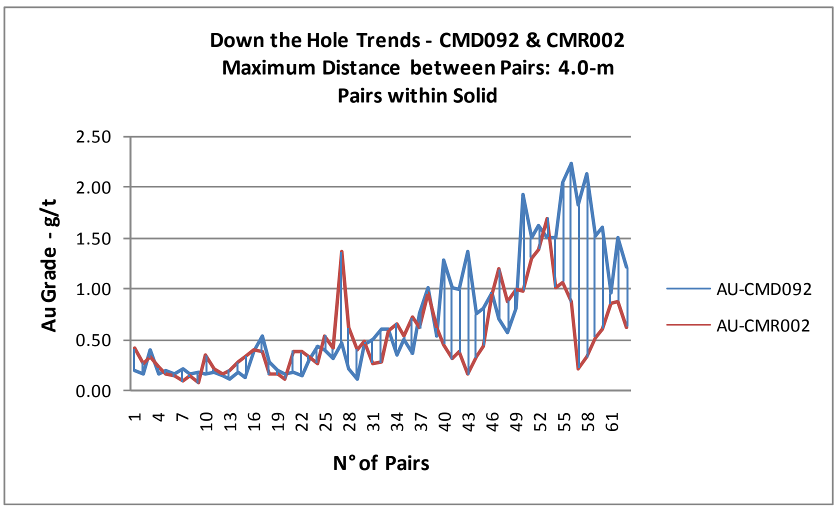


Figure 12-24 Trend Plots – Au Grades

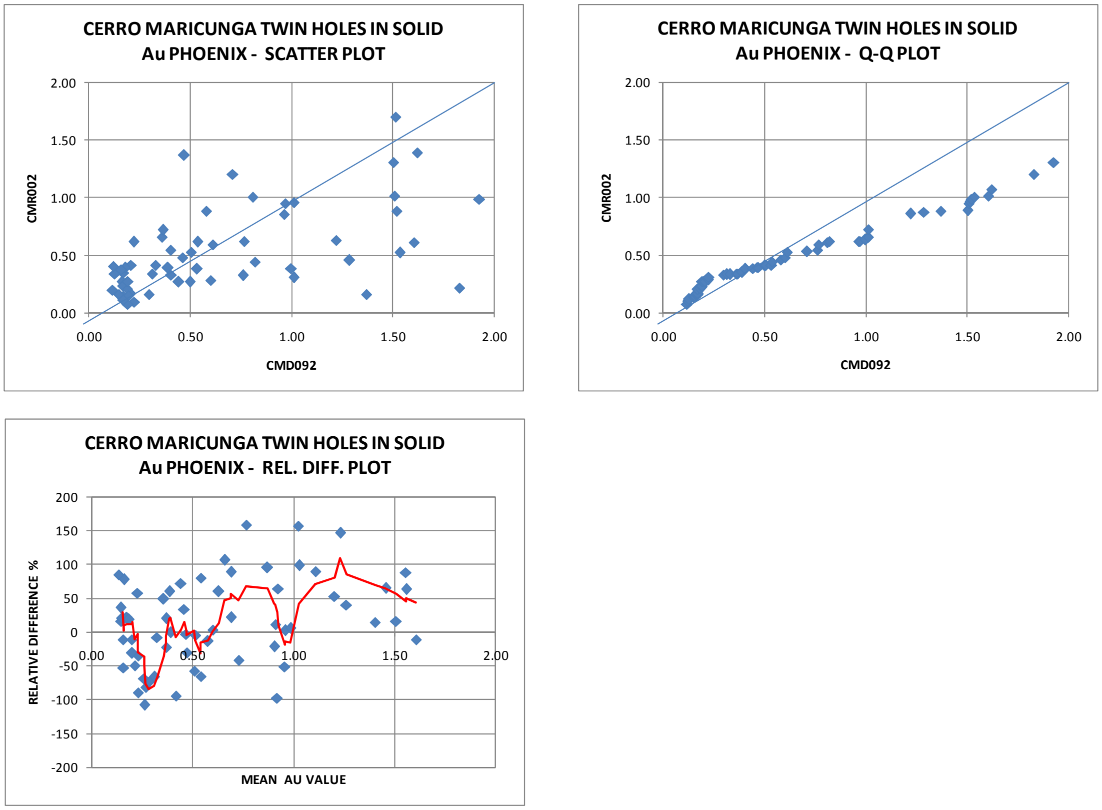


Figure 12-25 Scatter, Q-Q and Relative Difference Plots – Au

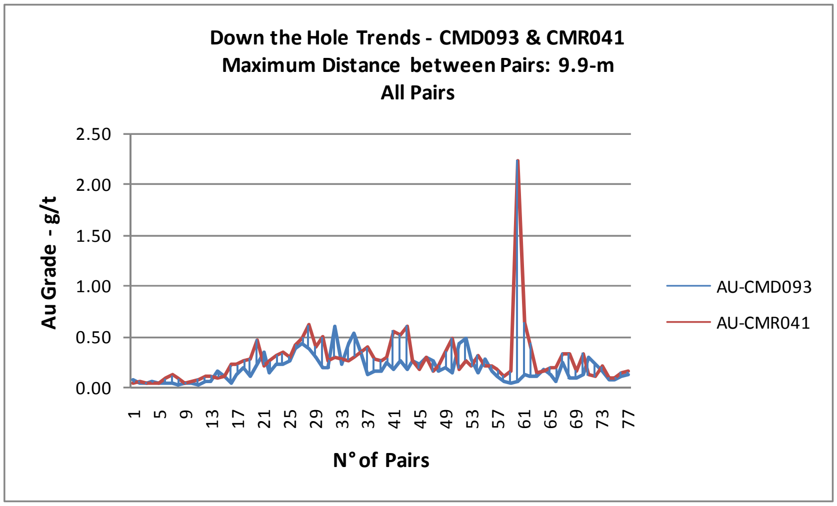
12.6.5 Statistics and Graphs for CMD093-CMR041-Phoenix Zone

The following comments were pertinent regarding data shown in Table 12-17, Figure 12-26 and Figure 12-27.

- The T statistic (-2.33) showed that the difference between means was significant, being the limit value equivalent to 1.96. The global bias was -49.36%, being RC gold values considerably higher than DDH gold values
- The trend-plot (Figure 12-26) showed lack of consistency between gold values, even though the values were consistently low except for a very high RC peak
- The scatter-plot (Figure 12-27) had a large dispersion as shown by the correlation coefficient (0.162) and the pair-wise mean relative error of -49.36%
- The Q-Q plot (Figure 12-27) showed that RC gold values were consistently higher than DDH
- The pair-wise relative difference v/s mean grade plot with a 10-term moving average (red line) showed a similar trend to the Q-Q plot (Figure 12-27).

Table 12-17 Summary Data – CMD093 – CMR041– Phoenix Zone

Parameter	Summary of Data	
	CMD093	CMR041
Number	46	46
Minimum	0.056	0.125
Maximum	0.603	2.243
Mean	0.247	0.369
Estándar Deviation	0.125	0.313
T Test	-2.330	
Mean Rel. Err.	45.634	
BIAS (%)	-49.363	
Correlation-r	0.162	



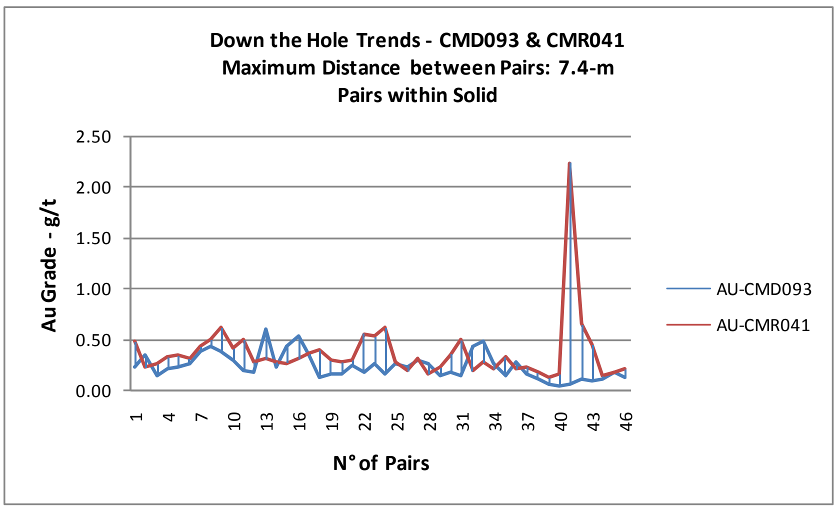


Figure 12-26 Trend Plots – Au Grades

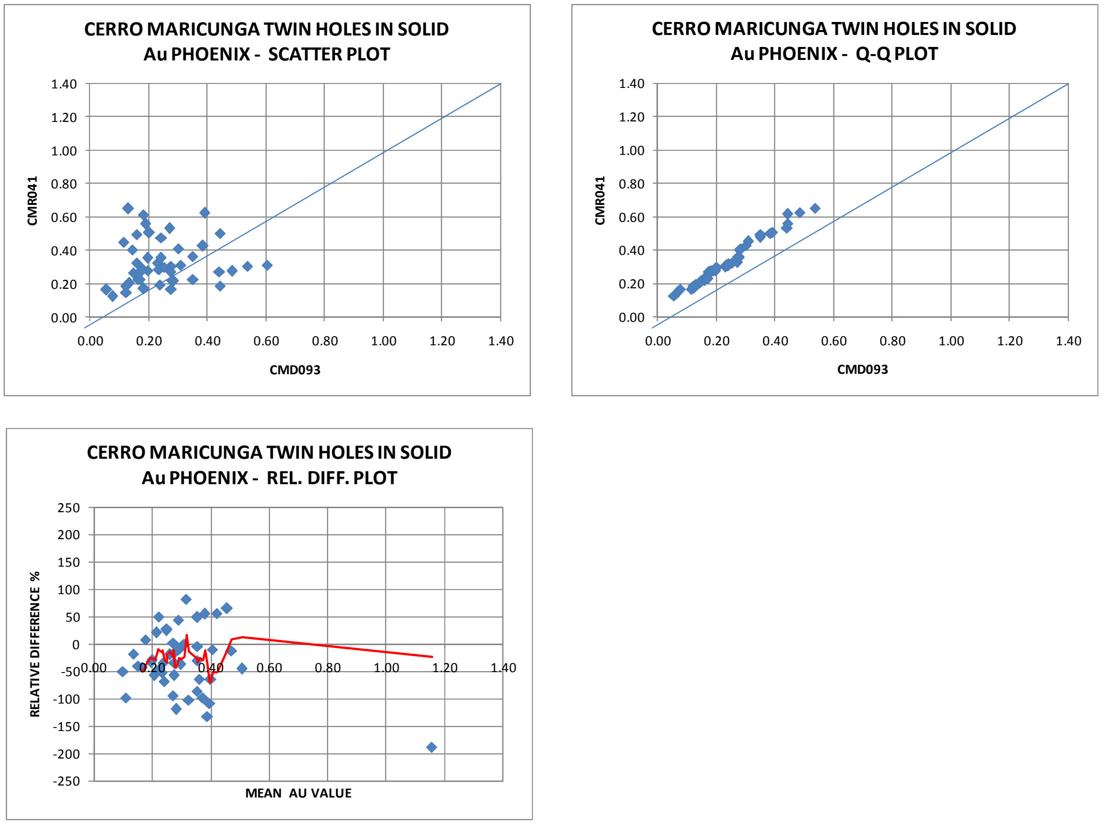


Figure 12-27 Scatter, Q-Q and Relative Difference Plots – Au

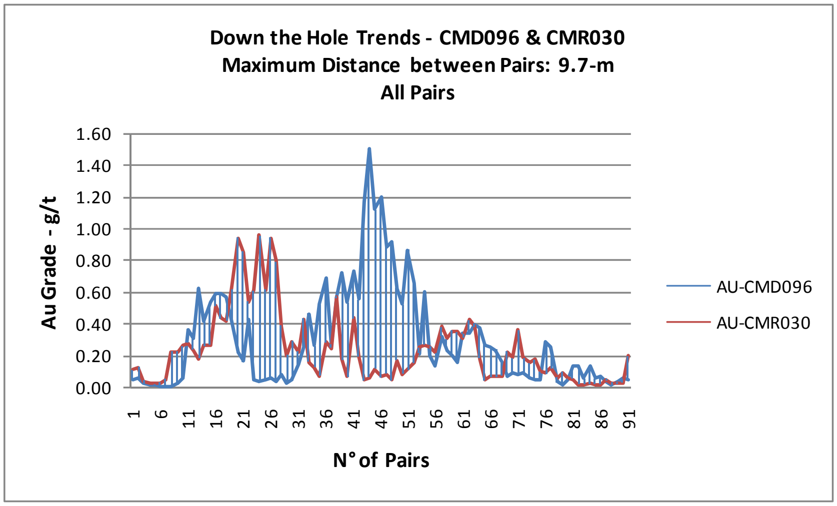
12.6.6 Statistics and Graphs for CMD096-CMR030-Phoenix Zone

The following comments were pertinent regarding data shown in Table 12-18, Figure 12-28 and Figure 12-29.

- The T statistic (1.69) showed that there was no significant difference between means however the global was 24.79%
- The trend-plot (Figure 12-28) showed anomalous behavior since the trends were completely different
- The scatter-plot (Figure 12-29) had a large dispersion as shown by the low correlation coefficient (0.434) and the large pair-wise mean relative error (79.31%), even though the distance between pairs varied between 3.4 m and 6.0 m
- The Q-Q plot and the pair-wise relative difference v/s mean grade plot with a 10-term moving average (Figure 12-29) showed that DDH gold values were higher than RC for grades lower than 0.3 ppm. Thereafter the trend was reversed.

Table 12-18 Summary Data – CMD096 – CMR030– Phoenix Zone

Parameter	Summary of Data	
	CMD096	CMR030
Number	61	61
Minimum	0.012	0.049
Maximum	1.504	0.964
Mean	0.418	0.315
Estándar Deviation	0.327	0.235
T Test	1.693	
Mean Rel. Err.	79.307	
BIAS (%)	24.785	
Correlation-r	0.434	



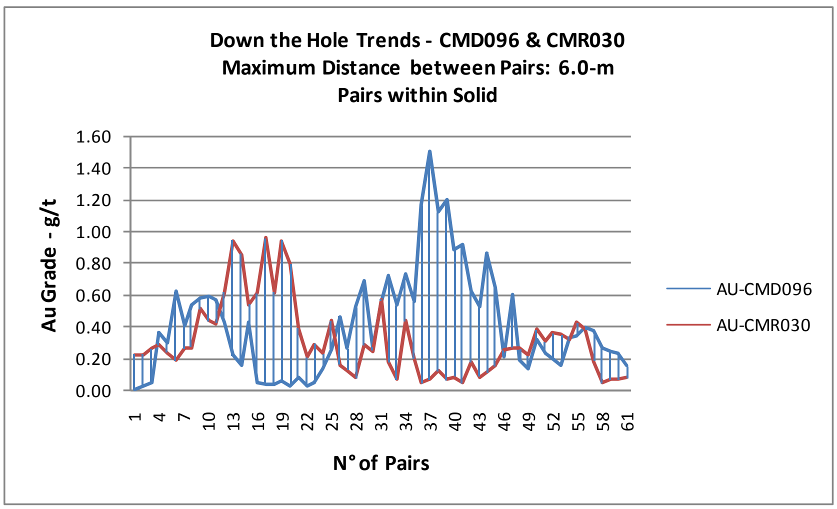


Figure 12-28 Trend Plots – Au Grades

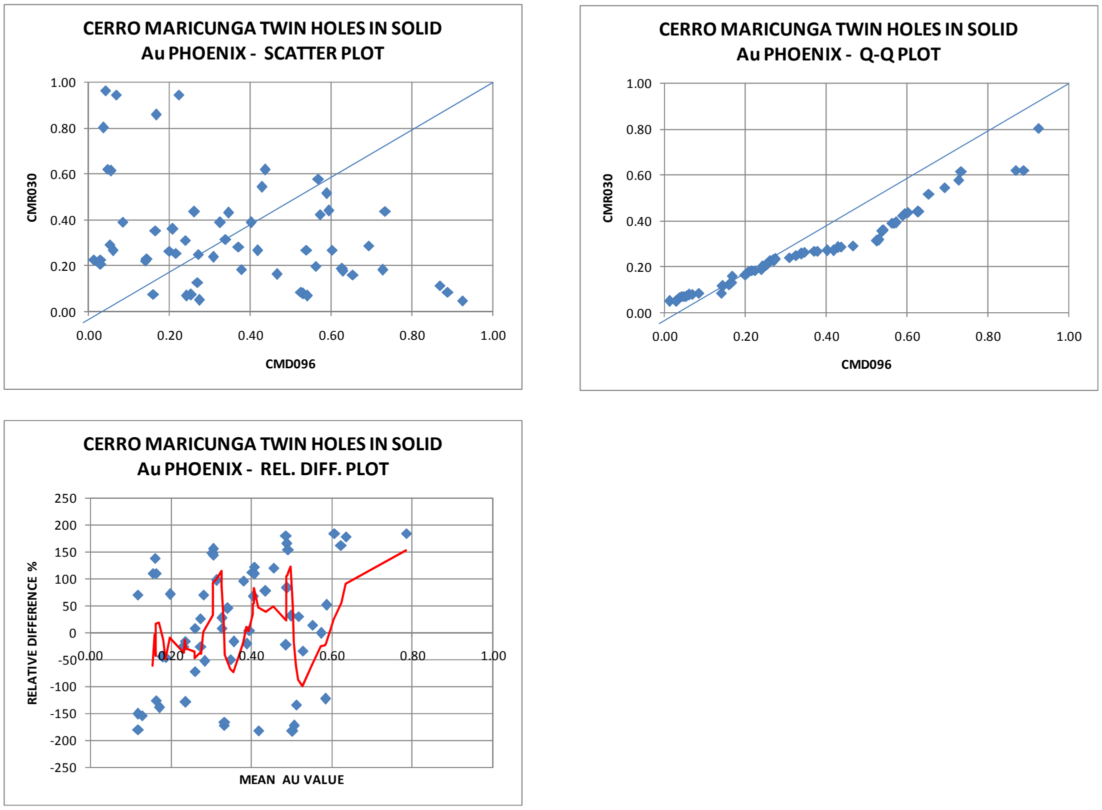


Figure 12-29 Scatter, Q-Q and Relative Difference Plots – Au

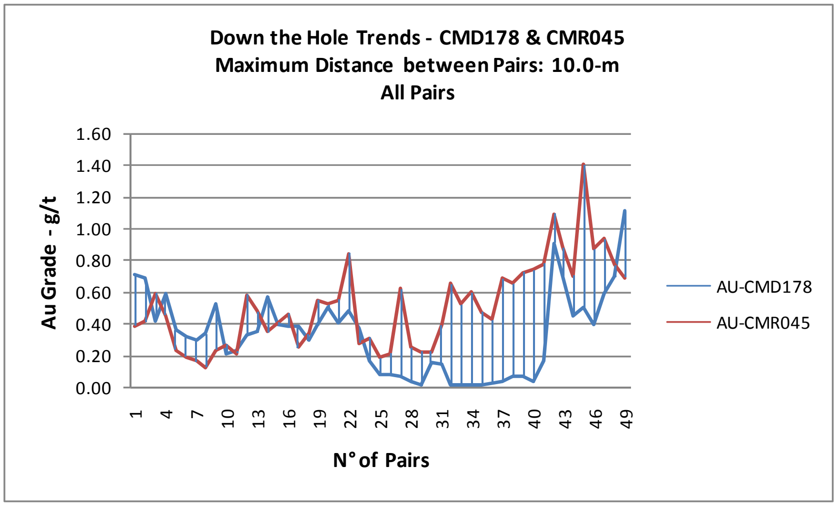
12.6.7 Statistics and Graphs for CMD178-CMR045-Phoenix Zone

The following comments were pertinent regarding data shown in Table 12-19, Figure 12-30 and Figure 12-31.

- The T statistic (-4.12) showed that there was a very significant difference between means, being the global bias -55.03%
- The trend-plot (Figure 12-30) showed anomalous behavior since the trends were completely different
- The scatter-plot (Figure 12-31) had a large dispersion as shown by the low correlation coefficient (0.310) and the large pair-wise mean relative error (69.84%)
- The Q-Q plot and the pair-wise relative difference v/s mean grade plot with a 10-term moving average (Figure 12-31) showed that RC gold values were consistently higher than the DDH gold values.

Table 12-19 Summary Data – CMD178 – CMR045– Phoenix Zone

Parameter	Summary of Data	
	CMD178	CMR045
Number	49	49
Minimum	0.016	0.131
Maximum	1.114	1.406
Mean	0.331	0.513
Estándar Deviation	0.258	0.269
T Test	-4.124	
Mean Rel. Err.	69.842	
BIAS (%)	-55.035	
Correlation-r	0.310	



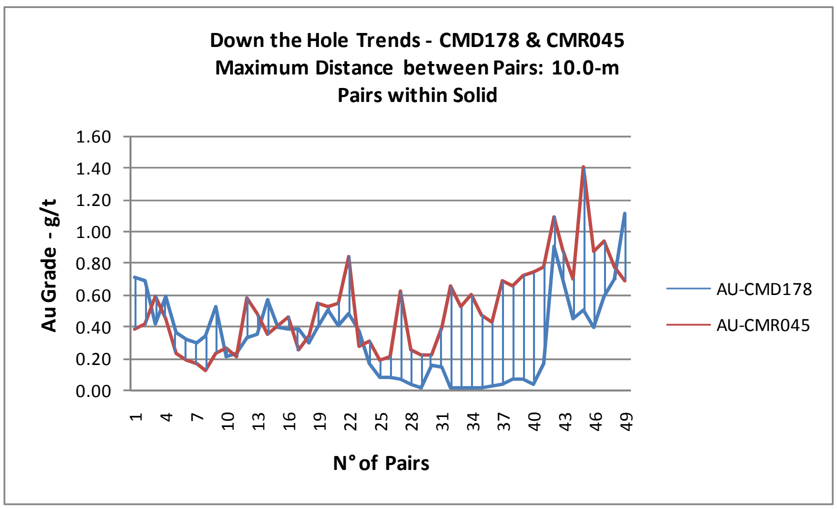


Figure 12-30 Trend Plots – Au Grades

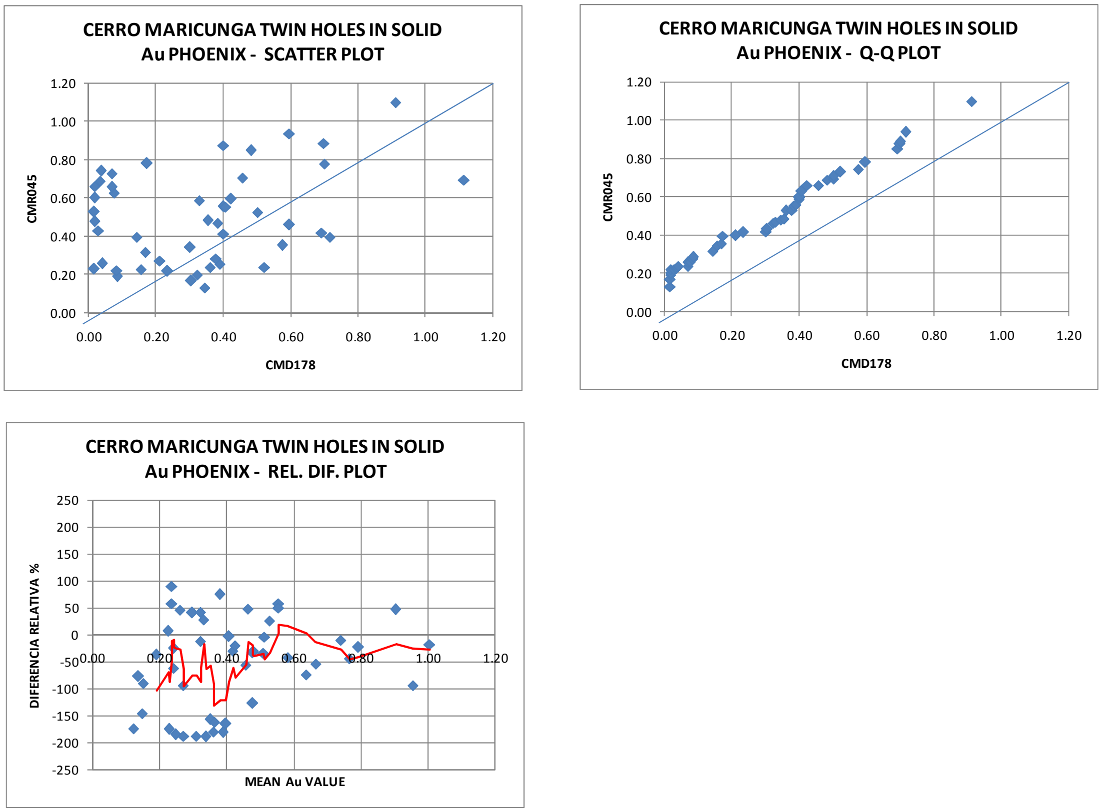


Figure 12-31 Scatter, Q-Q and Relative Difference Plots – Au

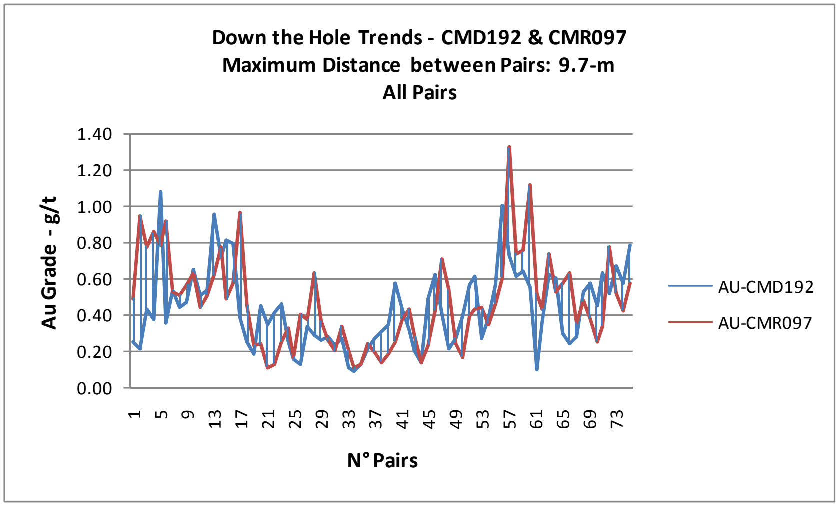
12.6.8 Statistics and Graphs for CMD192-CMR097-Crux Zone

The following comments were pertinent regarding data shown in Table 12-20, Figure 12-32 and Figure 12-33.

- The T statistic (-1.36) showed that there was no significant difference between means however the global was -10.26%
- The trend-plot (Figure 12-32) showed similar trends for both holes
- The scatter-plot (Figure 12-33) had a large dispersion as shown by the low correlation coefficient (0.468) and a significant pair-wise mean relative error (37.55%), even though the distance between pairs vary between 3.3 m and 8.1 m
- The Q-Q plot showed very low bias for gold values below 0.6 g/t. Thereafter, RC gold values were higher.

Table 12-20 Summary Data – CMD192 – CMR097– Crux Zone

Parameter	SUMMARY OF DATA	
	CMD192	CMR097
Number	63	63
Minimum	0.096	0.117
Maximum	1.084	1.326
Mean	0.425	0.469
Estándar Deviation	0.223	0.265
T Test	-1.361	
Mean Rel. Err.	37.552	
BIAS (%)	-10.257	
Correlation-r	0.468	



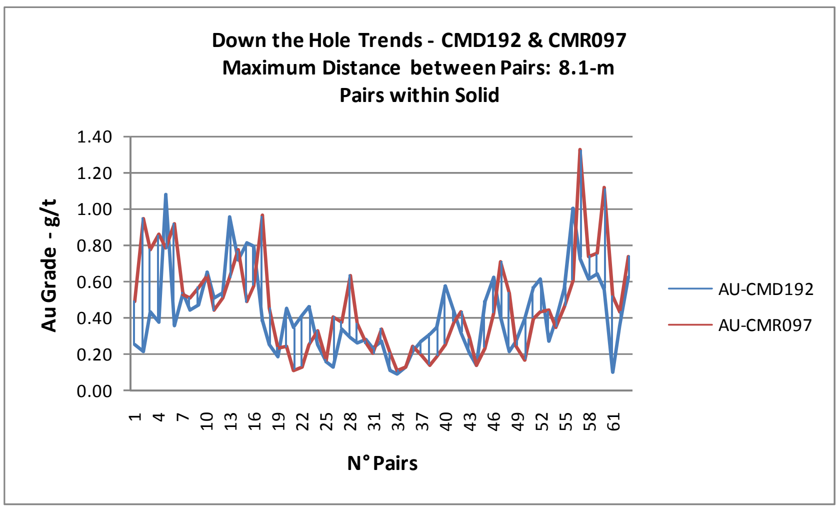
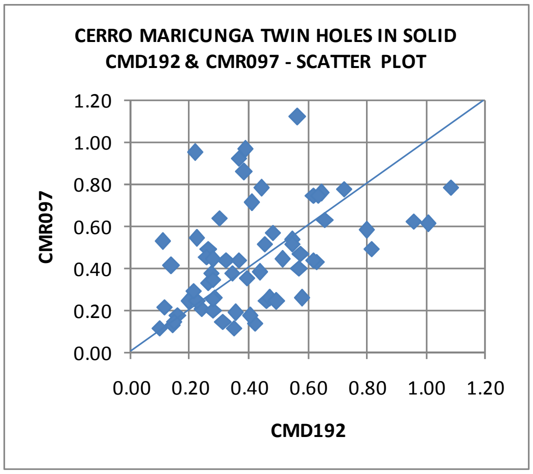
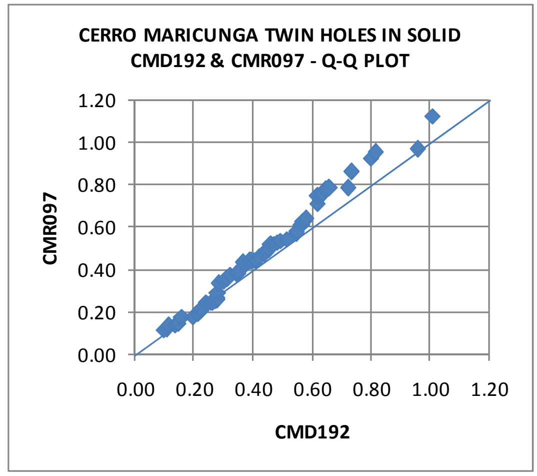


Figure 12-32 Trend Plots – Au Grades





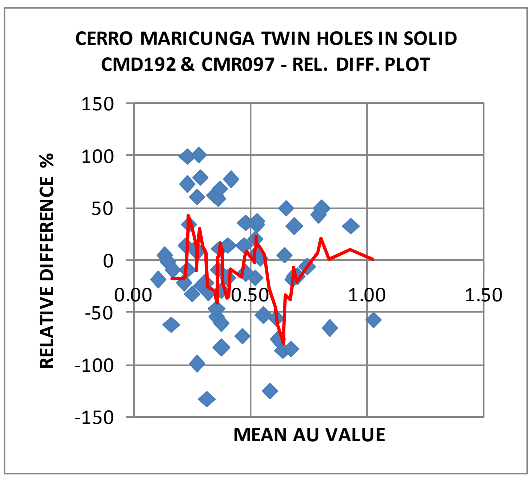


Figure 12-33 Scatter, Q-Q and Relative Difference Plots – Au

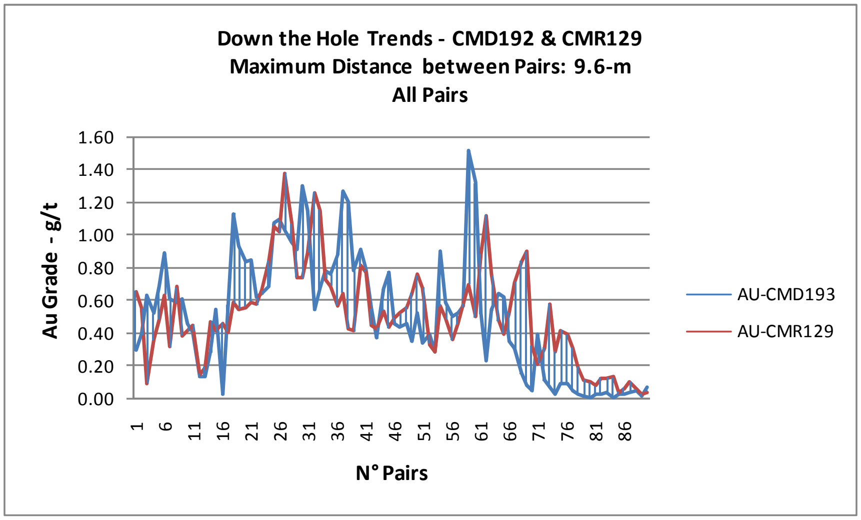
12.6.9 *Statistics and Graphs for CMD193-CMR129-Crux Zone*

The following comments were pertinent regarding data shown in Table 12-21, Figure 12-34 and Figure 12-35.

- The T statistic (1.43) showed that there was no significant difference between means and the global bias was 8.86%
- The trend-plot (Figure 12-34) showed different trends for both holes, although the overall RC and DDH means were close
- The scatter-plot (Figure 12-35) had a large dispersion as shown by the low correlation coefficient (0.349) and a significant pair-wise mean relative error (36.97%), even though the distance between pairs varied between 3.5 m and 6.1 m
- The Q-Q plot showed that gold values below 0.6 g/t fell fairly close to the first bisector. Thereafter, DDH gold values were higher.

Table 12-21 Summary Data – CMD193 – CMR129– Crux Zone

Parameter	Summary of Data	
	CMD193	CMR129
Number	63	63
Minimum	0.025	0.097
Maximum	1.515	1.379
Mean	0.673	0.613
Estándar Deviation	0.315	0.257
T Test	1.434	
Mean Rel. Err.	36.968	
BIAS (%)	8.862	
Correlation-r	0.349	



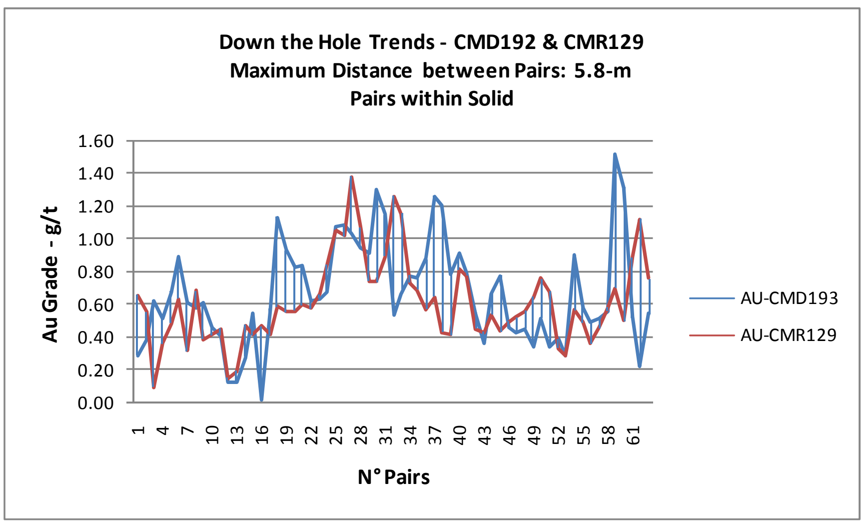


Figure 12-34 Trend Plots – Au Grades

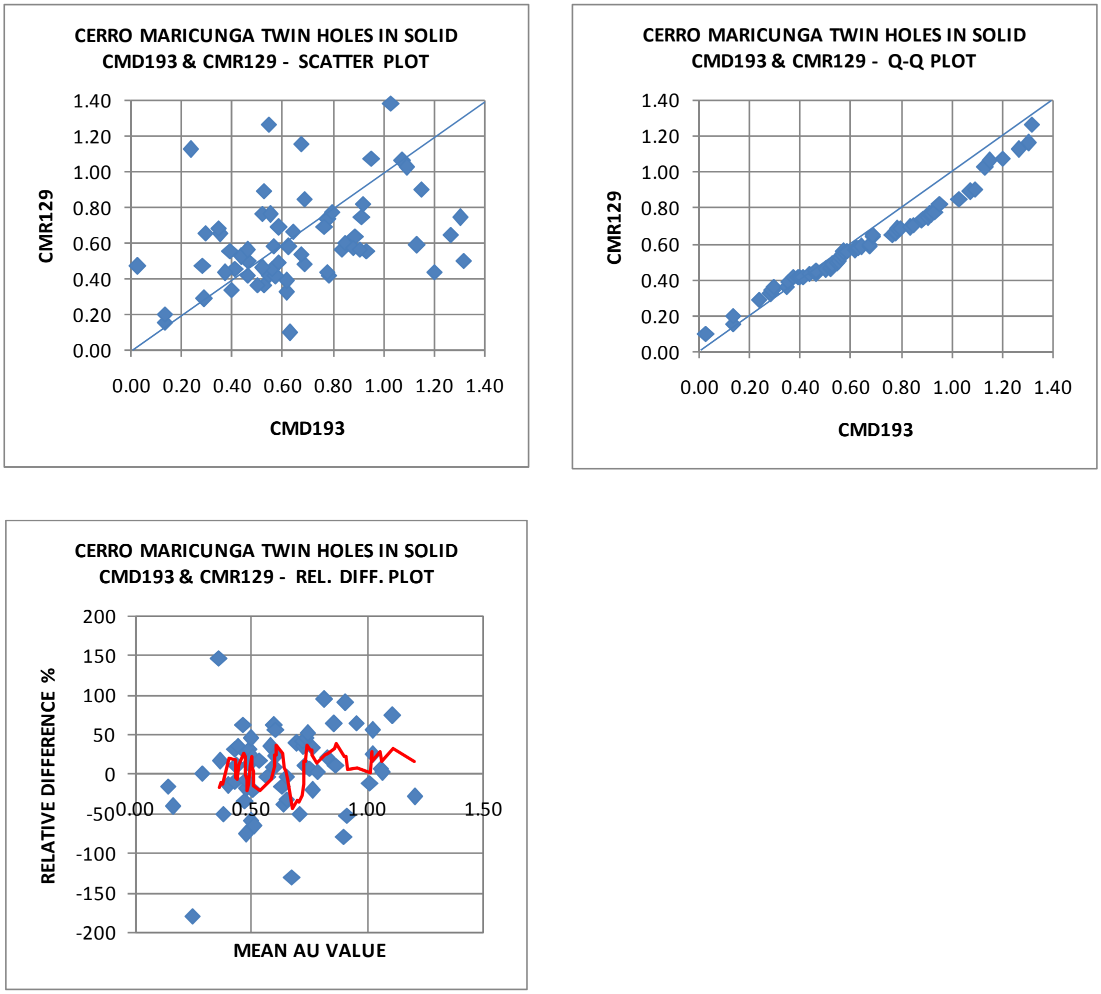


Figure 12-35 Scatter, Q-Q and Relative Difference Plots – Au

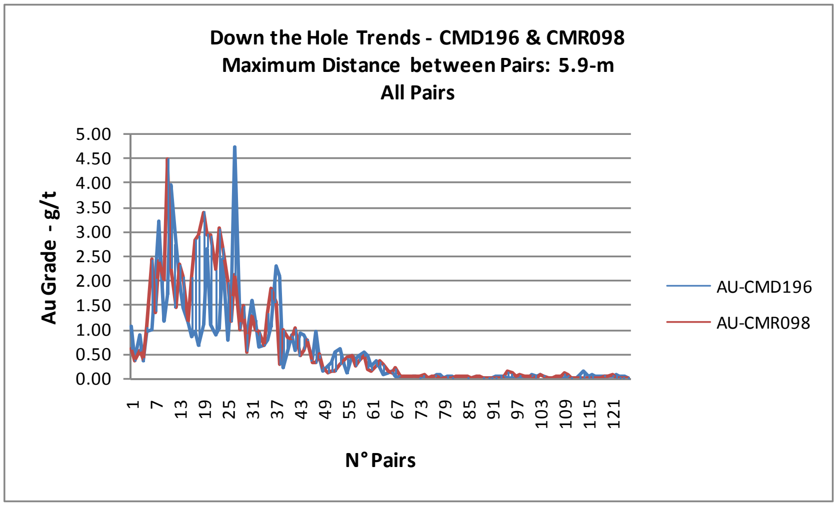
12.6.10 Statistics and Graphs for CMD196-CMR098-Crux Zone

The following comments were pertinent regarding data shown in Table 12-22, Figure 12-36 and Figure 12-37.

- The T statistic (-1.24) showed that there was no significant difference between means, the global bias being -13.62%
- The trend-plot (Figure 12-36) showed similar trends for both holes, except between 30.0 to 50.0 m depth, where RC gold values were much higher than DDH gold values
- The scatter-plot (Figure 12-37) had a large dispersion as shown by the fairly low correlation coefficient (0.512) and a significant pair-wise mean relative error (44.79%)
- The Q-Q plot showed that gold values below 0.8 g/t fall fairly close to the first bisector. Thereafter, RC gold values were higher.

Table 12-22 Summary Data – CMD193 – CMR129– Crux Zone

Parameter	Summary of Data	
	CMD196	CMR098
Number	63	63
Minimum	0.121	0.118
maximum	4.724	4.491
Mean	1.090	1.239
Standar Deviation	0.904	1.016
T Test	-1.236	
Mean Rel. Err.	44.792	
BIAS (%)	-13.618	
Correlation-r	0.512	



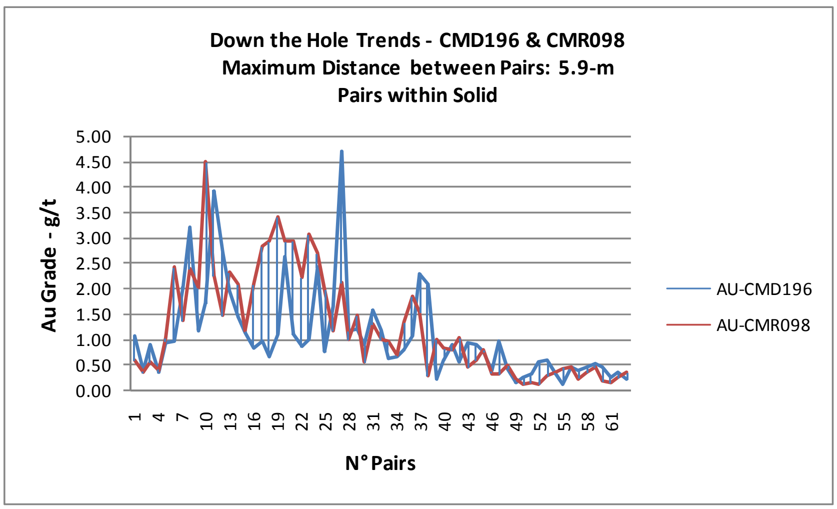
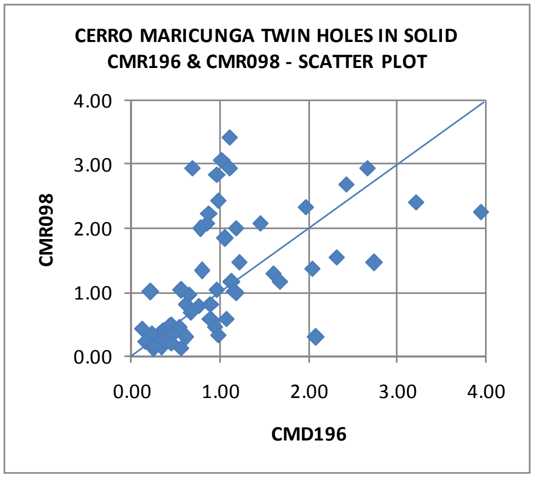
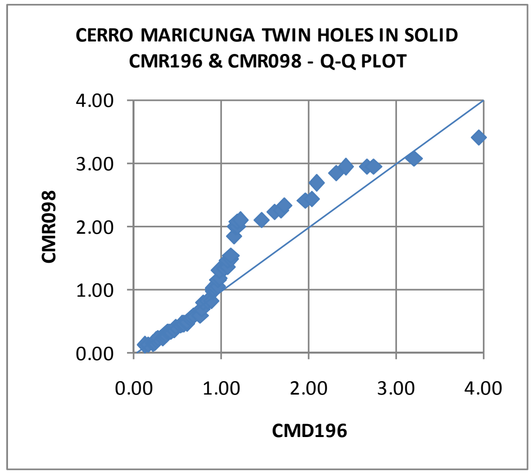


Figure 12-36 Trend Plots – Au Grades





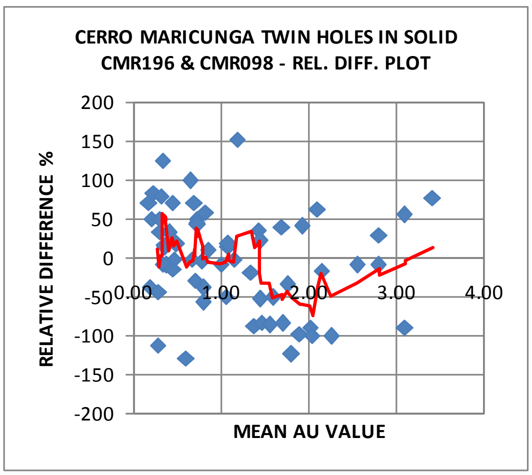


Figure 12-37 Scatter, Q-Q and Relative Difference Plots – Au

12.6.11 Statistics and Graphs for all Twin Holes

The analyses carried out in the previous sections showed a variable behavior; global biases varied from 8.86 (DDH higher than RC) to -55.03% (RC higher than DDH).

The analyses that follow show the overall behavior within the mineralized envelopes.

The methodology was analogous to the one applied for individual sets of twin holes.

The following comments were made regarding statistics and graphical analyses (Figure 12-23, Figure 12-38 and Figure 12-39).

- The T statistic was not significant (-0.15) and the average gold grades for RC (0.545 g/t) and DDH (0.542 g/t) were very close being the overall bias -0.56%
- The trend-plots were replaced by a graph consisting of:
    - RC – DDH paired values were sorted from low to high according to the RC gold grades and plotted against an arbitrary sequence number ranging from 1 to 581 (total number of pairs)
    - DDH grades, as well as a ten-term moving average were plotted against the same sequence number.

This graph (Figure 12-38) showed that DDH gold values varied around the sorted RC values for grades that fell between 0.4 and 0.5g/t Au, and that DDH gold values tended to be higher within the 0.0 – 0.4 g/t Au interval, and lower for grades above 0.5 g/t Au.

- The scatter-plot (Figure 12-39) had a fairly large dispersion as shown by the correlation coefficient (0.602) and the pair-wise mean relative error of 50.54%
- The Q-Q plot (Figure 12-39) showed the following trends:
    - RC-Au > DDH Au values in 0.003 to 0.4 g/t range
    - RC-Au and DDH-Au values ranging from 0.4 to 1.8 g/t fall close to the first bisector
    - RC-Au > DDH Au for values between 1.8 and 3.0 g/t
    - DDH-Au > RC-Au values above 3.0 g/t.

The pair-wise relative difference v/s mean grade plot with a 10-term moving average (red line) shows a similar trend to that of the QQ plot.

Table 12-23 Summary Data – All Twinholes

Parameter	Summary of Data	
	ALL DDH	ALL RC
Number	581	581
Minimum	0.008	0.003
maximum	4.724	4.491
Mean	0.542	0.545
Standar Deviation	0.549	0.521
T Test	-0.154	
Mean Rel. Err.	50.542	
BIAS (%)	-0.563	
Correlation-r	0.602	

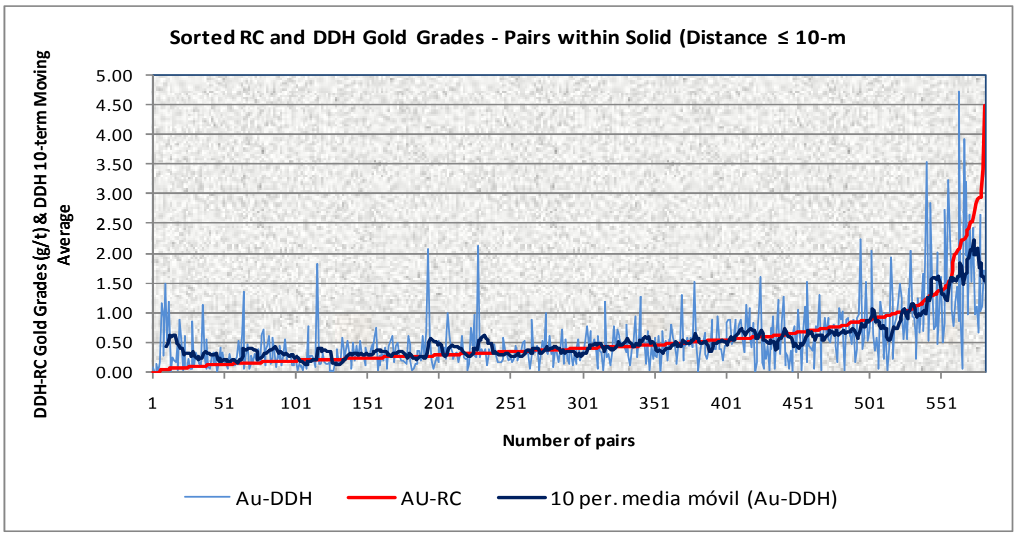


Figure 12-38 Sorted RC Au Grades, DDH Au Grades and DDH 10-term Moving Average

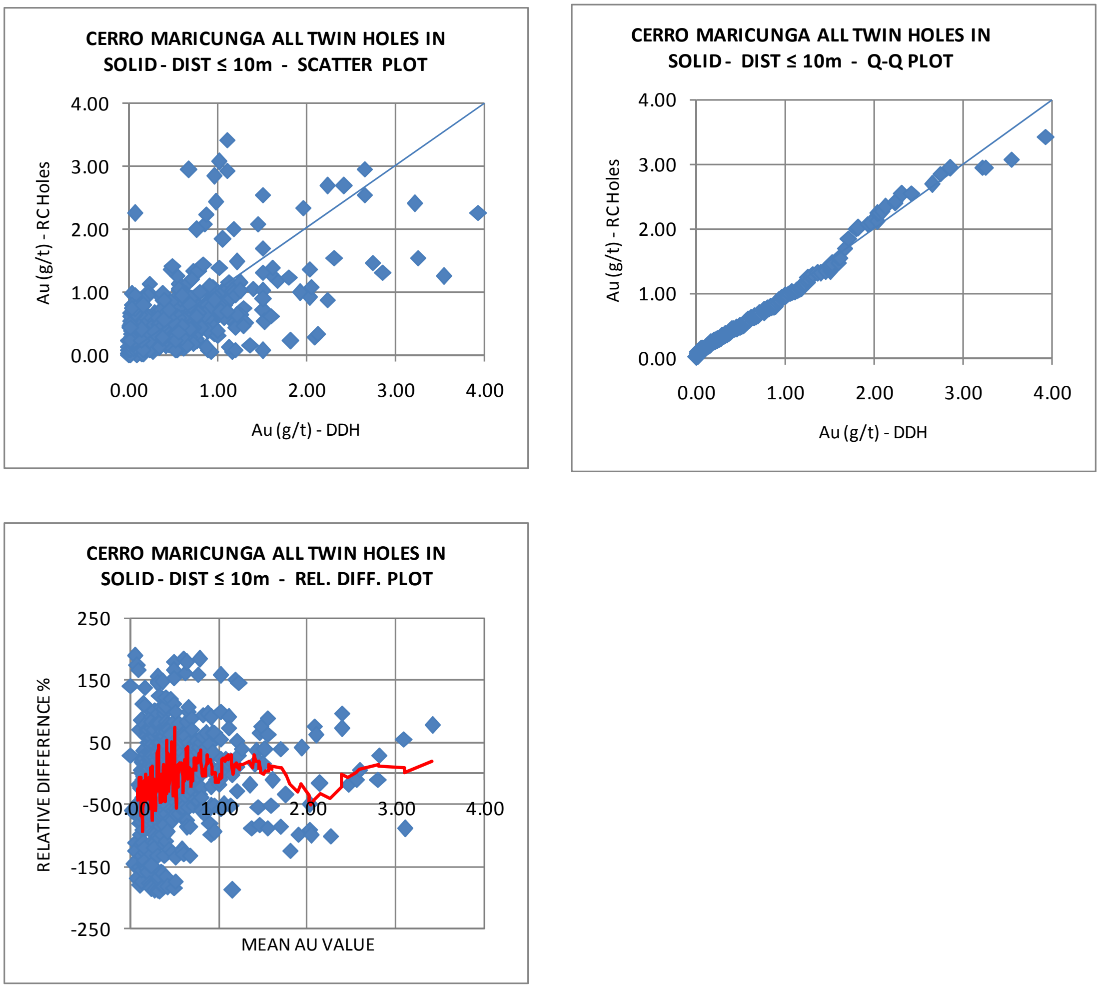


Figure 12-39 Scatter, Q-Q and Relative Difference Plots - Au

12.7 Estimation of Nugget Effect

It is well known that theoretically the nugget effect is inversely proportional to the sample support (volume or weight).

The sample mass for diamond drill holes (DDH) and reverse circulation drill holes (RC) was calculated and is presented in Table 12-24.

Table 12-24 Sample Mass for DDH and RC Holes

Drillhole Type	Diameter	Diameter (cm)	Length (cm)	Density	Weight (kg)
RC	5.5"	13.97	200	2.43	74.5
DDH	HQ	6.35	200	2.43	15.4

The sample collection and preparation protocols are as follows:

Diamond Drill Holes:

1. Two metre core samples were cut lengthwise in two halves using a diamond saw. One half was kept in the core storage facility and the second half, weighing some 8.2 kg was sent for sample preparation and assaying.
2. The 8.2 kg sample was crushed to 10# (2.0 mm) by means of a jaw crusher.
3. A one kg sample split was obtained using a rotary divider.
4. The 1000 g sample was pulverized to 150# (0.106 mm).
5. Two 250 g and one 500 g envelopes were obtained by increments. One envelope was sent for 50g fire assay with AA finish.

Reverse Circulation Drill Holes

1. One quarter of the approximately 80 kg sample was obtained on the field by means of a riffle splitter (approximately 20 kg).
2. The rest of the sample preparation protocol was basically the same as points 2 to 5 of the protocol used for diamond drill holes.

The inverse proportionality of the nugget effect to the sample mass would be true if the two samples were prepared and assayed in the same way, i.e. the complete sample were reduced to 10# and a 1000 split were obtained, pulverized and assayed.

As the sample preparation protocols were quite different for the two kinds of samples, the nugget effects were no longer related and it was best to calculate them empirically.

The nugget effects for DDH and RC samples were estimated as follows:

- All samples were 2.0 m in length
- Two metre samples were assigned individual coordinates using a GEMCOM routine
- Samples were selected using the following criteria:
    - Only samples within mineralized envelopes were used
    - DDH and RC sample pairs separated by at most 10m were used.
- Traditional variograms (with standardized sills) and (1-correlograms) were calculated for each type of sample
- The lag distance used was 2.0 m with a lag tolerance of 1.0
- Average down hole variograms and (1-correlograms) were calculated for both types of samples.

Results are presented in the following figures, where the red experimental variogram corresponds to DDH and the black experimental variogram corresponds to the RC holes.

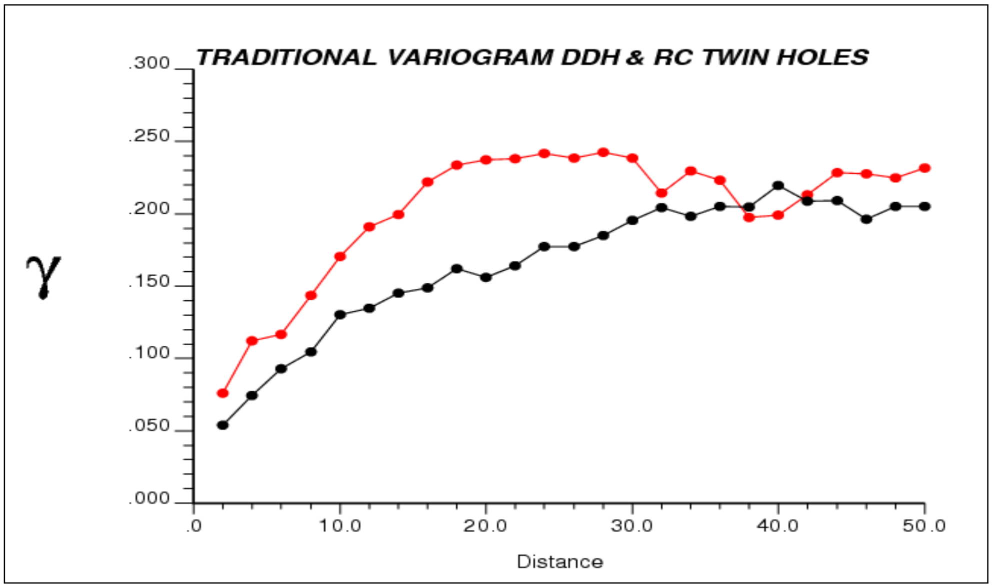


Figure 12-40 Variogram for DDH & RC Twin Holes

The following conclusions can be derived from this analysis:

- The variogram shapes were similar for the two types of drill holes
- The nugget effects were practically identical for both types of drill holes and are on the order of 5% to 10% of the total sill, reflecting good short range continuity and very good sample preparation and assaying protocols
- Both types of drill holes were equivalent in quality and therefore they can be used jointly for resource estimation.

12.8 Drill Hole Collar Coordinate Field Validation

- Dr. Eduardo Magri visited the Cerro Maricunga deposit on November 27th and November 28th, 2012. During his visit, he verified recorded drill hole collar coordinates within the database against hand-held GPS readings
- A total of 15 drill hole collar were measured which correspond to approximately 5% of total drill holes. Differences in collar coordinate position varied between 4.75 and 12.10 metres, which were within the precision of a hand-held GPS. Results were considered satisfactory.

12.9 Conclusions

Overall conclusions drawn from the QA-QC analyses are as follows:

- Analyses of duplicates show good precision, indicating that the protocols used for sample preparation and assaying were adequate
- Analyses of standards used during exploration show good accuracy
- Analyses of blanks show no serious contamination problems between samples.

QA-QC data generated throughout the 2011 – 2013 drilling campaigns at Cerro Maricunga meets acceptability criteria and therefore the exploration data used complies with required confidence for resource modeling and estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Since 2010, a number of metallurgical test campaigns have been undertaken with different mineralized material from Cerro Maricunga, with the objective of characterize the ore in terms of its composition and its amenability to be processed under standard metallurgical technologies. The main focus of these campaigns is to study the material behavior under different leaching and crushing conditions, in order to define the optimum scenario to recover the precious metals contained in it.

The leaching study covered from bottle roll tests to column tests, which were conducted to material crushed under a wide range of particle sizes, including a fine range for the bottle roll test and a coarser one for the column. In parallel, several leaching conditions were tested, in order to find the optimum one in terms of reaction pH, cyanide and lime consumption, cement and others.

On the other hand, the crushing angle was also covered, conducting crushing test to the samples, to define the design criteria that allows selecting the proper equipment for the project. Also alternatives techniques to the standard ones were also tested, in particular the high pressure grinding roll technology ("HPGR").

In these campaigns, a series of samples were tested from different zones of the deposit, to assure that the results would be as representative as possible of the project material. The tests were carried out between 2010 and 2013, and included over 30 column and over 100 bottle roll tests. In addition, between 2013 and 2014 a new metallurgical test campaign was undertaken, in order to explore new possibilities for the mineral processing. The totality of these campaigns has been completed by the following established mineral processing laboratories: Kappes, Cassiday and Associates ("KCA"), Reno, Nevada; and, AMTEL (Advanced Mineral Technology Laboratory Ltd), London, Ontario, Canada.

The following chapter summarizes the results obtained from the metallurgical tests completed on twelve composites samples of the Cerro Maricunga mineralized material, during the last 4 years. The results summarized in this report, support the design criteria considered for the proposed recovery circuit, and the involved composite samples represent the spatial variability of the mineralized material through all the Cerro Maricunga deposit.

Further information on the metallurgical test programs is provided in the "Technical Report of the Cerro Maricunga Gold Project, Region III Chile" date November 9, 2012 and "Preliminary Economic Assessment for the Cerro Maricunga Oxide Gold Project, III Region, Chile – Technical Report NI 43-101" dated March 15, 2013.

13.1 Characterization of Composite Samples

The following table summarizes the composite samples that have been considered in the current analysis, and include corresponding gold (Au) and silver (Ag) head grades.

Table 13-1 Composite Samples Identification

Zone	AMTEL ID	KCA avg. head assays (Au g/t)	KCA head assays (Ag g/t)	KCA head assays (Cu g/t)
Crux	Composite 1	1.08	0.40	376
Phoenix-Lynx	Composite 2	0.78	0.40	247
Phoenix	Composite 4	0.28	-	129
Phoenix	Composite 5	0.49	-	260
Surface trench	Composite 6	0.55	-	195
Crux-Phoenix-Lynx	Composite 7	0.23	-	139
Crux	Composite Crux 0.25	0.25	0.88	124
Crux	Composite Crux 0.45	0.44	1.41	148
Lynx	Composite Lynx 0.25	0.24	0.73	139
Lynx	Composite Lynx 0.45	0.47	0.82	203
Phoenix	Composite Phoenix 0.25	0.24	1.06	133
Phoenix	Composite Phoenix 0.45	0.45	0.95	284

Spatial location of individual composites is shown in Section 13.1.3; Figure 13-1.

Physical and mineralogical analyses were conducted on some of the composites listed above. Results are presented in the following sections.

13.1.1 Physical Characterization

Hardness estimation (BWi) and abrasion characterization tests were carried out in 2010. Results are summarized in the following table.

Table 13-2 Hardness and Abrasion Indexes

Composite ID	Bwi (kWh/t)	Ai
Composite 1	10.6	0.0669
Composite 2	11.3	0.1239
Composite 4	10.7	-
Composite 5	10.5	-
Composite 6	9.8	-
Comp. Crux 0.25	10.0	0.1033
Comp. Crux 0.45	10.1	0.1223
Comp. Lynx 0.25	10.9	0.1251
Comp. Lynx 0.45	10.7	0.1080
Comp. Phoenix 0.25	9.7	0.0888
Comp. Phoenix 0.45	9.7	0.0925

Hardness and abrasion indexes indicate that mineralized material at the Cerro Maricunga deposit may be easily crushed.

13.1.2 Mineralogical Characterization

Three composite samples (# 4, 5, and 6) from the Phoenix Zone of the deposit were submitted to gold deportment analyses at AMTEL.

The analyses were carried out with conventionally crushed (P_{80} of 11-13mm) and milled (P_{80} of 80-100 μm) sub-samples of the composites mentioned above.

Summaries of the two deportment studies are detailed in Table 13-3 and Table 13-4.

Table 13-3 Gold Deportment in Crushed Ore

Composite ID	Head grade (Au g/t)	Crush size P_{80} (mm)	Gold grains (%)			Recovery (%)	
			Exposed Attached	Enclosed CN-Able	Refractory	BRT	CLT
Composite 4	0.287	11.5	81	14	5	75	80
Composite 5	0.466	11.5	79	17	4	81	86
Composite 6	0.505	13.0	74	13	13	63	80

Table 13-4 Gold Deportment in Milled Ore

Composite ID	Head grade (Au g/t)	Grind size P_{80} (mm)	Free gold (%)		Attached (%)			Enclosed CN-Able (%)	Refractory (%)	Recovery, % at P_{80} 75
			> 10 µm	< 10 µm	To FeOx	To Comp.	To Rock			BRT
Composite 4	0.287	87	3	47	12	6	16	7	8	86
Composite 5	0.466	110	14	43	9	3	21	5	5	89
Composite 6	0.505	83	12	51	6	1	8	9	13	79

Relevant conclusions drawn from data shown in these tables are the following:

- Gold deportment studies indicate that free and/or attached gold grains range from 74 to 81% in crushed ore and 78 to 90% in milled ore
- Refractory ore in Cerro Maricunga consists of very fine grained gold, contained within microcrystalline quartz
- Gold deportment studies were also carried out in 16 column leach test residues. In general, results indicate that gold residues consist of very small amounts of water soluble gold (< 1%). Proportional contents of exposed-enclosed-refractory gold are in the order of 1:4:5.

13.1.3 Spatial Distribution of Individual Samples

Composites were generated with mineralized material obtained from each mineralized zone of the deposit; Crux, Phoenix and Lynx. Location and mean grade of each composite are depicted in the longitudinal section shown Figure 13-1.

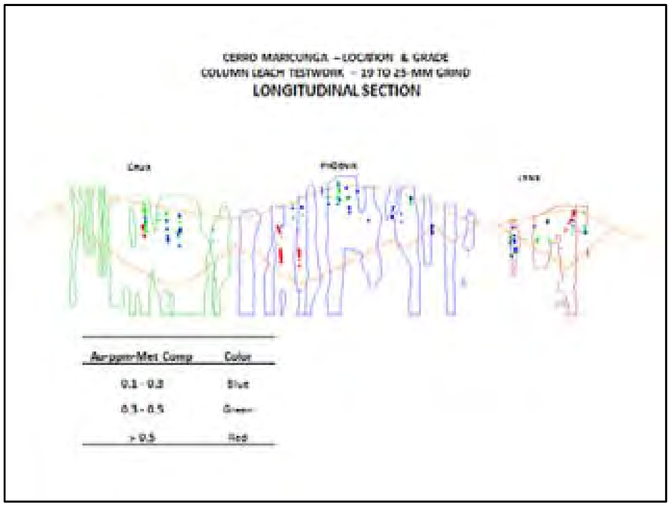


Figure 13-1 Location of Composites within each Mineralized Zone

13.2 Cyanidation Tests

Cyanidation tests were conducted in the form of column percolation and bottle roll tests. The following sections present a summary of the most relevant results obtained from these tests.

13.2.1 Column Percolation Tests

The relevant column percolation leach tests were conducted to 12 different composite samples, which are representative of the project deposit. The primary objective of these tests is to study the deposit behaviour for a crushing size of P_{80} 19 mm (P_{100} 25 mm), though some of the samples were tested under other crushing size in order to assess its impact in gold recovery.

The following table shows a summary of the relevant conditions under which the column tests were carried out

Table 13-5 Summary of Selected Column Percolation Leach Test Parameters

<table>
<tr><th>Composite ID</th><th>Target P_{80}/P_{100} crush size
(mm)</th><th>Column diameter
(mm)</th><th>Initial charge height/weight
(m)</th><th>Initial NaCN concentration
gpl NaCN</th><th>Maintained NaCN concentration
gpl NaCN</th><th>Maintained pH value</th><th>Cement dosage
(kg/t)</th></tr>
<tr><td>Composite 1</td><td>19.0</td><td rowspan="3">127</td><td>1.66</td><td rowspan="17">1.0</td><td rowspan="17">0.5</td><td rowspan="17">9-11</td><td rowspan="11">1.0</td></tr>
<tr><td>Composite 2</td><td>19.0</td><td>1.59</td></tr>
<tr><td>Composite 2</td><td>9.5</td><td>1.67</td></tr>
<tr><td>Composite 4</td><td>19.0</td><td rowspan="4">152</td><td>1.55</td></tr>
<tr><td>Composite 4</td><td>19.0</td><td>1.58</td></tr>
<tr><td>Composite 5</td><td>19.0</td><td>1.52</td></tr>
<tr><td>Composite 5</td><td>19.0</td><td>1.58</td></tr>
<tr><td>Composite 6</td><td>100.0</td><td>445</td><td>3.01</td></tr>
<tr><td>Composite 6</td><td>50.0</td><td>292</td><td>2.50</td></tr>
<tr><td>Composite 6</td><td>19.0</td><td rowspan="2">152</td><td>1.68</td></tr>
<tr><td>Composite 7</td><td>19.0</td><td>1.60</td></tr>
<tr><td>Comp. Crux 0.25</td><td>25 (P_{100})</td><td rowspan="6">150</td><td rowspan="6">50 kg</td><td rowspan="6">0.0</td></tr>
<tr><td>Comp. Crux 0.45</td><td>25 (P_{100})</td></tr>
<tr><td>Comp. Lynx 0.25</td><td>25 (P_{100})</td></tr>
<tr><td>Comp. Lynx 0.45</td><td>25 (P_{100})</td></tr>
<tr><td>Comp. Phoenix 0.25</td><td>25 (P_{100})</td></tr>
<tr><td>Comp. Phoenix 0.45</td><td>25 (P_{100})</td></tr>
</table>

The following table shows a summary of the results obtained from the column leach tests.

Table 13-6 Summary of Column Percolation Leach Results

Composite ID	Target P_{80}/P_{100} (mm)	KCA avg. head assays (Au g/t)	Calc. head grade (Au g/t)	Method for head grade calculation	Extracted, Au (%)	Days of leach	Consumption NaCN (kg/t)	Addition hydrated (kg/t)
Composite 1	19.0	1.08	1.13	GAC	89	57	1.03	3.08
Composite 2	19.0	0.78	0.76	GAC	79	57	1.06	3.07
Composite 2	9.5		0.79	GAC	80	57	1.19	3.06
Composite 4	19.0	0.28	0.31	GAC	80	87	0.82	2.51
Composite 4	19.0		0.31	GAC	82	87	0.52	2.51
Composite 5	19.0	0.49	0.53	GAC	86	87	0.74	2.01
Composite 5	19.0		0.50	GAC	84	87	0.97	2.03
Composite 6	100.0	0.55	0.58	GAC	77	87	0.09	6.61
Composite 6	50.0		0.54	GAC	78	87	0.10	6.66
Composite 6	19.0		0.58	GAC	80	87	0.44	6.53
Composite 7	19.0	0.23	0.22	GAC	78	82	0.57	4.01
Comp. Crux 0.25	25 (P_{100})	0.25	0.25	SA	80	113	0.62	5.10
Comp. Crux 0.45	25 (P_{100})	0.44	0.44	SA	78	113	0.75	5.09
Comp. Lynx 0.25	25 (P_{100})	0.24	0.24	SA	79	113	0.82	5.05
Comp. Lynx 0.45	25 (P_{100})	0.47	0.45	SA	81	113	0.75	5.09
Comp. Phoenix 0.25	25 (P_{100})	0.24	0.24	SA	82	113	0.85	5.08
Comp. Phoenix 0.45	25 (P_{100})	0.45	0.45	SA	79	113	0.96	5.11

In the previous table, "GAC" stands for Granular Activated Carbon, while "SA" stands for Solution Assays, both corresponding to the chosen grade measurement method.

The following can be established from the column leach tests.

- Gold extraction for composite 1, which has the highest grade, was 89% at P_{80} 19 mm, after 57 days of leach
- Gold extractions for composite 2 only increased from 79% to 80% when the crush size was decreased from 19 to 9.5 mm
- The average gold extraction for composite 4 was 81% at P_{80} 19 mm, after 87 days of leach
- The average gold extraction for composite 5 was 85% at P_{80} 19 mm, after 87 days of leach
- Gold extraction for composite 6 was 80% at P_{80} 19 mm, after 87 days of leach
- Gold extraction for composite 7 was 78% at P_{80} 19 mm, after 82 days of leach
- Gold extraction for composite 6 only decreases from 80% to 78% and to 77% when the crush size was increased from 19 to 50 and to 100 mm, respectively
- Gold extractions for the last six composites range between 78% and 82%, at P_{100} 25 mm and after 113 days of leach. There is no apparent dependence between head grade and gold extraction.

13.2.2 Additional column percolation tests

Some additional column tests were conducted during 2012 in order to assess the effect of different test condition in gold extraction. The following table summarizes the main results from the additional tests made.

Table 13-7 Summary of additional column tests results

Composite ID	P_{80} sized crush size (mm)	KCA avg. head assays (Au g/t)	Aggl. cement (kg/t)	Alkalinity cement (kg/t)	Extracted, Au (%)
C,L,P (AG) 1:1:1	16.1	0.44	0.0	0.0	83
	17.6		12.5	0.0	79
	19.0		0.0	1.0	79
C,L,P (AG+LG) 1:1:1 (HPGR)	8.8	0.32	0.0	0.0	85
	9.2		12.5	0.0	79
Surface Trench 2011	46.0	0.46	0.0	0.0	78
	47.0		12.5	0.0	77
	21.0		0.0	0.0	83
	21.0		12.5	0.0	80
Surface Trench 2012	139.0	0.39	0.0	1.0	76
	129.0		0.0	0.0	68

The following can be established from the additional column leach tests.

- Coarsening the crush size (P_{80}) from 21 to 139 mm in the surface trench composite resulted in a 7% loss in gold recovery
- Agglomeration with 12.5 kg cement/t has a detrimental effect on gold recovery
- The effects of blending and agglomerating 1 kg/t of cement on gold recovery are unclear
- When is used HPGR as a mode of crushing, the improvement in gold recovery is only modest.

13.2.3 Bottle Roll Tests (BRT)

Bottle roll leach tests were also conducted on the 12 composite samples selected for the column tests. Different particle sizes were covered. Table 13-8 shows relevant results. The tests were conducted under the following main conditions: solids 50% (w/w), 1 gpl of NaCN and pH 10 to 11. The following table shows a summary of the results obtained from the bottle roll tests.

Table 13-8 Summary - Bottle Roll Leach Test Results

Tests N°	Composite ID	Target P_{80} crush size (mm)	Calc. head grade (Au g/t)	Extracted, Au (%)	Days of leach	Consumption NaCN (kg/t)	Addition hydrated lime (kg/t)
45206 A/B	Composite 1	19.0	1.10	81	10	0.25	2.50
45208 A/B		9.5	1.05	86	10	0.12	3.00
45210 A/B		1.0	1.06	88	10	0.12	3.50
45238 A/B		0.1	1.24	92	4	0.12	3.50
45168 A/B	Composite 2	19.0	0.80	76	10	0.16	2.50
45170 A/B		9.5	0.78	78	10	0.07	3.00
45172 A/B		1.0	0.76	80	10	0.06	4.00
45238 C/D		0.1	0.76	82	4	0.19	3.00
60010 A/B	Composite 4	50.0	0.28	66	6	0.12	1.00
60012 A/B		12.5	0.29	75	6	0.02	2.25
60014 A/B		1.0	0.27	83	6	0.06	2.50
60015 A/B		0.1	0.30	87	6	0.01	3.00
60016 A/B	Composite 5	50.0	0.50	65	6	0.08	1.00
60018 A/B		12.5	0.46	81	6	0.08	1.50
60020 A/B		1.0	0.50	77	6	0.08	2.50
60021 A/B		0.1	0.55	89	6	0.02	1.50
60004 A/B	Composite 6	50.0	0.57	55	6	0.07	4.10
60006 A/B		12.5	0.57	63	6	0.03	5.25
60008 A/B		1.0	0.58	79	6	0.03	6.00
60009 A/B		0.1	0.63	82	6	-	7.00
60027 A/B	Composite 7	19.0	0.20	71	6	0.19	2.70
60028 A/B		12.5	0.21	73	6	0.04	3.00
60029 A/B		6.3	0.21	75	6	0.05	3.50

It must be noted that results for the Crux, Phoenix and Lynx composites are not shown. These composites were only tested at sizes 0.075 and 0.5 mm, which are not considered relevant for a heap leach operation.

The following conclusions can be obtained from the summarized bottle roll test results:

- For composites 1 and 2, gold extractions obtained after 10 days of leach, and at P_{80} 19 mm, are lower than the column extractions, the latter being 89% and 79% after 57 days of leach
- For composites 4, 5, 6 and 7, gold extractions obtained after 6 days of leach and at P_{80} 12.5 mm are much lower than the column extractions achieved after 87 days or leach at a P_{80} of 19 mm, especially for composite 6
- Based on the last two findings it can be concluded that the BRT are not useful for the estimation of a maximum gold extraction, instead, the conducted BRT seem to be useful to elucidate the impact of

fine grinding on gold extraction. This impact seems to be positive, with a slight increase in the gold extraction for smaller grind sizes, but since the proposed recovery method is heap leaching this is not a relevant finding

- NaCN consumptions reported in the column tests are much higher than those reported in the bottle roll tests; however, it is KCA laboratory opinion that NaCN consumption of the column tests is approximately 3 times higher than the value to be expected in the operation, mainly due to an evaporation effect
- It must be noted that only the bottle roll tests that are relevant for the current processing plant design are covered in the present report: additional information can be found in the previous technical report that have been issued for the Cerro Maricunga Project.

13.2.4 Implications of the Cyanidation Tests Results

The following metallurgical implications are highlighted based on the cyanidation tests results:

Effect of grade in the gold extraction

In order to study the empirical effect of the grade in gold extraction, the column leach tests made at P_{80} 19 mm, or at similar size, were analyzed. The extraction results used in this analysis correspond to those obtained after 57 days of leach, interpolated with other data obtained in the longer-running column leach tests. Table 13-9 shows tests used.

Table 13-9 Selected results from column leach tests

Tests N°	Composite ID	Target P_{80}/P_{100} crush size (mm)	Calc. head grade (Au g/t)	Method for head grade calculation	Extracted, Au (%)	Days of leach
45640	Composite 1	19.0	1.13	GAC	89.0	57
45643	Composite 2	19.0	0.76	GAC	79.0	57
60042/45	Composite 4	19.0	0.31	GAC	80.2	57
60048/51	Composite 5	19.0	0.51	GAC	82.0	57
60039	Composite 6	19.0	0.58	GAC	79.7	57
60054	Composite 7	19.0	0.22	GAC	77.2	57
60074	Comp. Crux 0.25	25 (P_{100})	0.25	SA	77.5	57
60071	Comp. Crux 0.45	25 (P_{100})	0.44	SA	77.5	57
60080	Comp. Lynx 0.25	25 (P_{100})	0.24	SA	78.0	57
60077	Comp. Lynx 0.45	25 (P_{100})	0.45	SA	80.5	57
60086	Comp. Phoenix 0.25	25 (P_{100})	0.24	SA	80.0	57
60083	Comp. Phoenix 0.45	25 (P_{100})	0.45	SA	78.0	57

It must be noted that tests on composites 1, 2, 4, 5, 6 and 7 were carried out using cement for pH control. Results obtained from the analyses of these six tests were used in the design stage of the leach plant. Results obtained in the remaining six composites (Crux-Phoenix-Lynx) were later incorporated in order to confirm trends observed in the initial analyses.

The trend observed in Figure 13-2 shows a slight increment of gold leach extraction rates for higher head gold grades.

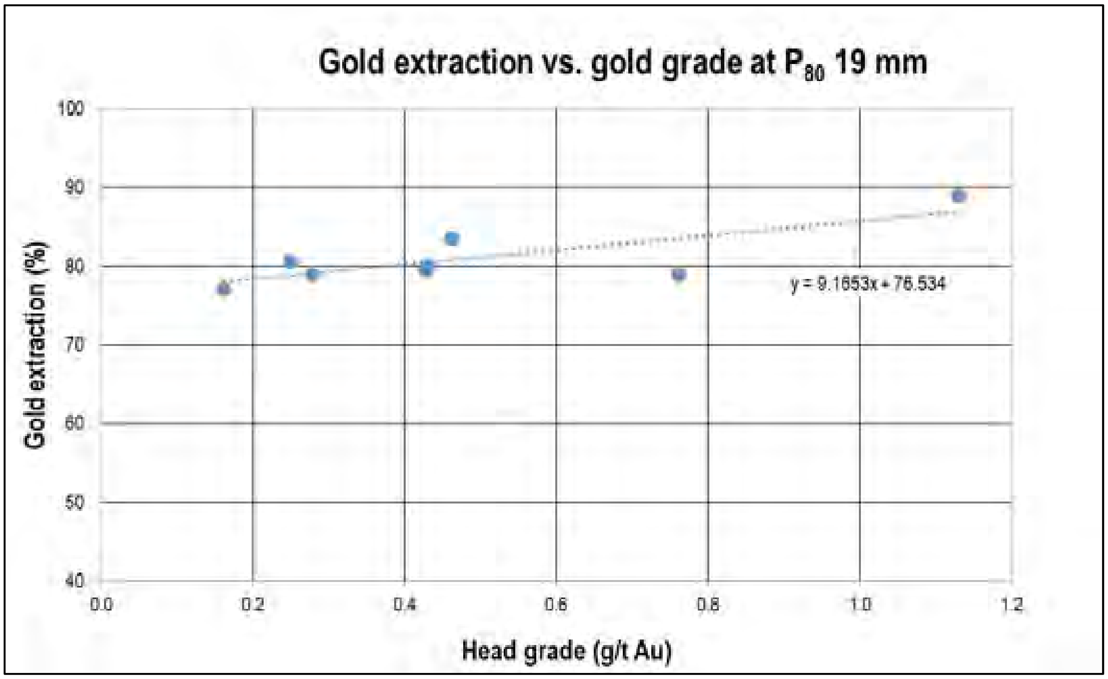


Figure 13-2 Gold Extraction v/s Gold Grade

The following comments are made based on data shown in Table 13-9 and Figure 13-2:

- Based on the available experimental data, the model represents the best estimation of gold extraction for different gold grades
- This model allows predicting gold extraction in column percolation leach tests with mineralized material with gold grades ranging from 0.22 to 1.13 g/t.

The following figure presents the correlation when including the 6 tests done without cement (Crux-Phoenix-Lynx).

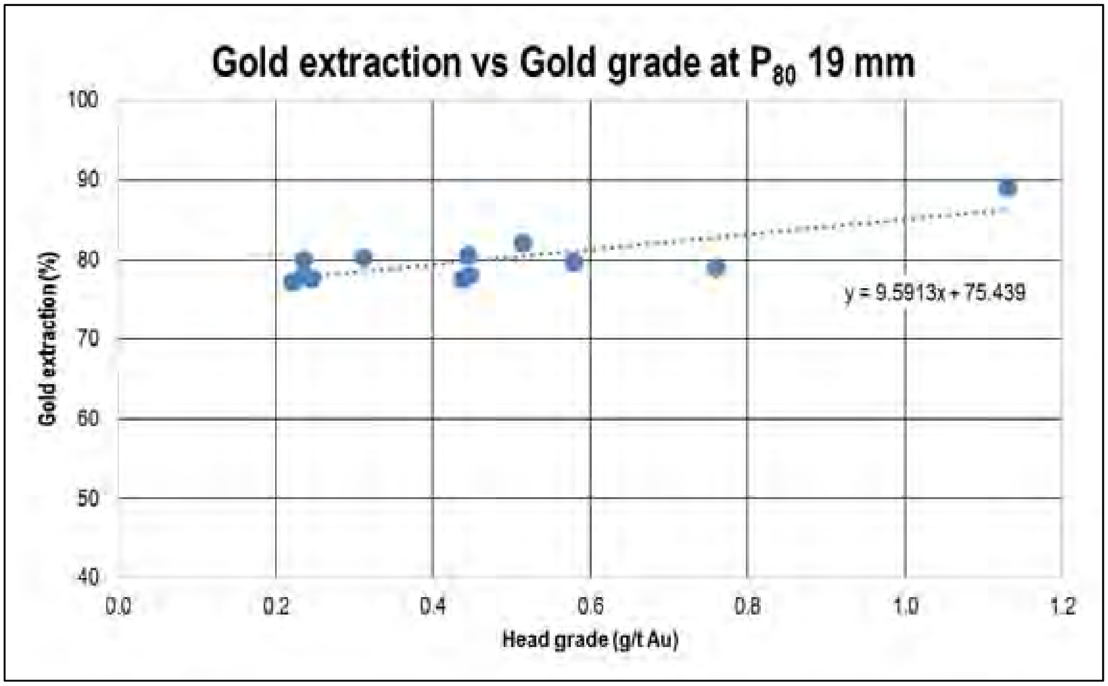


Figure 13-3 Gold Extraction v/s Gold Grades – All Tests

As may be seen, the initial trend is maintained.

Effect of particle size in gold extraction

Based on the column percolation leach tests results, the effect of the crush size in gold extraction is quite low; in fact, the extraction obtained for composite 2 at P_{80} 9.5 mm is very similar to that obtained at a P_{80} 19 mm. Furthermore, recoveries for different crush sizes in composite 6 remain practically the same:

- P_{80} -100 mm 77%
- P_{80} - 50 mm 78%
- P_{80} - 19 mm 80%

This is an expected outcome that agrees with the information obtained from the gold deportment studies. The column tests results indicate that in metallurgical terms there are three forms of gold: (1): exposed (free and attached) (exposed CN-able), (2): locked within rock and FeOx (enclosed CN-able) and (3): refractory gold. Forms (2) and (3) are not sensible to crush size.

Effect of residence time in gold extraction

The effect of leaching time in the gold extraction can be easily assessed through the kinetic curves. Curves for different composites are shown in Figure 13-4 (Comp 1 and 2), Figure 13-5 (Comp 4, 5, 6 and 7) and Figure 13-6 (Crux-Phoenix-Lynx composites).

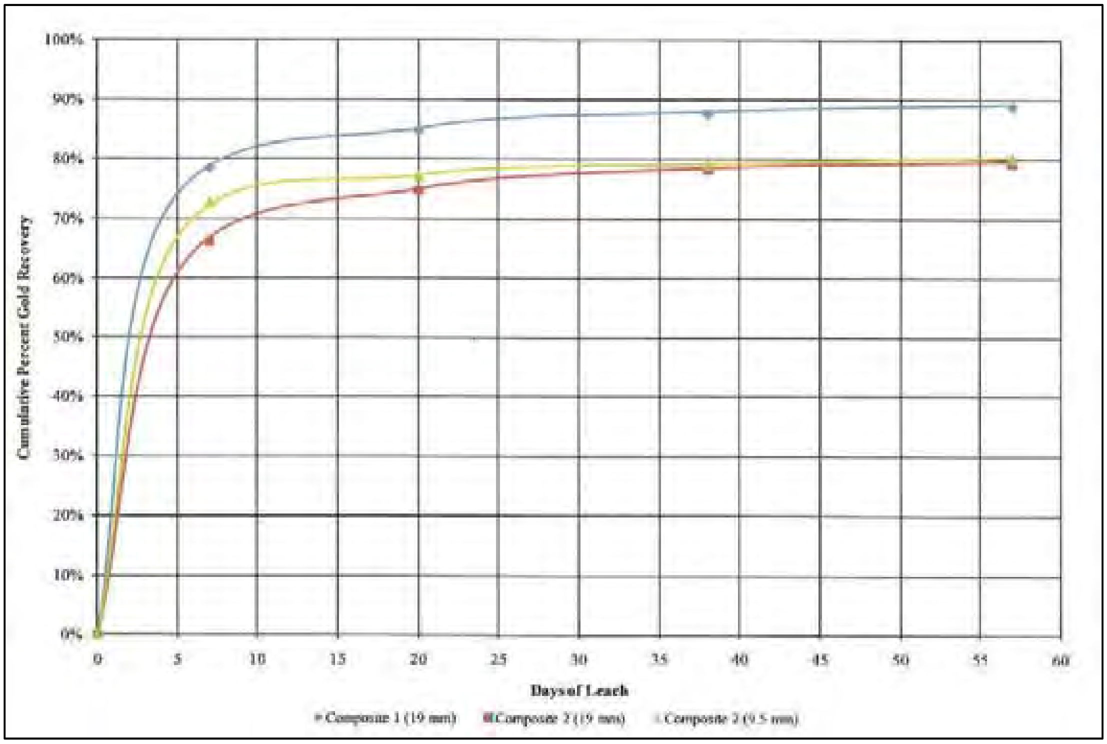


Figure 13-4 Kinetics Curves Composites 1 and 2

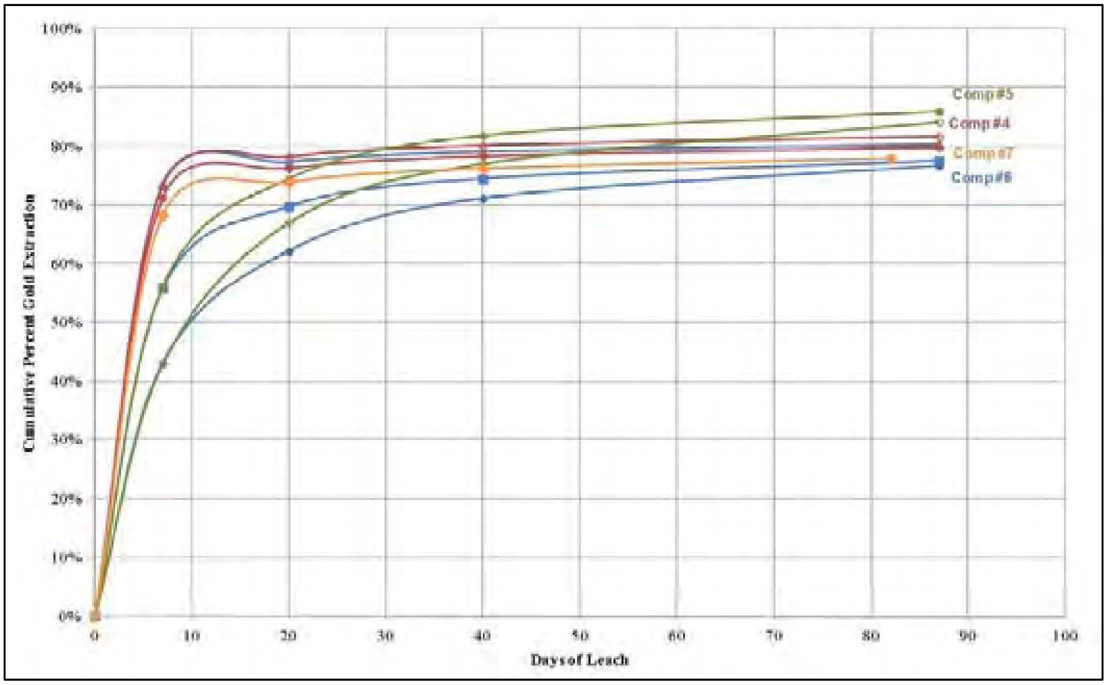


Figure 13-5 Kinetics Curves Composites 4, 5, 6 and 7

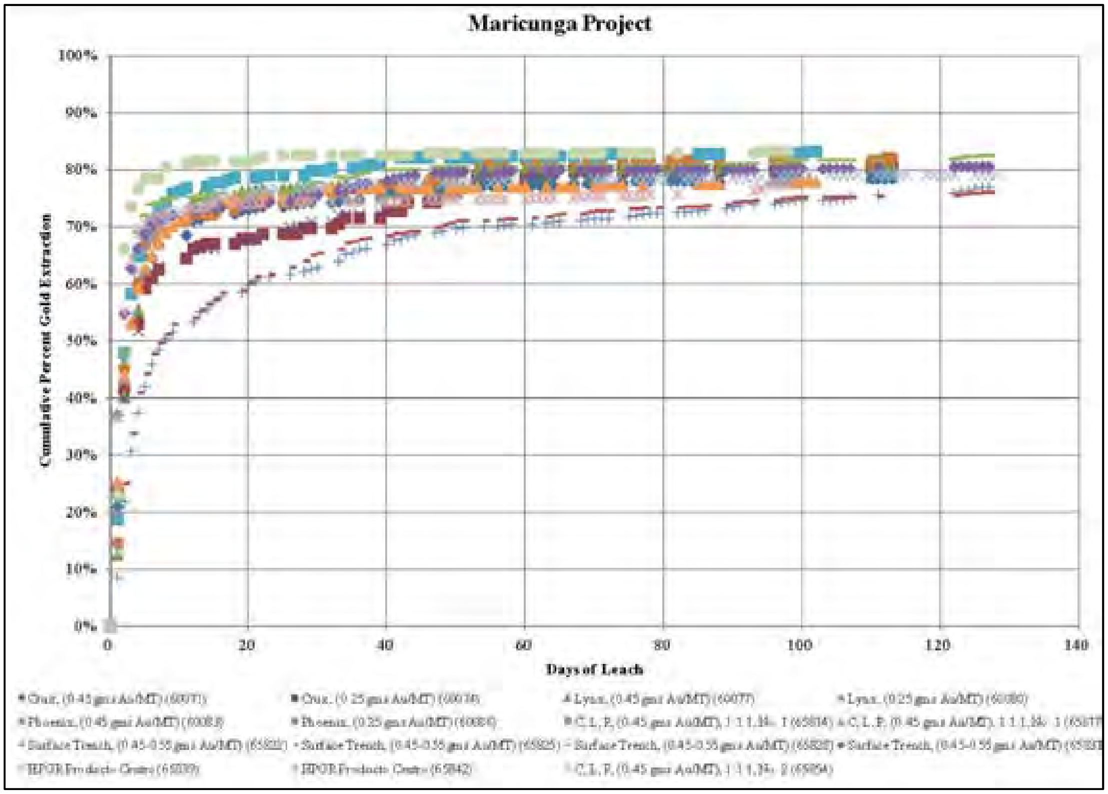


Figure 13-6 Kinetics Curves Composites Crux, Lynx and Phoenix

These figures allow concluding that there are gold forms that are recovered quite fast and other with slower kinetics. These findings also agree with results obtained in the gold deportment studies.

<u>Cyanide and lime consumption</u>

The following table presents the cyanide and lime consumption results obtained from the column percolation leach tests.

Table 13-10 Cyanide and Lime Consumption in Column Tests

Tests N°	Composite ID	Target P_{80}/P_{100} crush size (mm)	Extracted, Au (%)	Days of leach	Consumption NaCN (kg/t)	Corrected value of NaCN (kg/t)	Addition hydrated lime (kg/t)
45640	Composite 1	19.0	89	57	1.03	0.34	3.08
45643	Composite 2	19.0	79	57	1.06	0.35	3.07
60042	Composite 4	19.0	80	87	0.82	0.27	2.51
60045	Composite 4	19.0	82	87	0.52	0.17	2.51
60048	Composite 5	19.0	86	87	0.74	0.25	2.01
60051	Composite 5	19.0	84	87	0.97	0.32	2.03
60039	Composite 6	19.0	80	87	0.44	0.15	6.53
60054	Composite 7	19.0	78	82	0.57	0.19	4.01
60074	Comp. Crux 0.25	25 (P_{100})	80	113	0.62	0.21	5.10
60071	Comp. Crux 0.45	25 (P_{100})	78	113	0.75	0.25	5.09
60080	Comp. Lynx 0.25	25 (P_{100})	79	113	0.82	0.27	5.05
60077	Comp. Lynx 0.45	25 (P_{100})	81	113	0.75	0.25	5.09
60086	Comp. Phoenix 0.25	25 (P_{100})	82	113	0.85	0.28	5.08
60083	Comp. Phoenix 0.45	25 (P_{100})	79	113	0.96	0.32	5.11
45206 A/B	Composite 1	19.0	81	10	0.25		2.50
45168 A/B	Composite 2	19.0	76	10	0.16		2.50
60012 A/B	Composite 4	12.5	75	6	0.02		2.25
60018 A/B	Composite 5	12.5	81	6	0.08		1.50
60006 A/B	Composite 6	12.5	63	6	0.03		5.25
60027 A/B	Composite 7	19.0	71	6	0.19		2.70

Is KCA laboratory experience that cyanide consumption in operational heaps is around 1/3 of the consumption obtained in laboratory column tests, hence a “corrected” consumption has been calculated corresponding to 1/3 of the consumption reported for each test.

The average NaCN corrected consumption is 0.26 kg/t and the average of hydrated lime addition is 4 kg/t. It must be noted that these values only give an idea of the consumptions that can be expected in an industrial process.

Refractory behavior of copper

The average head grade of copper in the twelve composites is 198 g/t. Copper extraction was measured in all composites and results reveal a refractory behavior of this element against the cyanide leaching. The following sentences summarize the findings about copper.

- Copper extraction in column leach tests on composites 1 and 2, is very low, ranging from 1.0% to 2.6%

- Copper extraction on composites 4, 5, 6 and 7, is also very low, with values under 2%
- Copper extraction on composites Crux, Lynx and Phoenix is 1.5%, also a low value.

The low copper extractions obtained from the column tests are in agreement with the preliminary findings about the copper minerals that explain its grade in the mineral. These findings reveal that copper occurs mainly as chalcopyrite, a very refractory mineral under cyanide conditions.

In conclusion, an approximate copper extraction of 2% is expected in an industrial cyanidation process, which means that a SART process is not required.

13.3 Conclusions

The following conclusions are drawn based on results presented herein:

- Gold extractions of 80% can be achieved, for material with 0.40 g/t Au at P_{80} 19 mm
- The following model can predict the gold extraction at a P_{80} of 19 mm and ore with gold grades ranging between 0.22 and 1.13 g/t:
    - Gold extraction (%) = 9.1653*(Gold head grade in Au g/t) + 76.534.
- Crushed ore contains three forms of gold; (1): exposed CN-able gold and hence easily recoverable, (2): enclosed CN-able gold and not particularly sensitive to crush size, and (3): refractory gold, which accounts for less than 10% of total gold
- Mineralogical characterization findings were contrasted with the metallurgical tests results, confirming that crush size does not have an important impact in gold extraction and that most of the extractable gold is recoverable quite fast. These findings allow considering new possibilities for crushing sizes and residence times, which could have a positive impact in capital and operating costs, but with low impact in gold extraction.

14 MINERAL RESOURCE ESTIMATE

14.1 Modeling Procedure

14.1.1 Data Used

The resource model was generated using the following data:

- Surface maps containing lithological units, structures and trenches with assays
- Geological descriptions (logging) of 86 diamond drill holes totaling 29,534 metres of core specimens
- Lithological descriptions (quick-logging) of 234 reverse circulation holes totaling 75,496 metres of RC cuttings
- Assays for a total 21,385 two metre DDH and RC samples
- Lithological descriptions of 5 trenches totaling 266 metres
- Assays for a total of 131 two metre trench samples.

14.1.2 Section and Plan Interpretation

A total of 48 sections (300 NE to 2,650 NE) and 14 plan views (4,950 masl down to 4,300 masl) were interpreted by hand. Section and plan spacing was 50 metres using a ± 25 m influence.

Structures mapped at surface were interpolated in sections and plans.

A grade-shell of 150 ppb was contoured and interpolated, as were known barren porphyry units.

Figure 14-1 corresponds to a three dimensional view of the Cerro Maricunga model.

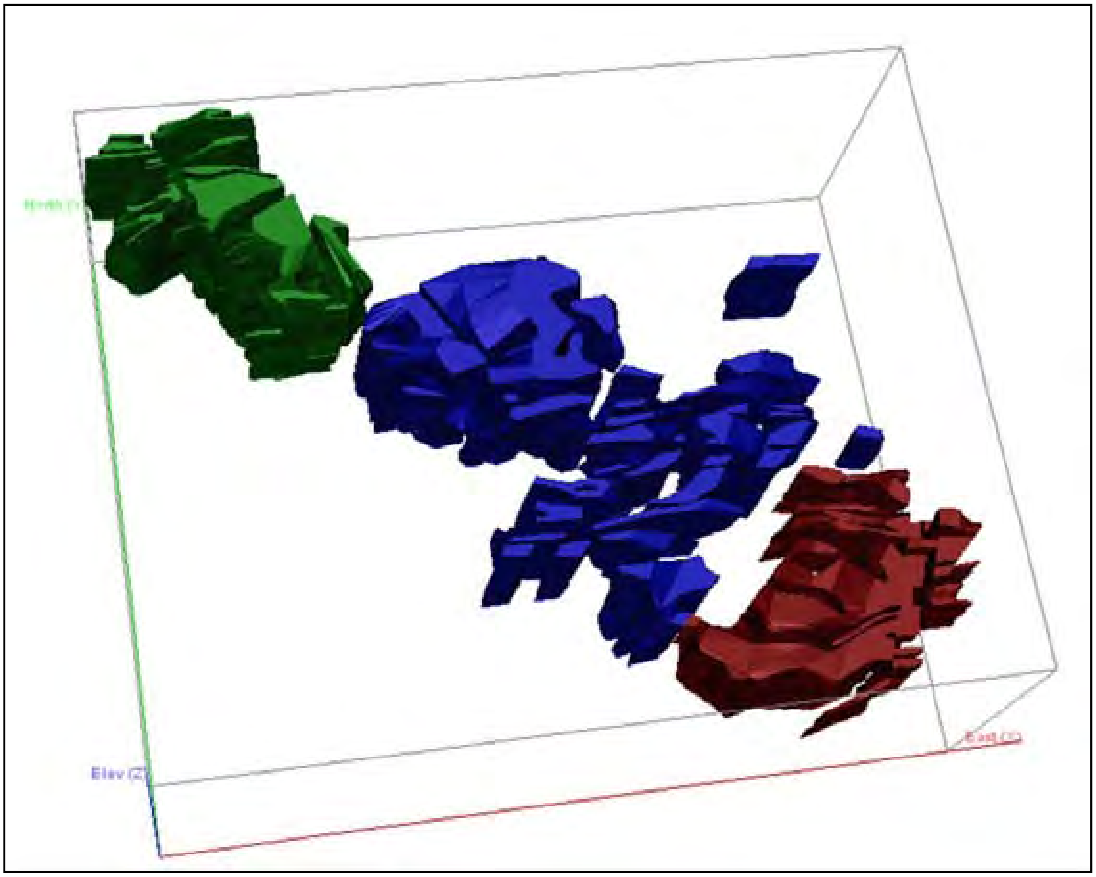


Figure 14-1 3-D View of Cerro Maricunga's Mineralized Zones

14.1.3 Definition of Estimation Domains

Three estimation domains were defined in Cerro Maricunga based on major fault systems which have displaced the mineralized EW structures (Figure 14-2). These are:

- Lynx with predominant NW faults
- Northern Phoenix with predominant NW faults
- Crux and southern Phoenix with predominant NNE faults.

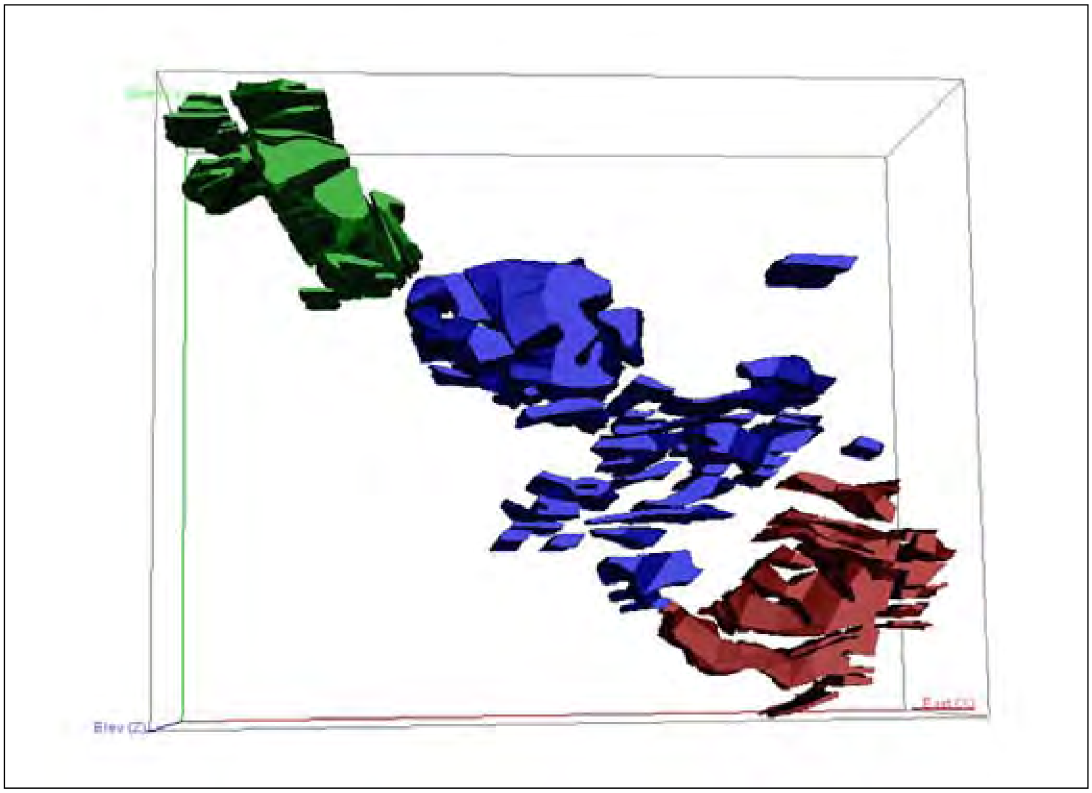


Figure 14-2 Cerro Maricunga Estimation Domains

14.2 Exploratory Data Analyses

Data used for the resource estimation consisted of reverse circulation and diamond drill hole samples are described in Table 14-1. Minor intervals within RC holes were not assayed due to poor cutting recovery, however, in general diamond core and RC chip recoveries were very good.

Table 14-1 Cerro Maricunga Drilling Phases – Metres Drilled & Metres Assayed

Zone	Phase	DDH HOLES			RC HOLES			DDH + RC HOLES		
		Drilled m	Assayed m	Not Assayed m	Drilled m	Assayed m	Not Assayed m	Drilled m	Assayed m	Not Assayed m
Crux	I	202.0	202.0	0.0	542.0	542.0	0.0	744.0	744.0	0.0
	II	1,068.0	1,068.0	0.0	452.0	452.0	0.0	1,520.0	1,520.0	0.0
	III	1,482.0	1,482.0	0.0	4,700.0	4,700.0	0.0	6,182.0	6,182.0	0.0
	IV	362.0	362.0	0.0	2,086.0	2,084.0	2.0	2,448.0	2,446.0	2.0
	Sub-Total	3,114.0	3,114.0	0.0	7,780.0	7,778.0	2.0	10,894.0	10,892.0	2.0
Phoenix	I	263.1	263.1	0.0	564.0	564.0	0.0	827.1	827.1	0.0
	II	1,069.4	1,069.4	0.0	7,966.0	7,964.0	2.0	9,035.4	9,033.4	2.0
	III	3,369.3	3,369.3	0.0	3,894.0	3,890.0	4.0	7,263.3	7,259.3	4.0
	IV	2,440.0	2,440.0	0.0	3,880.0	3,874.0	6.0	6,320.0	6,314.0	6.0
	Sub-Total	7,141.8	7,141.8	0.0	16,304.0	16,292.0	12.0	23,445.8	23,433.8	12.0
Lynx	I	161.9	161.9	0.0	0.0	0.0	0.0	161.9	161.9	0.0
	II	813.1	813.1	0.0	2,200.0	2,200.0	0.0	3,013.1	3,013.1	0.0
	III	1,280.0	1,280.0	0.0	1,916.0	1,916.0	0.0	3,196.0	3,196.0	0.0
	IV	860.0	860.0	0.0	972.0	972.0	0.0	1,832.0	1,832.0	0.0
	Sub-Total	3,115.0	3,115.0	0.0	5,088.0	5,088.0	0.0	8,203.0	8,203.0	0.0
Out	I	93.4	93.4	0.0	316.0	316.0	0.0	409.4	409.4	0.0
	II	3,923.2	3,923.2	0.0	13,946.0	13,932.0	14.0	17,869.2	17,855.2	14.0
	III	8,226.2	8,226.2	0.0	21,074.0	21,040.0	34.0	29,300.2	29,266.2	34.0
	IV	3,920.6	3,920.6	0.0	10,906.0	10,886.0	20.0	14,826.6	14,806.6	20.0
	Sub-Total	16,163.4	16,163.4	0.0	46,242.0	46,174.0	68.0	62,405.4	62,337.4	68.0
All zones	I	720.4	720.4	0.0	1,422.0	1,422.0	0.0	2,142.4	2,142.4	0.0
	II	6,873.7	6,873.7	0.0	24,564.0	24,548.0	16.0	31,437.7	31,421.7	16.0
	III	14,357.5	14,357.5	0.0	31,584.0	31,546.0	38.0	45,941.5	45,903.5	38.0
	IV	7,582.6	7,582.6	0.0	17,844.0	17,816.0	28.0	25,426.6	25,398.6	28.0
	Total	29,534.2	29,534.2	0.0	75,414.0	75,332.0	82.0	104,948.2	104,866.2	82.0

14.3 Database Description

The drill hole database consisted of the following tables:

- Collar Table: Variables contained in this table are the following:
  - DHID: Drill hole identification
  - X: East collar coordinate
  - Y: North collar coordinate
  - Z: Collar elevation.
- Survey Table: Variable within this table were:
  - From: Beginning of the interval
  - To: End of the interval
  - Azimuth: Azimuth of the interval
  - Dip: Dip of the interval.
- Assay Table: This table contained the following variables:
  - From: Initial point of the sample
  - To: Final point of the sample

- Au ppm: Gold grade in ppm.

- Specific Gravity Table: This table contained the following variables:
  - From: Initial point of the core specimen
  - To: Final point of the core specimen
  - SG: Specific gravity of the core specimen (g/cc).

14.4 Compositing, Statistics and Outliers

Since sampling was carried out almost consistently at 2 m intervals, coordinates were assigned to the center of individual samples via a method available in GEMCOM, which preserved the original sample length. Figure 14-3 shows a scatterplot between sample length and gold values for all mineralized samples within the three mineralized envelopes. This figure shows that there are only 26 samples (out of 52,612) in which the sample length differed from 2.0 m. Furthermore, there is an almost complete lack of correlation between sample length and gold grades.

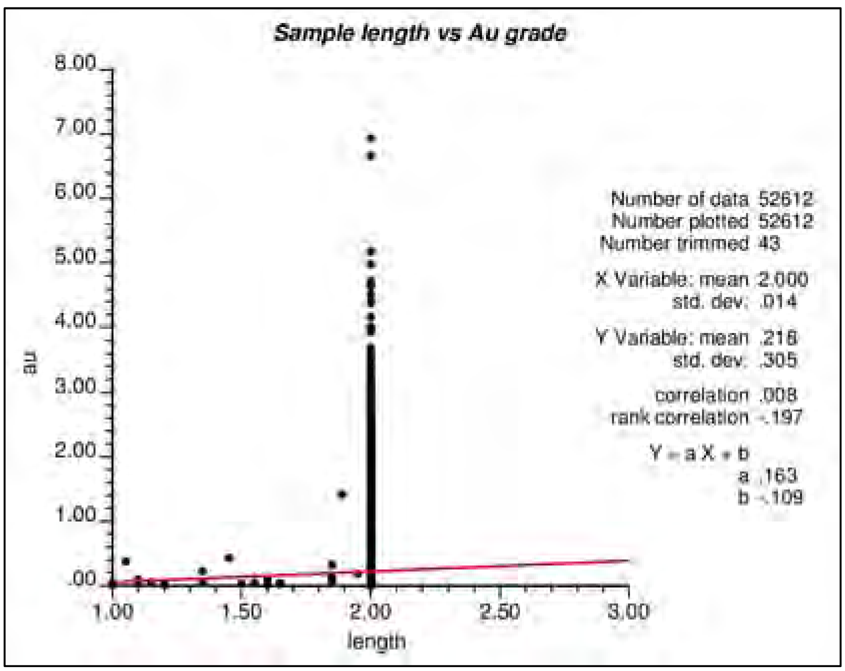


Figure 14-3 Sample Length vs. Gold Grades

Since practically all the samples were 2.0 m long, it was decided to use the samples directly in the resource estimation rather than calculating equal length composites, thus avoiding unnecessary smoothing.

Table 14-2 shows basic sample statistics for each mineralized envelope and also for the samples lying outside the mineralized envelopes (code 4).

Table 14-2 Basic Sample Statistics (Each Zone Separately)

Zone	Code	Variable	Valid N	Mean	Median	Min	Max	Q25	Q75	STD	CV
Lynx	1	Au g/t	4,102	0.458	0.297	0.003	6.940	0.185	0.543	0.493	1.077
North Phoenix	2	Au g/t	7,227	0.456	0.355	0.003	3.679	0.233	0.552	0.365	0.801
South Phoenix + Crux	3	Au g/t	10,056	0.368	0.273	0.003	5.179	0.178	0.431	0.350	0.951
All Mineralized	1+2+3	Au g/t	21,385	0.415	0.304	0.003	6.940	0.194	0.495	0.389	0.937
Outside	4	Au g/t	31,227	0.083	0.061	0.003	2.927	0.031	0.109	0.089	1.081

The following equal weighted histograms of gold grades are presented in this report:

Figure 14-4 – Lynx (1), North Phoenix (2), South Phoenix plus Crux (3), and Outside (4).

Figure 14-5 – All mineralized zones and samples outside de mineralized envelopes.

From Figure 14-3 and the above mentioned figures, it was apparent that mean grades within the mineralized envelopes ranged from 0.368 to 0.458 g/t Au, while data lying outside the mineralized envelopes had a mean grade of 0.083 g/t Au. Although, the means and the distributions of gold grades within the three mineralized envelopes were very similar, the zones were estimated separately since there were significant differences in their structural patterns.

The area lying outside the mineralized envelopes was estimated independently in order to have an estimation of the dilution material even though the mean grade is almost negligible.

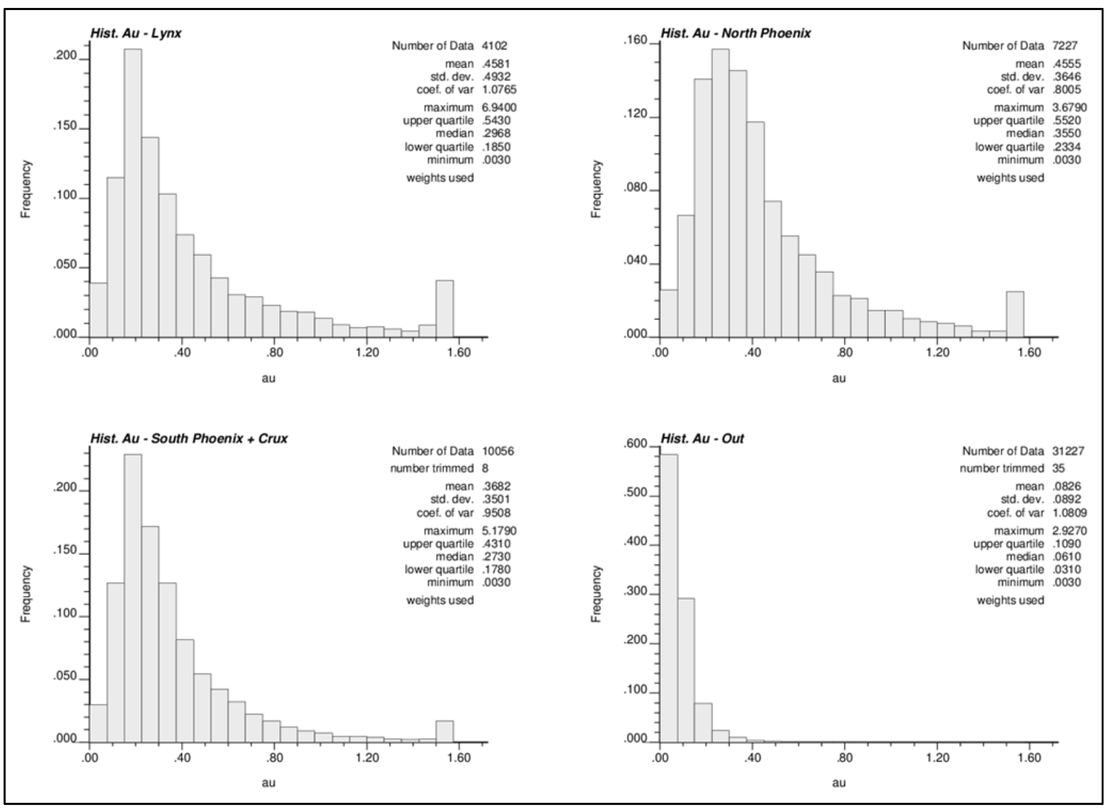


Figure 14-4 Equal Weighted Histograms for Each Mineralized Envelope and Samples Outside

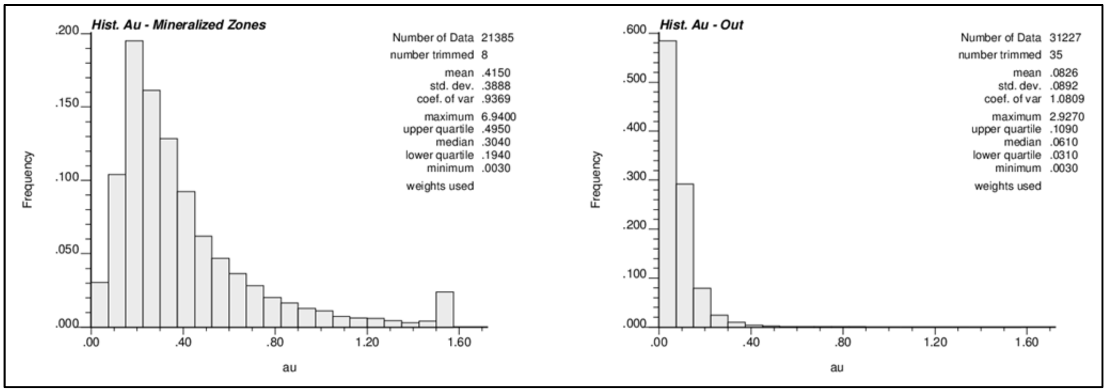


Figure 14-5 Equal Weighted Histograms for all Mineralized Envelopes and Samples Outside

The following gold grade log-probability plots are presented:

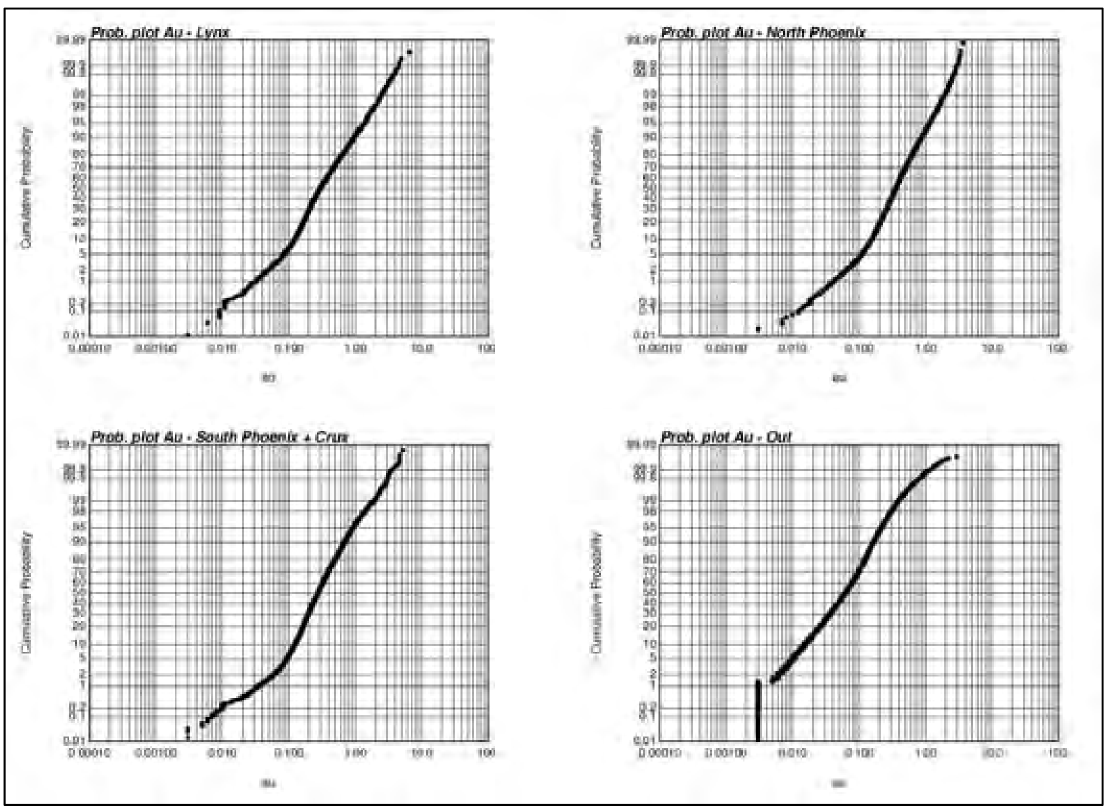


Figure 14-6 Log-Probability Plots for the Mineralized Envelopes

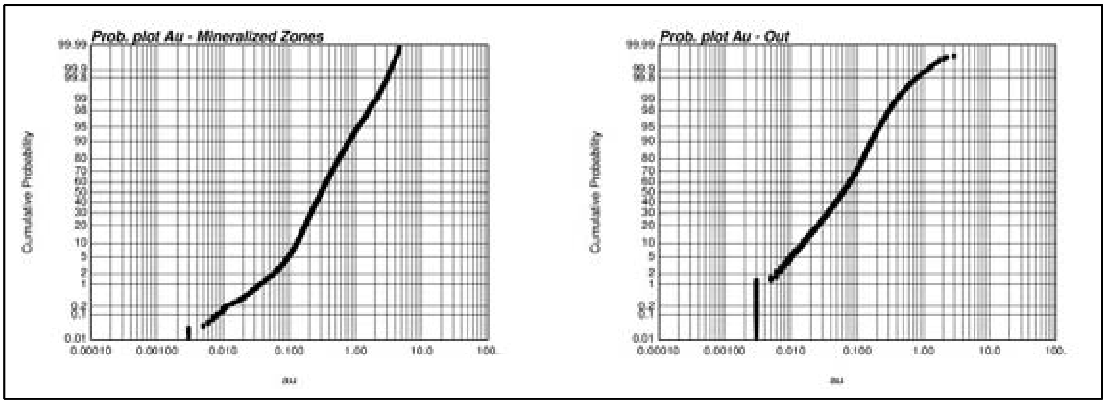


Figure 14-7 Gold Grades Log-Probability Plots for all Mineralized Envelopes and Outside

Log-probability plots showed reasonable linear trends except for very low values where a departure from log-normality was observed. No obvious high grade outliers were apparent. Therefore, no capping or high grade restrictions were applied within the mineralized envelopes.

It is well known that isolated high grades (outliers) may cause overestimation during the kriging process. To avoid this effect in the areas lying outside the mineralized envelopes, grades above 0.3 g/t were limited within a 15 m search radius.

14.5 Cell Declustering

It is typical that higher grade zones are usually more densely drilled than low grade zones. Therefore, equal weighted sample means generally produce a biased estimate of the global distribution average. These estimates are normally too high.

To avoid the effect of high grade clustering, the cell declustering technique was used. The following mean versus cell size graphs are presented in Figure 14-8:

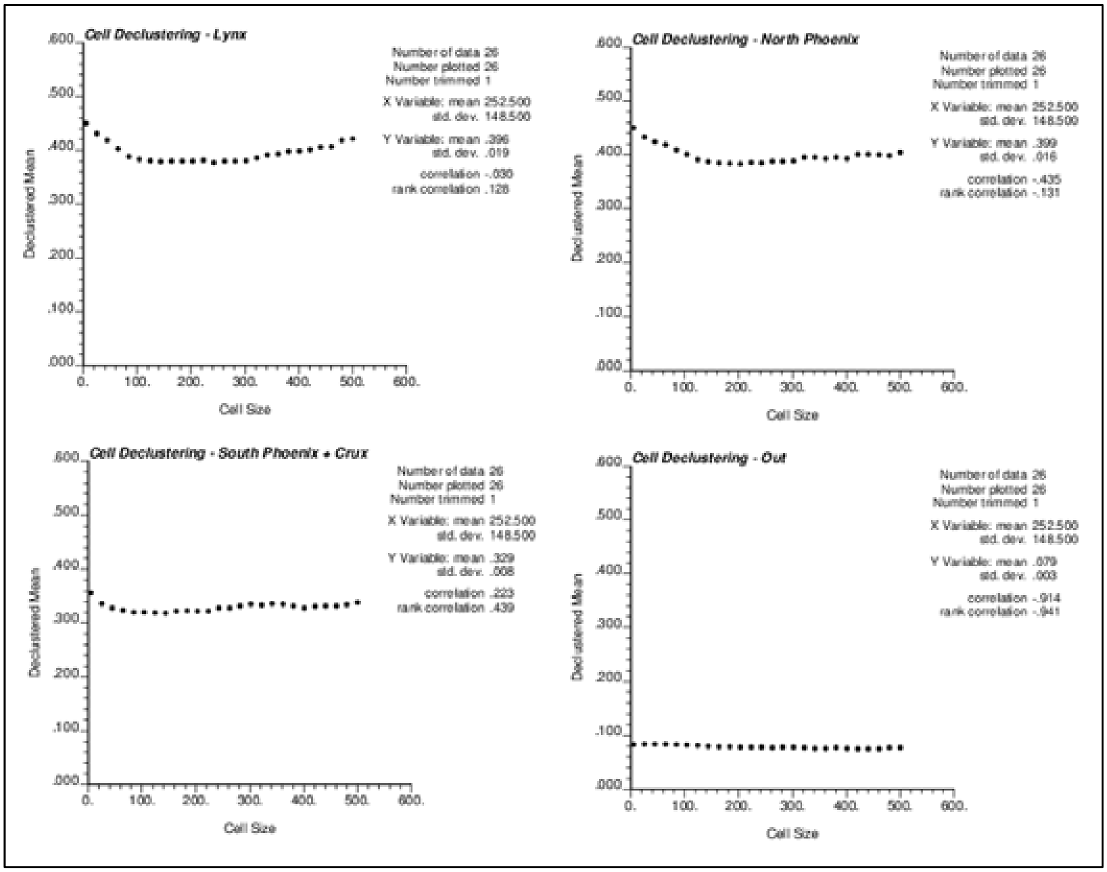


Figure 14-8 Cell Declustering: Lynx, North Phoenix, South Phoenix plus Crux, and the Outside

Table 14-3 shows the declustered sample statistics for each zone. It can be seen that the declustered means were considerably lower than the un-weighted means.

Table 14-3 Basic Sample Statistics (Declustered)

Zone	Code	Variable	Valid N	Dec-Mean	Un-Weight Mean	Median	Min	Max	STD	CV
Lynx	1	Au g/t	4,102	0.377	0.458	0.254	0.003	6.940	0.399	1.057
North Phoenix	2	Au g/t	7,227	0.383	0.456	0.3	0.003	3.679	0.303	0.789
South Phoenix + Crux	3	Au g/t	10,056	0.244	0.368	0.244	0.003	5.179	0.275	0.865
Outside	4	Au g/t	31,227	0.054	0.083	0.054	0.003	2.927	0.081	1.075

It is well known that kriging produces declustering, however, kriging declustering may differ from cell declustering. It is expected that the average grade of estimated blocks through kriging should lie somewhere between the declusterized mean and the un-weighted mean. This was be used as a global validation for kriging estimates.

14.6 Kriging Boundaries

Hard kriging boundaries were used between all kriging domains since there was a very abrupt decrease in gold grades between the mineralization lying inside the mineralized envelopes and the material lying outside. Figure 14-9, Figure 14-10 and Figure 14-11 show contact profiles between each mineralized envelope and the samples lying outside.

Contact profiles between Lynx and North Phoenix and South Phoenix and Crux are not shown since there are no mineralized samples between these estimation domains due to faulting.

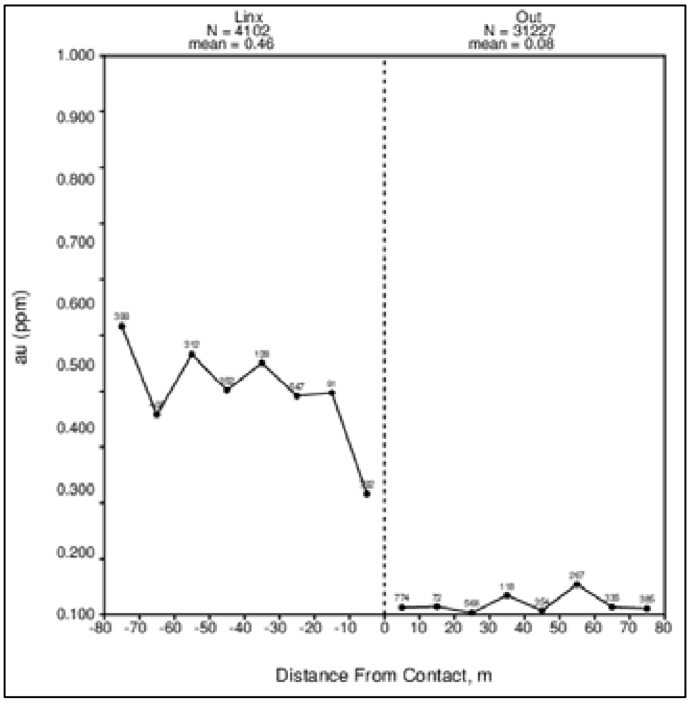


Figure 14-9 Contact Plot Between Samples in Lynx and Samples Lying Outside the Mineralized Envelopes

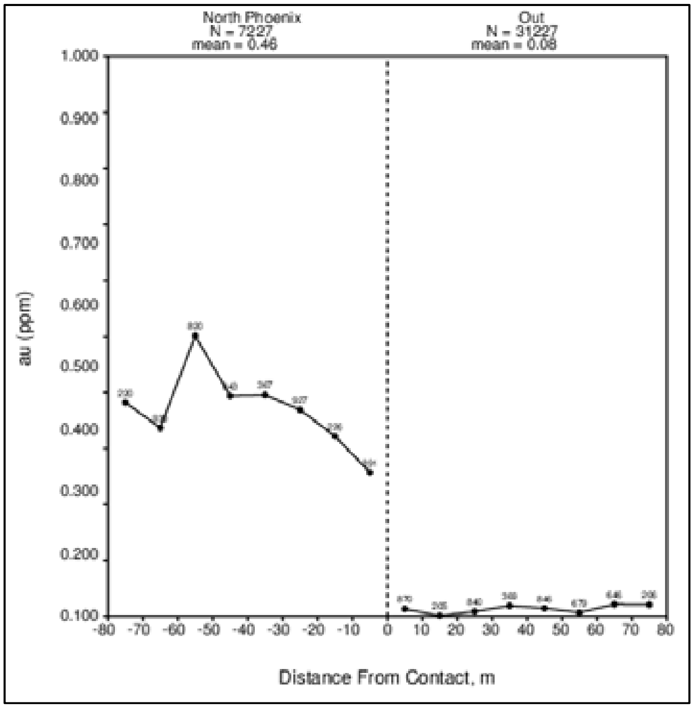


Figure 14-10 Contact plot Between Samples in North Phoenix and Samples Lying Outside the Mineralized Envelopes

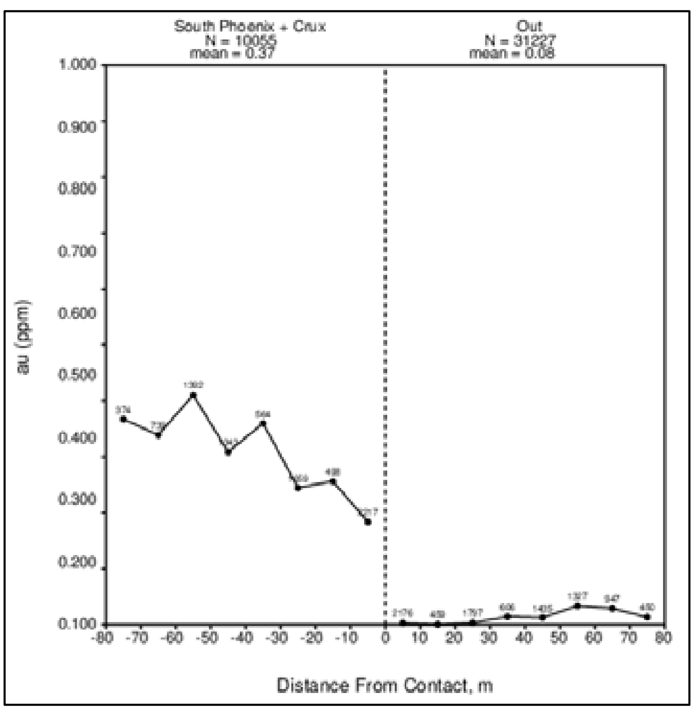


Figure 14-11 Contact Plot Between Samples in South Phoenix Plus Crux and Samples Lying Outside the Mineralized Envelopes

14.7 Variography

Variography for each domain was approached through the use of correlograms, since they are more stable than traditional variograms in the presence of outliers and mild trends that usually exist. Variogram maps (with color coded variance) were calculated in the three main planes and are shown in Figure 14-12, Figure 14-13, Figure 14-14 and Figure 14-15.

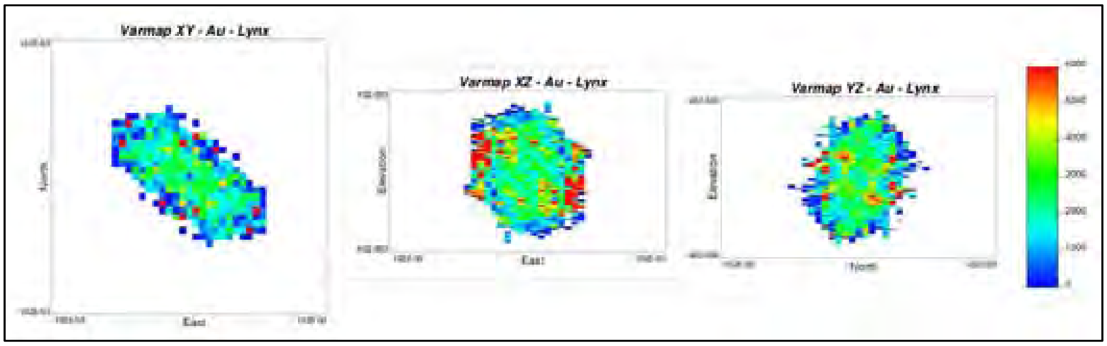


Figure 14-12 Variogram Map for Lynx (Code 1)

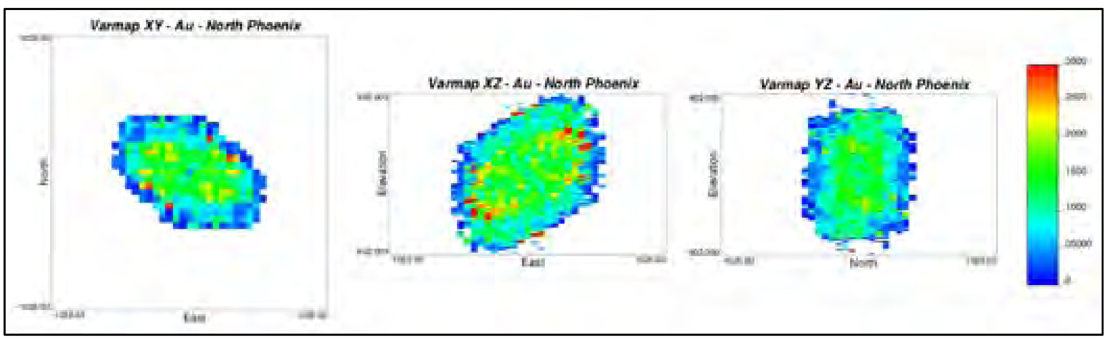


Figure 14-13 Variogram Map for North Phoenix (Code 2)

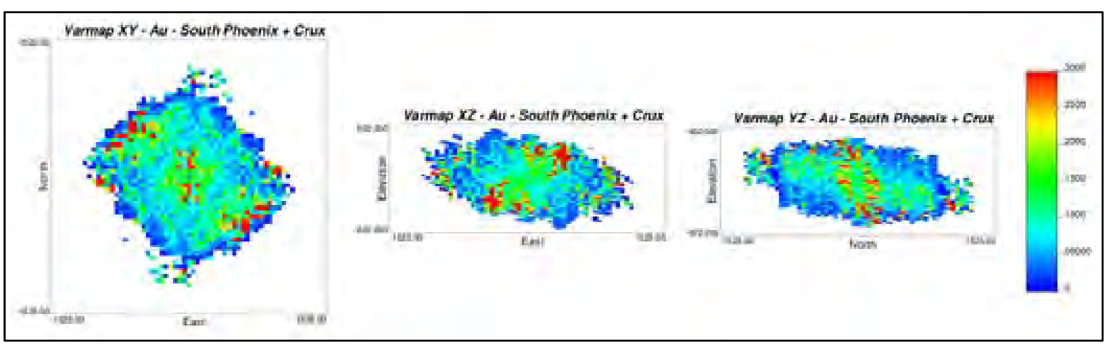


Figure 14-14 Variogram Map for South Phoenix Plus Crux (Code 3)

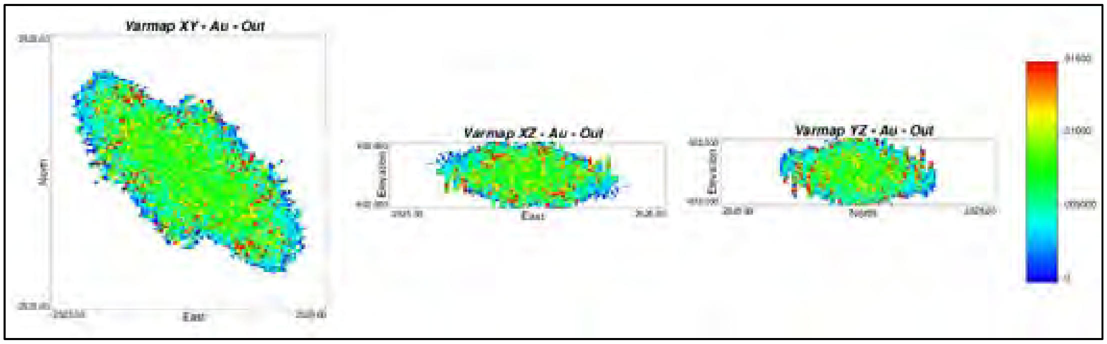


Figure 14-15 Variogram Map for Outside (Code 4)

It was observed that no strong anisotropies were revealed by the variogram maps.

The estimation of the nugget effect was done calculating and plotting "down the hole" correlograms. Anisotropy was investigated through the calculation of directional correlograms. Table 14-4 shows correlogram calculation parameters and Table 14-5 shows final correlogram models. Figure 14-16, Figure 14-17, Figure 14-18 and Figure 14-19 show experimental as well as fitted models for the estimation domains. Weak horizontal anisotropies are apparent for North Phoenix and outside. Omni-directional horizontal correlograms, shown in red, were modeled for Lynx and South Phoenix plus Crux.

Table 14-4 Cerro Maricunga Correlogram Calculation Parameters

Zone	Color	Direction	Azimuth	Az Tolerance	Az Band	Dip	Dip Tolerance	Dip Band	N Lags	Lag	Lag Tolerance
ALL	Green	East	0.0	22.5	100.0	0.0	15.0	15.0	20	50	25
	Red	North	90.0	22.5	100.0	0.0	15.0	15.0	20	50	25
	Black	Vertical	0.0	90.0	30.0	-90.0	15.0	30.0	100	8	4

Table 14-5 Cerro Maricunga Correlogram Modeling and Plotting Parameters

Zone	Color	Direction	Nugget	Sill-1	Range-1	Sill-2	Range-2	Sill-3	Range-3
Lynx	Red	East			19		70		220
	Red	North	0.15	0.65	19	0.10	70	0.10	220
	Black	Vertical			23		220		320
North Phoenix	Red	East			30		100		8
	Red	North	0.15	0.49	20	0.30	75	0.06	160
	Black	Vertical			25		80		8
South Phoenix + Crux	Red	East			20		70		250
	Red	North	0.15	0.35	20	0.30	70	0.02	250
	Black	Vertical			15		230		230
Outside	Green	East			10		20		-
	Red	North	0.20	0.62	18	0.18	210	-	-
	Black	Vertical			18		210		-

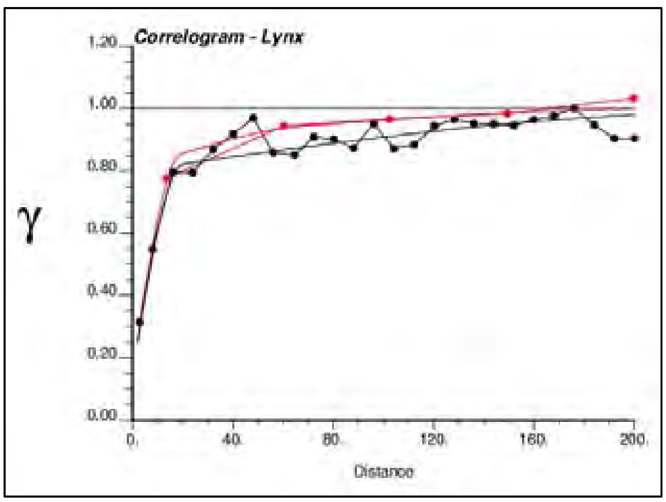


Figure 14-16 Lynx Correlogram Model

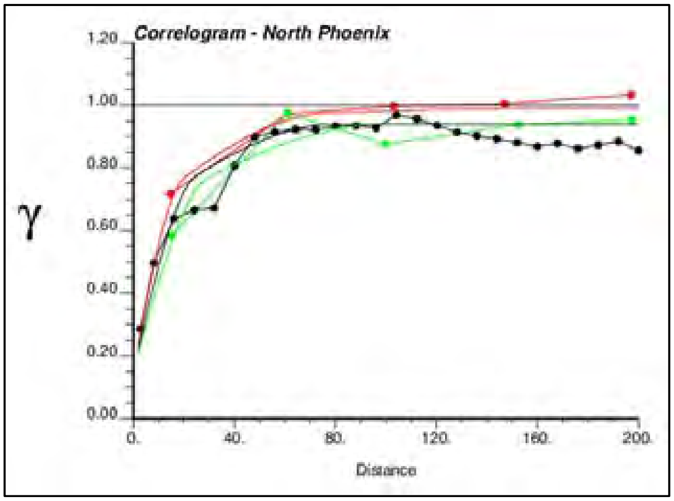


Figure 14-17 North Phoenix Correlogram Model

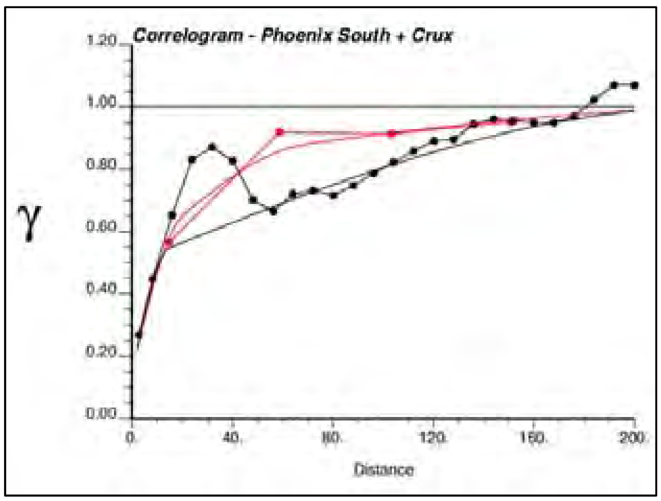


Figure 14-18 South Phoenix Plus Crux Correlogram Model

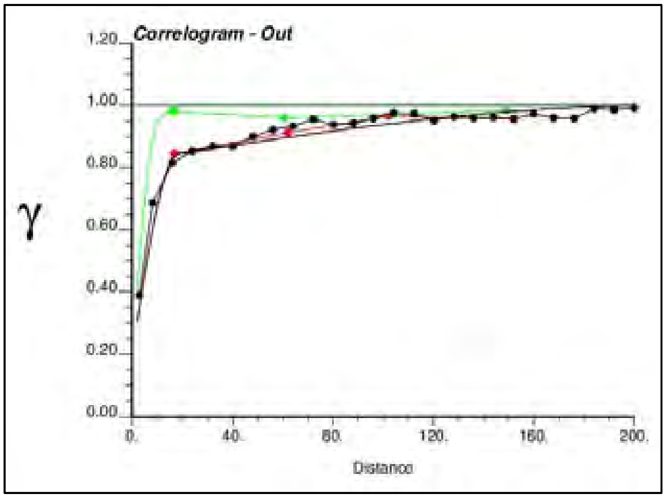


Figure 14-19 Outside Correlogram Model

Correlograms showed that spatial continuity was somewhat limited but this is frequent for this type of gold deposit. Graphs rose sharply and practical ranges in the horizontal and vertical directions were in the order of 40 and 100 metres respectively. The continuity of gold values in the vertical direction was slightly stronger than in the horizontal direction.

14.8 Block Model and Resource Estimation Plan

A gold block model consisting of 10 x 10 x 10 m blocks was created. Block model parameters are given below:

- X Origin: 480,000
- Y Origin: 7,011,600
- Z Origin: 5,150 (origin at the top of the model)
- Bearing: 45° (anti clockwise starting from x-axis)
- Plunge: 0°
- Dip: 0°
- Model Size X-Axis: 2,000 m
- Model Size Y-Axis: 3,000 m
- Model Sixe Z-Axis: 1,000 m
- Block Size X: 10 m
- Block Size Y: 10 m
- Block Size Z: 10 m
- Block Discretization: 3 x 3 x 3 (in X, Y and Z directions).

The most important variables of the model are:

- Au: Estimated Au grade in g/t

- Density: Block density
- Corrida: Au estimation pass
- NN: Number of samples used in Au estimation
- Var Au: Au kriging variance
- Categ: Resource classification category.

The grade estimation plan for Cerro Maricunga Project was carried out in three (3) passes. General settings are detailed below:

- The search radii for the first kriging pass were set at approximately the variogram ranges that correspond to 90% of the total sill in each direction
- The search radii for the third kriging pass were set quite large in order to avoid leaving too many blocks un-estimated
- All estimations were performed using the Ordinary Kriging method, including the low grade zone lying outside the mineralized envelopes
- No anisotropy rotation angles were used for search ellipsoids
- High grade restriction was used in the estimation of the material outside the mineralized zones, grades above 0.3 g/t were restricted in a 15 m search radius.

The estimation plan parameters are shown in Table 14-4.

Table 14-6 Au Estimation Plan Parameters

Zone	Range (m)				Samples		Max Samples per Hole	Discretization
	Pass	X	Y	Z	Minimum	Maximum		
Lynx	1	50	50	120	8	16	6	3 x 3 x 3
	2	180	180	200	8		6	
	3	180	180	400	4		-	
North Phoenix	1	80	50	80	8	16	6	3 x 3 x 3
	2	80	150	80	8		6	
	3	80	150	400	4		-	
South Phoenix + Crux	1	70	70	90	8	16	6	3 x 3 x 3
	2	180	180	140	8		6	
	3	180	180	400	4		-	
Outside	1	50	50	80	8	16	6	3 x 3 x 3
	2	50	100	100	8		6	
	3	150	200	400	4		-	

Statistics of gold mean grades and tonnages estimated in each kriging pass are shown in Table 14-7.

Table 14-7 Estimated Block Model Statistics

Zone	Pass	Total Blks	Estimated Blks	Non-Est Blks	% Estimated	Krig Au	Declus-Min	Declus-Max
Lynx	1	46,187	27,709	18,478	60.0	0.446	0.377	0.458
	2	18,478	18,352	126	99.7	0.328		
	3	126	126	0	100.0	0.495		
	Total	46,187	46,187	0	100.0	0.399		
North Phoenix	1	58,463	45,610	12,853	78.0	0.439	0.383	0.456
	2	12,853	5,729	7,124	87.8	0.336		
	3	7,124	7,124	0	100.0	0.346		
	Total	58,463	58,463	0	100.0	0.418		
South Phoenix + Crux	1	97,258	66,543	30,715	68.4	0.336	0.275	0.368
	2	30,715	26,646	4,069	95.8	0.281		
	3	4,069	4,069	0	100.0	0.351		
	Total	97,258	97,258	0	100.0	0.322		
Outside	1	5,798,092	137,295	5,660,797	2.4	0.078	0.075	0.083
	2	5,660,797	169,588	5,491,209	5.3	0.077		
	3	5,491,209	914,597	4,576,612	21.1	0.062		
	Total	5,798,092	1,221,480	4,576,612	21.1	0.066		

It can be seen that the percentage of estimated blocks in the first 2 passes were 99.7%and 87.7% and 95.8% for the mineralized zones, which was considered reasonable. Total estimated block percentages in all passes amounted to 100% for the mineralized zones. The total percentage of blocks estimated outside the mineralized envelopes was 21.1%.

Table 14-7 shows that the average of estimated block grades fell within the declusterized minimum and maximum grades, which corresponded to the declusterized mean and the un-weighted mean, respectively. This is true for three zones except blocks found outside the mineralized envelopes, where this condition was met for the first two kriging passes.

14.9 Validations

A series of block model validations were carried out. Details are given below.

14.9.1 Global Bias

As mentioned in previous sections, Au block grades were estimated using ordinary kriging in three passes. Additionally, declustered means obtained by nearest neighbor (NN) estimates were calculated. These consist of assigning to each block the grade of the nearest sample to the block's center.It should be noted that this validation procedure was carried out for measured and indicated resources, as will be defined in Section 14.11.

Global bias was assessed by comparing the means of the two estimates mentioned above. This validation was carried out within and outside the mineralized envelopes. Results are shown in Table 14-8. For completeness, this table also shows the cell declustered mean range obtained earlier.

As mentioned earlier, it is well known that kriging produces declustering; therefore, comparison against other declustering techniques was appropriate. Results are presented in Table 14-8. It can be seen that there is good agreement between the kriging and NN estimators for North Phoenix (2) and Outside (4). Results are not so favorable for Lynx (1) and South Phoenix plus Crux (3), where the average of the kriging estimators was higher than the NN estimator. In all cases the average of ordinary kriging estimators fell within the cell declustered mean range. Figure 14-20 shows a graphic representation of the data contained in Table 14-8.

Table 14-8 Global Bias Validation

Zone	Category	Total Blks	Krig Au	Dec Min	Dec Max	NN Au
Lynx	M + I	43,288	0.402	0.377	0.458	0.358
North Phoenix	M + I	49,162	0.432	0.383	0.456	0.416
South Phoenix + Crux	M + I	85,830	0.326	0.318	0.368	0.301
Outside	M + I	274,576	0.077	0.075	0.083	0.079

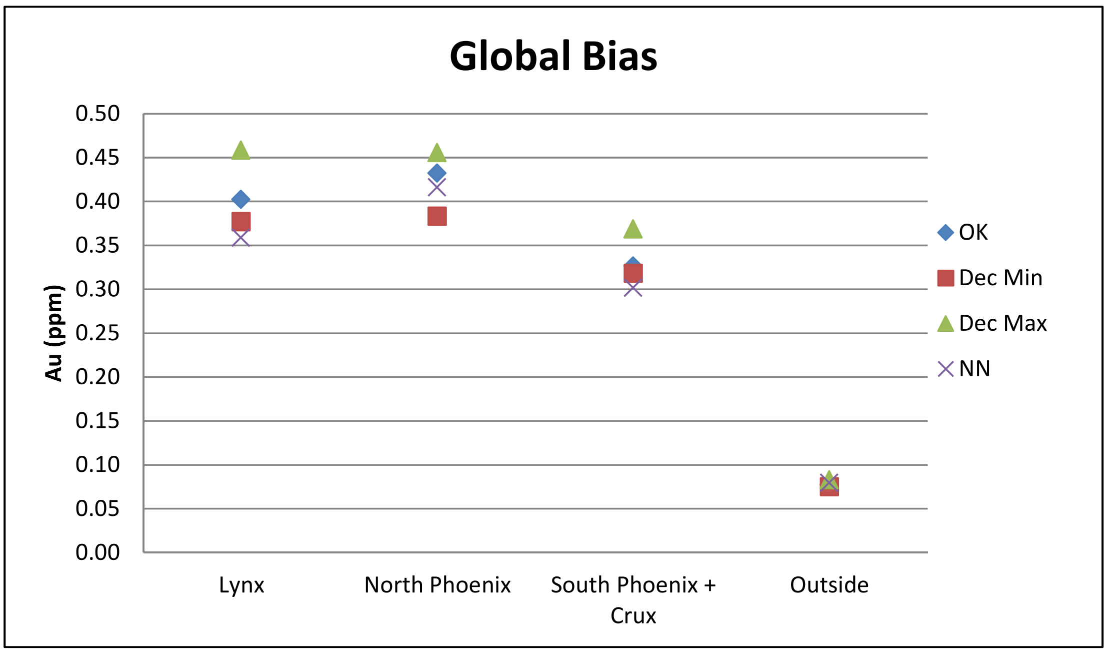


Figure 14-20 Au Global Bias

It can be concluded that kriging global validations perform as expected.

14.9.2 Drift Analysis

Drift analyses were carried out only for measured plus indicated resources by comparing the average kriging estimated block grades against the average NN estimates along 50 m slices in the X, Y and Z directions. Since the block model was rotated 45° NW, the slices are as shown in Figure 14-21 below.

Drift analyses for the entire deposit are shown in graphs; Figure 14-22, Figure 14-23 and Figure 14-24 Figure 14-25 to Figure 14-33 correspond to drift graphs for each zone in indifferent directions.

These graphs showed that kriging estimates (red) had very similar behavior to the declustered or NN estimates (green) since both curves follow very similar trends and therefore, results were considered satisfactory.

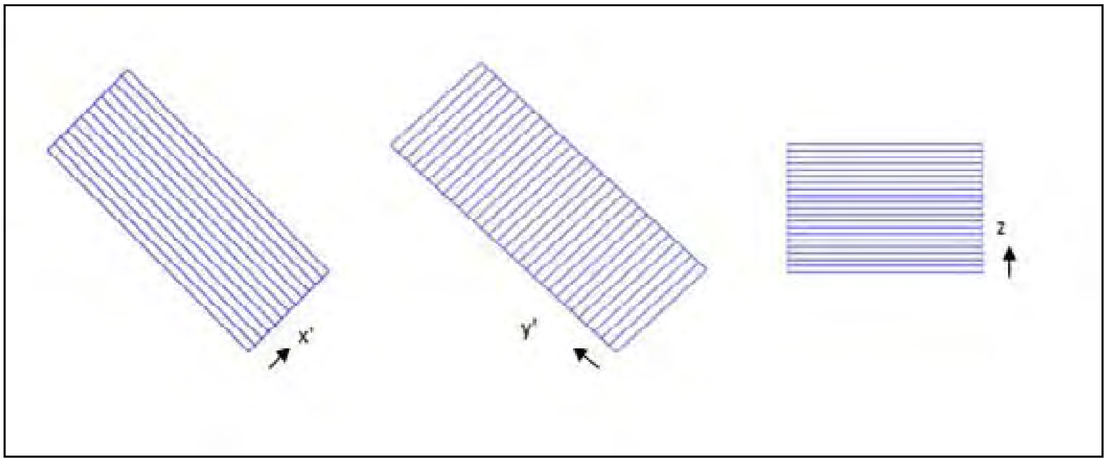


Figure 14-21 Slice Rotation According to Block Model

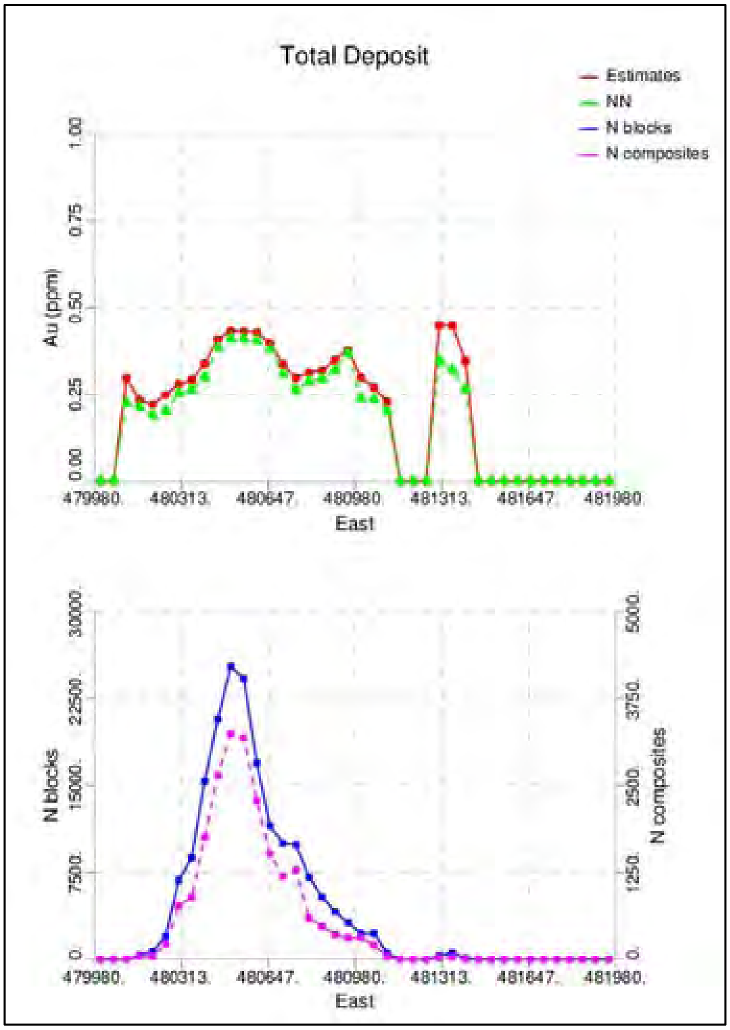


Figure 14-22 Drift Analysis – Entire Deposit (Along X)

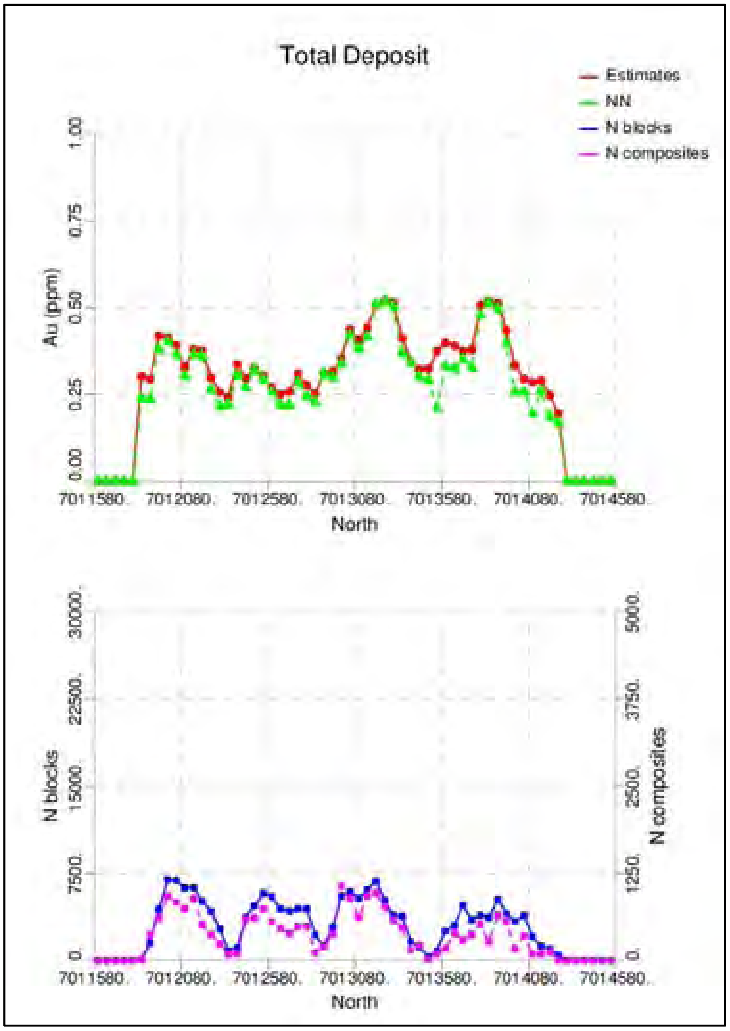


Figure 14-23 Drift Analysis – Entire Deposit (Along Y)

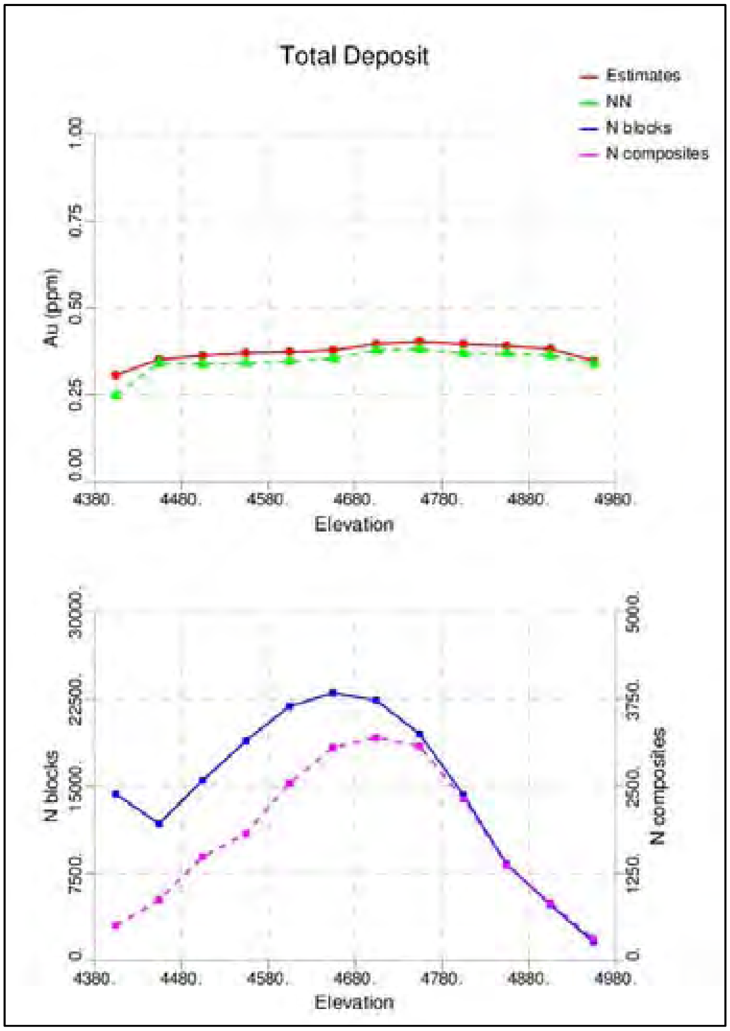


Figure 14-24 Drift Analysis – Entire Deposit (Elevation)

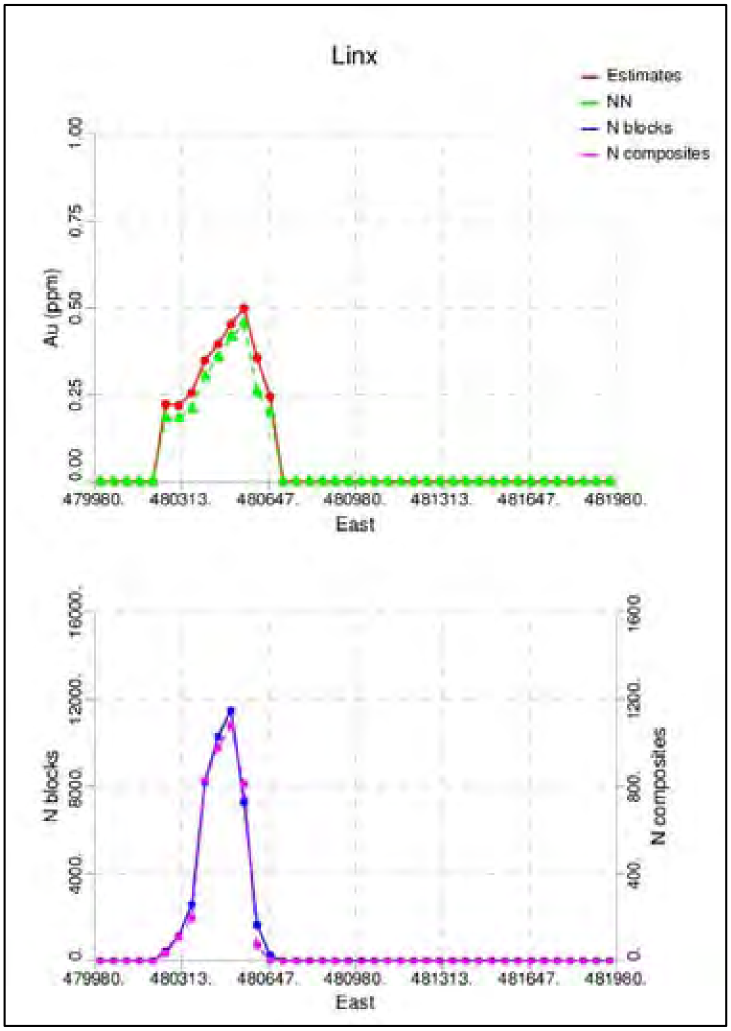


Figure 14-25 Drift Analysis – Lynx (Along X)

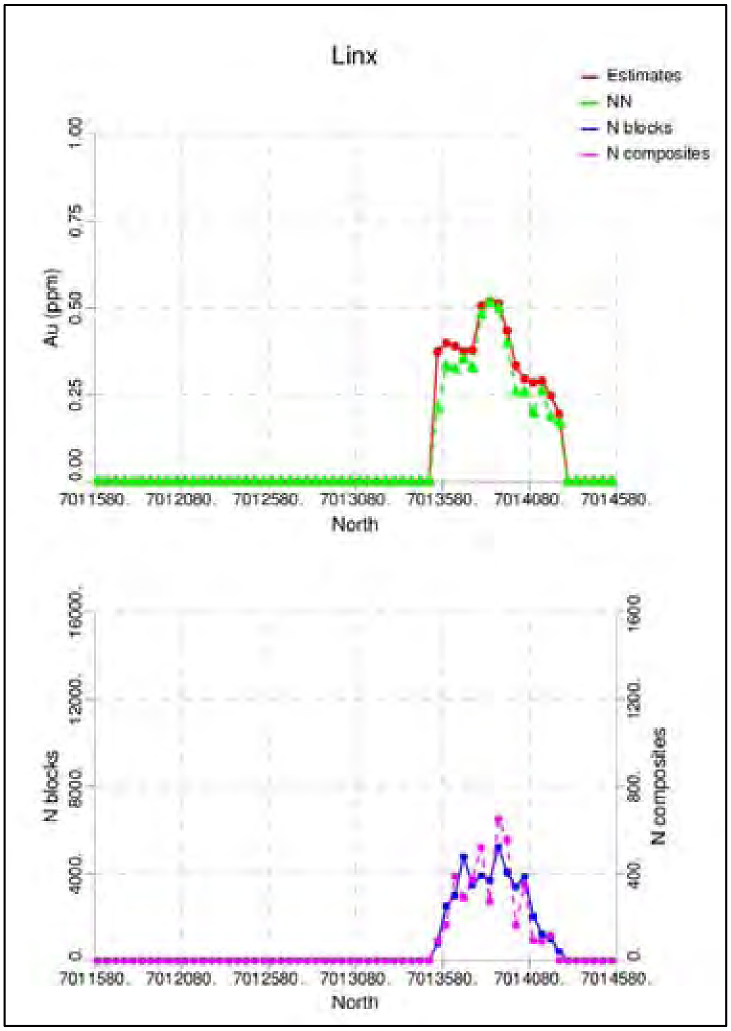


Figure 14-26 Drift Analysis – Lynx (Along Y)

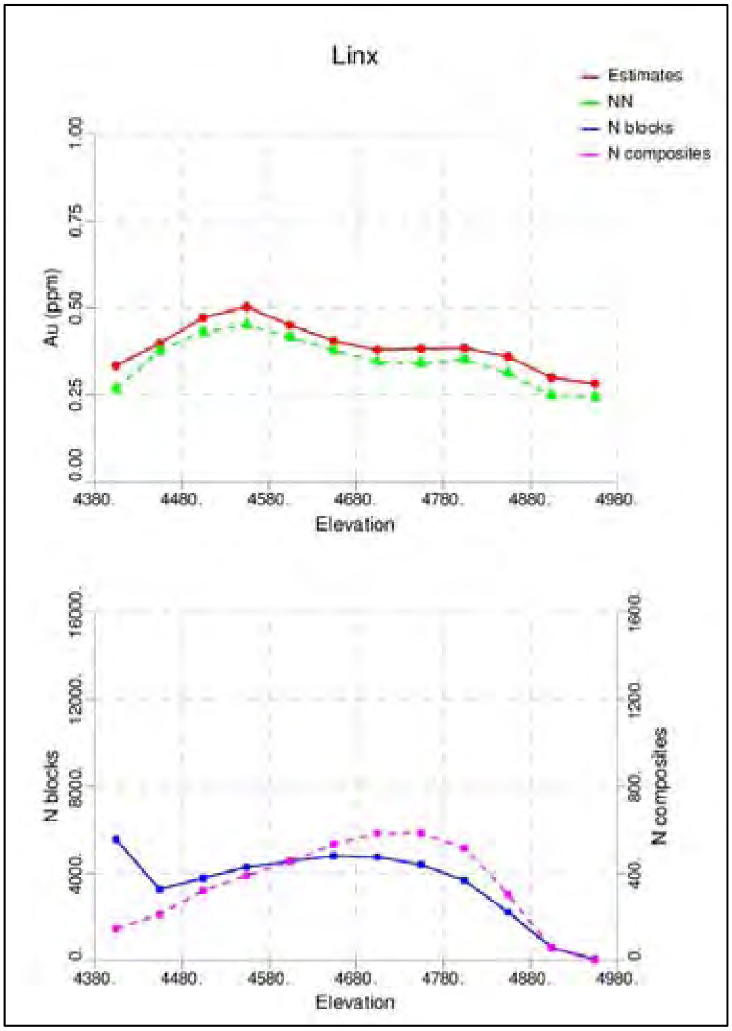


Figure 14-27 Drift Analysis – Lynx (Elevation)

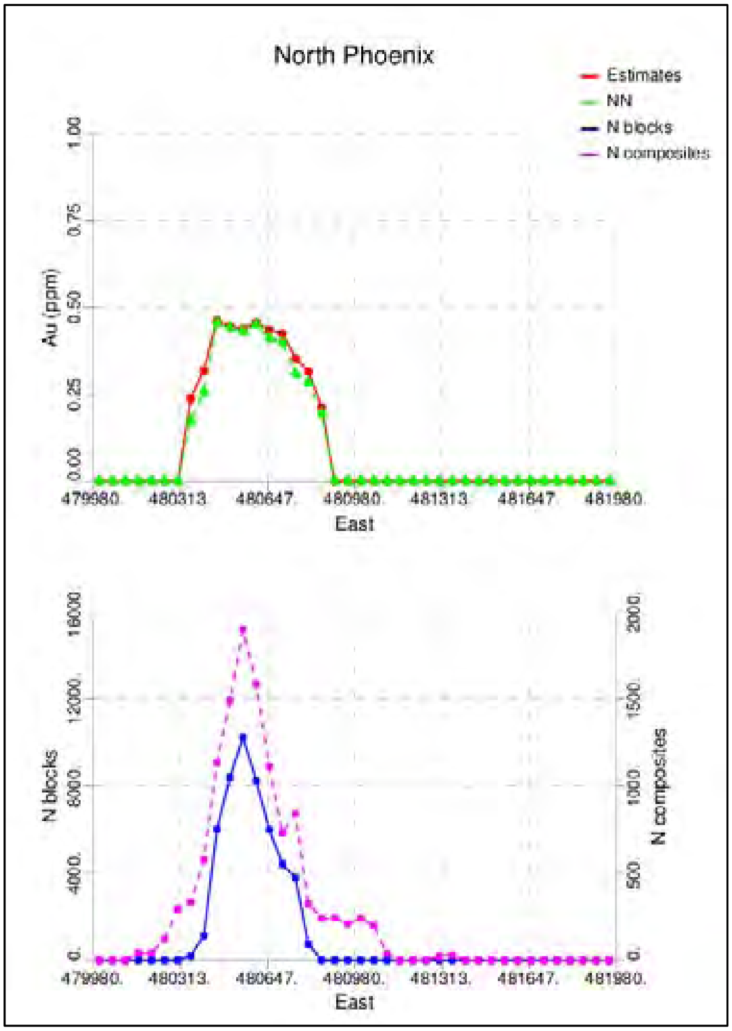


Figure 14-28 Drift Analysis – North Phoenix (Along X)

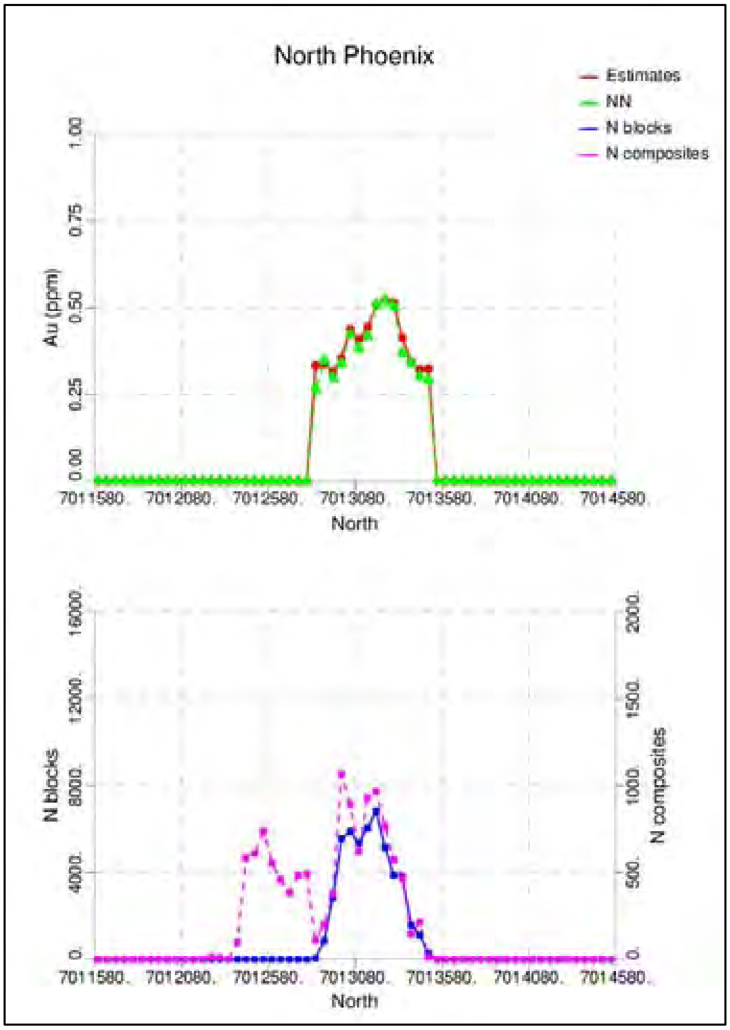


Figure 14-29 Drift Analysis – North Phoenix (Along Y)

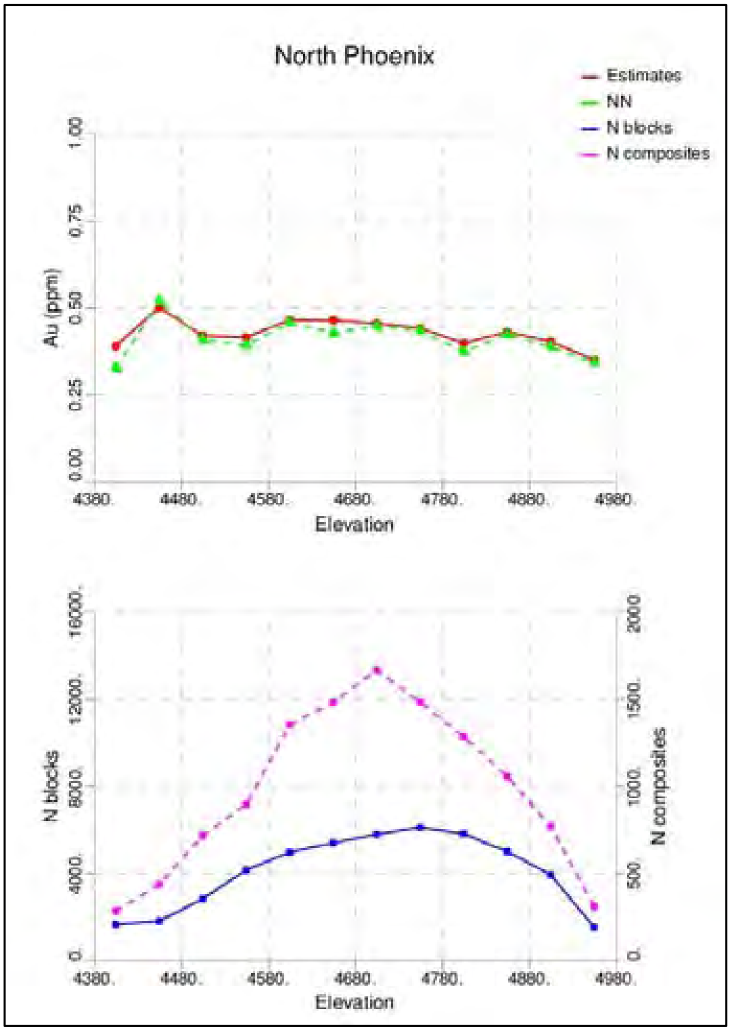


Figure 14-30 Drift Analysis – North Phoenix (Elevation)

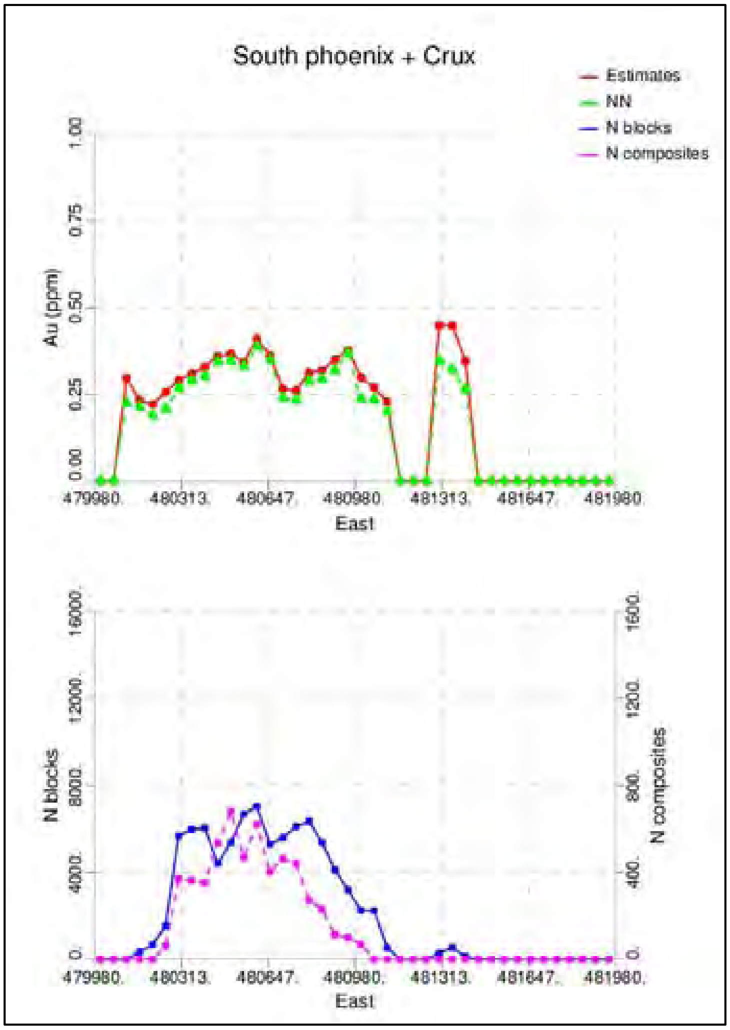


Figure 14-31   Drift Analysis – South Phoenix + Crux (Along X)

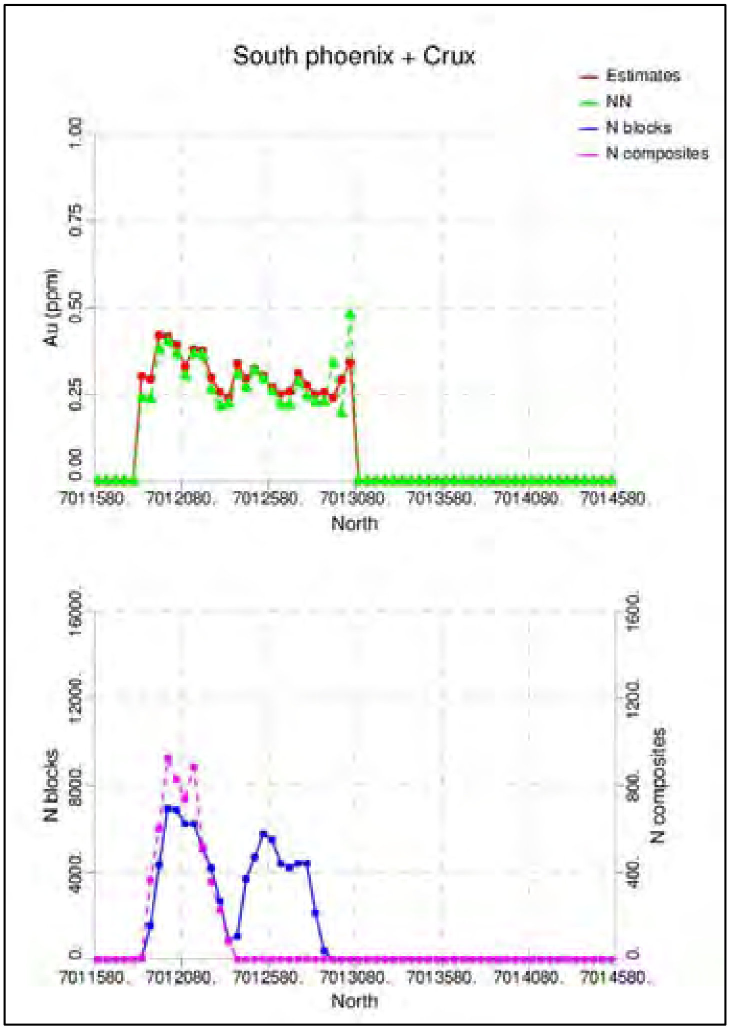


Figure 14-32 Drift Analysis – South Phoenix + Crux (Along Y)

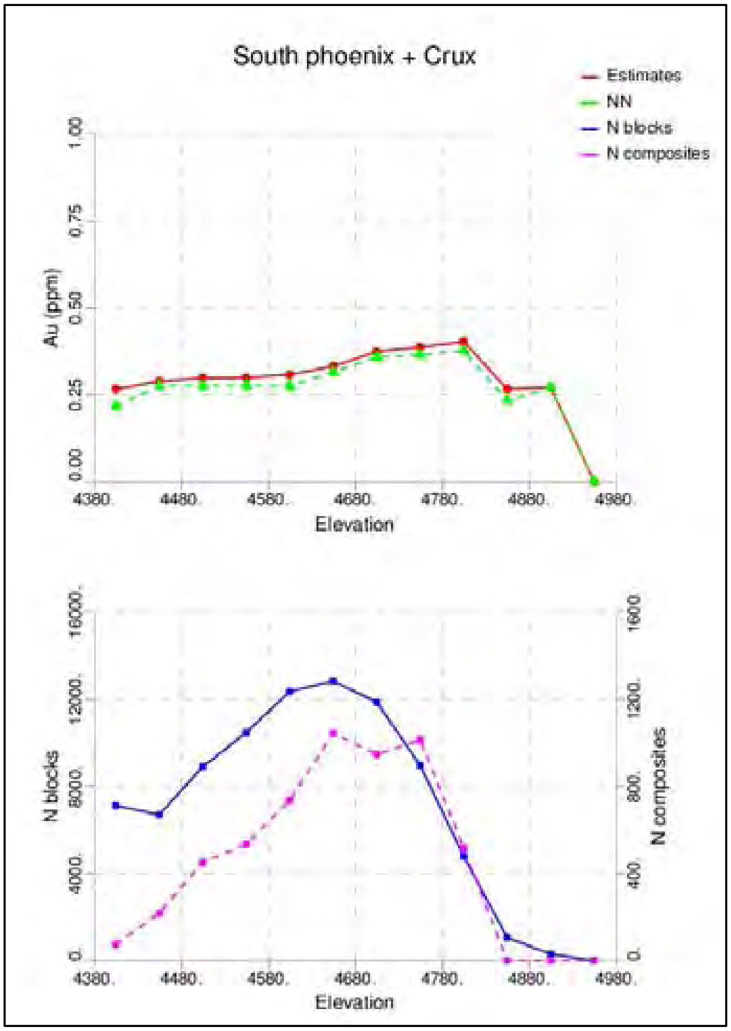


Figure 14-33 Drift Analysis – South Phoenix + Crux (Elevation)

Drift analysis results are considered adequate.

14.9.3 Graphic Validation

Four cross sections were prepared in order to compare block estimates against drill hole sample grades using the same color scheme. One section was chosen for each of the following zones: Lynx, North Phoenix, South Phoenix and Crux. The four cross sections are depicted on plan view 4650 in Figure 14-34.

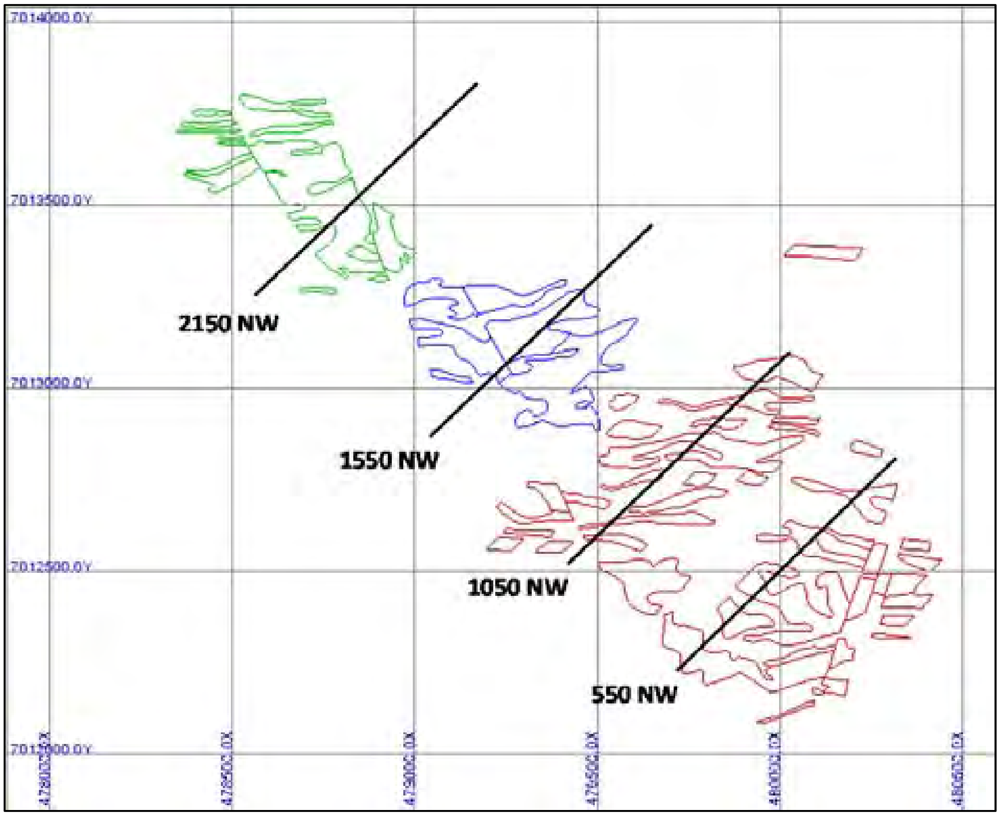


Figure 14-34 View of Cross Sections on Plan View 4650

The gold grade color scheme used is shown in Table 14-9.

Table 14-9 Color Scheme Used for Cross Sections

Gold Grade Interval		Color
From	To	
0.00	0.15	Gray
0.15	0.18	Green
0.18	0.30	Yellow
0.30	1.00	Red
1.00	100	Purple

Individual sections are shown in Figure 14-35, Figure 14-36, Figure 14-37 and Figure 14-38. Generally drill hole high and low grade zones are well reproduced in the block model. Results were considered satisfactory.

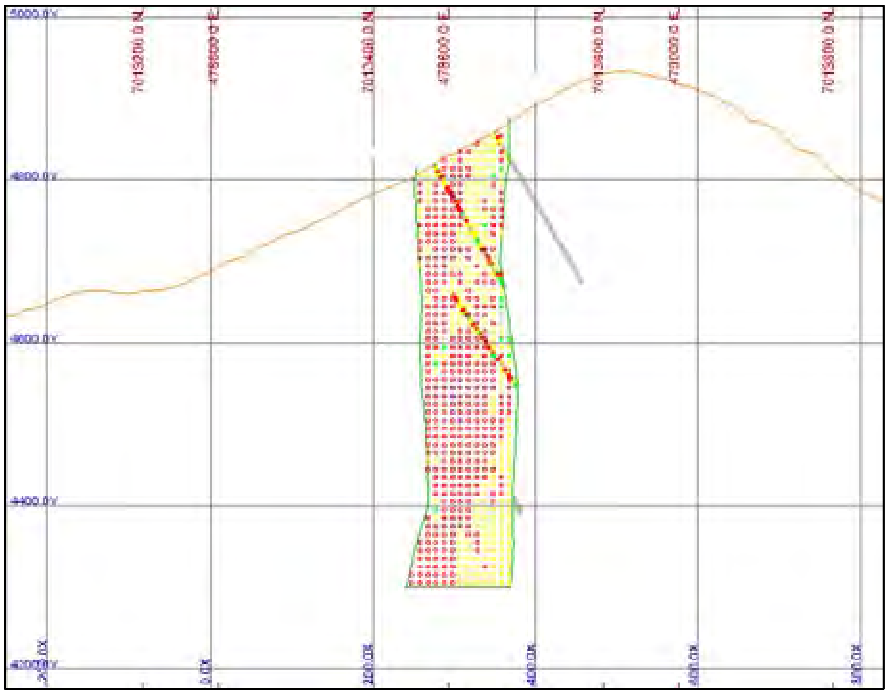


Figure 14-35 Lynx Cross Section (2150)

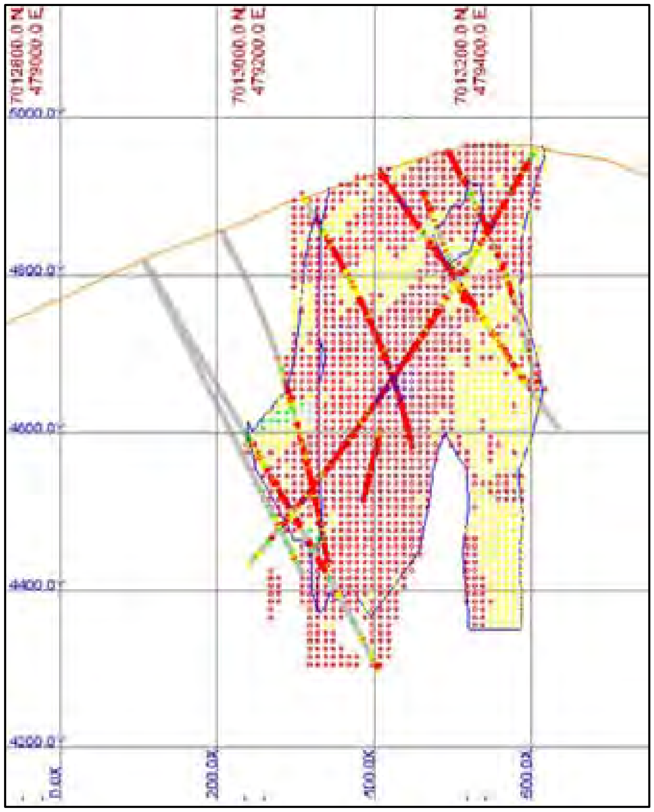


Figure 14-36 Phoenix Cross Section (1550)

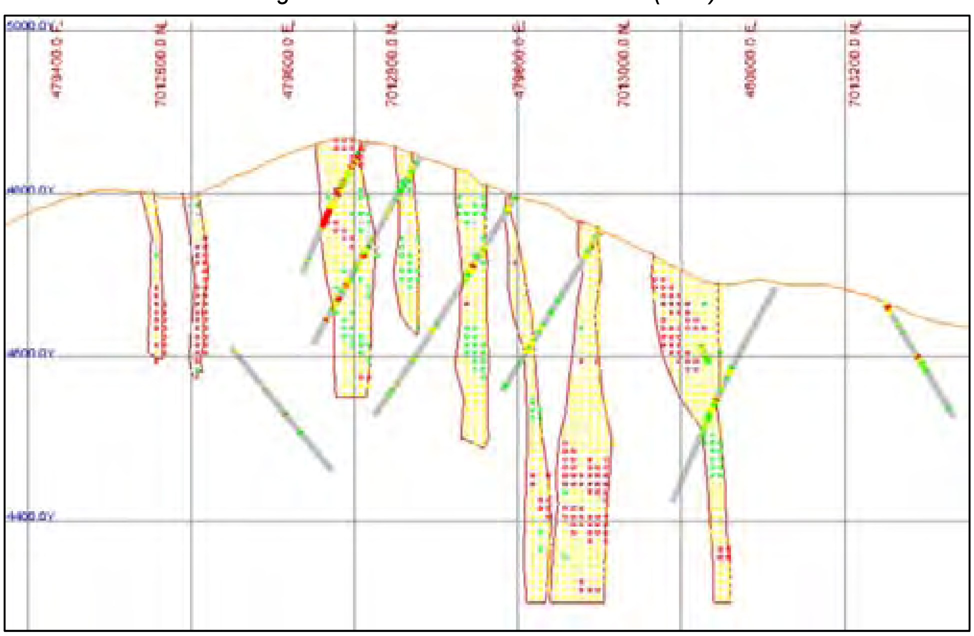


Figure 14-37 Phoenix Cross Section (1050)

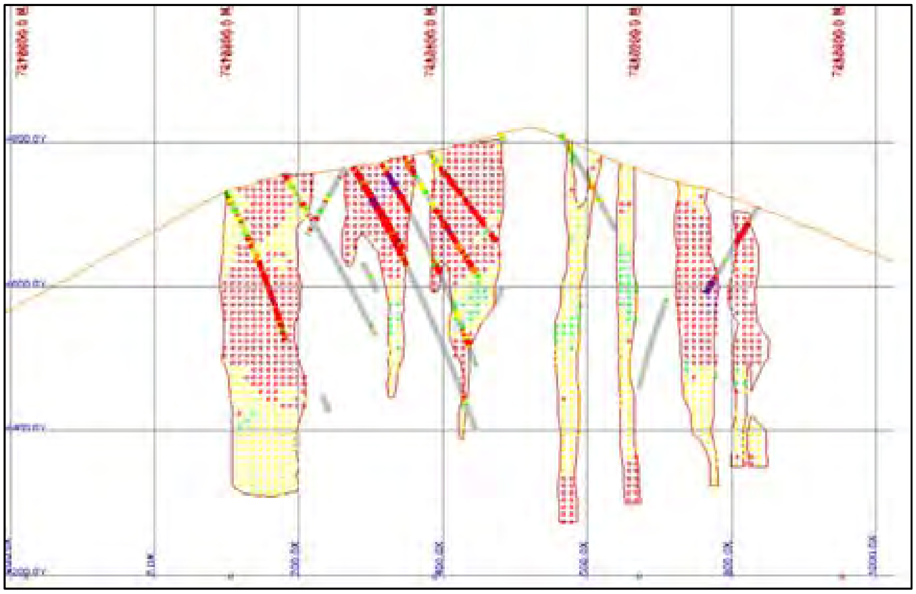


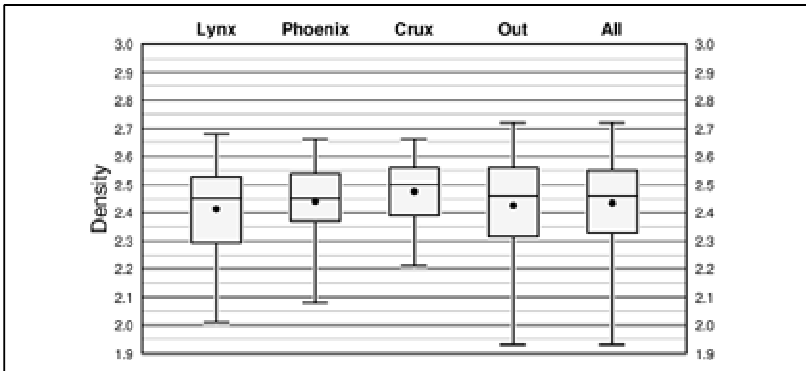
Figure 14-38 Crux Cross Section (0550)

14.10 Specific Gravity Model

A total of 527 10 cm core specimens were tested for specific gravity via the wax coated method.

All core specimens were photographed and described in detail for future use.

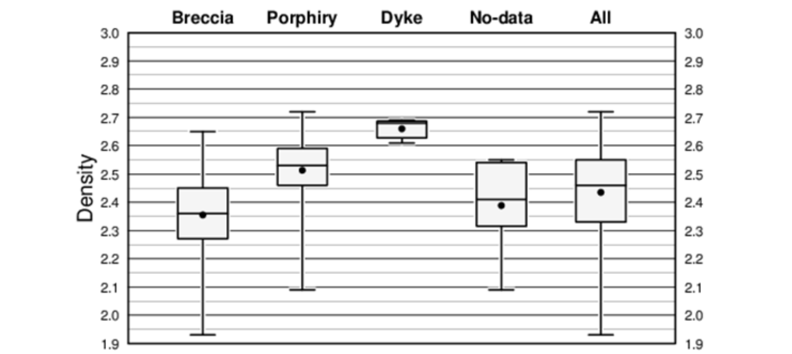
The first attempt was estimating a specific gravity to each mineralized envelope, however, no clear trend was found, as is shown in Figure 14-39.



	Lynx	Phoenix	Crux	Out	All	
Number of data	59	154	59	255	527	Number of data
Mean	2.4134	2.4406	2.4744	2.4271	2.4348	Mean
Maximum	2.68	2.66	2.66	2.72	2.72	Maximum
Upper quartile	2.5275	2.54	2.56	2.56	2.55	Upper quartile
Median	2.45	2.45	2.5	2.46	2.46	Median
Lower quartile	2.2925	2.37	2.39	2.3125	2.33	Lower quartile
Minimum	2.01	2.08	2.21	1.93	1.93	Minimum
Variance	0.0225	0.0157	0.0122	0.0272	0.0219	Variance
CV	0.0622	0.0514	0.0446	0.0679	0.0608	CV
Skewness	-0.3488	-0.5176	-0.5838	-0.6365	-0.6536	Skewness

Figure 14-39 Density within the Mineralized Envelopes and Outside

The second attempt was the use lithological units assigned to each core specimen. Figure 14-40 shows a box plot of specific gravities within each lithological domain.



	Breccia	Porphiry	Dyke	No-data	All	
Number of data	254	258	3	12	527	Number of data
Mean	2.3548	2.513	2.66	2.3883	2.4348	Mean
Maximum	2.65	2.72	2.69	2.55	2.72	Maximum
Upper quartile	2.45	2.59	2.6875	2.54	2.55	Upper quartile
Median	2.36	2.53	2.68	2.41	2.46	Median
Lower quartile	2.27	2.46	2.6275	2.315	2.33	Lower quartile
Minimum	1.93	2.09	2.61	2.09	1.93	Minimum
Variance	0.0199	0.011	0.0013	0.0226	0.0219	Variance
CV	0.0599	0.0417	0.0134	0.063	0.0608	CV
Skewness	-0.3675	-0.9177	-0.6655	-0.6563	-0.6536	Skewness

Figure 14-40   Density for the Lithological Domains

Figure 14-40 shows that the average density within the porphyry was slightly higher than that found within the breccia domain. However, since density specimens were only 10 cm long some discrepancies occurred between the lithological characterization of specimens and the three dimensional lithological model that contained them. Therefore, it was decided to estimate a specific gravity for the entire deposit using the ordinary kriging method without lithological control.

Figure 14-41 shows the experimental as well as fitted variogram model for density. Table 14-10 shows variogram model parameters.

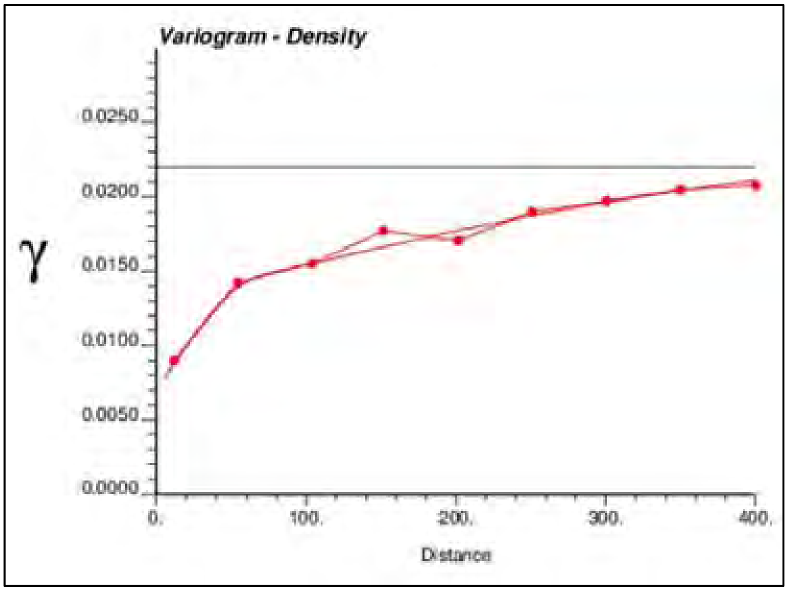


Figure 14-41 Variogram Model for Density

Table 14-10 Variogram Model Parameters for Density

Variable	Color	Direction	Nugget	Sill-1	Range-1	Sill-2	Range-2
Specific Gravity	Red	Omni-Directional	0.007	0.006	70	0.009	550

Estimation parameters used for the estimation of the specific gravity model are shown in Table 14-11. General considerations are:

- Density was estimated using ordinary kriging
- Density was estimated in 1 pass.

Table 14-11 Density Estimation Plan

Zone	Pass	Range (m)			Samples		Discretization
		X	Y	Z	Minimum	Maximum	
ALL	1	1000	1000	1000	3	8	3 x 3 x 3

The density model was validated via drift analysis shown in Figure 14-42, Figure 14-43 and Figure 14-44.

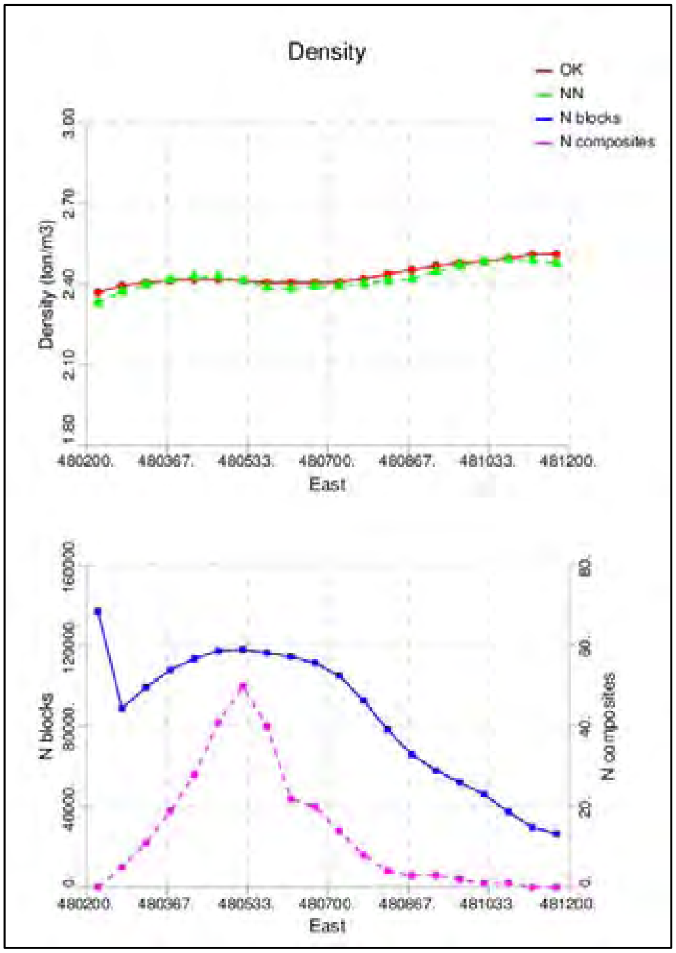


Figure 14-42   Drift Analysis – Density (Along X)

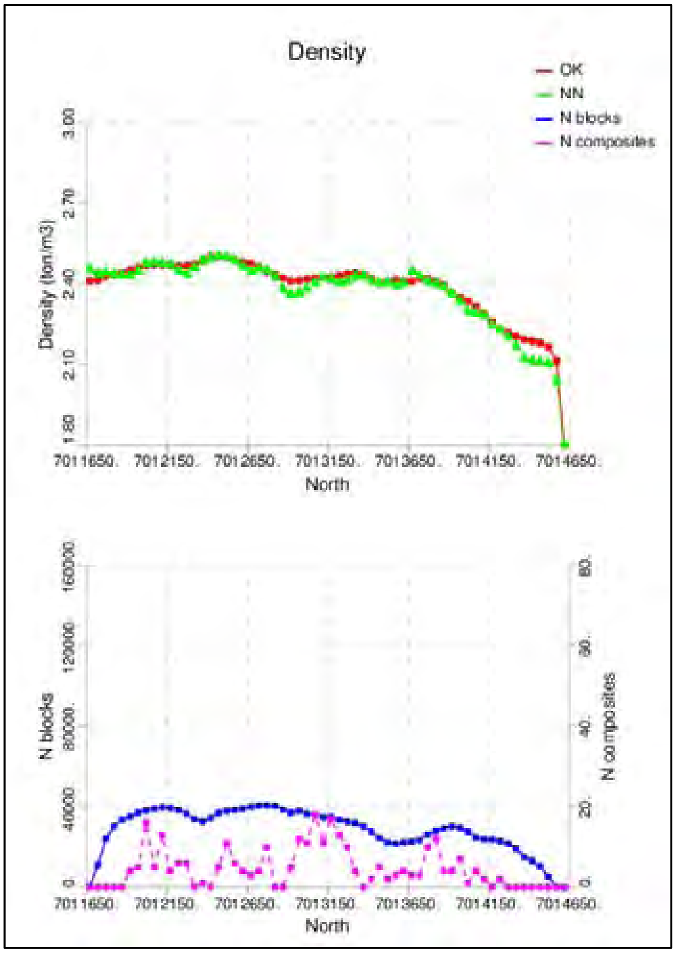


Figure 14-43 Drift Analysis – Density (Along Y)

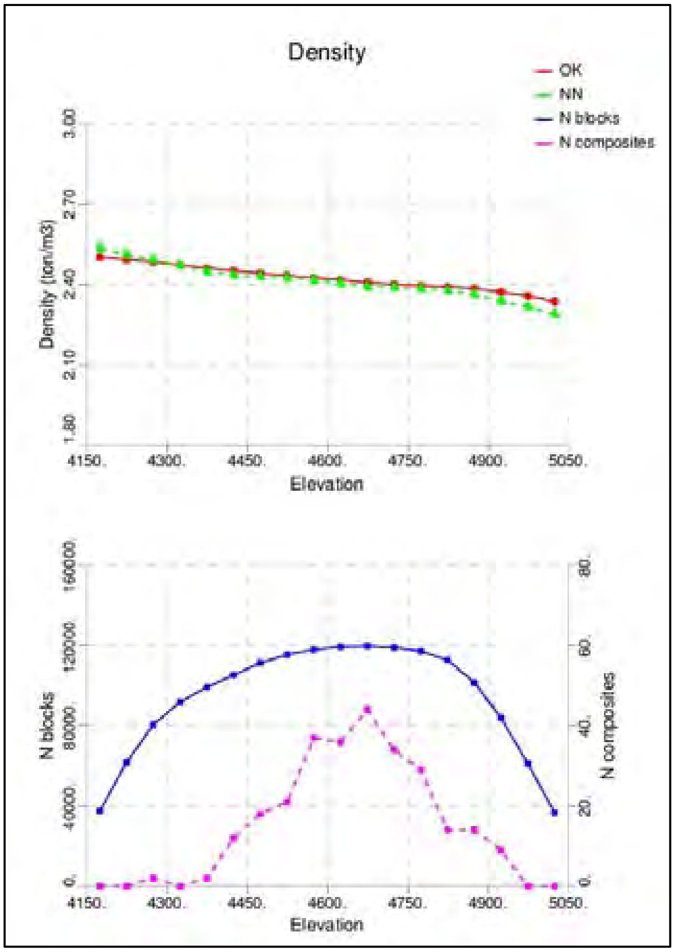


Figure 14-44 Drift Analysis – Density (Elevation)

14.11 Resource Categorization

Resource categorization consisted in assigning categories of measured, indicated and inferred to the estimated blocks within the block model. Obviously, denser drilling grids were associated to the more reliable category (measured) and very sparse drilling grids generated blocks classified within the inferred category.

In order to associate drilling grid configurations to the measured, indicated and inferred categories, the following statistical approach was used. This approach was considered to be compatible with the Canadian NI-43-101 Code.

Annual ore production grade and tonnage should be known with an error of ±15% with 90% confidence in order for the resource to be classified as indicated.

Quarterly production should be known with an error of ±15% with 90% confidence in order for the resource to be classified as measured.

Using these guidelines, idealized blocks approximating quarterly and annual production targets were estimated using a single ordinary kriging calculation for different sampling grids. Gold correlograms were used to estimate the ideal blocks. The resulting kriging variances were multiplied by the population variance and then divided by the population mean squared in order to obtain relative variances. Two independent loading points were assumed to obtain the final confidence limits. These are expressed as percentages and are given by the following expression (assuming errors to be normally distributed):

$$\mathbf{90\% \cdot \textit{Central Limit} = 1.646 \cdot 100 \cdot \sqrt{\frac{\textit{Kriging Variance} \cdot \textit{Variance}}{\textit{Mean}^2}}}$$

Equation 14-1

Grid spacing which produced confidence limits less than 15.0 percent were selected as the basis for the classification scheme.

A production target of 80,000 tonnes per day was recommended for this analysis. Other parameters are shown in Table 14-12.

Table 14-12 Additional Data Used for Resource Categorization

Variable	Parameter
Bench Height	10 m
Production Block Height	10 m and 20 m
Average Specific Gravity	2.44 t/m^3
N° of Independent Loading Points	2
Tonnes per day	80,000

Drilling grids of 50 x 50 metres, 50 x 100 metres and 100 x 100 metres were used. The analysis was carried out for 10 m and 20 m high mining blocks in order to compare results. Samples along the drill holes were located every 2.0 m. The 50 x 50 m and 50 x 100 m grids used were similar to the actual drilling grid used in the exploration campaigns: drill hole lines oriented in an N-E direction were spaced every 50 m and drill holes within the lines were separated every 50 or 100 m. All drill holes were assumed with a 60° dip angle. As an example, Figure 14-45 and Figure 14-46 show the sample data used for the 50 x 100 m grid, a production level of 80,000 tonnes per day and 20 m high mining blocks. Yearly (blue) and quarterly (red) production blocks are also shown.

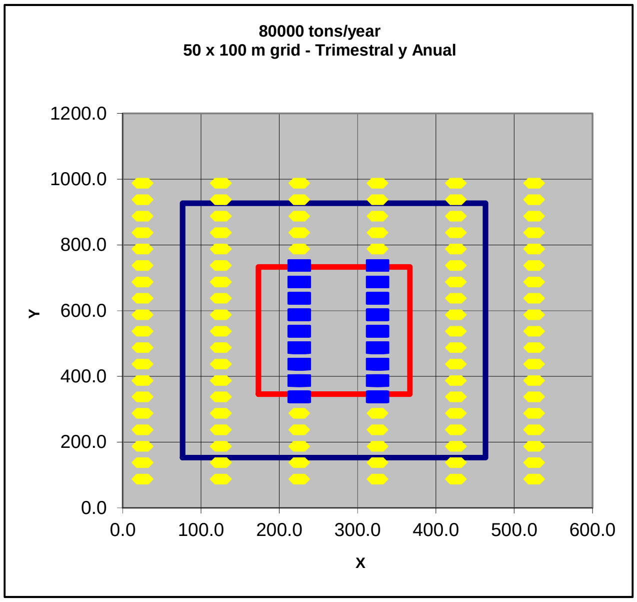


Figure 14-45 50 x 100m Drilling Grid for 80,000 Tonnes / Day-Plan View

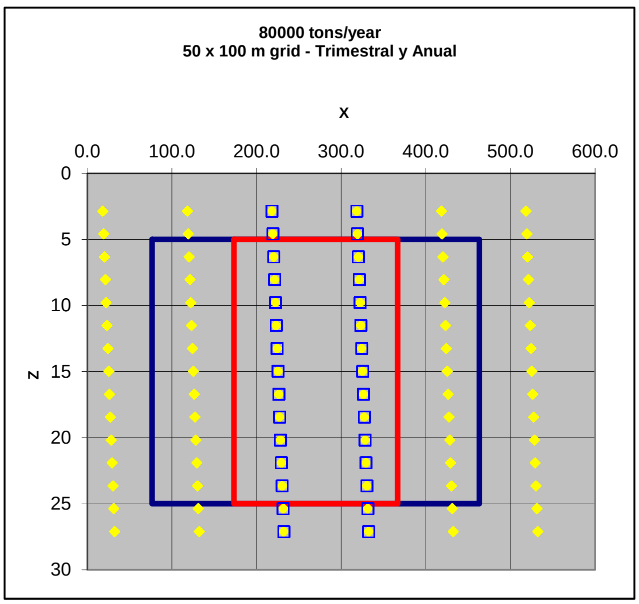


Figure 14-46 50 x 100m Drilling Grid for 80,000 Tonnes / Day – Vertical View

A single ordinary kriging calculation was performed for each block size and drilling grid. The kriging variance was calculated in each case and the 90% central confidence limits were calculated using Equation 14-1. Results for the three mineralized zones are presented in Table 14-13 for annual and quarterly production targets which correspond to measured and indicated resources respectively. Also included are relative error v/s drilling grid graphs for measured and indicated resources enhancing the 15% relative error threshold (Figure 14-47 and Figure 14-38).

Table 14-13 Kriging Errors for 50 x 50, 50 x 100 and 100 x 100 grids

Zone	N°	Drilling-Grid	Error (%)	Period	N°	Kriging	Mean	Variance	Corrected	Error (%)
Lynx (North)	2	50 x 50	4.06	Annual	2,831	0.001944102	0.377	0.1592	0.000608383	4.06
		50 x 100	5.73		1,710	0.003877917			0.001213564	5.73
		100 x 100	7.71		935	0.007024157			0.002198123	7.71
		50 x 50	7.90	Quarter	540	0.007377751			0.002308776	7.90
		50 x 100	11.96		270	0.016887770			0.005284818	11.96
		100 x 100	14.30		255	0.024150640			0.007557643	14.30
North Phoenix (Central)	2	50 x 50	4.17	Annual	2,831	0.002050549	0.383	0.0918	0.000641694	4.17
		50 x 100	5.74		1,710	0.003897094			0.001219547	5.74
		100 x 100	8.62		935	0.008771146			0.002744821	8.62
		50 x 50	7.92	Quarter	540	0.007402978			0.002316670	7.92
		50 x 100	11.92		270	0.016769470			0.005247797	11.92
		100 x 100	15.82		255	0.029549470			0.009247139	15.82
South Phoenix Crux (South)	2	50 x 50	3.60	Annual	2,831	0.001280245	0.318	0.0756	0.000478712	3.60
		50 x 100	6.15		1,710	0.003737601			0.001397572	6.15
		100 x 100	8.88		935	0.007789494			0.002912664	8.88
		50 x 50	6.95	Quarter	540	0.004778311			0.001786716	6.95
		50 x 100	12.36		270	0.015092700			0.005643494	12.36
		100 x 100	16.65		255	0.027390350			0.010241860	16.65
Lynx (North)	1	50 x 50	3.82	Annual	2,925	0.001726733	0.377	0.1592	0.000540360	3.82
		50 x 100	5.24		1,575	0.003241271			0.001014315	5.24
		100 x 100	6.92		882	0.007789494			0.001770675	6.92
		50 x 50	6.38	Quarter	882	0.004778311			0.001503021	6.38
		50 x 100	9.51		378	0.015092700			0.003340618	9.51
		100 x 100	14.15		162	0.027390350			0.007402056	14.15
North Phoenix (Central)	1	50 x 50	3.75	Annual	2,925	0.001658094	0.383	0.0918	0.000518880	3.75
		50 x 100	5.17		1,575	0.003159687			0.000988785	5.17
		100 x 100	7.45		882	0.006557210			0.002051999	7.45
		50 x 50	6.33	Quarter	882	0.004727165			0.001479308	6.33
		50 x 100	9.43		378	0.010504410			0.001064443	9.43
		100 x 100	14.76		162	0.025726520			0.002024011	14.76
South Phoenix Crux (South)	1	50 x 50	3.35	Annual	2,925	0.001061140	0.318	0.0756	0.000413601	3.35
		50 x 100	5.37		1,575	0.002846696			0.003287225	5.37
		100 x 100	7.40		882	0.005412921			0.008050795	7.40
		50 x 50	5.79	Quarter	882	0.003311916			0.001238399	5.79
		50 x 100	9.67		378	0.009244959			0.003456895	9.67
		100 x 100	14.52		162	0.020849160			0.007795262	14.52

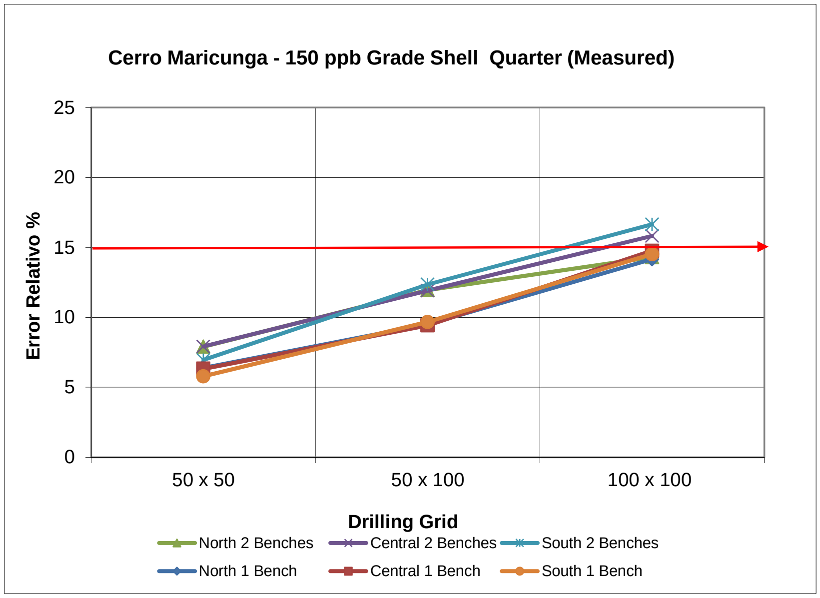


Figure 14-47 Relative Error v/s Drilling Grid – Measured Resources

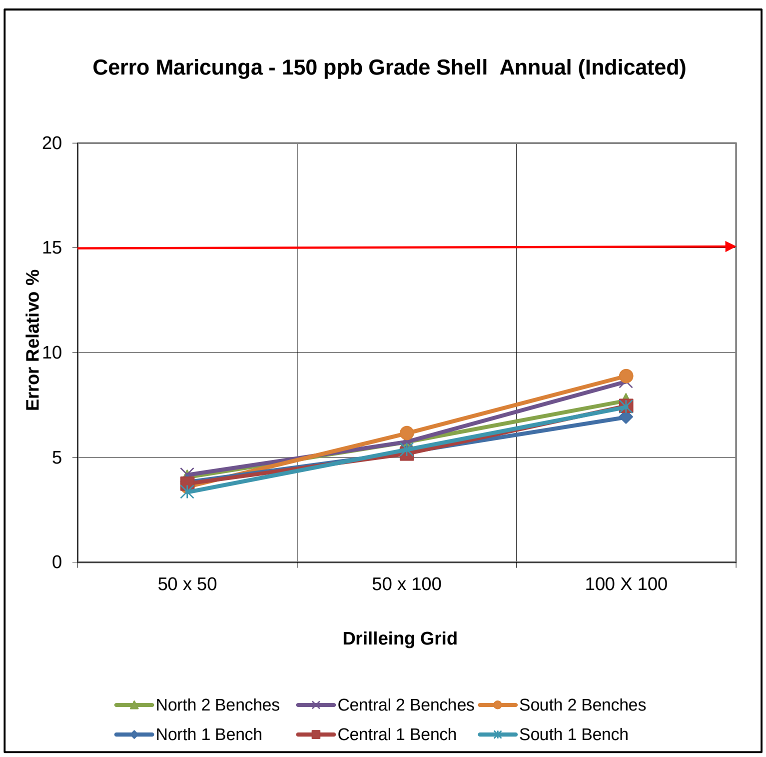


Figure 14-48 Relative Error v/s Drilling Grid – Indicated Resources

The following may be concluded based on Table 14-13:

- Drilling grids of 50 x 50 m and 50 x 100 m were sufficient to defining measured resources, however, a 100 X 100 m grid was not sufficient for this category
- A drilling grid of 100 x 100 m is sufficient for the indicated resource category.

The conclusion is that the Cerro Maricunga deposit is well drilled and needs no further infill drilling campaigns.

As previously noted, assay data used for the resource estimation were mainly drill holes but also including 266 m of quality assayed trench samples.

In addition to these data, exploration trench samples were also used for resource categorization in order to define indicated resources near surface. It should be stressed this decision was taken due to the fact that trenches were mapped and assayed and therefore were clear evidence of the extension of mineralized envelopes toward surface. The location of the trenches is shown in Figure 14-49.

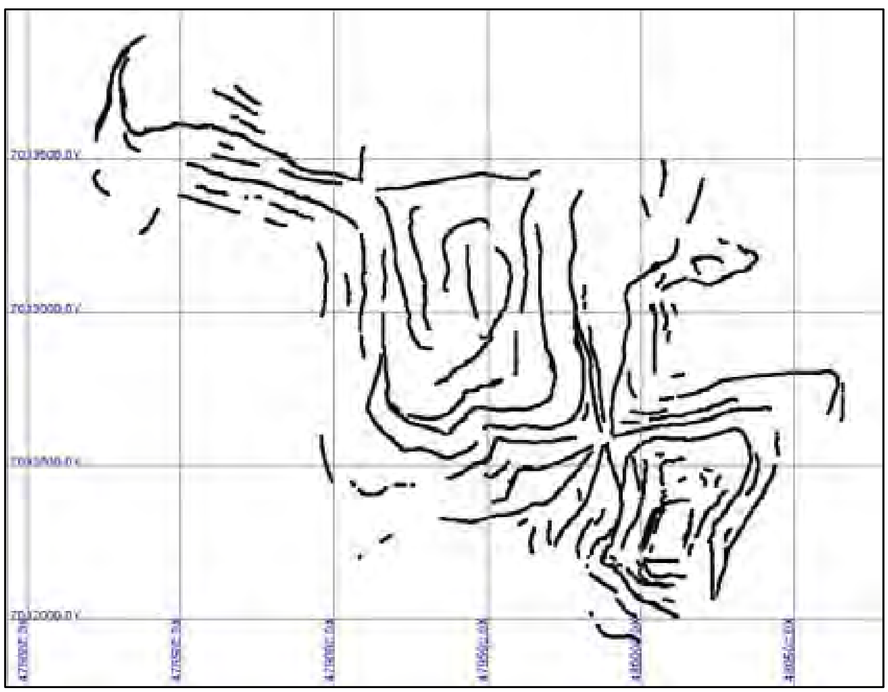


Figure 14-49 Cerro Maricunga Exploration Trenches

The following procedure was developed in order to "paint" the 10 x 10 x 10 m blocks that were estimated within a 50 x 50 m or 50 x 100 m drilling grids (measured resources) or 100 x 100 m grid (indicated resources). Plans and section showing estimated blocks and their kriging estimation variances were inspected. The highest kriging estimation variances encountered in zones drilled using approximate 50 x 50 m or 50 x 100 m grids were noted for the three mineralized zones and outside separately. The procedure was repeated considering zones drilled using approximate 100 x 100 m grids. The kriging variances determined are shown in Table 14-14.

Table 14-14 Kriging Estimation Variances for 50 x 50 and 100 x 100 grids

Category	Kriging Variance			
	Lynx	North Phoenix	South Phoenix + Crux	Outside
Measured	0.000 - 0.580	0.000 - 0.580	0.000 - 0.500	0.000 - 0.500
Indicated	0.580 - 0.725	0.580 - 0.850	0.500 - 0.750	0.500 - 0.570
Inferred	> 0.725	> 0.850	> 0.750	> 0.570

Finally, blocks estimated with kriging variances within the ranges shown in Table 14-14 were categorized accordingly. Categories assigned to blocks were smoothed by means of an algorithm in order to avoid "islands" of, for example, indicated blocks within inferred zones.

Results of the resource categorization procedure are shown graphically in the same cross sections considered in Section 14.9 – *Graphic Validation*; cross sections are shown from Figure 14-50 Figure 14-50, Figure 14-51 and Figure 14-52. Green colored blocks represent the measured category, blue colored blocks are indicated and red represent block within inferred category.

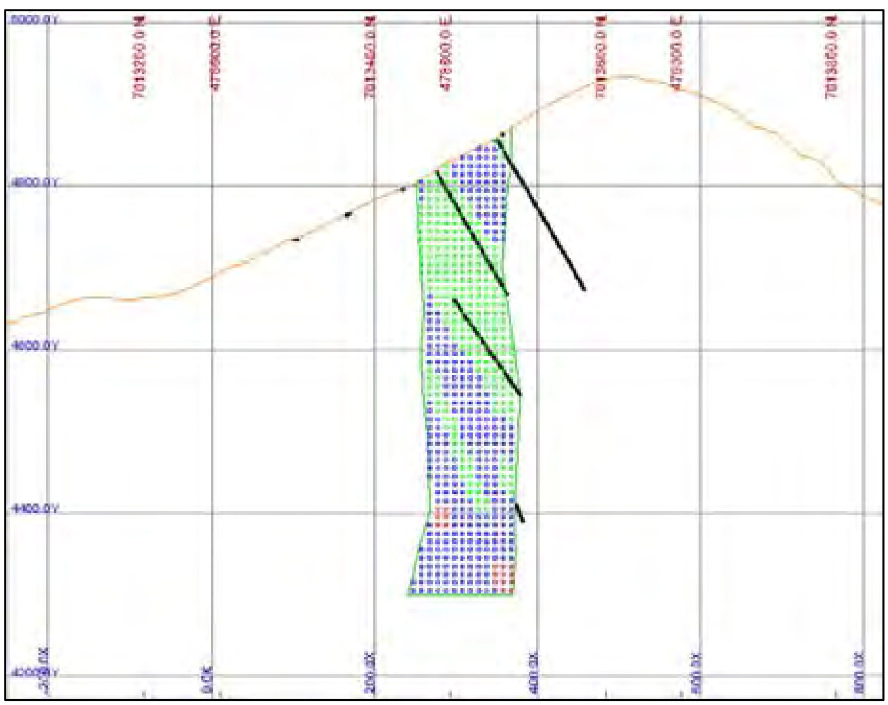


Figure 14-50 Lynx Resource Categorization Cross Section (2150)

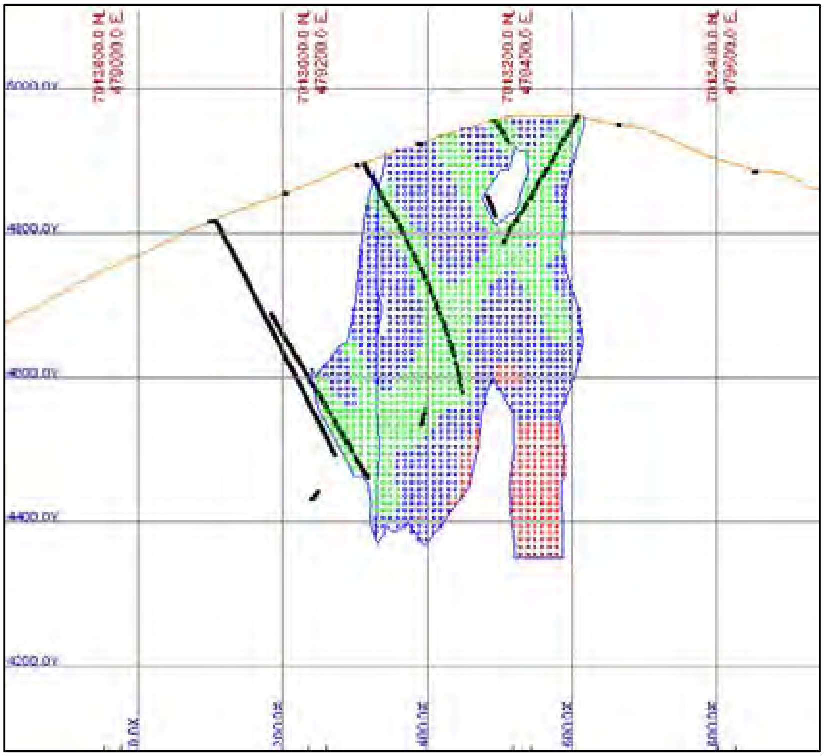


Figure 14-51   Phoenix Resource Categorization Cross Section (1550)

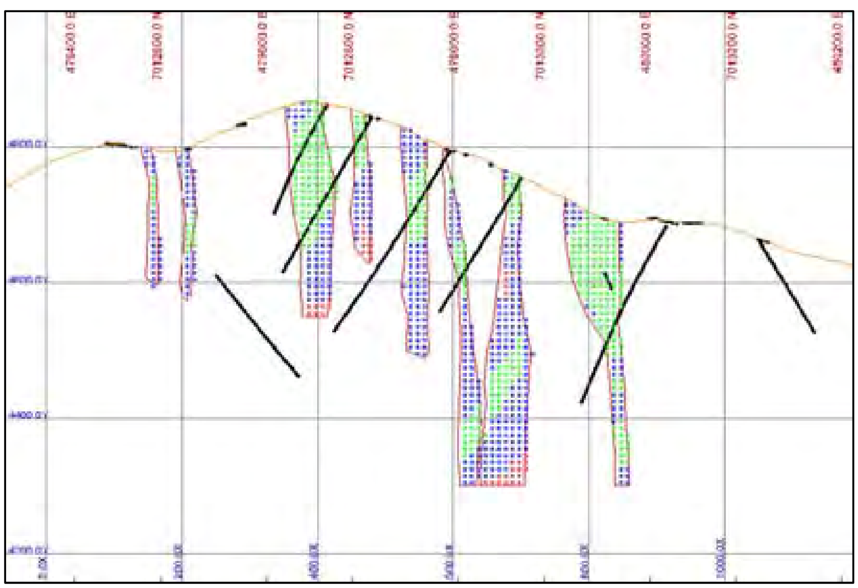


Figure 14-52   Phoenix Resource Categorization Cross Section (1050)

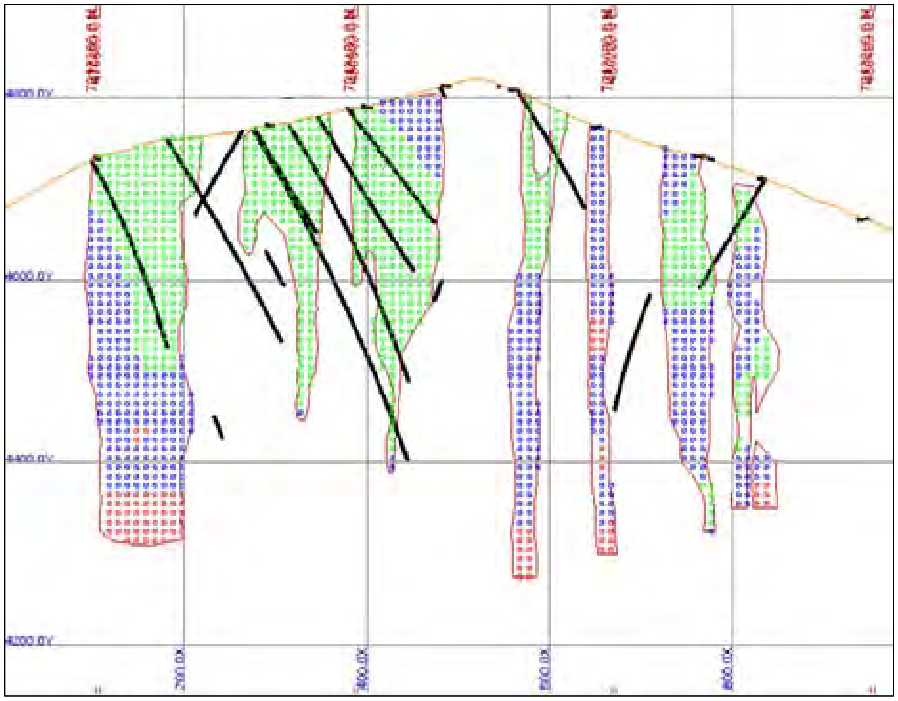


Figure 14-53 Crux Resource Categorization Cross Section (550)

14.12 Resource Tabulation

Unconstrained measured, indicated, measured plus indicated and inferred resources for Crux, Phoenix, and Lynx Outside are shown in Table 14-15. At a cut-off grade of 0.15 g/t Au, the Cerro Maricunga deposit hosts measured and indicated resources of 5.249 million ounces gold in 433.8 million tonnes grading 0.38 g/t Au with a further 0.603 million ounces of gold in the inferred category at a grade of 0.32 g/t Au.

Table 14-15 Cerro Maricunga Project – Geological Resources 2013

PHOENIX	Measured		Indicated		M + I		M + I	Inferred		Inf
Cut-Off	(Mt)	Mean Grade	(Mt)	Mean Grade	(Mt)	Mean Grade	Moz	Mean Grade	(Mt)	(Moz)
0.400	20.4	0.61	33.7	0.59	54.1	0.60	1.036	0.61	33.7	0.111
0.350	25.9	0.56	45.0	0.53	70.9	0.54	1.238	0.56	45.0	0.142
0.300	31.8	0.52	57.6	0.49	89.5	0.50	1.432	0.52	57.6	0.173
0.250	36.9	0.48	70.2	0.45	107.1	0.46	1.588	0.48	70.2	0.209
0.200	39.8	0.47	77.2	0.43	117.0	0.44	1.662	0.47	77.2	0.242
0.150	40.7	0.46	79.1	0.42	119.8	0.44	1.678	0.46	79.1	0.253
0.100	40.8	0.46	79.5	0.42	120.4	0.43	1.680	0.46	79.5	0.253
0.050	40.9	0.46	79.7	0.42	120.6	0.43	1.681	0.46	79.7	0.253
0.000	40.9	0.46	79.7	0.42	120.6	0.43	1.681	0.46	79.7	0.253
LYNX	Measured		Indicated		M + I		M + I	Inferred		Inf
Cut-Off	(Mt)	Mean Grade	(Mt)	Mean Grade	(Mt)	Mean Grade	Moz	Mean Grade	(Mt)	(Moz)
0.400	9.2	0.66	30.6	0.60	39.7	0.62	0.789	0.66	30.6	0.041
0.350	11.1	0.61	39.4	0.55	50.5	0.57	0.920	0.61	39.4	0.050
0.300	13.7	0.56	48.0	0.51	61.7	0.52	1.036	0.56	48.0	0.057
0.250	16.3	0.51	59.9	0.46	76.2	0.48	1.164	0.51	59.9	0.068
0.200	18.6	0.48	75.1	0.42	93.7	0.43	1.291	0.48	75.1	0.080
0.150	20.1	0.46	82.8	0.39	102.9	0.41	1.344	0.46	82.8	0.084
0.100	20.4	0.45	84.0	0.39	104.4	0.40	1.351	0.45	84.0	0.084
0.050	20.4	0.45	84.0	0.39	104.5	0.40	1.351	0.45	84.0	0.084
0.000	20.4	0.45	84.0	0.39	104.5	0.40	1.351	0.45	84.0	0.084
TOTAL	Measured		Indicated		M + I		M + I	Inferred		Inf
Cut-Off	(Mt)	Mean Grade	(Mt)	Mean Grade	(Mt)	Mean Grade	Moz	Mean Grade	(Mt)	(Moz)
0.400	52.1	0.61	84.3	0.59	136.4	0.60	2.617	0.61	84.3	0.191
0.350	69.1	0.55	115.6	0.53	184.6	0.54	3.196	0.55	115.6	0.274
0.300	90.6	0.50	153.7	0.48	244.3	0.49	3.818	0.50	153.7	0.359
0.250	115.8	0.45	202.2	0.43	318.0	0.44	4.468	0.45	202.2	0.477
0.200	139.7	0.41	253.5	0.39	393.1	0.40	5.014	0.41	253.5	0.572
0.150	152.8	0.39	281.0	0.37	433.8	0.38	5.249	0.39	281.0	0.603
0.100	154.3	0.39	283.7	0.37	438.0	0.37	5.268	0.39	283.7	0.606
0.050	154.4	0.39	284.0	0.37	438.4	0.37	5.269	0.39	284.0	0.606
0.000	154.4	0.39	284.0	0.37	438.4	0.37	5.269	0.39	284.0	0.606

- Mineral resources are reported as global unconstrained resources
- Mineral resources are not confined within a pit using mining parameters
- Rounding may result in apparent summation differences between tonnes, grade and contained gold ounces
- Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

15 MINERAL RESERVES ESTIMATE

15.1 Summary

A mine plan was developed for the Cerro Maricunga Oxide Gold Project to process 80,000 tpd. The total material (ore and waste) movement required was variable during the life of mine, fluctuating between 94 million tonnes and 50 million tonnes annually during the first 8 years of production. The mine is scheduled to work seven days per week, 350 days per year (considering 15 days of weather delays). Each day will consist of two 12 hour shifts with four mining crews required to cover the operation.

The study is based on operating the Cerro Maricunga mine with 42 m^3 shovels and 290 t capacity trucks. This type of equipment can achieve the productivity required for an annual maximum total material movement of 94 million tonnes; and also to provide good mining selectivity with the shovels as required for grade control.

Using the resource estimate, NCL performed pit optimization and mine planning introducing factors to account for dilution and ore loses. Because of the importance of the head grades to the processing plant, significant grade control efforts must be made during mining to minimize sub-grade material being fed to the plant.

Mineral Reserves are defined within an open pit mine plan generated considering diluted Measured and Indicated Mineral Resources.

Mineral Resources were converted to Mineral Reserves recognizing the level of confidence in the Mineral Resource estimate and reflecting any modifying factors. The Proven Mineral Reserve is based on Measured Mineral Resources and Probable Mineral Reserve is based on Indicated Mineral Resources after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Project.

The Mineral Reserve is that part of the Mineral Resource which can be economically mined by open pit mining methods. As stated above, dilution of the Mineral Resource model and an allowance for ore loss was included in the Mineral Reserve estimate.

15.2 Block Model

NCL was provided with the February 2014 updated resource block model, developed by Magri Consultores Limitada (Magri), including the results from Atacama's Phase IV drilling campaign results (2012 - 2013). A three dimensional block model was generated to estimate grades. The selected block size was based on the geometry of the domain interpretation and the data configuration. A block size of 10 m E x 10 m N x 10 m RL was selected. Sufficient variables were included in the block model construction to enable grade estimation and reporting.

Resource estimation was undertaken using Ordinary Kriging (OK) by zone type as the principal estimation methodology for gold. Estimate for bulk density was also carried out by OK, but without any lithological or zone control.

The following are the variables contained in the data in the block model received:

- Zone code
- Bulk density (t/m^3)
- Gold grade in grams per tonne (Au g/t)
- A class code to distinguish measured, indicated, and inferred resource blocks.

Atacama personnel provided the initial topography and this was added to the model by NCL. NCL did not audit the sampling data or the block model. Mineral resources based on the models are tabulated at various cut-off grades in Geology and Mineral Resources section of the Pre-Feasibility Study.

The February 2014 block model included resources classified as measured, indicated or inferred. All the activities of pit optimization, mine design, mine planning and reserves estimate were carried out using this block model and did not include the inferred resources as part of the available resources (only measured and indicated resources can be converted into mineral reserves). Inferred resources were treated as waste.

15.3 Parameters for Mine Design

15.3.1 Base Parameters

Table 15-1 summarizes the base case economic parameters used for Lerchs-Grossman economic shells analysis and mine design.

The mining cost estimate for the pit optimization process is based on studies developed by NCL during 2012 as part of the PEA study. The estimated average project mining cost was separated into various components such as fuel, explosives, tires, parts, salaries and wages as at similar current operations in Chile. Each component was updated for 1st quarter 2014 prices and the exchange rate from Chilean Pesos to US dollars. This resulted in the mining cost estimate of approximately $1.45/t shown in Table 15-1. The metal price, processing costs, refining costs and processing recoveries were provided to NCL by Atacama or Alquimia.

Table 15-1 Pit Optimisation Parameters

Item	Units	Value
Metal Price		
Gold	$/oz	1,300
Recovery to Dore		
Gold	%	79.50%
Off-Site Costs		
Gold refining charge	$/oz	10
Operating Cost		
Average mining cost	$/tonne mined	1.45
Processing + G&A	$/t proc	3.09
Average Overall Pit Slope Angle		
Sector	North	40°
	Central	41°
	South	40°
Other		
Grade factor (1-Dilution)	%	100
Mining recovery	%	100
Discount Rate	%	8

Pit shells were generated for several gold prices; from $520/oz to $1,560/oz, using the discounted technique, applying a discount rate of 8% every 40 vertical metres (4 mining benches per year). Measured and indicated mineral resources were used as only these can be converted into reserves. Inferred mineral resources are not converted to reserves and are instead treated as waste for mine planning purposes.

15.3.2 Slope Angles

Slope angles used for pit optimisation were a result from an analysis performed on the pit design developed for the PEA study. Several measurements were made in different directions, getting values between 40° and 41° for the overall angles, taking into account ramps and geotechnical catch berms.

The total area of the pit was divided into three zones: North, Central and South, assigning overall angles for pit optimisation of 40°, 41° and 40°, respectively, as shown in Figure 15-1.

These values were later on validated with the geotechnical study carried out by Derk, a Chilean geotechnical consultancy company, based in Santiago (www.derk.cl), developed for the Pre-Feasibility Study and summarised in Section 18.

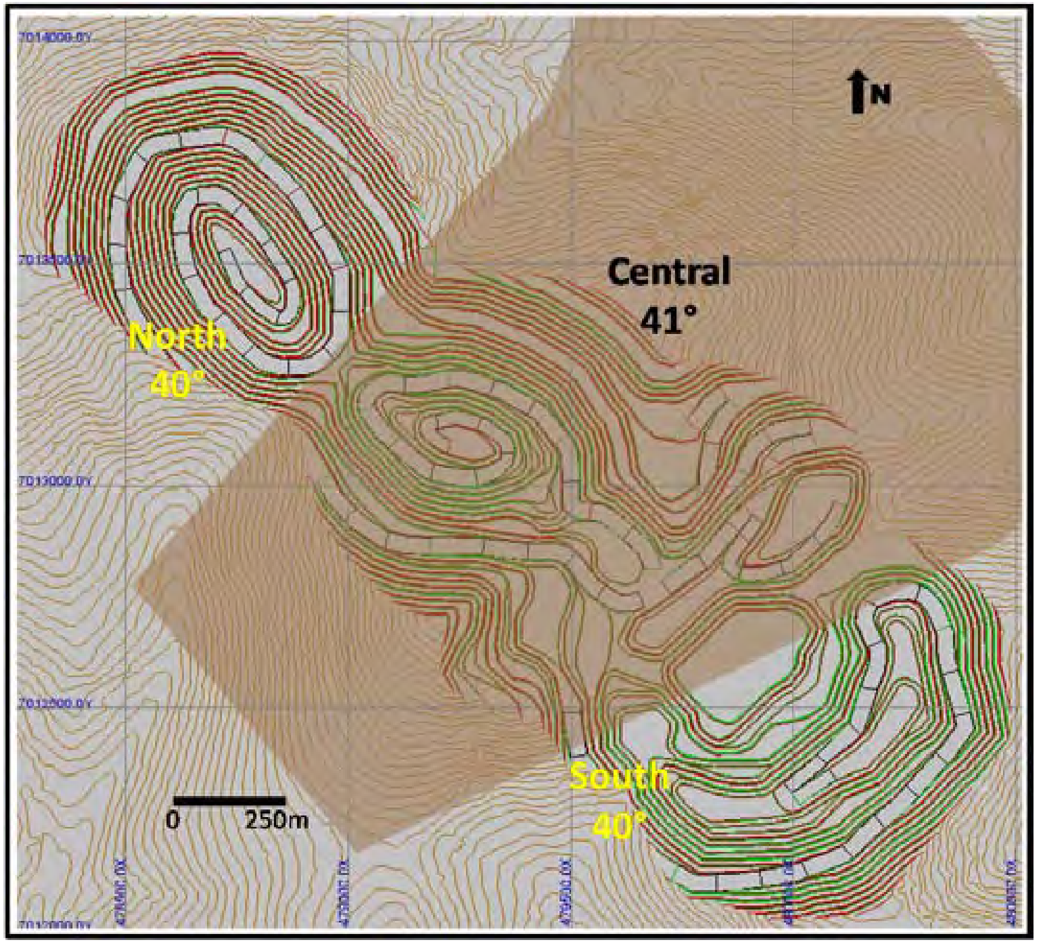


Figure 15-1 Slope Domains for Pit Optimisation

15.3.3 Dilution and Ore Losses

The selected block size of 10 m E x 10 m N x 10 m RL of the resource model is compatible with the loading mining equipment and a good selectivity is expected.

Nevertheless, a mining dilution was estimated by analysing the grade continuity of the block model. To every ore block (with a gold grade above cut-off) within the final pit, the four neighbours blocks were identified, interrogating the gold grade of each of them and accounting for those below the cut-off grade (dilution).

It was assume an over-excavation of the loading equipment of 1 m. The over excavated material below cut-off grade represents 3% of the ore tonnage. If the assumed 1 m of over-excavation is considered also as under-excavation (ore loss), the ore tonnage remains and the average grade is affected.

As a result of the analysis, a factor of 97% is applied to the in-situ grades, which represents a 3% of ore loss and 3% of dilution.

NCL notes that careful grade control will need to be practiced during mining operations to avoid sending sub-grade material to the plant, because of the important effect of head grade on gold production. These efforts should include the following standard procedures:

- Implement an intense and systematic program of sampling, mapping, laboratory analyses and reporting
- Utilize specialized in-pit, bench sampling drills for sampling well ahead of production drilling and blasting
- Use of shovels to selectively mine ore zones
- Maintain high quality laboratory staff, equipment and procedures to provide accurate and timely assay reporting
- Utilize trained geologists and technicians to work with shovel operators in identifying, marking and selectively mining and dispatching ore and waste.

15.3.4 Cut-Off Grades

Considering the base parameters shown in Table 15-1, the internal cut-off corresponds to 0.09 g/t Au. Operational cut-off grades of 0.18 g/t Au during the first three years and 0.15 g/t Au from year four onwards were used as a strategy to improve the grade of the plant feed during production.

The material with gold grades between the internal cut-off and the yearly operational cut-off was sent to the waste dump.

15.4 Mineral Reserves Statement

It is the opinion of NCL that the mine production schedule defines the mineral reserve for a mining project. Table 5.3-9 reports the mineral reserve of the Cerro Maricunga Project based on the production schedule used for the PFS study.

Mineral Reserves are summarized have an effective date of 20th August, 2014. The Qualified Person for the estimate is Mr. Carlos Guzman, Registered Member of the Chilean Mining Commission and FAusIMM, an NCL Principal and Project Director.

Table 15-2 Mineral Reserves

Category	Tonnage (kt)	Gold (g/t)	Contained Gold (koz)
Proven Reserves	126,876	0.39	1,603
Probable Reserves	167,555	0.4	2,140
Total Mineral Reserves	294,431	0.4	3,743

Comments pertinent to results shown in Mineral Reserve Table are as follow:

1. Mineral Reserves are reported as constrained within Measured and Indicated pit design, and supported by a mine plan featuring a constant throughput rate. The pit design and mine plan were optimized using the following economic and technical parameters: gold price of $1300/oz Au;

recovery to dore assumptions 79.5% for gold; $10.0/oz of Au refining charges; ore and waste average mining cost of $1.45/t, and process and G&A costs of $3.09/t processed; average pit slope angles that range from 40° to 48°; an assumption of 97% mining recovery and 3% of mining dilution.

2. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
3. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

15.5 Comment on Mineral Reserves

The opinion of the QP is that the mineral reserves for the Cerro Maricunga deposit have been prepared using industry best practices and conform to the requirements of Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), CIM Standards on Mineral Resources and Reserves - Definitions and Guidelines (2014).

The main factors that may affect the Mineral Reserve estimate are metallurgical recoveries and operting costs (fuel, energy and labour). The base price of the gold, even though the most important factor for revenue calculation, has a lower impact on the Mineral Reserve estimate because the selected Lerchs-Grossman shell used as the guide for practical mine design was obtained using the discounted technique and the mine plan considers operational cut-offs higher than the internal cut-off.

16 MINING METHODS

16.1 Pit Design

Initial pit design considerations are included in Section 15.

16.1.1 Final Pit Design

The final pit design was based on the economic shells obtained at revenue factor 1.0. Table 16-1 shows the key open pit design parameters.

The road width of 38 m will accommodate the selected 290 t trucks. NCL used a 10% road gradient which is common in the industry for this type of trucks. The current mine plan is designed with 10 m benches stacked to 20 m (i.e. double benching). Mining costs for this report are based on blasting 10 m benches for the waste zones and for the ore.

Additional 30 m wide safety berms were included in the design when the slope height exceeds 140 m, accordingly to the geotechnical recommendations.

Table 16-1 Mine Design Parameters

Parameter	Unit	Value
Haul Road Width	m	38
Haul Road Grade	%	10
Bench Height	m	10
Stacked Bench Height with 2 Benches Stacked	m	20
Nominal Minimum Mining Phase Width	m	100
Batter Angle	°	75°
Berm Width	m	9.8
Inter-ramp angle	°	53°
Security Berm Width every 140 m of Pit Wall	m	30

Figure 4 shows the final pit design. There are planned two exits on the west of the pit which give accesses to the primary crusher and to the waste storage areas.

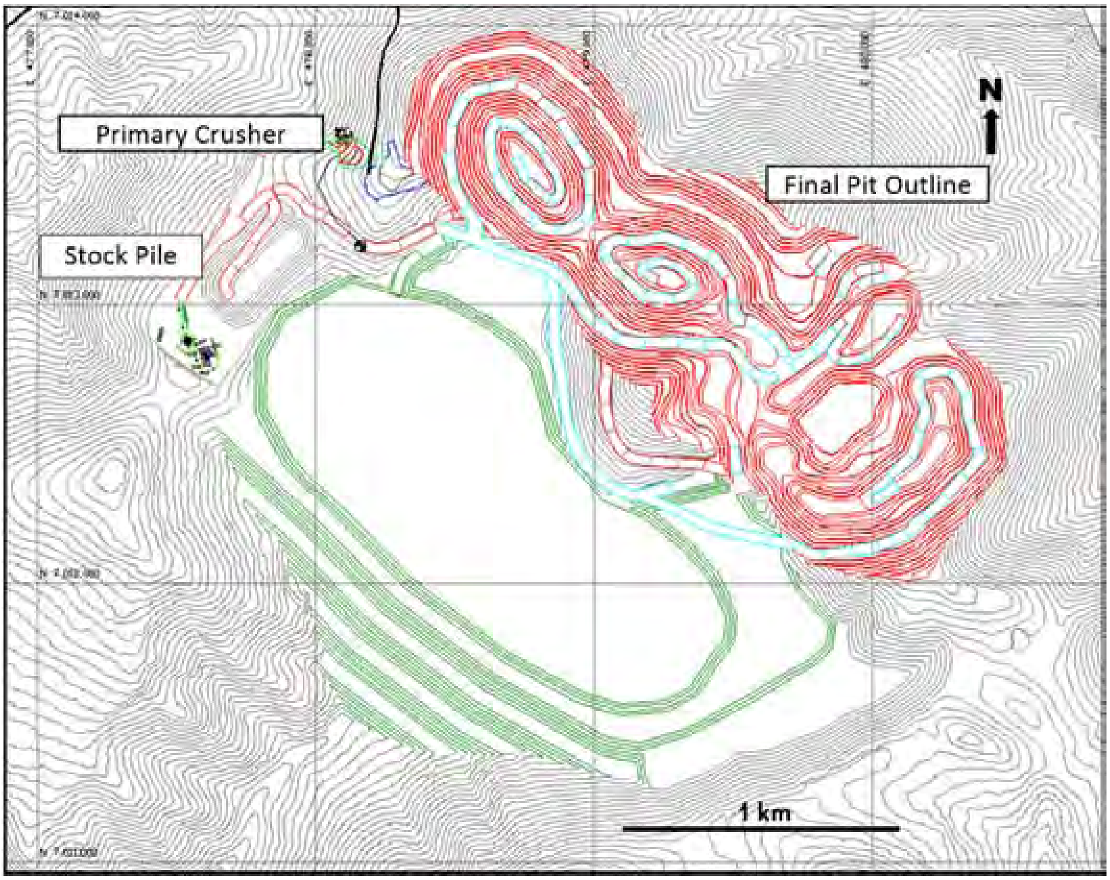


Figure 16-1 Final Pit Design

16.1.2 Mining Phases Design

NCL designed a set of seven mining phases, or pushbacks were designed by analysing the Whittle Four-X series of nested shells. Pit bottoms were selected to project them to surface, applying recommended slopes. Figure 5 shows the phases outlines on benches 4,800, 4,680, 4,630 and 4,590.

Phase 1 targets the ore with the highest grade and lowest strip ratio in the south area, down to 4,620 masl elevation. Phase 2 is in the north area, independent from Phase 1, and goes down to 4,680 masl elevation. Phase 3 is another independent pit in the central/east area, down to 4,570 masl elevation.

Phases 4 and Phase 5 correspond to expansions of Phase 2 and Phase 1, going down to 4,570 and 4,490 masl respectively. Phase 6 corresponds to an expansion of the central zone, down to 4,460 masl and finally Phase 7 is an expansion to the north, down to 4,450 masl.

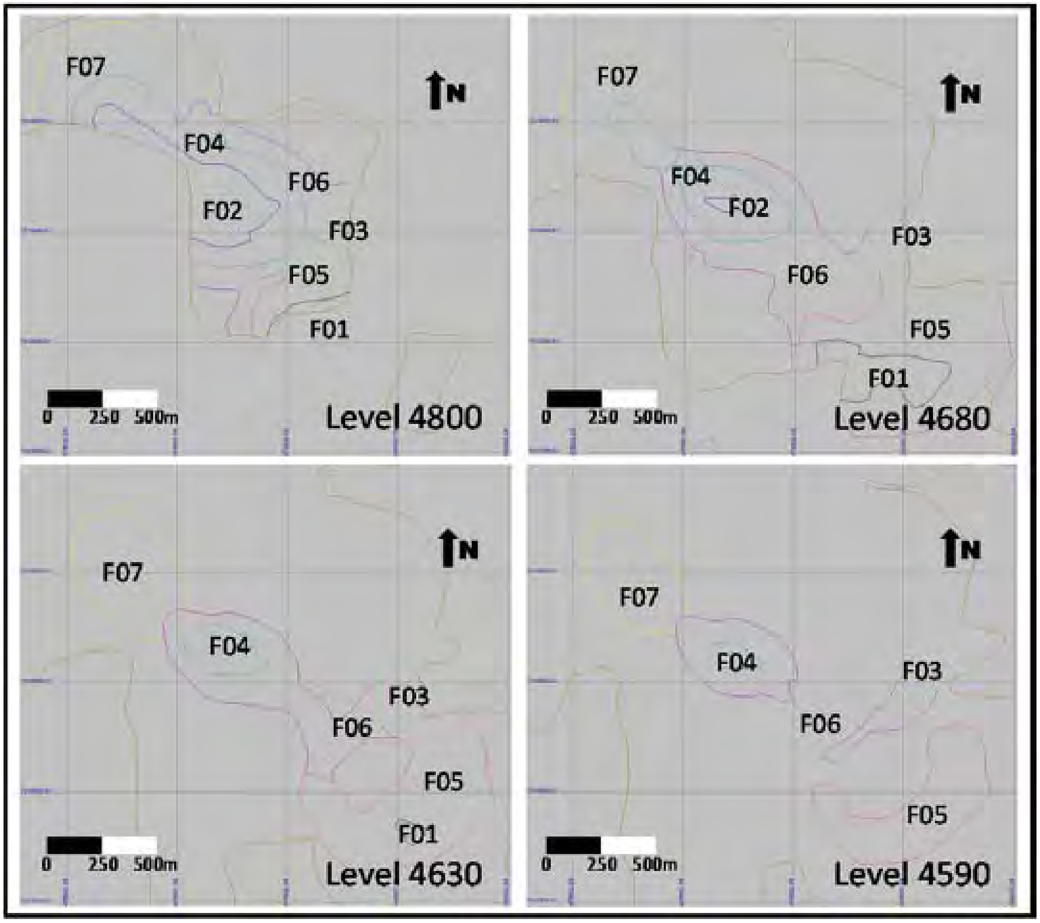


Figure 16-2 Mining Phases Design

16.1.3 Dilution and Ore Losses

The selected block size of 10 m E x 10 m N x 10 m RL of the resource model is compatible with the loading mining equipment and a good selectivity is expected.

Nevertheless, a mining dilution was estimated by analysing the grade continuity of the block model. To every ore block (with a gold grade above the cut-off grade) within the final pit, the four neighbours blocks were identified, interrogating the gold grade of each of them and accounting for those below the cut-off grade (dilution).

It was assume an over-excavation of the loading equipment of 1 m. The over excavated material below cut-off grade represents 3% of the ore tonnage. If the assumed 1 m of over-excavation is considered also as under-excavation (ore loss), the ore tonnage remains and the average grade is affected.

As a result of the analysis, a factor of 97% is applied to the in-situ grades, which represents a 3% of ore loss and 3% of dilution.

16.1.4 Tabulation of Pit Contained Resources

Table 16-2 summarizes the pit contained resources for the final design pit at several different gold cut-off grades. Table 16-3 shows the resources for the individual mining phases. The tables include only Measured and Indicated Resources using the updated February 2014 block model and the dilution criteria. Inferred Resources are considered to be waste material.

At 0.15 g/t Au cut-off, the final pit contains 297.2 Mt of ore at 0.40 g/t Au and 514.7 Mt of waste for 811.9 Mt of total material (ore + waste).

NCL does not consider Table 16-2 and Table 16-3 a statement of Mineral Reserves. It is included to provide a distribution of ore grades in the pit and various phases. The Mineral Reserve is based on the mine production schedule and is shown in Section 16.2.

Table 16-2 Final Pits at Various Gold (g/t) Cut-Offs – Including Dilution

Cut-off	Mineralized Material		Strip Ratio
(Au g/t)	(Mt)	(Au g/t)	
0.40	111.7	0.60	6.27
0.30	186.3	0.50	3.36
0.20	272.6	0.42	1.98
0.18	236.6	0.42	2.43
0.15	297.2	0.40	1.73
0.10	368.7	0.34	1.20
Total Material (Ore+Waste)		811.9	Mt

Table 16-3 Mining Phases at Various Gold (g/t) Cut-Offs – Including Dilution

Cut-off Au g/t	(Mt)	(Au g/t)	Strip Ratio	Cut-off Au g/t	(Mt)	(Au g/t)	Strip Ratio
Mining Phase 1:		Total (Mt)	48.9	Mining Phase 5:		Total (Mt)	181.3
0.40	15.8	0.65	2.09	0.40	13.9	0.52	12.09
0.30	22.0	0.57	1.22	0.30	31.2	0.43	4.82
0.20	26.7	0.51	0.83	0.20	56.1	0.35	2.23
0.18	27.2	0.50	0.80	0.18	60.0	0.34	2.02
0.15	27.6	0.50	0.77	0.15	63.5	0.33	1.86
0.10	32.2	0.44	0.52	0.10	80.7	0.28	1.25
Mining Phase 2:		Total (Mt)	84.6	Mining Phase 6:		Total (Mt)	169.5
0.40	24.9	0.57	2.40	0.40	14.4	0.56	10.74
0.30	40.0	0.48	1.11	0.30	27.0	0.46	5.27
0.20	49.6	0.44	0.71	0.20	45.1	0.38	2.76
0.18	50.3	0.43	0.68	0.18	0.3	0.37	656.12
0.15	50.9	0.43	0.66	0.15	50.6	0.35	2.35
0.10	56.7	0.40	0.49	0.10	66.3	0.30	1.56
Mining Phase 3:		Total (Mt)	33.8	Mining Phase 7:		Total (Mt)	164.8
0.40	1.5	0.53	21.70	0.40	17.8	0.68	8.25
0.30	4.4	0.40	6.75	0.30	25.9	0.57	5.36
0.20	9.3	0.32	2.65	0.20	36.5	0.48	3.51
0.18	10.0	0.31	2.37	0.18	38.8	0.46	3.25
0.15	11.0	0.29	2.07	0.15	42.0	0.44	2.93
0.10	15.1	0.25	1.24	0.10	54.5	0.36	2.03
Mining Phase 4:		Total (Mt)	128.8	Total		Total (Mt)	811.9
0.40	23.4	0.61	4.51	0.40	111.7	0.60	6.27
0.30	35.8	0.52	2.60	0.30	186.3	0.50	3.36
0.20	49.2	0.44	1.62	0.20	272.6	0.42	1.98
0.18	50.1	0.44	1.57	0.18	236.6	0.42	2.43
0.15	51.5	0.43	1.50	0.15	297.2	0.40	1.73
0.10	63.2	0.37	1.04	0.10	368.7	0.34	1.20

16.2 Mine Production Schedule

A mine production schedule was developed to show the ore tonnes, metal grades, waste material and total material by year, throughout the life of the mine. The distribution of ore and waste contained in each of the mining phases was used to develop the schedule, ensuring that criteria such as continuous ore exposure, mining accessibility, and consistent material movements were met.

NCL used an in-house developed system to evaluate several potential production mine schedules. The required annual ore tonnes and user-specified annual total material movements are provided to the algorithm, which then calculates the mine schedule. Several runs at various proposed total material movement rates were done to determine a good production schedule strategy. This program is not a simulation package, but a tool for calculation of the mine schedule and haulage profiles for a given set of phases and constraints that must be set by the user.

NCL included dilution and ore losses for mine planning purposes, which considers a constant factor of 97% of the in-situ grades.

Table 16-4 shows mine production of ore for each mining year. The schedule is based on process plant throughput of 80,000 tpd through Year 9 (29.2 Mtpy) and then reduces through to Year 13 due to lower availability of ore at the bottom of the pit. Table 16-4 also shows the total mined material movement by year, which peaks at 94 Mtpy during Year 4. The limit on the ore production is the number of benches that it is possible to mine in a year in a single phase, or vertical development per phase.

As stated in Section 15, it is the opinion of NCL that the mine production schedule defines the mineral reserve for the mining project. Table 16-5 reports the mine production schedule by mineral reserve category for the Cerro Maricunga Project.

The plant feed is shown in

Table 16-6; this indicates that Year 1 feed to the plant is made up of material mined during pre-production and Year 1.

Table 16-4 Mining Production Schedule

Period	Ore from Mine			Waste (kt)	Total Mined (kt)
	Tonnage (kt)	Gold (g/t)	Contained Gold (koz)		
PP	5,652	0.44	80	5,348	11,000
1	23,548	0.46	345	26,131	49,679
2	29,200	0.42	391	47,163	76,363
3	29,200	0.41	382	54,038	83,238
4	29,200	0.37	343	54,475	83,675
5	29,200	0.38	354	64,949	94,149
6	29,200	0.36	340	60,669	89,869
7	29,200	0.35	323	55,310	84,510
8	29,200	0.36	336	46,328	75,528
9	20,025	0.35	227	45,579	65,604
10	15,480	0.42	209	31,820	47,300
11	12,778	0.47	194	18,614	31,392
12	8,374	0.55	148	5,626	14,000
13	4,174	0.54	72	1,370	5,544
Total	294,431	0.4	3,743	517,419	811,850

Table 16-5 Mining Production Schedule by Reserve Category

Period	Mine Scedule by Reserve Category					
	Proven			Probable		
	Tonnage (kt)	Au (g/t)	(koz)	Tonnage (kt)	Au (g/t)	(koz)
PP	2,372	0.42	32	3,280	0.45	48
1	12,039	0.47	182	11,509	0.44	163
2	13,037	0.44	183	16,163	0.40	208
3	15,188	0.40	197	14,012	0.41	184
4	13,622	0.36	156	15,578	0.37	186
5	12,774	0.37	153	16,426	0.38	201
6	13,776	0.35	156	15,424	0.37	184
7	15,700	0.34	171	13,501	0.35	152
8	11,899	0.35	135	17,302	0.36	201
9	6,343	0.37	75	13,682	0.35	153
10	4,849	0.46	71	10,631	0.40	138
11	3,049	0.49	48	9,729	0.47	147
12	1,630	0.62	32	6,744	0.53	115
13	599	0.57	11	3,575	0.53	61
Total	126,876	0.39	1,603	167,555	0.40	2,140

Table 16-6 Plant Feed

Period	Plant Feed		
	Tonnage (kt)	Gold (g/t)	Contained Gold (koz)
PP			
1	29,200	0.45	424
2	29,200	0.42	391
3	29,200	0.41	382
4	29,200	0.37	343
5	29,200	0.38	354
6	29,200	0.36	340
7	29,200	0.35	323
8	29,200	0.36	336
9	20,025	0.35	227
10	15,480	0.42	209
11	12,778	0.47	194
12	8,374	0.55	148
13	4,174	0.54	72
Total	294,431	0.4	3,743

16.3 Waste Storage Area

One waste rock storage area at the west and south of the pits was designed for the life of the project. The final configuration is shown in Figure 16-1.

The geotechnical study carried out by Derk, a Chilean geotechnical consultancy company, based in Santiago (www.derk.cl), developed for the Pre-Feasibility Study and summarised in Section 18, also includes the stability analysis for the waste dumps. The geotechnical recommendation for the waste dump design is summarized in Table 16-7.

Table 16-7 Waste Dump Design Parameters

Parameter	Unit	Value
Layer Height	m	60
Batter Angle	°	37°
Berm Width	m	30
Overall angle	°	29°
Berm Width every 60 m layer	m	30

16.4 Mine Equipment

16.4.1 General

Mine equipment requirements were calculated based on the annual mine production schedule, the mine work schedule and equipment hourly production estimates. Table 16-8 summarizes the peak number of units required for pre-production and commercial production. Table 16-9 provides the fleet requirements by year during the mine life. This represents the equipment necessary to perform the following duties:

- The pre-production development required to expose ore for initial production
- Mine and transport ore to the ROM pad area
- Mine and transport waste from the pit to the waste storage areas
- Maintain all the mine work areas, in-pit haul roads and external haul roads and the waste storage areas
- Rehandle the ore (load, transport and auxiliary equipment) from the ROM pad and stockpiles to the primary crusher.

Table 16-8 Equipment Requirement

Type of Equipment	Pre-Production	Peak Requirement
FEL WA 1200	1	1
Hydraulic Shovel PC 8000	1	4
Haul Truck 930E-4SE	5	17
Diesel Drill PV 275	2	5
Support Drill	1	2
Bulldozer 1 D 375A-6R	2	3
Bulldozer 2 D475A-5E0	1	1
Wheel dozer 1 WD 600-3	1	2
Wheel dozer 2 WD 900-3A	1	1
Motor grader 1 GD 825A-2	1	3
Water Truck HD 785-7	1	2
Backhoe	1	1
Lube Truck	1	1
Support Truck	1	1
Mobile Crane 200t	1	1
Lowboy Truck CXU 613 / 100 T	1	1
Tire Handler WD 600-3	1	1
Lighting Plant MOTOR LDW 1003 GE	7	14

Table 16-9 Equipment Requirement per Year

Type of Equipment	PP	1	2	3	4	5	6	7	8	9	10	11	12	13
FEL WA 1200	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic Shovel PC 8000	1	2	3	3	3	4	4	4	3	3	2	2	1	1
Haul Truck 930E-4SE	5	11	15	17	17	17	17	17	17	13	8	6	3	2
Diesel Drill PV 275	2	3	4	5	5	5	5	5	4	4	3	2	1	1
Support Drill	1	1	2	2	2	2	2	2	2	2	1	1	1	1
Bulldozer 1 D 375A-6R	2	3	3	3	3	3	3	3	3	3	3	3	1	1
Bulldozer 2 D475A-5E0	1	1	1	1	1	1	1	1	1	1	1	1		
Wheel dozer 1 WD 600-3	1	1	2	2	2	2	2	2	2	2	1	1	1	1
Wheel dozer 2 WD 900-3A	1	1	1	1	1	1	1	1	1	1	1	1		
Motor grader 1 GD 825A-2	1	3	3	3	3	3	3	3	3	3	3	3	1	1
Water Truck HD 785-7	1	1	2	2	2	2	2	2	2	1	1	1	1	1
Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Support Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Crane 200t	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lowboy Truck CXU 613 / 100 T	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire Handler WD 600-3	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant MOTOR LDW 1003 GE	7	9	12	13	13	14	14	14	12	12	9	8	6	6

16.4.2 Base Production and Operating Parameters

The type and quantity of mining equipment required to satisfy the mine production schedule are presented in this section.

<u>Equipment Selection Criteria</u>

The study is based on operating the mine with 20m^3 front-end-loaders, 42 m^3 capacity hydraulic excavators (shovels) and trucks with a capacity of 290 t. This type of equipment is able to develop the required productivity to achieve an annual total material movement of 94 million tonnes; and also to obtain good mining selectivity with the excavators as defined by the grade control activities.

The fleet is complemented with drilling rigs for ore and waste. Auxiliary equipment includes tracked dozers, wheel dozers, motor graders and a water truck. A small drill rig is also included for pre-splitting purposes.

The mine fleet also includes the necessary equipment to re-handle the ore from the stockpiles to the primary crusher. This operation will be carried out using a front end loader and the same 290 t trucks operating in the mine.

<u>Production Requirement</u>

The mine major equipment was selected based on the mine production schedule, 6 months of pre-production and approximately 13 years of commercial mining operations. The pre-production period will include preparing roads, preparing bench openings and pre-production stripping. The total material mined during pre-production is 11 Mt.

Re-handling ore will be required in Year 1 for material mined during pre-production to complete the plant feed requirement. Additional 10% of the plant feed on a yearly basis was assumed for re-handling.

Main Characteristics of the Rock

Table 16-10 shows the material characteristics used for the equipment productivity calculations. An average dry bank density of 2.447 t/m^3 was used for ore and 2.403 t/m^3 for waste. The density values are based on the resource block model values for the various materials tabulated from the mine production schedule. The material handling swell was estimated at 30%.

NCL assumed a moisture content of 2%, which represents the weight percentage of the wet weight of the material. The density of wet, loose material was used to calculate truck allowable payload limits.

All equipment production is reported in dry metric tonnes. This corresponds to the units of dry measurement contained in the computer model, the stated mineable resource and the mine material movements summarized in Table 16-4.

Table 16-10 Material Characteristics

Parameter	Unit	Ore	Waste
Dry Bank Density	t/m^3	2.447	2.403
Material Handling Swell	%	30%	30%
Moisture Content	%	2%	2%
Dry Loose Density	t/m^3	1.882	1.848
Wet Loose Density	t/m^3	1.921	1.886

Operating Time Definition

The mine is scheduled to work seven days per week, 350 days per year (15 days down due to weather conditions). Each day will consist of two 12 hour shifts. Four mining crews will rotate to cover the operation (2 working and 2 on time off).

Figure 16-3 shows the definitions used for equipment time allocation and calculation of the main operational indices.

<table>
<tr><td colspan="4">Total Annual Hours</td></tr>
<tr><td colspan="3">Available Hours</td><td>Scheduled / Non Scheduled Maintenance</td></tr>
<tr><td colspan="2">Operative Hours</td><td>Scheduled Losses Reserve</td><td></td></tr>
<tr><td>Effective Hours</td><td>Operational Losses</td><td></td><td></td></tr>
<tr><td colspan="4">Availability (%) = Available Hours / Total Annual Hours</td></tr>
<tr><td colspan="4">Use of Availability (%) = Operative Hours / Available Hours</td></tr>
<tr><td colspan="4">Operational Losses (%) = Effective Hours / Operative Hours</td></tr>
</table>

Figure 16-3 Operating Time and Indices Definition

The following definitions apply for the operating time calculation:

1. Maintenance time: Applies when the equipment is:
    a. In maintenance.
    b. Waiting for maintenance personnel.
    c. Waiting for maintenance equipment or spare parts.
    d. Travel time to workshops.
    e. Waiting time within the workshops.
2. Scheduled losses: Time allocated for:
    a. Meal breaks.
    b. Shift change.
    c. Blasting.
    d. Refuelling.
    e. Meetings.
    f. Weather conditions.
3. Reserve: The equipment is available but has not been allocated to a face
4. Operational losses: Time allocated for:
    a. Working face preparation.
    b. Training.
    c. Accidents/incidents.
    d. Equipment movement.
    e. Equipment inspection.

The general concept is that operational hours correspond to all the time when the odometer is working. It is a management issue to avoid scheduled loss or reserve going to operational losses. The operational indices above have been estimated for all major units of equipment based on NCL's experience and experience at similar operations.

A job efficiency factor (operational losses) of 83.3%, to allow for operational losses, was used to estimate all major units of equipment and productivities; this corresponds to 50 minutes per operating hour. A job efficiency of 85% was used for the haul trucks.

Table 16-11 summarizes the use of time based on the criteria adopted.

Table 16-11 Summary of Operating Time and Utilization Factors

Time Factor	Unit	Hydraulic Shovel	FEL	Haul Truck	Drill
Calendar time	d/y	365	365	365	365
Scheduled shutdown	d/y	0	0	0	0
Unscheduled days down	d/y	15	15	15	15
Mine work days	d/y	350	350	350	350
Shift per day	shift/d	2	2	2	2
Hours per shift	h/shift	12	12	12	12
Calendar time	h/y	8,760	8,760	8,760	8,760
Availability	%	85%	85%	85%	85%
Available time	h/y	7,439	7,404	7,468	7,483
Standby					
Lunch	min/shift	60	60	46	60
Shift start-up / Instructions	min/shift	15	15	15	15
To the pit	min/shift	5	5	5	5
From the pit	min/shift	5	5	0	5
Operator change	min/shift	1	1	1	1
Equipment inspection	min/shift	5	5	5	5
Blasting	min/shift	8	8	0	15
Diesel/Lube	min/shift	9	9	9	9
Face change	min/shift	10	16	0	49
Security talks	min/shift	5	5	5	5
Total Internal Standby	min/shift	123	129	85.5	169
Total Internal Standby	h/shift	2.05	2.15	1.43	2.82
Total internal standby	h/y	1,435	1,505	998	1,972
Weather	d/y	15	15	2	2
Total external standby	d/y	15	15	2	2
Total external standby	h/y	360	360	12	48
Total Standby	h/y	1,795	1,865	1,010	2,020
Gross Operating Hours	h/y	5,644	5,539	6,458	5,463
Efficiency	%	83%	83%	85%	83%
Net Operating Hours	h/y	4,685	4,597	5,490	4,534
Utilization %	%	75.9%	74.8%	86.5%	73.0%
Utilization hours/day	h/d	18.2	18.0	20.8	17.5

16.4.3 Equipment Operating Requirement

Drilling

The drilling equipment will consist of diesel units capable of drilling 10 ⅝” diameter holes for ore and waste. Additionally, support units capable of drilling 6½” diameter holes for pre-splitting are included.

A general design for the drilling and blasting patterns has been carried out, using NCL experience on similar projects. Table 16-12 shows the drilling parameters.

As a result of these calculations, daily production capacity has been estimated for each period of the mine plan; according to the required tonnages, the number of units was estimated for every time period, as shown in Table 16-13. Two units will be required for the pre-production period; during commercial production the requirement gradually increases up to five units from Year 3 through Year 7. Support unit requirements are one during pre-production and two during the life of mine. From Year 10 the requirement for both type of equipment decreases due to the reduced material mined.

Blasting Design Parameters

According to the drill pattern specified, a blasting powder factor between 178 g/t and 368 g/t were estimated for ore and waste, as shown in Table 16-12. Both estimated values are common for fresh rock material.

Table 16-12 Drilling and Blasting Design Parameters

Pattern Code	Drill Size	10 5/8 ORE	10 5/8 WASTE	10 5/8 Buffer	6 1/2 Pre-splitting
Drilling Pattern	Unit				
Drilling Diameter	mm	269.9	269.9	269.9	165.1
Bench Height	m	10	10	10	10.5
Sub-drill	m	1	1		
Stemming	m	4	3	4	2
Burden	m	7.0	9.0	8.4	5.0
Spacing	m	8.0	10.0	9.2	3.0
Specific drilling	m^3/m	50.9	81.8	77.6	15.1
Re-drill	%	3%	3%	3%	3%
Penetration rate	m/h	30	35	30	40
BCM per hole	m^3/hole	560.0	900.0	776.2	158.8
Bit Life	m	5,800	5,800	5,800	500
Blasting Pattern	Unit				
Dry/wet	Code	Dry	Dry	Dry	Dry
Average in-situ density	t/m^3	2.43	2.43	2.43	2.43
Tonnage per hole	t/hole	1,361	2,187	1,886	386
Explosive column length	m	7.0	8.0	6.0	8.5
Specific drilling	t/m	123.7	198.8	188.6	36.7
Specific drilling inc re-drill	t/m	120.1	193.0	183.1	35.7
Tonnes per blast	kt	*105*	*105*	*33*	*10*
Holes per blast	#	77	48	17	25
Explosive Distribution					
Blasted with ANFO	%	80%	80%	80%	
Blasted with Emulsion	%	20%	20%	20%	
Blasted with Enaline	%				100%
Explosive Density					
ANFO density	t/m^3	0.85	0.85	0.85	0.85
Emulsion density	t/m^3	1.25	1.25	1.25	1.25
Enaline density	kg/m	1.45	1.45	1.45	0.83
Column Charge					
ANFO column charge	kg	340.4	389.0	291.7	
Emulsion column charge	kg	500.5	572.0	429.0	
Enaline column charge	kg				7.1
Powder Factor					
ANFO powder factor	kg/t	0.250	0.178	0.155	
Emulsion powder factor	kg/t	0.368	0.262	0.227	
Enaline powder factor	kg/t				0.018
Powder Factor					
ANFO powder factor	kg/m^3	0.61	0.43	0.38	
Emulsion powder factor	kg/m^3	0.89	0.64	0.55	
Enaline powder factor	kg/m^3				0.04
Explosive Consumption	kg/kt	273.65	194.60	169.24	18.36
ANFO	kg/kt	200.09	142.29	123.74	
Emulsion	kg/kt	73.56	52.31	45.49	
Enaline	kg/kt				18.36
Blasting Accessories Consumption					
Explosive cord	m/hole	12	12	12	13
Starter line	unit/blast	1	1	1	1
Boosters per hole	#	2	2	2	
Delays per hole	#	2	2	2	1
Holes per surface connector	#	26	16	6	9

Table 16-13 Drilling Requirement Estimate

	Unit	PP	1	2	3	4	5	6	7	8	9	10	11	12	13
Production rate (t/op h) Material															
Ore	t/op h	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,700	2,520
Waste	t/op h	4,725	4,725	4,725	4,725	4,725	4,725	4,725	4,725	4,725	4,725	4,725	4,725	5,062	4,725
BU	t/op h	3,842	3,842	3,842	3,842	3,842	3,842	3,842	3,842	3,842	3,842	3,842	3,842	4,116	3,842
Period production Material															
Ore	kt/p	5,762	14,021	14,021	14,021	14,021	14,021	13,860	13,860	13,860	13,860	13,860	13,860	14,850	13,860
Waste	kt/p	10,804	26,290	26,290	26,290	26,290	26,290	25,988	25,988	25,988	25,988	25,988	25,988	27,844	25,988
BU	kt/p	8,785	21,377	21,377	21,377	21,377	21,377	21,131	21,131	21,131	21,131	21,131	21,131	22,641	21,131
Material	kt	11,000	49,679	76,363	83,238	83,675	94,149	89,869	84,510	75,528	65,604	47,300	31,392	14,000	5,544
Ore	kt	5,409	21,680	26,940	26,546	26,706	27,331	27,311	27,066	26,118	18,192	14,297	11,924	7,699	3,496
Waste	kt	5,118	24,058	43,512	49,127	49,823	60,792	56,743	51,269	41,439	41,407	29,388	17,370	5,173	1,148
BU	kt	473	3,942	5,911	7,565	7,146	6,026	5,815	6,175	7,971	6,005	3,616	2,097	1,128	900
Number of Required Equipment	#	2	3	4	5	5	5	5	5	4	4	3	2	1	1
Ore	#	0.9	1.5	1.9	1.9	1.9	1.9	2.0	2.0	1.9	1.3	1.0	0.9	0.5	0.3
Waste	#	0.5	0.9	1.7	1.9	1.9	2.3	2.2	2.0	1.6	1.6	1.1	0.7	0.2	0.0
BU	#	0.1	0.2	0.3	0.4	0.3	0.3	0.3	0.3	0.4	0.3	0.2	0.1	0.0	0.0

Loading

The performance of the loading units has been calculated on the basis of the operational indices and detailed estimate of the times involved in the loading activity.

Table 16-14 shows the performance calculation for each unit.

Table 16-14 Loading Productivity Estimate

Item	Unit	FEL WA-1200	Hydraulic Shovel PC 8000
Bucket Capacity	yd^3	26.0	55.0
Bucket Capacity	m^3	19.9	42.1
Filling factor	%	95%	92%
Wet Load	t	36.3	73.0
Maximum truck capacity - Weight	t	290	290
Maximum truck capacity - Volume	m^3	180	268
Truck capacity	t	290	290
Required number of passes - weight	#	8.0	4.0
Required number of passes - volume	#	9.5	6.9
Number of passes	#	8	4
Time per pass	min	0.750	0.600
Loading time per truck	min	6.0	2.4
Time waiting for trucks	min	0.5	0.5
Time manoeuvring	min	0.5	0.4
Hourly instant productivity	t/hr ef	2,486	5,273
Operational Factor	%	83.3%	83.3%
Hourly effective productivity	t/hr op	2,071	4,394
Availability	%	89.0%	89.0%
Utilization	%	74.81%	75.87%
Hours per shift	hrs	12	12
Shift per day	#	2	2
Period Production (wet basis)	Mt/year	12.08	25.99

Using the above productivities, the loading equipment requirements shown in Table 16-15 have been estimated, assuming that the ore and waste material will be loaded with the shovel and the front end loader will be used for as support and re-handling the ore to the primary crusher.

Table 16-15 Loading Requirement Estimate

Mine schedule - wet		PP	1	2	3	4	5	6	7	8	9	10	11	12	13
	kt	11,224	56,461	77,921	84,937	85,383	96,070	91,703	86,235	77,069	66,943	48,265	32,032	14,286	5,657
FEL															
Material	Unit														
Ore to Crusher	kt		2,403	2,980	2,980	2,980	2,980	2,980	2,980	2,980	2,043	1,580	1,304	854	426
Ore to Stock	kt	577													
Stock to Crusher	kt		5,767												
Waste to waste dump	kt	546	2,666	4,813	5,514	5,559	6,627	6,191	5,644	4,727	4,651	3,247	1,899	574	140
Short term Stockpile	kt		2,403	2,980	2,980	2,980	2,980	2,980	2,980	2,980	2,043	1,580	1,304	854	426
Hydraulic Shovel															
Material	Unit														
Ore to Crusher	kt		21,626	26,816	26,816	26,816	26,816	26,817	26,816	26,816	18,390	14,217	11,735	7,690	3,833
Ore to Stock	kt	5,191													
Stock to Crusher	kt														
Waste to waste dump	kt	4,911	23,998	43,313	49,627	50,028	59,647	55,716	50,795	42,546	41,858	29,222	17,094	5,167	1,258
Short term Stockpile	kt														
Required loading units															
FEL		1	1	1	1	1	1	1	1	1	1	1	1	1	1
Material	Unit														
Ore to Crusher	#		0.20	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.17	0.13	0.11	0.07	0.04
Ore to Stock	#	0.11													
Stock to Crusher	#		0.47												
Waste to waste dump	#	0.11	0.22	0.40	0.46	0.47	0.55	0.52	0.48	0.40	0.39	0.27	0.16	0.05	0.01
Short term Stockpile	#		0.20	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.17	0.13	0.11	0.07	0.04
Hydraulic Shovel		1	2	3	3	3	4	4	4	3	3	2	2	1	1
Material	Unit														
Ore to Crusher	#		0.82	1.03	1.03	1.04	1.04	1.06	1.06	1.06	0.72	0.56	0.46	0.30	0.15
Ore to Stock	#	0.48													
Stock to Crusher	#														
Waste to waste dump	#	0.45	0.91	1.67	1.91	1.95	2.32	2.19	2.00	1.67	1.65	1.15	0.67	0.20	0.05
Short term Stockpile	#														

Hauling

i. Haulage Distance Calculation

Using an in-house system the haulage distances were calculated on a yearly basis for the pits, waste dumps and stockpiles, for all mining phase and for ore and waste. The distances were divided between ramp (normally at 10% gradient) and horizontal transport. Table 16-16 summarizes the distances.

Table 16-16 Hauling Distances Summary

Period	Tonnage kt	Horizontal m	Uphill m	Downhill m	Total m
PP	11,000	430	243	2,003	2,676
1	57,686	480	197	1,800	2,477
2	79,283	517	174	1,957	2,647
3	86,158	760	265	1,823	2,848
4	86,595	980	323	1,714	3,017
5	97,069	861	361	1,329	2,551
6	92,789	1,326	612	822	2,760
7	87,430	1,598	903	539	3,039
8	78,448	1,164	1,381	642	3,187
9	67,607	1,164	846	600	2,610
10	48,848	1,176	783	315	2,274
11	32,669	1,119	1,119	228	2,466
12	14,837	887	1,627	138	2,652
13	5,961	783	2,016	25	2,825
Total	846,380	1,006	621	1,128	2,756

Figure 16-4 shows graphically the weighted average total hauling distance per mining period, and the percentage of the uphill distance. The total distance varies from a minimum of 1.5 km to a maximum of 6.9 km. The crests and valleys observed in the uphill percentage curve are generated by the opening of new phases where the stripping of the waste at the upper mining benches reduces the uphill haulage.

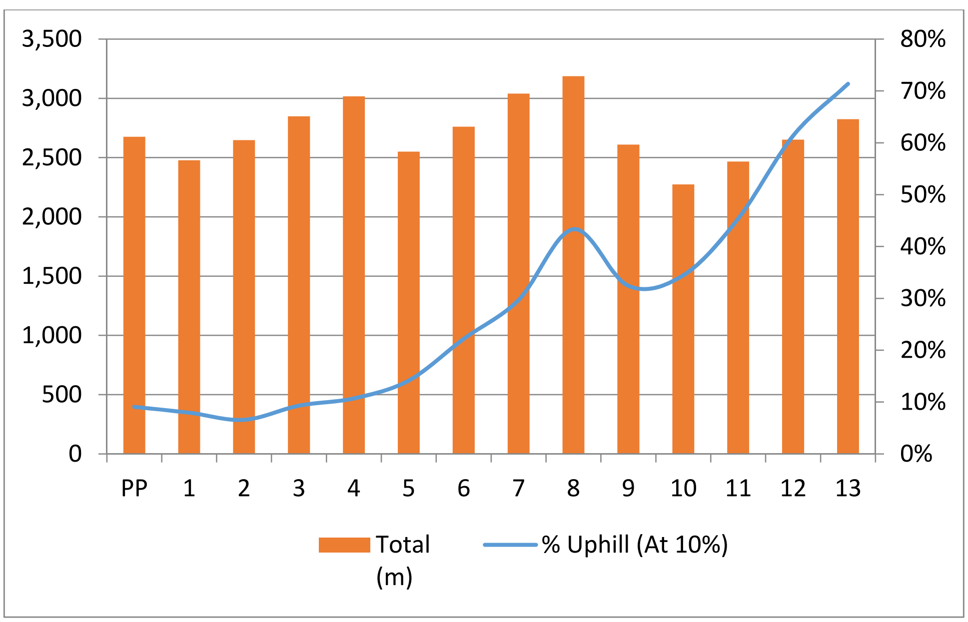


Figure 16-4 Total Hauling Distance

ii. Truck Speeds

Truck speeds were determined using typical values obtained from supplier information and similar operations. The values used are shown in Table 16-17.

Table 16-17 Average Truck Speeds

Type	Unit	Loaded	Empty
Flat in Bench	km/hr	25	25
Flat in Pit	km/hr	40	45
Up In Pit	km/hr	12	27
Down in Pit	km/hr	25	40
Flat Out Pit	km/hr	40	45
Up Out Pit	km/hr	12	27
Down Out Pit	km/hr	25	40

iii. Fixed Times in Truck Cycle

The truck cycles include fixed times for loading, dumping and queuing. The values for loading are shown in the tables for loader performance (Table 16-14). Two and a half minute has been added to every cycle for dumping and queuing.

iv. Trucks Requirement

The number of units required was obtained by dividing the annual capacity of transport of a truck for each combination and period, by the corresponding tonnage according to the defined assignment per loading unit.

Truck operating hours were calculated per period, type of material and loading unit dividing the tonnage that has to be transported by the hourly productivity of each combination. A summary of units required is presented in Table 16-18.

Operational indices considered for the trucks were:

- Availability (MA): Variable profile according to vendor and fleet life
- Use of availability (UA): 86%
- Operational efficiency: 85% (accounting for operator factor, inspection, training).

The number of required trucks during pre-production is 5. The requirement gradually increases from 11 units in Year 1 to a maximum of 17 units in Years 3 to 8, then decreases to the end of mine life as less material is mined.

Table 16-18 Summary of Total Required Truck (290 tonnes)

	Unit	PP	1	2	3	4	5	6	7	8	9	10	11	12	13
TOTAL REQUIRED TRUCKS	#	5	11	15	17	17	17	17	17	17	13	8	6	3	2
	#	4.94	10.01	14.08	15.97	16.58	16.71	16.48	16.74	16.92	12.03	7.83	5.78	2.95	1.29
Trucks loaded by FEL															
Material	Unit	0.58	1.65	1.65	1.86	1.93	1.98	1.94	1.94	1.93	1.41	0.94	0.68	0.34	0.15
Ore to Crusher	#		0.50	0.54	0.54	0.57	0.49	0.57	0.69	0.75	0.45	0.32	0.25	0.19	0.10
Ore to Stock	#	0.32													
Stock to Crusher	#		0.54												
Waste to waste dump	#	0.26	0.60	1.11	1.32	1.36	1.49	1.36	1.26	1.18	0.97	0.62	0.43	0.15	0.04
Short term Stockpile	#		0.23	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.19	0.15	0.12	0.08	0.04
Trucks loaded by Hydraulic Shovel PC 8000															
Material	Unit	4.35	8.37	12.43	14.11	14.65	14.73	14.54	14.79	14.99	10.62	6.90	5.10	2.61	1.14
Ore to Crusher	#		3.82	4.04	4.03	4.27	3.55	4.31	5.34	5.91	3.45	2.39	1.86	1.44	0.82
Ore to Stock	#	2.43													
Stock to Crusher	#														
Waste to waste dump	#	1.92	4.55	8.39	10.07	10.38	11.18	10.23	9.46	9.08	7.17	4.50	3.24	1.17	0.32
Short term Stockpile	#														

Auxiliary Equipment

Major auxiliary equipment refers to the major mine equipment that is not directly responsible for production, but which is scheduled on a regular basis. Equipment operating requirements, operating hours and personnel requirements were estimated for this equipment. The primary function of the auxiliary equipment is to support the major production units, and provide safe and clean working areas. Equipment types included in the auxiliary mine fleet are:

- Komatsu D375A-6R Track Dozer (525 HP)
- Komatsu D475A-5E0 Track Dozer (860 HP)
- Komatsu WD600-3 Wheel Dozer (485 HP)
- Komatsu WD900-3A Wheel Dozer (853 HP)
- Komatsu GD825A-2 Grader (280 HP)
- Komatsu Water Truck HD 785-7 (85 m^3)
- Sandvik DR560 Support Drill (6½").

The primary duties assigned to the auxiliary equipment are as follows:

- Mine development including access roads, drop cuts, temporary service ramps, safety berms
- Waste rock storage area control; this includes maintaining access to the dumping areas and maintaining the travel surfaces
- Ore stockpile storage area control; this includes maintaining access to the stockpile areas and maintaining the travel surfaces
- Maintenance and clean-up in the mine and waste storage areas

- Drilling for pre-splitting.

The values in Table 16-19 were used to estimate the number of required operating hours for the mine auxiliary equipment.

Table 16-20 shows the assignment criteria for the different types of auxiliary equipment. For example, the 525 HP track dozer is assigned as one unit for every two units of loading equipment, plus one base units. For Year 1 this translates to 1.5 units (3 loading units), plus 1 base units, equals 3 track dozers.

Table 16-20 shows the estimated quantities for the major auxiliary equipment. In general, four track dozers, three wheel dozers, three motor-graders and two water trucks are required.

Table 16-19 Operating Hours per Type of Auxiliary Equipment

Equipment	Unit	Value
Bulldozer 1 D 375A-6R		
Hours per shift	h/sft	12
Shifts per day	sft/d	2
Availability	%	83%
Utilization	%	60%
Period capacity	h/p	4,362
Bulldozer 2 D475A-5E0		
Hours per shift	h/sft	12
Shifts per day	sft/d	2
Availability	%	83%
Utilization	%	60%
Period capacity	h/p	4,362
Wheeldozer 1 WD 600-3		
Hours per shift	h/sft	12
Shifts per day	sft/d	2
Availability	%	83%
Utilization	%	60%
Period capacity	h/p	4,362
Wheeldozer 2 WD 900-3A		
Hours per shift	h/sft	12
Shifts per day	sft/d	2
Availability	%	83%
Utilization	%	60%
Period capacity	h/p	4,362
Motorgrader 1 GD 825A-2		
Hours per shift	h/sft	12
Shifts per day	sft/d	2
Availability	%	83%
Utilization	%	65%
Period capacity	h/p	4,726
Water Truck HD 785-7		
Hours per shift	h/sft	12
Shifts per day	sft/d	2
Availability	%	83%
Utilization	%	65%
Period capacity	h/p	4,726

Table 16-20 Major Auxiliary Equipment Requirement

Ancillary		PP	1	2	3	4	5	6	7	8	9	10	11	12	13
	Total fleet	7	12	14	14	14	14	14	14	13	12	11	11	5	4
Bulldozer 1 D 375A-6R	1 every 2 loading equipment	2	3	3	3	3	3	3	3	3	3	3	3	1	1
Base requirement		1	1	1	1	1	1	1	1	1	1	1	1		
Loading units per Bulldozer 1		2	2	2	3	3	3	3	3	3	2	2	2	2	2
Total required units		2.0	2.5	3.0	2.3	2.3	2.7	2.7	2.7	2.3	3.0	2.5	2.5	1.0	1.0
Bulldozer 2 D475A-5E0	1 every 6 loading equipment	1	1	1	1	1	1	1	1	1	1	1	1		
Base requirement															
Loading units per Bulldozer 2		6	6	6	6	6	6	6	6	6	6	6	6		
Total required units		0.3	0.5	0.7	0.7	0.7	0.8	0.8	0.8	0.7	0.7	0.5	0.5		
Wheeldozer 1 WD 600-3	1 every 2 loading equipment	1	1	2	2	2	2	2	2	2	2	1	1	1	1
Base requirement															
Loading units per Wheeldozer 1		3	3	3	3	3	3	3	3	3	3	3	3	3	3
Total required units		0.7	1.0	1.3	1.3	1.3	1.7	1.7	1.7	1.3	1.3	1.0	1.0	0.7	0.7
Wheeldozer 2 WD 900-3A	1 every 6 loading equipment	1	1	1	1	1	1	1	1	1	1	1	1		
Base requirement															
Loading units per Wheeldozer 2		6	6	6	6	6	6	6	6	6	6	6	6		
Total required units		0.3	0.5	0.7	0.7	0.7	0.8	0.8	0.8	0.7	0.7	0.5	0.5		
Motorgrader 1 GD 825A-2	As a function of the lengths roads	1	3	3	3	3	3	3	3	3	3	3	3	1	1
Base requirement															
Required units Motorgrader 1		1	3	3	3	3	3	3	3	3	3	3	3	1	1
Total required units		1.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	1.0	1.0
Water Truck HD 785-7	As a function of the lengths roads	1	1	2	2	2	2	2	2	2	1	1	1	1	1
Base requirement		1	1	1	1	1	1	1	1	1	1	1	1		
Required units Water Truck		1	1	2	2	2	2	2	2	2	1	1	1	1	1
Total required units		1.0	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0	1.0	1.0	1.0	1.0

Support Equipment

Additional equipment to support mining activities was estimated. The estimation is detailed in Table 16-21.

Table 16-21 Support Equipment Requirement

Type of Equipment	No. of Units
Backhoe	1
Lube Truck	1
Support Truck	1
Mobile Crane 200 t	1
Lowboy Truck CXU 613 / 100 t	1
Tire Handler WD 600-3	1
Lighting Plant MOTOR LDW 1003 GE	14 (max)

An estimate for the operating hours requirement for the support equipment was developed, based on the operating criteria in Table 16-22.

Table 16-22 Support Equipment Operating Hours Criteria

Support Equipment	Hours per shift hr/sft	Shifts per day sft/d	Availability %	Utilization %	Period capacity hr/period
Backhoe	12	2	83.0%	60.0%	4,362
Lube Truck	12	2	83.0%	60.0%	4,362
Support Truck	12	2	83.0%	50.0%	3,635
Mobile Crane 200t	12	2	83.0%	40.0%	2,908
Lowboy Truck CXU 613 / 100 T	12	2	83.0%	30.0%	2,181
Tire Handler WD 600-3	12	2	83.0%	40.0%	2,908
Lightning Plant MOTOR LDW 1003 GE	12	2	83.0%	20.0%	1,454

Total Mine Fleet Requirement

The total main equipment requirements for the project are summarized in Table 16-23 for every period of the plan. Table 16-24 shows the operating hours requirement for all the pieces of equipment; this is the basis for the operating cost estimate.

Table 16-23 Total Mining Equipment Requirement

Type of Equipment	PP	1	2	3	4	5	6	7	8	9	10	11	12	13
FEL WA 1200	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic Shovel PC 8000	1	2	3	3	3	4	4	4	3	3	2	2	1	1
Haul Truck 930E-4SE	5	11	15	17	17	17	17	17	17	13	8	6	3	2
Diesel Drill PV 275	2	3	4	5	5	5	5	5	4	4	3	2	1	1
Support Drill	1	1	2	2	2	2	2	2	2	2	1	1	1	1
Bulldozer 1 D 375A-6R	2	3	3	3	3	3	3	3	3	3	3	3	1	1
Bulldozer 2 D475A-5E0	1	1	1	1	1	1	1	1	1	1	1	1		
Wheel dozer 1 WD 600-3	1	1	2	2	2	2	2	2	2	2	1	1	1	1
Wheel dozer 2 WD 900-3A	1	1	1	1	1	1	1	1	1	1	1	1		
Motor grader 1 GD 825A-2	1	3	3	3	3	3	3	3	3	3	3	3	1	1
Water Truck HD 785-7	1	1	2	2	2	2	2	2	2	1	1	1	1	1
Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Support Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Crane 200t	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lowboy Truck CXU 613 / 100 T	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire Handler WD 600-3	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant MOTOR LDW 1003 GE	7	9	12	13	13	14	14	14	12	12	9	8	6	6

Table 16-24 Mine Equipment Operating Hours Requirement Summary

Mine Equipment	PP	1	2	3	4	5	6	7	8	9	10	11	12	13
FEL WA-1200	542	6,392	5,200	5,539	5,560	6,076	5,865	5,601	5,159	4,218	3,093	2,176	1,102	479
Hydraulic Shovel PC 8000	2,299	10,383	15,960	17,397	17,489	19,678	18,783	17,663	15,786	13,712	9,886	6,561	2,926	1,159
Haul Truck 930E-4SE	13,522	68,257	95,714	107,928	111,430	112,314	111,279	113,016	114,229	81,206	53,006	39,142	20,104	8,798
Diesel Drill DR 460	3,353	14,721	21,438	22,901	23,003	25,281	24,361	23,199	21,210	17,546	12,834	8,954	4,147	1,865
Support Drill	426	2,427	3,690	4,335	4,230	4,198	4,025	3,982	4,242	3,428	2,273	1,424	689	399
Bulldozer 1 D 375A-6R	3,586	10,906	13,087	10,179	10,179	11,633	11,633	11,633	10,179	13,087	10,906	10,906	4,362	4,362
Bulldozer 2 D475A-5E0	598	2,181	2,908	2,908	2,908	3,635	3,635	3,635	2,908	2,908	2,181	2,181	0	0
Wheeldozer 1 WD 600-3	1,195	4,362	5,817	5,817	5,817	7,271	7,271	7,271	5,817	5,817	4,362	4,362	2,908	2,908
Wheeldozer 2 WD 900-3A	598	2,181	2,908	2,908	2,908	3,635	3,635	3,635	2,908	2,908	2,181	2,181	0	0
Motorgrader 1 GD 825A-2	1,942	14,178	14,178	14,178	14,178	14,178	14,178	14,178	14,178	14,178	14,178	14,178	4,726	4,726
Water Truck HD 785-7	1,942	4,726	9,452	9,452	9,452	9,452	9,452	9,452	9,452	4,726	4,726	4,726	4,726	4,726
Backhoe	1,793	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362
Lube Truck	1,793	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362	4,362
Support Truck	1,494	3,635	3,635	3,635	3,635	3,635	3,635	3,635	3,635	3,635	3,635	3,635	3,635	3,635
Mobile Crane 200t	1,195	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908
Lowboy Truck CXU 613 / 100 T	896	2,181	2,181	2,181	2,181	2,181	2,181	2,181	2,181	2,181	2,181	2,181	2,181	2,181
Tire Handler WD 600-3	1,195	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908	2,908
Lightning Plant MOTOR LDW	4,183	13,087	17,450	18,904	18,904	20,358	20,358	20,358	17,450	17,450	13,087	11,633	8,725	8,725

16.5 Mine Personnel

16.5.1 General

Mine personnel includes all the salaried supervisory and other staff working in mine operations, maintenance and engineering/geology departments, and the hourly paid employees required to operate and maintain the drilling, blasting, loading, hauling and mine support activities.

16.5.2 *Salaried Staff*

Mine salaried staff requirements over the project life are shown in Table 16-25. The staff consists of 53 during pre-production and 60 during commercial production. Of the 60 persons assigned for Years 1 through 11, 13 are in mine operations, 19 in mine maintenance and 28 in technical services.

Annual costs for the personnel, including fringe benefits, are also shown on Table 16-25. The personnel costs used for the project were provided by Atacama and were developed from costs obtained from benchmarking of other Chilean mining operations.

Table 16-25 Salaried Staff Labour Requirements

	$/year	PP	1	2	3	4	5	6	7	8	9	10	11	12	13
Mine Operations Overhead		**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**12**
Mine Manager	200,004	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Analyst	60,000	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Operations Superintendent	80,004	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shift Boss	60,000	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Dispatch Engineer	60,000	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dispatch Operator	30,000	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Trainer	60,000	1	1	1	1	1	1	1	1	1	1	1	1	1	
Mine Maintenance Overhead		**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**19**	**13**
Mine Maintenance Superintendent	80,004	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant	30,000	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Workshop boss	60,000	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Chief Maintenance Planning	60,000	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planning Engineer	60,000	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Maintenance Supervisor (workshop)	60,000	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Maintenance Field Chief	60,000	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Programmers	60,000	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Maintenance Shift Boss	60,000	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Technical Services		**21**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**21**	**19**
Engineering Superintendent	80,004	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planning Engineer (Long Term / Reserves)	60,000	1	2	2	2	2	2	2	2	2	2	2	2	1	1
Geologist (Long Term / Reserves)	60,000	1	2	2	2	2	2	2	2	2	2	2	2	1	1
Mine Planning Assistant (Long Term / Reserves)	60,000	1	2	2	2	2	2	2	2	2	2	2	2	1	1
Mine Planning Engineer (Short term)	60,000	1	2	2	2	2	2	2	2	2	2	2	2	2	
Surveyor	60,000	1	2	2	2	2	2	2	2	2	2	2	2	1	1
Surveyor Assistant	30,000	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Mine Geologist (Short Term)	60,000	1	2	2	2	2	2	2	2	2	2	2	2	1	1
Geologist helper	30,000	1	2	2	2	2	2	2	2	2	2	2	2	1	1
Core Shed Chief	60,000	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Samplers Chief	60,000	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Samplers	30,000	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Total Salaried Staff		**53**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**53**	**44**

16.5.3 *Hourly Labour*

a) General

Mine total hourly personnel requirements are shown in Table 16-26. The required number of personnel is 169 during pre-production. The maximum number of persons during commercial production is 356 in Years 5 and 6. Table 5.5-2 also shows the annual cost for hourly personnel, including fringe benefits. The hourly personnel costs were provided by Atacama and were developed from costs obtained from benchmarking of other Chilean mining operations.

b) Mine Operations

The majority of personnel in mine operations are equipment operators. The number of operators for major equipment was calculated as part of the equipment operating requirements above. The fleet size, the number of crews and the labour factor are used to calculate the number of operators as follows:

Operators = (Total Fleet) x (Crews) x (Labour Factor)

The labour factor (LF) was assumed to be 1.13 for this study and was provided to NCL by Atacama. This factor adds operators to account for vacation, sick leave and absenteeism. NCL rounds up to the nearest whole number if a fractional number of operators is indicated by the calculation.

There is no allowance for blasting personnel in the estimate. Blasting will be performed by a contractor and the personnel costs are included in the price of the service.

c) Mine Maintenance

Table 16-26 shows the number of maintenance personnel required for each time period. The ratio of maintenance personnel to operating personnel ranges from 72% to 79% during commercial production.

Table 16-26 Mine Hourly Labour Requirement

	$/year	PP	1	2	3	4	5	6	7	8	9	10	11	12	13
Direct Manpower		102	152	190	200	196	200	200	199	191	170	130	118	69	64
FEL WA 1200	39,996	3	3	3	3	3	3	4	4	4	4	4	4	4	4
Electric Rope Shovel	39,996														
Hydraulic Shovel PC 8000	39,996	6	11	15	15	15	19	18	18	14	14	9	9	5	5
Haul trucks	39,996	23	49	66	74	73	74	74	74	73	58	36	28	14	10
Electric Drill	39,996														
Diesel Drill PV 275	39,996	10	14	18	21	21	20	21	21	17	17	13	9	5	5
Support Drill	39,996	3	4	6	7	6	7	6	6	7	7	3	4	3	3
Bulldozer 1 D375	39,996	13	18	21	20	20	20	20	20	20	18	18	17	7	6
Bulldozer 2 D475	39,996	5	5	5	5	4	4	4	4	4	4	4	4		
Wheeldozer 1 WD600	39,996	5	5	9	9	9	8	8	8	8	8	4	4	4	4
Wheeldozer 2 WD900	39,996	5	5	5	5	5	5	5	4	4	4	4	4		
Motorgrader 1 GD825	39,996	5	13	13	12	12	12	12	12	12	12	12	12	4	4
Motorgrader 2	39,996														
Water Truck HD785	39,996	5	5	9	9	8	8	8	8	8	4	4	4	4	4
Backhoe	39,996	3	4	4	4	4	4	4	4	4	4	3	3	3	3
Fuel Truck 85 m3	39,996														
Lube Truck	39,996	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Support Truck	39,996	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mobile Crane 200t	39,996	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Lowboy Truck CXU 613 / 100 T	39,996	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Tire Handler WD 600-3	39,996	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Lightning Plant MOTOR LDW 1003 GE	39,996														
Required Mechanics		67	109	142	153	153	156	156	156	150	130	97	86	24	22
Site		12	18	27	30	30	33	33	33	27	27	18	15	6	6
Drills	39,996	9	12	18	21	21	21	21	21	18	18	12	9	4	4
Shovels	39,996	3	6	9	9	9	12	12	12	9	9	6	6	2	2
Workshop		55	91	115	123	123	123	123	123	123	103	79	71	18	16
Base	39,996	4	4	4	4	4	4	4	4	4	4	4	4	2	2
FEL	39,996	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Truck	39,996	20	44	60	68	68	68	68	68	68	52	32	24	6	4
Ancillary	39,996	28	40	48	48	48	48	48	48	48	44	40	40	8	8
Total Direct		169	261	332	353	349	356	356	355	341	300	227	204	93	86

17 MINERAL PROCESSING AND RECOVERY METHODS

Run of mine ore will be crushed in a primary, secondary and tertiary crushing circuit and then treated by a heap cyanidation process, in order to recover gold.

The processing plant has been designed for an annual working rate of 360 days for the crushing plant, and 365 days for the leaching and ADR plant, therefore the crushing plant considers a nominal throughput of 81,150 tpd, while the leaching and ADR plant 80,000 tpd. The average head grade considered for plant design is of 0.5 g/t of Au and 0.25 g/t of Ag. The resources available for the project are estimated to be 294 Mt, establishing the project life at 13 years.

The project includes the following unit operations or facilities:

- Crushing (primary, secondary and tertiary)
- Heap Leaching.
    - Solution handling.
- ADR plant (Adsorption, desorption and recovery).
    - Adsorption
    - Acid wash and Desorption
    - Recovery (Electrowinning and smelting)
    - Carbon reactivation.

A general diagram for the process route is shown in Figure 17-1. Design considers a gold recovery, in the leaching process, of 80%. The overall calculated recovery for the process is 79.2%, including ADR and electrowinning.

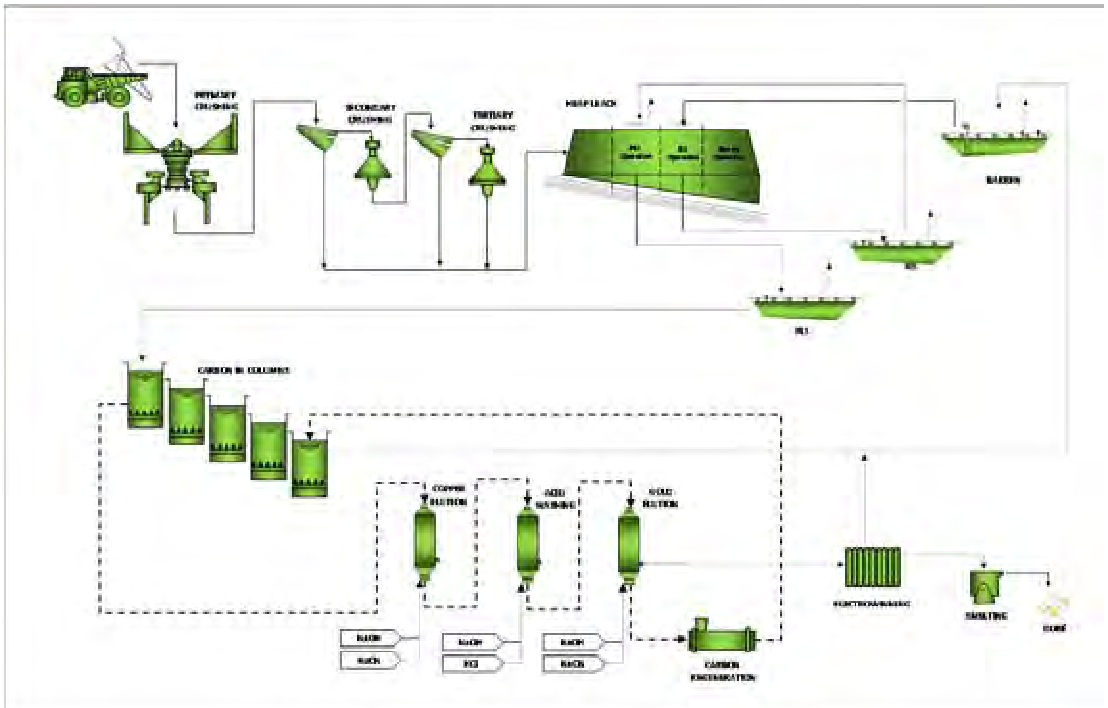


Figure 17-1 Conceptual Process Plant Flowsheet

The key process criteria used for plant design and operating costs calculations are provided in Table 17-1.

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Table 17-1 Main Design Criteria for Plant

Item	Unit	Value	Source
General			
Operating days for crushing plant	d/y	360	Defined by client
Operating days for ADR plant	d/y	365	Defined by client
Daily operating hours	h/d	24	Defined by client
Daily throughput for crushing plant	tpd	81,150	Obtained from mining plan
Daily throughput for ADR plant	tpd	80,000	Obtained from mining plan
Au head grade for design	g/t	0.50	Defined by client and by Alquimia Conceptos S.A.
Ag head grade for design	g/t	0.25	Defined by client and by Alquimia Conceptos S.A.
Cu head grade for design	ppm	240	Defined by client and by Alquimia Conceptos S.A.
Natural moisture of material	%	2	Defined by client and by Alquimia Conceptos S.A.
Utilization			
Primary crushing	%	72	Defined by client and by Alquimia Conceptos S.A.
Secondary crushing	%	85	Defined by client and by Alquimia Conceptos S.A.
Tertiary crushing	%	85	Defined by client and by Alquimia Conceptos S.A.
Solution handling	%	95	Defined by Alquimia Conceptos S.A.
ADR plant, Electrowinning and Smelting	%	92	Defined by Alquimia Conceptos S.A.
Distribution system for fresh water	%	95	Defined by Alquimia Conceptos S.A.
Heap Leaching			
Au recovery	%	80	Result from Maricunga testwork, defined by client
Ag recovery	%	65	Result from Maricunga testwork, defined by client
Cu recovery	%	1	Result from Maricunga testwork, defined by client
ADR Plant			
PLS Au concentration	g/m^3	0.35	Calculated through mass balance
PLS Ag concentration	g/m^3	0.14	Calculated through mass balance
PLS Cu concentration	g/m^3	66	Calculated through mass balance
BLS Au concentration	g/m^3	0.0032	Calculated through mass balance
BLS Ag concentration	g/m^3	0.0013	Calculated through mass balance
BLS Cu concentration	g/m^3	54.32	Calculated through mass balance
Au adsorbed	%	99	Industry practice standard or benchmarking
Ag adsorbed	%	99	Industry practice standard or benchmarking

17.1 Primary Crushing and Stock Pile

Run of mine (ROM) material is transported by trucks to the primary crushing facility at a rate of 81,150 tpd, where it is discharged directly into the dump pocket of the primary gyratory crusher, operating with an open side setting of 165 mm (6'') to achieve a product size P_{80} of 116 mm.

Primary crusher discharge is then transported by an overland conveyor system to a coarse material stockpile with a live capacity of 34,500 t, equivalent to 10 hours of operation. Primary crushing stage is illustrated in Figure 17-2.

A dust management system by suppression is considered for the primary crushing plant, which consists on wetting the dust through spray nozzles, avoiding dust contamination to the atmosphere.

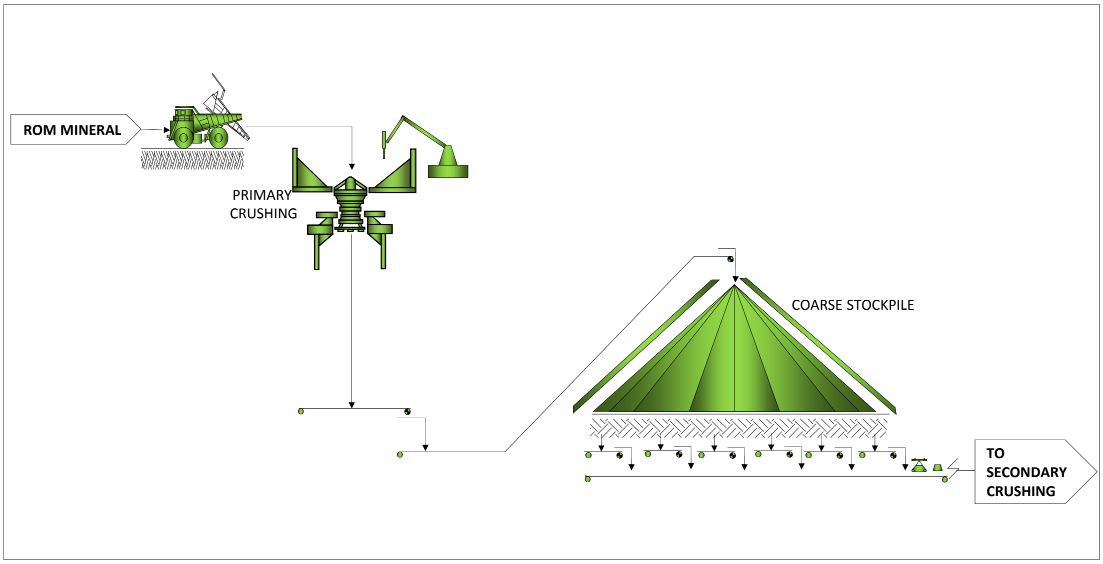

Figure 17-2 Crushing and Stockpile Stage

17.2 Secondary and Tertiary Crushing

Material from the coarse stockpile is conveyed to the secondary and tertiary crushing stage. The material is firstly classified with two conventional screens of 10 ft x 24 ft each and slot opening of 38 mm. The undersize material is conveyed directly to the fine stockpile, while the oversize material is fed to two cone crushers operating with a closed side setting ("CSS") of 35 mm. The product of the secondary crushing stage has a P_{80} of 44 mm.

The product of the secondary crushers is classified by three conventional screens of 10 ft x 20 ft each and slot opening of 20 mm. Undersize material is conveyed directly to the fine stockpile, while oversize material is processed by three tertiary cone crushers with a nominal opening (CSS) of 18 mm. The product from the tertiary crushers is conveyed to the fine stock pile, which has a live capacity of 16,250 t. From the fine stock pile the material is conveyed to the heap leach. The final product of the secondary and tertiary crushing stages has a P_{80} of 19 mm.

Secondary and tertiary crushing facility shall also have a dust collection system. All crushers are in a closed building to reduce the dust contamination. Secondary and tertiary crushing stages are illustrated in Figure 17-3.

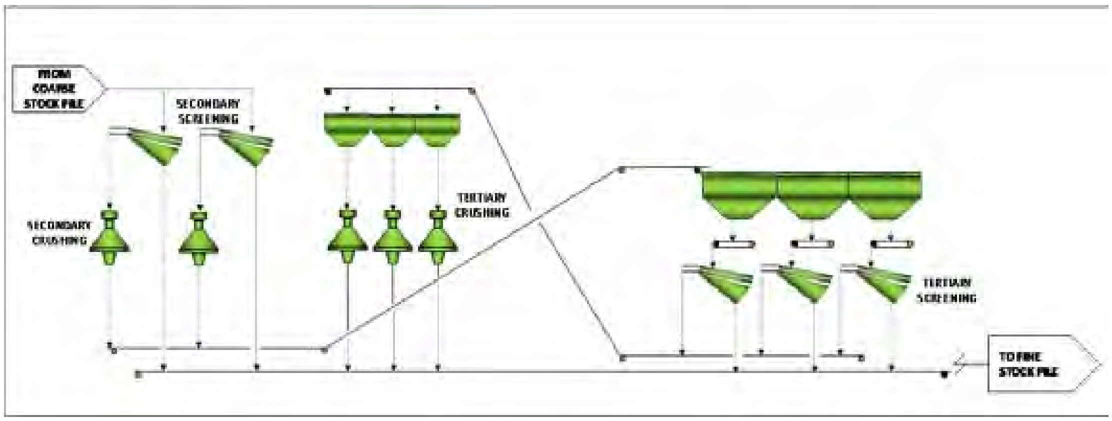

Figure 17-3 Secondary and Tertiary Crushing Stage

17.3 Heap Leaching

Crushed material is conveyed from the fine stockpile to the leaching heap initially by a tripper conveyor located at one side of the heap and then loaded onto the heap using a grasshopper and radial stacker system, which are illustrated in Figure 17-5. The material is arranged in modules of 10 m height x 90 m long and 40 m wide.

Before starting the material stacking, the pad area has to be conditioned to accomplish the heap stability requirements, and then covered by a geomembrane. The heap will have a final height of 100 metres.

Heap irrigation of the material is carried out through a two stage drip irrigation system, using different irrigation solutions in each stage to maximize the gold recovery. The irrigation rate considered is 10 $l/h/m^2$, with an evaporation rate of 2 $l/d/m^2$ for the heap and 4 $l/d/m^2$ for the process ponds.

Gold and silver are recovered through the use of a weak cyanide solution which irrigates the stacked material using drippers over a 160 day cycle. In this cycle the material is irrigated in two stages to maximize initial gold value in the pregnant solution. Fresh material is leached with intermediate leach solution (ILS) which contains a medium gold tenor, and pregnant leach solution (PLS) is obtained, which is sent directly to the ADR plant. The material that has been irrigated for more than half the cycle (80 days) is then leached with barren solution to obtain an ILS. The ILS solution irrigates and drains through the leach heap, reports to the PLS pond and from there is pumped to the ADR plant where gold is adsorbed onto activated carbon. Gold stripped solution exiting the ADR plant is pumped to the barren solution pond. This circuit minimizes solution flow volumes downstream of the heap leach and also increases gold tenor in solution reporting to the ADR plant. The diagram for the leaching stage is illustrated in Figure 17-4. An emergency pond has also been contemplated, with a residence time of 1 day. The emergency pond is located next to the PLS pond.

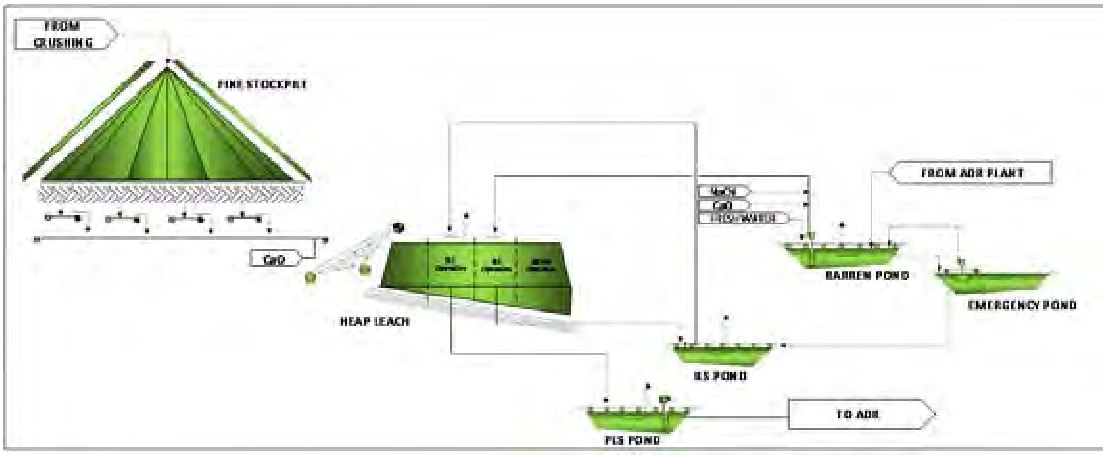

Figure 17-4 Heap Leaching Stage

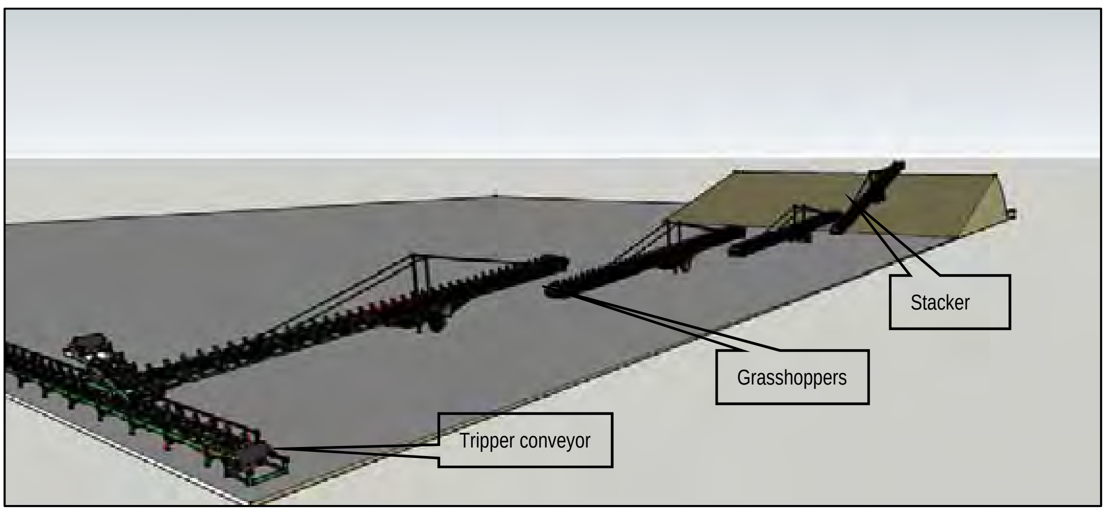

Figure 17-5 Heap Leach Stacking System

17.3.1 Heap leach pad

The heap leach pad is located 6 km from the pit, at an elevation of 4,250 m.a.s.l. as illustrated in Figure 17-6.

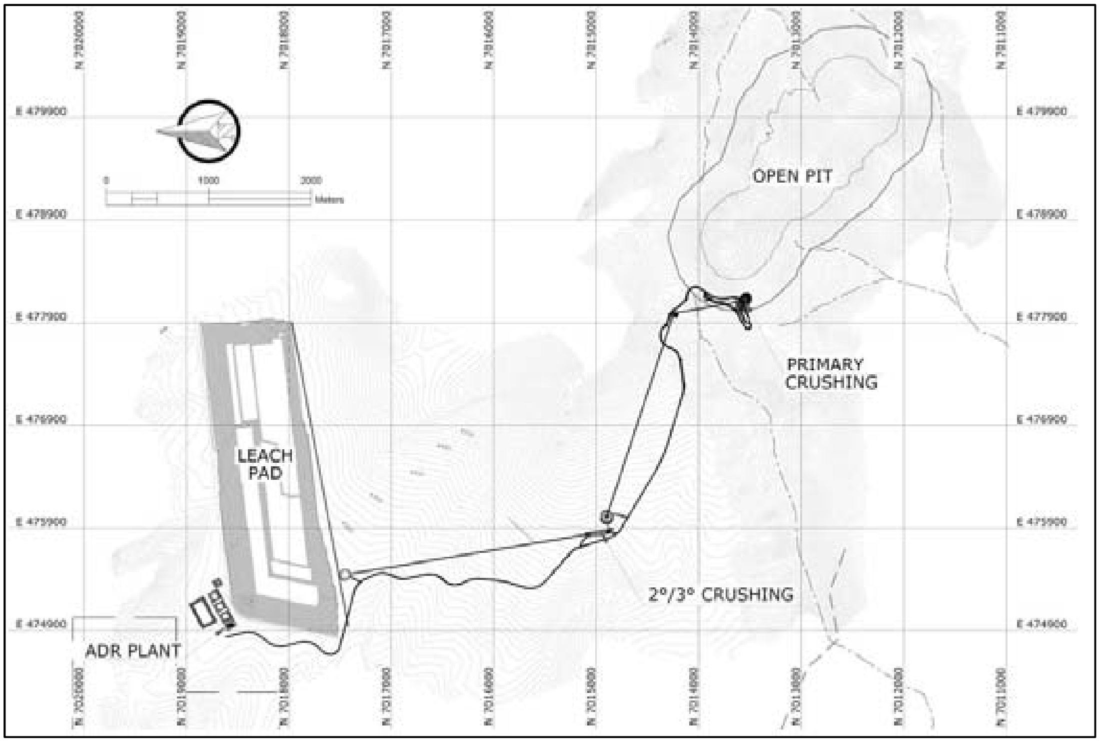

Figure 17-6 Heap Leaching Area

A conceptual design for the pad has been developed involving the irrigating and drainage systems as well as the liner system configuration.

17.3.2 Irrigation System

The irrigation system designed will evenly apply cyanide solution directly on the heap leach surface, to ensure leaching throughout the heap. For design purposes, an irrigation rate of 10 l/hr/m^2 and 160 days irrigation cycle, 80 days with barren solution and 80 with intermediate solution, were considered.

The modules are arranged in cells of 40 m width. The length of each cell is defined by the geometry of each lift plan area, which also determines the number of modules per cell. The heap leach cell configuration is as presented in Figure 17-7.

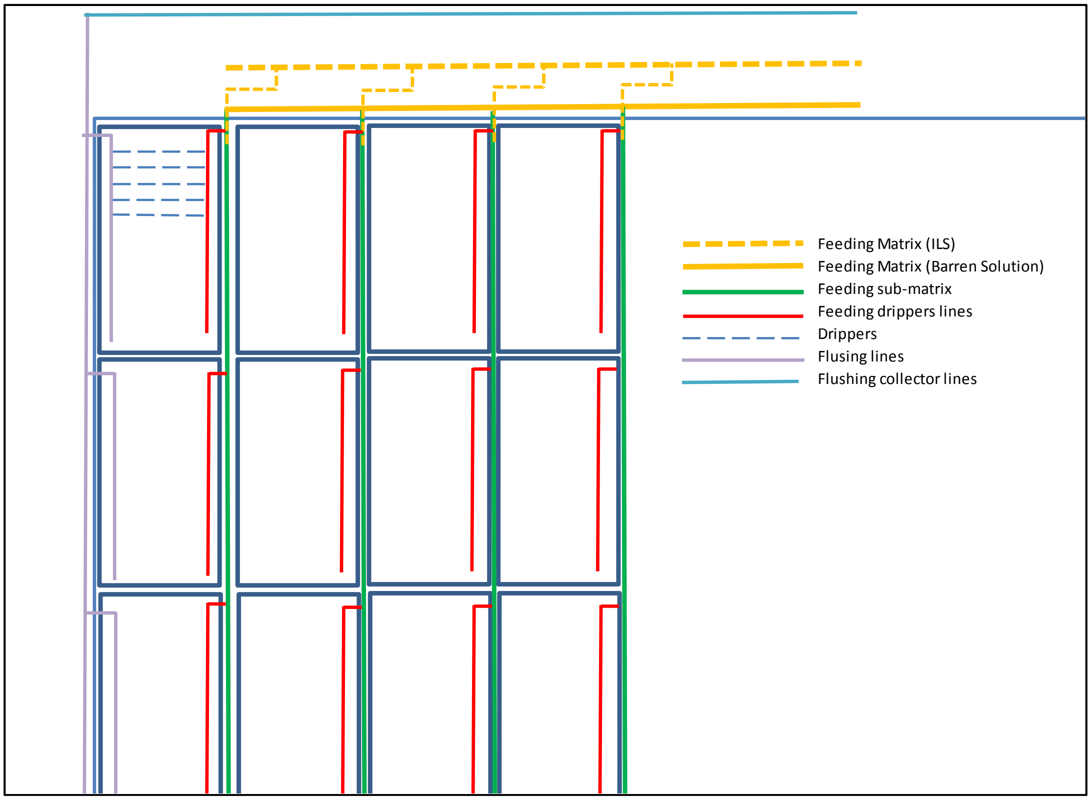

Figure 17-7 Irrigation Cell Configuration

The irrigation system consists of several irrigation subsystems subdivided in modules of 90 m x 40 m. The main components of each module are:

- Drippers or emitters
- Dripper lines
- Feed sub matrix
- Feed pipeline or matrix
- Flushing sub matrix
- Flushing pipeline or matrix.

The spacing between the drippers and the dripper lines was calculated based on the irrigation rate set for the project and the basic criteria to ensure a uniform irrigation through the heap leach surface and depth.

For operational purposes, and based on the suppliers experience in similar projects, a square arrangement is recommended, resulting the following 0.7 m of separation between drippers and dripper lines. Figure 17-8 shows a schematic module configuration.

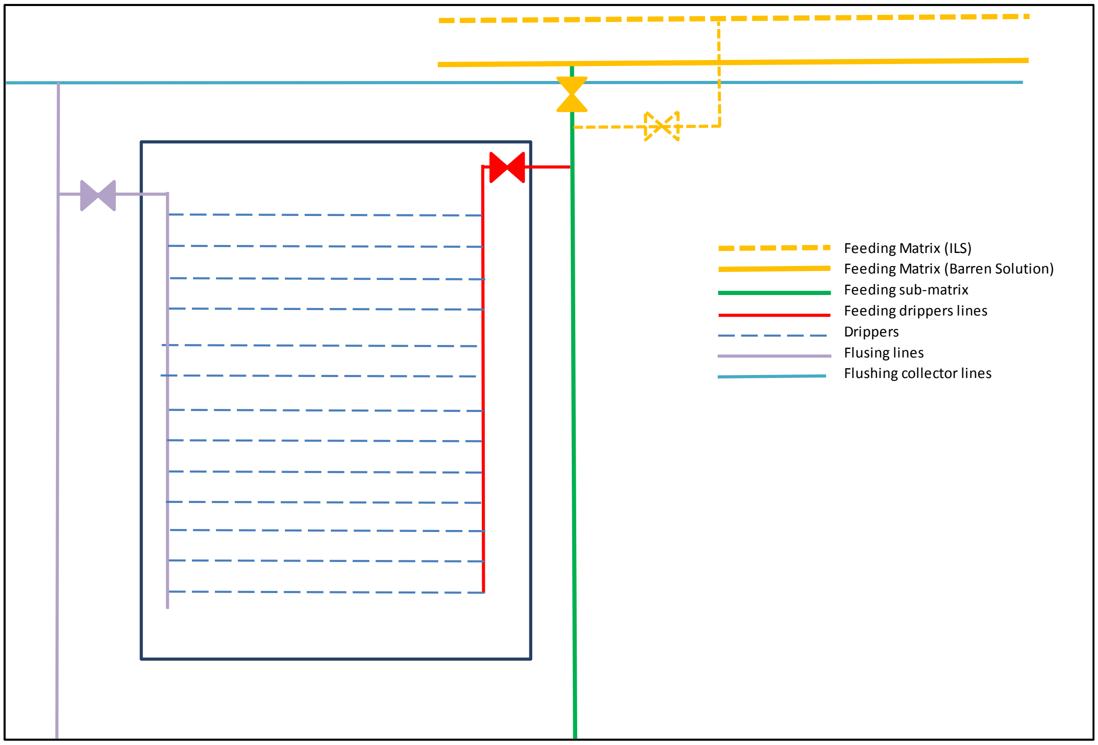

Figure 17-8 Module Configuration

This system allows the irrigation with ILS and Barren solution for each cell using the valve configuration showed in Figure 17-8, which depends on the leaching cycle being carried out.

17.3.3 Drainage System

The drainage system will consist of a network of perforated HDPE pipes, running in East to West direction towards the telescopic longitudinal collector pipelines placed at every 40 m. The longitudinal manifolds consist of perforated HDPE pipelines, running from south to north, transporting the flow into discharge chambers. Additionally, perimeter collector pipelines will be placed at the downstream end of each cell, and will also direct the solution flow into the discharge chambers. A plan view of the drainage system is presented in Figure 17-9.

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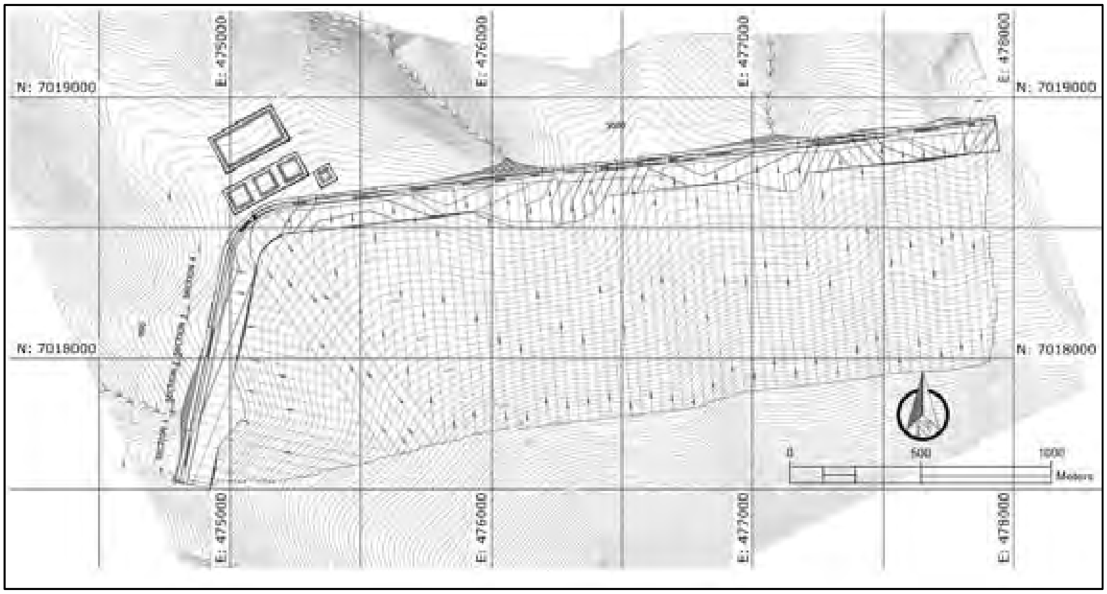

Figure 17-9 Drainage and Collection System (Plan view)

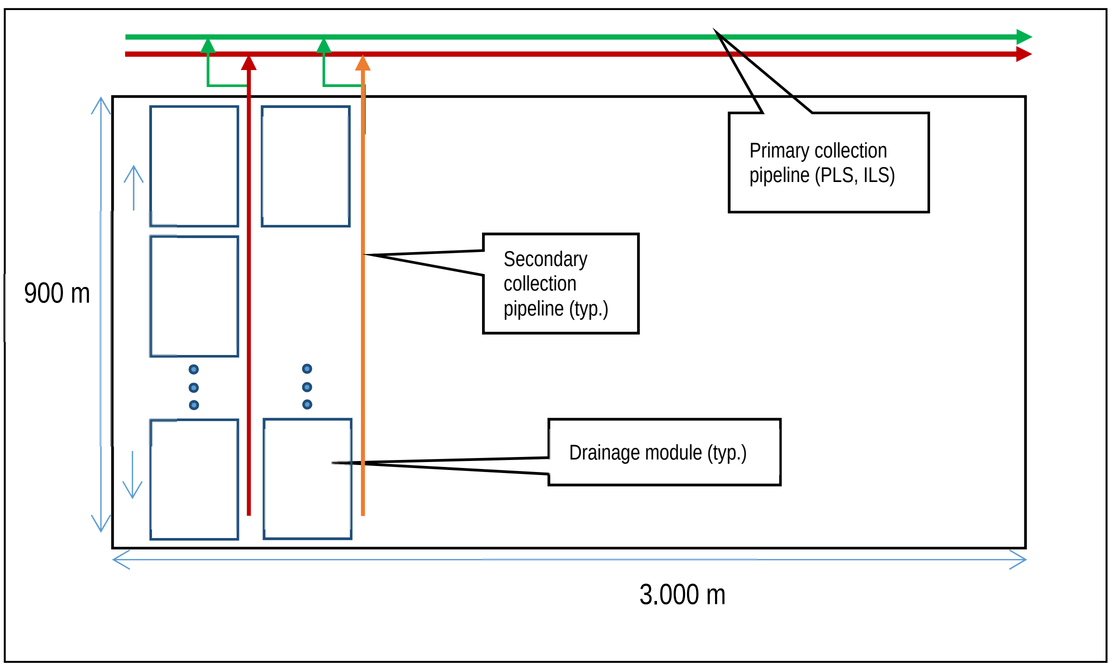

Figure 17-10 Drainage and Collection System Diagram

This system allows the collection of PLS or ILS for each cell using the valve configuration showed in Figure 17-10, which depends on gold concentration in the solution and leaching cycle being carried out.

The general cell configuration is illustrated in Figure 17-11.

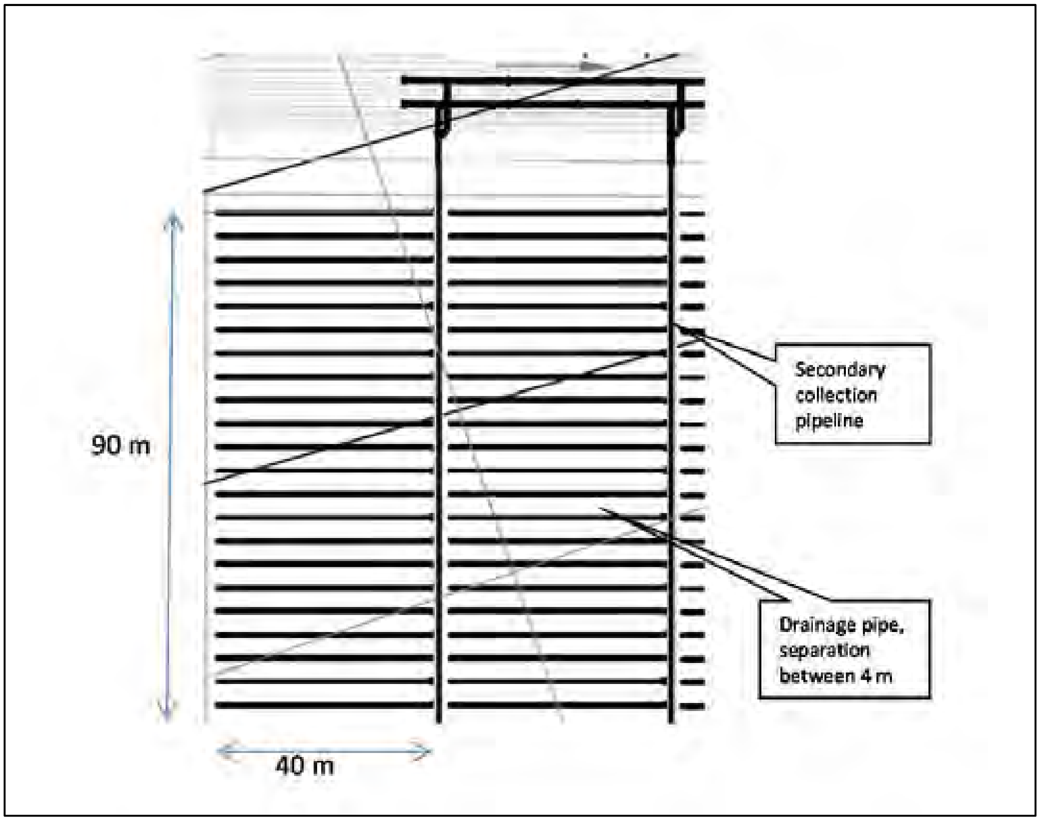

Figure 17-11 Drainage Cell – General Plan

Longitudinal Collector Pipelines

To optimize the drainage system design, the longitudinal collector pipelines were designed by sectors, from the north to south side of the heap leach, resulting in a telescopic pipeline arrangement.

A summary of the results is the following:

- Sector I (from primary collector to 270 m length):
  - Nominal diameter 280 mm.
- Sector II (from 270 m to 450 m length):
  - Nominal diameter 315 mm.
- Sector III (from 450 m to 630 m length):
  - Nominal diameter 355 mm.
- Sector IV (from 630 m to end of the heap pad):
  - Nominal diameter 400 mm.

Discharge Launder

The flows discharged to from secondary collection lines shall be directed to the main perimeter collector launder for ILS and PLS, which are located in east to west direction of the heap leach.

17.3.4 Liner System Configuration

Base Liner System

The leach pad base liner system, as the environmental barrier between the leach operations and existing ground, consists of the following components:

- 0.60 m thick overliner, consisting of granular material which provides permeability and protection to the HDPE geomembrane
- Primary barrier consisting of 2.0 mm thick HDPE geomembrane liner single textured
- 0.15 m thick fine material, which provides protection of the geomembrane and ground contact
- 0.5 m thick basal improvement.

This is presented in Figure 17-12.

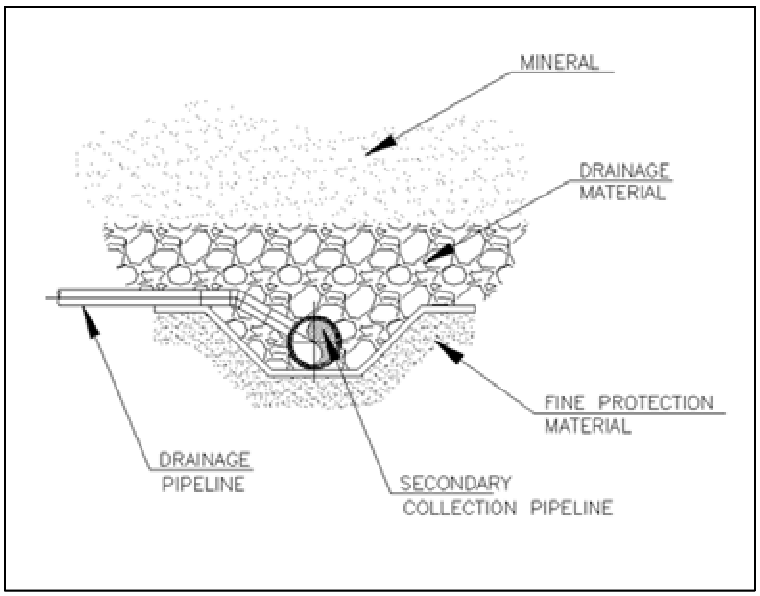

Figure 17-12 Base Liner System

Leach Pad Construction and Loading Plan

The leach pad loading plant considers that the material will be stacked from the lower to upper level and from the West to the East. Each platform is 10 m high and the maximum piezometric height is 100 m.

Figure 17-13 shows the construction of the leach pad at year 2, year 5, year 7 and 100% fill stages. Requirement for base liner system is related to the leach pad extension.

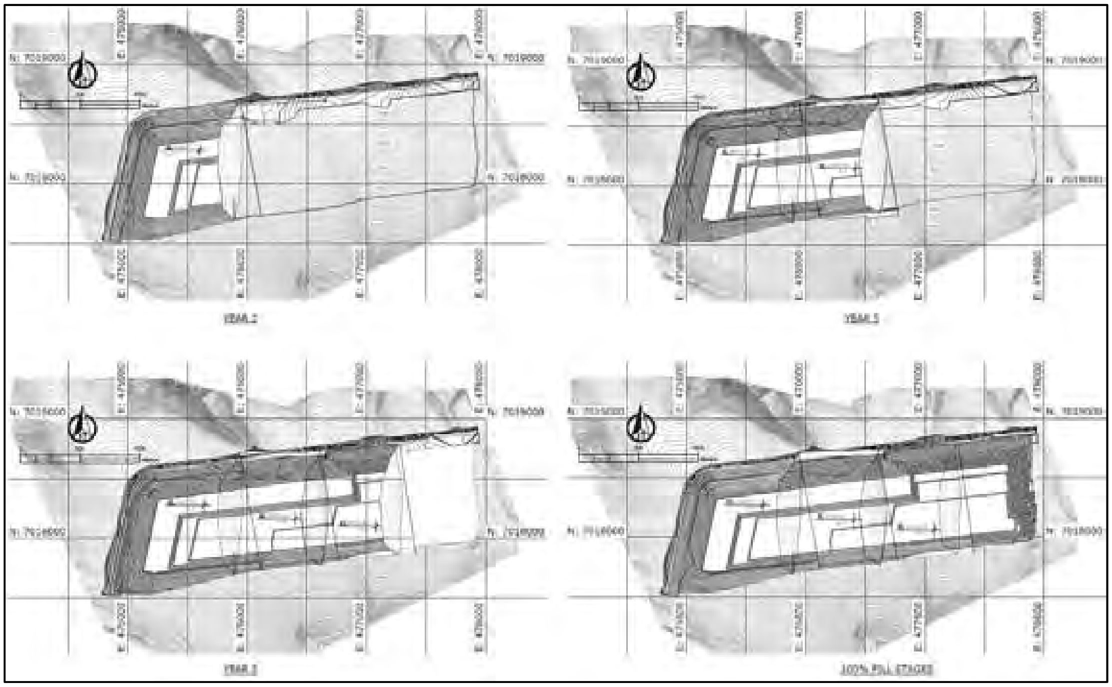

Figure 17-13 Leach Pad Construction

17.4 ADR, EW and Smelting

The process to recover gold and silver from PLS, ADR plant, involves three stages: adsorption, desorption and regeneration.

In the adsorption stage, gold and silver contained in the PLS solution are loaded onto activated carbon. The process considers three trains operating in parallel with five adsorption tanks each, where the pregnant solution passes in counter current by the activated carbon.

With the exception of the first adsorption tank, solution enters into the tanks from a central downcomer pipe and flows up through bubble caps in a distribution plate, preventing the backflow of carbon, and fluidizing the carbon bed before overflowing from the top of the adsorption tank. The solution overflows into a circumferential launder which leads to the next adsorption tank in the series.

Discharge solution (BLS) from the last tank passes through a vibrating carbon catch screen, which retains any carbon escaping from the adsorption circuit and into the barren solution tank, and then the solution is pumped to the barren solution pond. Fresh and regenerated carbon is added to the last stage of the adsorption circuit.

The carbon in the adsorption circuit is moved counter current to the solution flow until it reaches the first stage of the adsorption tanks, and is then transferred to the elution stage.

The carbon is loaded into the elution column by a carbon transfer pump from the first stage of the adsorption circuit. The carbon falls directly into the column and unnecessary moisture is allowed to drain to the floor, where it will be picked up by a vertical spindle spillage pump and discharged to the overflow launder at the last adsorption tank.

Loaded carbon is then pumped into the elution columns, were a modified Anglo American Research Laboratories (AARL) process is carried out. The carbon is pre-soaked with a caustic cyanide solution and eluted with cold water for copper removal, then acid washed to remove the contaminants in the carbon, and then water washed and pretreated with a caustic/cyanide solution, to be then eluted for gold and silver using hot water al 110° C. Details on each of the elution stages will be described below.

First the loaded carbon is presoaked in a 0.5% NaOH and 1% NaCN solution at a rate of 2 BV/h, for 20 minutes at ambient temperature, to solubilize the copper contained in the carbon. The importance of the temperature in this stage is to prevent any gold from transferring into the solution.

After the presoak, carbon is washed with water at ambient temperature at a rate of 2 BV/h for 90 minutes to remove solubilized copper and residual cyanide, and also to reduce the pH of the carbon to neutral to prepare for the next stage. The solution obtained from these two stages is sent directly to the stripped sector of the leach heap.

Following these stages, the carbon is acid washed to remove inorganic contaminants adsorbed onto the carbon during its contact with the solution and slurry in the adsorption circuits. A 3% HCl solution flows through the carbon at a rate of 2 BV/h for a period of 30 min, where the solution discharged from the top of the column flows by gravity to the acid neutralization tank.

Another water wash stage follows the acid wash, to remove the residual acid from the carbon bed. The carbon is water washed with water at a temperature between 80-90° C, at a rate of 2 BV/h, for a period of 120 min. The water used for washing is then directed to a neutralization tank, where it is diluted with water and neutralized with NaOH before being discharged to the barren pond. The water wash is conducted at higher temperatures to elevate the temperature of the carbon before the gold and silver elution begins.

Before elution, the carbon has to undergo a stage of pre-treatment, achieved with a 0.5% NaOH and 1% NaCN solution at a temperature of 110° C. This stage is carried out for 20 minutes at a rate of 2 BV/h of solution, passing the caustic cyanide solution through heat exchangers (primary and secondary) before entering the column to increase temperature to the desired value. The solution passes through the column, discharged from the top of the column and after passing through the column, is pumped through the reclaim secondary heat exchanger to reduce its temperature so that flashing is prevented. The primary heat source will be a gas fired boiler which heats thermal oil up to 130° C, fluid that is then pumped through the hot side of the primary heat exchanger.

The following stage is gold and silver elution. Solubilized gold and silver in the carbon is eluted by hot water at a temperature of 110° C, for 3 hours at a rate of 2 BV/h. The product of this stage is the electrolyte, which is pumped to the electrowinning stage.

The final stage of elution is the cooling of carbon inside the gold and silver column before transferring it from the column to prevent water from boiling when removing the carbon from the column, and to stop any further elution of gold and/or silver from the carbon. This stage is carried out for 20 min with water at ambient temperature, at a rate of 2 BV/h. The heating circuit if not operated in this stage.

Eluted carbon is hydraulically transferred from the elution column, dewatered and regenerated at 600-700° C every four cycles. A horizontal regeneration kiln is installed to regenerate the carbon after elution. Regenerated carbon passes through a carbon sizing circuit to remove fine carbon before returning it to the adsorption trains. The purpose of regeneration is to reactivate the carbon by volatilizing organic compounds, such as oils, that adsorb onto the carbon. Reactivation is necessary to maintain efficient gold recovery in the adsorption circuits.

Pregnant solution produced from elution process is stored in the electrolyte tank and circulated through electrowinning cells, where gold and silver are transferred onto cathodes. Facilities are designed to recirculate electrolyte back to the electrolyte tank and through the cells until solution tenor drops below a desired level of gold concentration, considering it spent solution. Spent electrolyte solution is then pumped to the barren pond, to be reused in heap leach.

Gold loaded cathodes are removed from the cells, calcined and smelted into doré bars, which are the final products of the process, and stored in a vault from their commercialization.

The process for the ADR plant, electrowinning and smelting is illustrated in Figure 17-14.

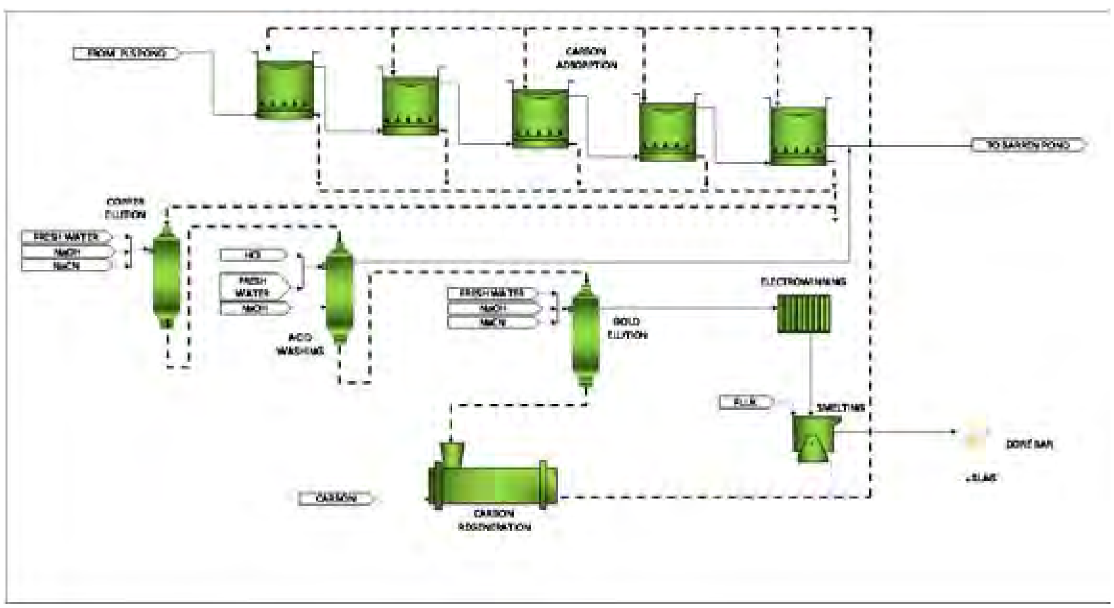

Figure 17-14   ADR, Electrowinning and Smelting Stage

The main equipment selected for the process is shown in Table 17-2, describing equipment dimensions and capacities.

Table 17-2 Main Process Equipment

Equipment	Qty	Characteristics
Primary crushing		
Giratory crusher	1	62 inch x 75 inch , 450 kW
Secondary crushing		
Secondary screen	2	Conventional, 10 ft x 24 ft
Cone crusher	2	nominal opening 35 mm , 746 kW
Tertiary crushing		
Tertiary screen	3	Conventional, 10 ft x 20 ft
Cone crusher	3	nominal opening 18 mm , 597 kW
Heap Leaching		
Barren pond	1	23,200 m^3
ILS pond	1	21,700 m^3
PLS pond	1	21,700 m^3
Emergency pond	1	103,800 m^3
ADR Plant / Smelting		
Adorption column	5	3 train, Capacity= 4,057 m^3/h
Copper elution column	1	28.6 m^3
Acid wash column	1	28.6 m^3
Elution column	1	28.6 m^3
Electrowinning cells	4	Capacity= 3.54 m^3
Smelting	1	Induction furnace for 0.05 t/d

Table 17-3 presents reagents and consumables consumption, including water and energy requirements.

Table 17-3 Reagents and Consumables Consumpion

Reagent / Consumable	Consumption	
	IP	Value
Crusher Liner	g/t	45
Sodium Cyanide	g/t	234
Lime	g/t	2,700
Carbon	t/d	1.3
Borax	kg/ t ppt*	300
Sodium Nitrate	kg/ t ppt*	1,500
Silice	kg/ t ppt*	1,250
Sodium Hydroxide	g/t	10
Hydrochloric Acid (32%)	g/d	2.5
Energy	MWh/LoM	1,358,842
Primary Crushing		71,952
Secondary/tertiary Crushing		304,460
Leaching		769,134
ADR Plant		106,655
Other		106,642
Water	m^3/d	7,877

(*) t ppt: Tonnes of precipitate to smelting

17.5 Leach Pad Geotechnical Study

17.5.1 Leach Pad Description and Location

In accordance to what was observed in the sector, the area selected for the construction the leach pad corresponds to a terrace or an old alluvial deposit (Quaternary deposits), mainly composed of granular materials like gravels and silty sands with medium to dense compactness. The filler is arranged over a volcanic bedrock probably tertiary-aged.

Figure 17-15 presents the general characteristics of the relief and the geotechnical setting of the area selected for the leach pad construction.

Figure 17-15 General Arrangement of the Leach Pad Site

17.5.2 Geotechnical Study of Selected Site

In accordance to the site exploration results performed in the previous study, developed in 2012 and contained in the "Geotechnical Study of Leaching Pad Platform – Cerro Maricunga" report, the soil in the sector can be generically classified as gravels and silty sands without plasticity. Its fine material content (silt), according to the particle size analysis performed, varies between 15% and 20%, with the highest content in the upper layers.

The natural soil stratum thickness could not been determined since it extends to the maximum depth known (around 5.5 m), where it was not detected the presence of sub-superficial waters upwelling or ground water level.

<u>Stratigraphy</u>

From the direct stratigraphy description of the soil samples observed in the study, it can be inferred a general categorization of the soils in the sector, which, from their genesis point of view, corresponds to an old alluvial filler composed of medium granular materials (gravels and sands of varied particle sizes) and a silty content slightly variable in the depth. The medium stratigraphic characteristics of the area are indicated in Table 17-4.

Table 17-4 Typical Stratigraphy for the Area

Horizon	Limit Points (m)	Thickeness (m)	Description
H1	0.0 - <1.60	< 1.60	Gravel and sandy natural soil with fines without plasticity, around 20%, color light brownish-grey, with some dispersed gravels and blocks of 10'-12' maximum size. Low humidity and medium compactness. No vegetal content.
H2	<1.60 – 5.50 +	4.00 <	Gravel and silty sands natural soils, color lightgray, containing larger gravel particles and blocks, up to 20', medium-thick compactness and fines (slit) of low plasticity. No vegetal content.

17.5.3 Soil Resistance Parameters

The resistance of any soil can be represented through a failure model (Mohr-Coulumb), where the intrinsic resistance is defined by means of the cohesion (c) and internal friction parameters (Φ). The direct determination of these parameters requires the realization of direct cut or triaxial laboratory experiments.

In the absence of these test results it is possible to estimate the value of these parameters using correlations with other parameters (plasticity in the case of clay or density in the case or granular soils). For the calculation of the design parameters it is necessary to previously estimate these resistance parameters (c, Φ).

The internal friction angle (Φ) can be estimated from the granulometric characteristics and natural density (measured in situ or in a laboratory) with respect to the lowest and highest values that can be obtained in standard test (Relative Density), using correlations included in the specialized geotechnical literature.

In accordance with these correlations, medium values for these parameters for natural soils stratum (H1 or H2) are:

- Φ : 35º
- c : 2 (t/m^2)
- ρ : 1,800 kg/ m^3 (wet natural density)

17.5.4 Leach Pad Stability Analysis

The verification of the slopes structural stability consists, in a first stage, in determining the equilibrium degree that the deposit would have under statics conditions (where only gravity is acting), with respect to the internal resistant forces of the material, established from the friction and cohesion parameters. The relationship between the total magnitudes of the acting forces, with respect to the internal resistance of the material, is represented through the Static Security Factor.

As a second verification stage, it is incorporated the additional acting force: the inertia forces that would take place during an earthquake, which can be quantified by means of a horizontal acceleration coefficient and another vertical one acting at the same time over the material. The relationship between all acting forces, with respect to the internal resistance of the material, is represented through the Seismic Safety Factor, used to define the security degree of the slope in such limit condition.

To execute the verification and calculation described above, it was used the computer program XSTABL, which is based in a routine originally developed in Purdue University (Indiana-USA), and upon which have been developed most of the software currently available for slopes stability analysis.

The analysis procedure basically involves the generation of a large amount of failure-prone surfaces inside the slope and the determination of the safety factors associated to each of these surfaces. The final equilibrium degree of the slope under analysis is represented by the minimum value obtained from all generated failure-prone surfaces (critical failure).

Modelling and Material Properties

For the combined modelling of the Deposit-Foundation pair, it has been considered to stablish a vertical stratification of the materials, which considers the following depth distribution:

- Crushed and leached material with low/medium humidity
- Draining interface composed by the lower layer of the leached mineral, the draining layer (cover) and the geomembrane. The interphase total thickness is considered to be 1.0 m, with the phreatic level on its surface
- Founding soil above which the cover-geomembrane pair is supported.

The properties of the materials that form the group mineral-draining interphase-foundation soil, according to the modelling that will be used in the stability verification, corresponds, for the mineral case, to the minimum values obtained from the direct cut tests conducted on crushed and agglomerated minerals from similar projects. For the draining interphase-geomembrane conservative characteristics values have been considered. The parameters considered are summarized in the following Table 17-5.

Table 17-5 Summary of Materials Properties and Parameters

MATERIAL	Wet Density (kN/m^3)	Saturated Density (kN/m^3)	Cohesion (kPa)	Friction Angle (Φ)
Crushed mineral on leaching pad (above phreatic level)	18	-	1	35
Cover-geomembrane interphase	20.3	21.5	0	25
Founding soil	18	19	20	35

17.5.5 Stability analysis of the critical section

With the objective of making a comprehensive stability analysis of the slope, section 2 of the projected leaching pad was used, which is considered the most unfavourable (higher basal gradient and slope height).

Table 17-6 presents a summary with the main results obtained from the stability analysis of the critical section.

Table 17-6 Results of Critical Section Analysis.

Verified type of failure	Static S.F.	Seismic S.F.	Comments/Observations
Surfaces with random failures beginning in the frontal slope (Random)	1.656	1.12	Laminar-like minor failures and main failure would compromise lower benches
Surface with interphase failure (2), compromising the entire height (Block)	2.16	1.36	Critical surfaces develop in the base of the leaching pad, by the cover-geomembrane interphase, potential failure compromises the entire height

17.5.6 Comments

From the stability analysis, the following conclusion and comments may be addressed:

- In terms of the static and seismic stability of the leaching pad, the medium slope considered 1:2.5 (equivalent to 22°) is stable under higher basal gradient conditions (6.6%) and all the properties contemplated for the materials that form the leaching pad
- The potentially most critical type of failure is a laminar kind one, which would result in a partial failure in the lower section of the slope. This would compromise approximately 25% of the entire leach height
- Considering the safety factors obtained from the analysis a massive failure compromising the entire height of the leaching pad has a low probability of occurrence
- Given the geometric conditions, the system is stable up to a gradient of 10%
- A maximum thickness of 5 m is recommended for initial/lower pad layer
- The basal gradient can reach values up to 5% without affecting the stability of the initial layer
- As a supplementary safety measure, it is recommended to perform an outline analysis of the perimeter surface of the leaching pad foundation platform, for a 3% gradient and 100 m width
- Site soil material may be used as base material for the construction of the geomembrane support material (silty sand) and the draining layer (cover). The results, in terms of particle size and properties, indicate that the most convenient alternative is to screen material under 5 mm mesh, which would allow obtaining the two required aggregates.

18 PROJECT INFRASTRUCTURE

The Cerro Maricunga Project requires significant infrastructure associated to the operation of the plant. Infrastructure includes: roads, electrical supply, water supply, workshops, warehouse, offices, laboratories, reagent plant, camp accommodation among others facilities as shown in Figure 18-1.

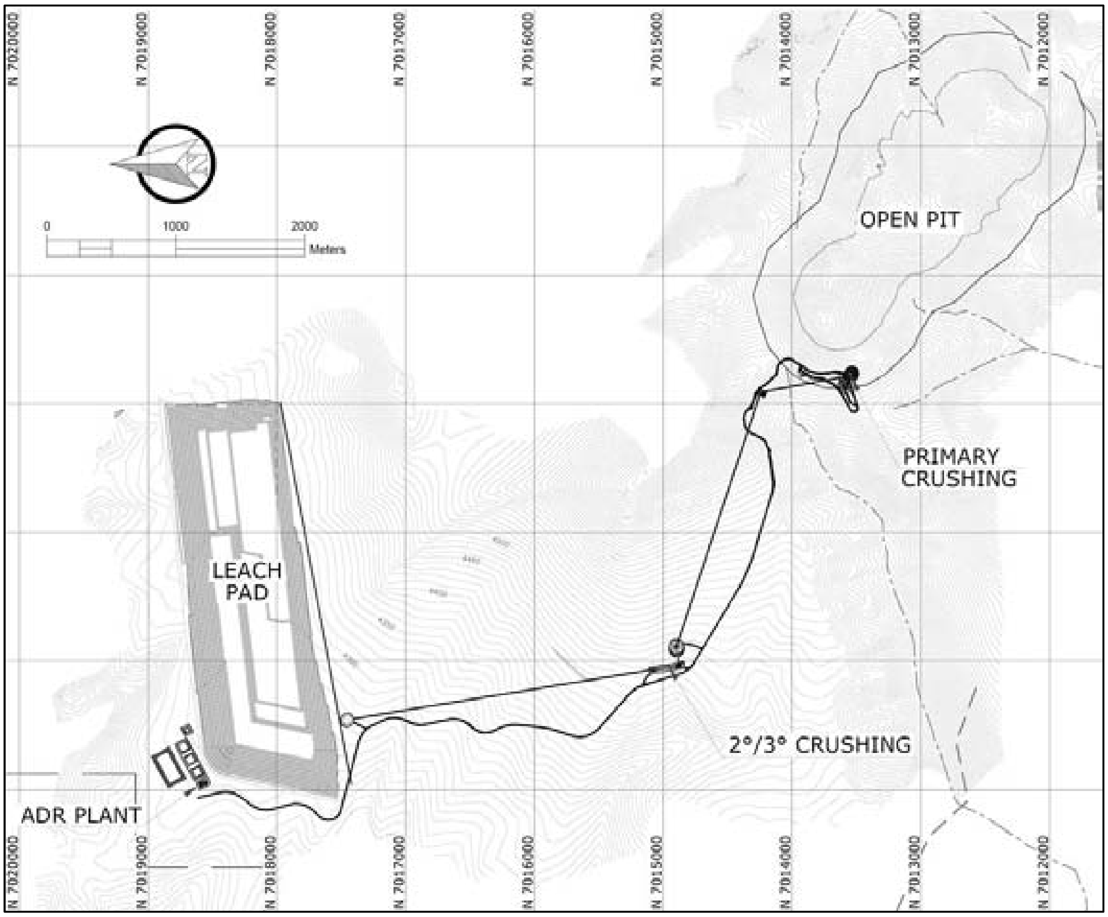

Figure 18-1 Main Infrastructure Facilities

18.1 Water Supply

Cerro Maricunga project has a fresh water make up requirement of 96 l/s. Atacama has acquired a water supply by contracting Aguas Chañar, the largest water treatment operator used in the city of Copiapo, at a flow of 80 litres per second. Water will be transported from Aguas Chañar plant (Copiapo) to Cerro Maricunga by a pump system, which consists of a pipeline with four intermediate pump stations located on a specific points along the route.

Water flow is transported through a pipeline 149 km long, built out of 12 in diameter carbon steel. The pipeline route is illustrated in Figure 18-2.

Figure 18-2 Fresh water pipe route

The pipeline route begins at Aguas Chañar plant (Copiapo), where water flow is pumped from an accumulation tank. The flow is transported from one pump station to another by means of centrifugal horizontal pumps, until the final stage which consists of the fresh water pond on Cerro Maricunga site. Figure 18-3 shows a general sketch of the pump system:

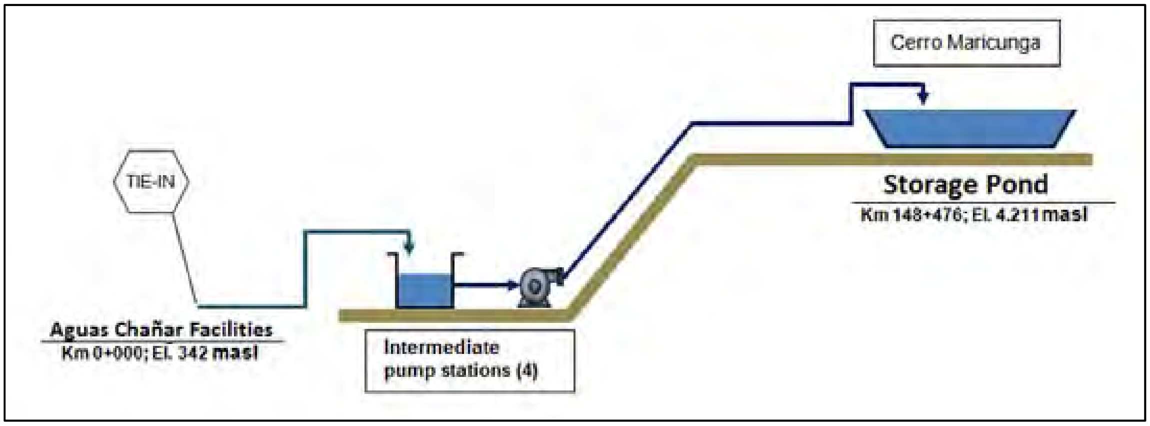

Figure 18-3 Industrial water pump system sketch

Four pump stations were calculated, each one equipped with 2 multistage centrifugal horizontal pumps of 1,800 kW installed power (one operating, one stand-by). Table 18-1 details the characteristics of each pump station for the system:

Table 18-1 Pump station location and Fitting classes

	Pump Station N°1	Pump Station N°2	Pump Station N°3	Pump Station N°4
Type	Multistage centrifugal horizontal pump, split casing			
Operating pumps	1	1	1	1
Stand-by pumps	1	1	1	1
Pump station class	ANSI 900	ANSI 900	ANSI 900	ANSI 900
Station Location (km)	0	60.2	92.7	129.5
Station Elevation (msnm)	342	1,183	2,179	3,147
Power installed (kW)	1,800	1,800	1,800	1,800

Pump station N°1 is powered by Aguas Chañar plant. Electrical substations are projected for pump stations (2, 3, 4). The installed power for each station is shown on Table 18-2.

Table 18-2 Pumps Stations Power

Pump Station	Demand (kW)	Installed (kW)
N°2	1,754	3,911
N°3	1,754	3,911
N°4	1,754	3,911

18.2 Power Supply

The energy for Cerro Maricunga project will be supplied by 110 kV transmission line connected to main Chilean electrical transmission grid or “Sistema Interconectado Central [SIC]”, at “Carrera Pinto” electrical substation. The Carrera Pinto substation is installed between “Cardones” and “Diego de Almagro“ electrical line 1x220 kV, property of TRANSELEC.

Transmission line supplies energy for the process plant, water pump stations (2, 3, 4) and project facilities. The total demand of the project is 26.7 MVA where power system is designed to transmit a maximum of 40 MVA The increase of 2,000 MW in Chilean power grid (SIC) and the connection between the two main Chilean power grids (SIC and SING) (Ref: Informe tecnico definitive 2014), ensures energy availability for Cerro Maricunga project by the first year of operation.

The design of the high voltage power line follows existing roads and the water pipeline, so that the power line extends over pump stations (2, 3, 4). The power line is shown in Figure 18-4 from the substation Carrera Pinto to substation Cerro Maricunga.

The power supply infrastructures required for the project are presented as follows:

- High voltage power line 110 kV, 110.5 km length from substation Carrera Pinto to substation Cerro Maricunga
- Substation Carrera Pinto 220 kV / 100kV
- Substation Cerro Maricunga 110 kV / 23 kV
- Substation for water pump station N°1, 110/23 kV
- Substation for water pump station N°2, 110/23 kV
- Substation for water pump station N°3, 110/23 kV
- Substation for water pump station N°4, 110/23 kV.

Figure 18-4 Power Line Route Carrera Pinto Substation to the Cerro Maricunga Substation

18.3 Roads

18.3.1 Access Roads

The location of the Cerro Maricunga deposit provides major advantages for the project's construction and future exploitation. The existing Ch 31 International road (Copiapo to Argentinean border) is in the vicinity of the project. The access from Ch 31 to the project requires constructing an 11.5 km road. Figure 18-5 shows the projected access road. The road will have a 7 m width, maximum slope 10% and will be compacted, rolled bare over its entire length, as shown as Figure 18-6.

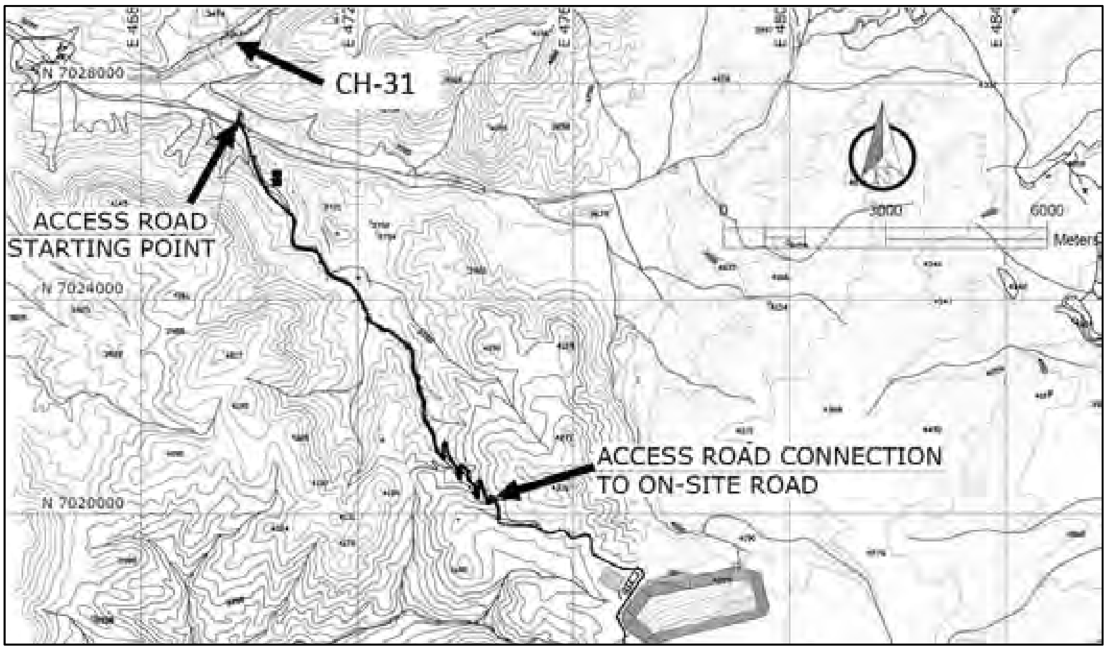

Figure 18-5 Access roads drawing

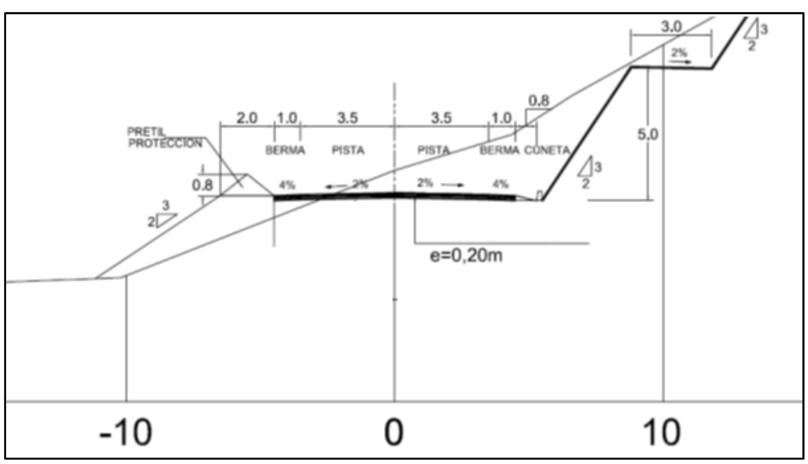

Figure 18-6 Access Road – section view

18.3.2 On-Site Roads

Project design includes all on-site roads that connect the main facilities as shows Figure 18-7. On site roads are connected to project access road. All on-site roads will be constructed using the same standard as the access road. On site roads go to the following areas and cover a total distance of 12.4 km:

- Mine
- Primary crushing
- Secondary and tertiary crushing
- Leach pad
- ADR plant.

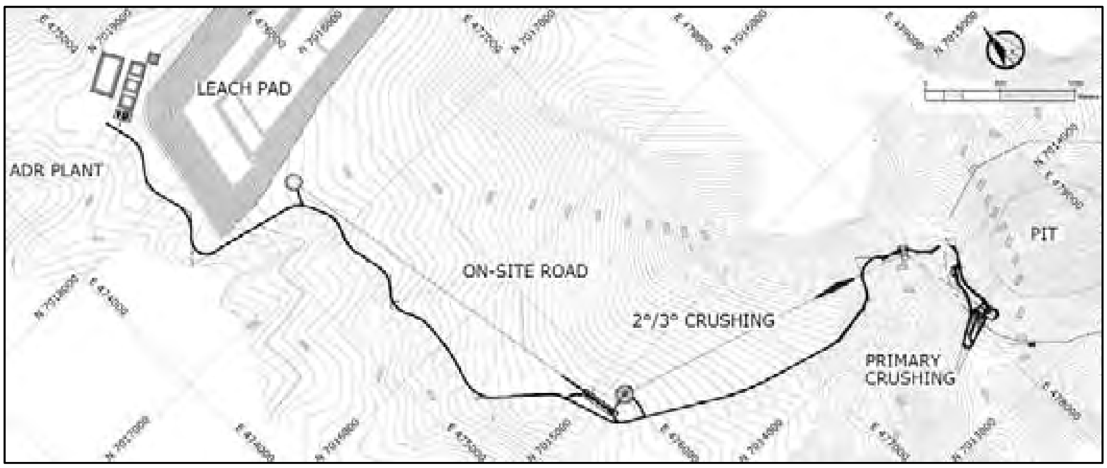

Figure 18-7 On-site roads drawing

18.4 Site Accommodation

The facilities associated with personnel housing comprises the following infrastructure: dining room, individual dorms modules, bathroom facilities, collective dorms modules, polyclinic and recreational room. These accommodation facilities will be used initially for construction contractors and thereafter for operation and administrative personnel.

This operation camp will be located approximately 12 km northwest by project access road at 3,415 masl as shows Figure 18-8. The camp has been designed to accommodate a maximum of 350 people over a 13 year period and will cover an area of approximately 50,000 m^2. The camp will be a prefabricated modular structure.

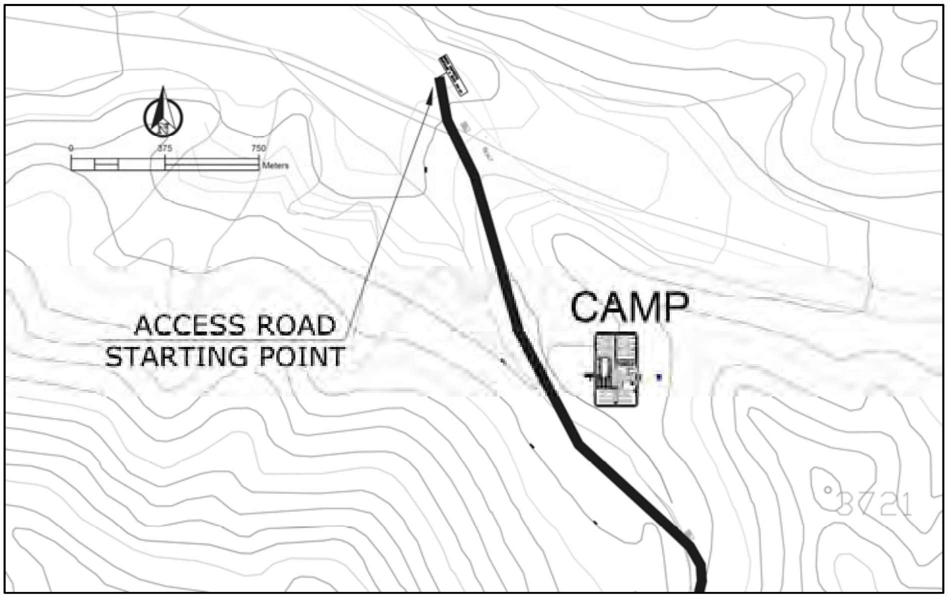

Figure 18-8 Operation Camp Location

18.5 Potable Water Supply

The Project considers a potable water treatment plant to process the water from Aguas Chañar.

18.6 Sewage Treatment

Sewage from the camp will be routed to a sewage treatment plant sized for effluent of 350 employees.

Septic tanks will collect sewage from the administration office, security gatehouse and truck shop. The septic sludge will be pumped out and transported by truck to the treatment facilities.

18.7 Waste Management

All solid domestic wastes, industrial wastes and toxic wastes, generated by the plant, will be temporally stored in a warehouse.

Suitable areas will be designated for the storage of common household waste, which will be produced according to the development of life in the mining operations.

18.8 Administration Building

The administration building will be a single storey building. This building will have offices for plant senior management, besides administrative and technical staff.

Offices are also considered for secretarial and accounting personnel. In addition, the building will have a reception area, a training room, a dining room with kitchen, a large conference hall, smaller meeting rooms, male and female rest rooms and office supply and equipment rooms.

The building will have a central air conditioning system and be equipped with swing windows. Utilities provided include electric power, potable water and a sewage system. Main telephone switchboard and network servers for the operation will be located in the administration building.

18.9 Laboratory

A laboratory for sample analysis will be located in the electrowinning and smelting area.

The assay laboratory will be a modular building type structure fully fitted with laboratory equipment and services so as to reduce the cost of construction and installation. The structure will include a sample preparation room, fire assay, wet laboratory, instrumentation, offices, restroom and electrical sections.

Bag houses, air handling units, cabinets and countertops, dust hoods, drying ovens, fume hoods and scrubbers will be part of the equipment.

18.10 Reagents Plant

A reagent plant has been considered for the storage and supply of reagents. A steel building with concrete foundation is considered for ADR plant reagents storage. In addition, Tanks, agitators and dosage pumps are considered for reagents distribution.

A lime silo is considered for storage and dosage, previous to the stage of loading of the leach pad.

18.11 Maintenance Facilities

The workshops contemplate the following facilities: maintenance shop, welding shop, warehouse, cleaning shop, oiling shop and tire change shop.

The buildings will have a steel structure with concrete foundations. The workshop areas will be assisted by an overhead traveling crane.

Utilities provided include electric power, potable water, compressed air and a sewage system.

18.12 Communications

18.12.1 Off-site Communications

Basic telephone service will be initially supplied via satellite communication. However as the project is carried out, cell phone and desktop communication will be developed to service the needs of the project during construction and operations. An emergency phone connected to the UPS in the control room will be provided on the satellite service. Internet will be accessible with connectivity to the intranet for the home office during construction and operation.

18.12.2 On-site Communication

All facilities will have internal telephone systems connected to every building on site. In addition a wireless systems will be installed, which will also be used by portable radios.

18.13 Mining Facilities

18.13.1 Magazine and Fuel Storage

Vendor will provide storage facilities for fuel, lubricants and explosives as part of their supply contract, thus the cost for these items are included in the delivered price.

18.13.2 Truck Workshop

Truck workshops used for mining truck maintenance and reparations will be located close to the primary crusher. A 4-bay truck shop with 10 m width manually operated gates is considered. Each bay will be 15 m wide, 24 m long and 17 m high and is equipped with all necessary operation equipment such as lifting devices, lubrication machines and ancillary machinery. Overhead traveling crane will be provided for maintenance purposes. The truck shop will be in accordance to the highest safety standard for working personnel.

The truck workshop building, outside storage facilities, work areas, parking and open space otherwise available for truck servicing will cover an area of approximately 150 m x 170 m.

The construction of the truck maintenance shop will be undertaken by ARRIGONI leaseback contract.

19 MARKET STUDIES AND CONTRACTS

19.1 Doré Metal Market

This market is very competitive and in many cases payable metals approaches 100% of the analytical contents and there is not a great deal of variance in terms between refineries. This situation is expected to continue in the foreseeable future.

Doré production from the Cerro Maricunga Project could be sold either on the spot market or under agreements with refineries. Sales and marketing considerations would be evaluated during feasibility-level studies. It is expected that any sales and refining agreements would be negotiated in line with industry norms.

Changes in market price for gold will affect both cash flows and profitability of the Project. Figure 19-1 shows the variability of gold price during the last 5 years (Sept 2009 to August 2014). Gold prices during this period ranged from a low of $ 955/oz (on Sept, 2009) to a high of $ 1,895/oz (on September, 2011) and the average price for the last two years (1,360 $/oz from January 1st 2013 to August 19th 2014) exceeds the assumptions utilized in the base case of financial modelling for the Project.

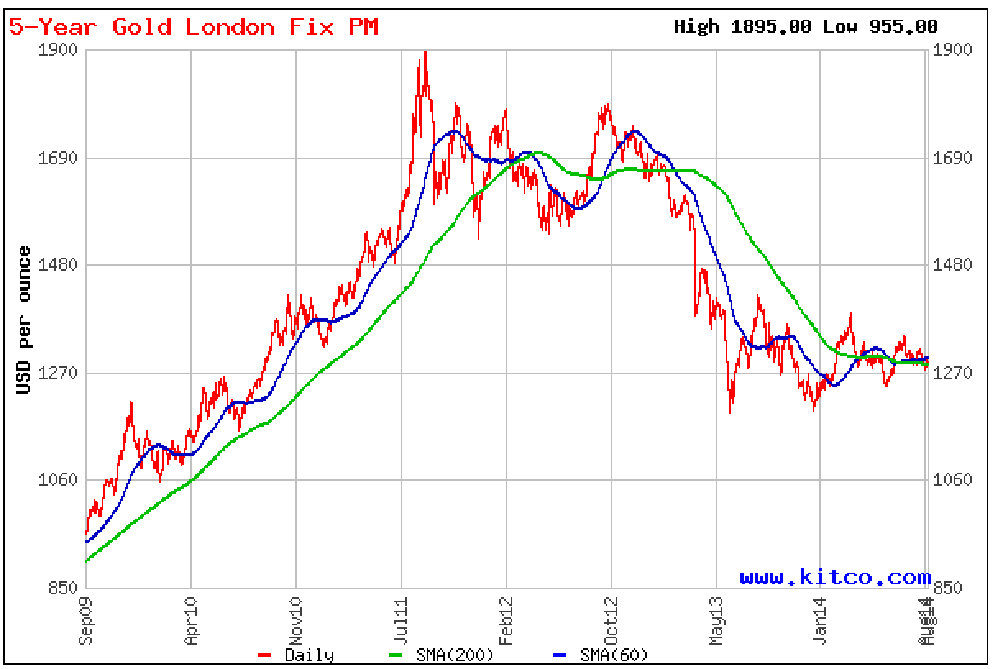

Figure 19-1 Gold Spot Price – August, 2004 to August 2014 (Source: www.kitco.com)

The long term gold price consensus assumptions seen in the market recently are in the $ 1,100 to $ 1,500 per ounce range.

19.2 Contracts

Cerro Maricunga has signed a contract with Aguas Chañar for the supply of water for the project with a minimum supply of 80 l/s.

The following leasing and budgetary quotes are considered:

- Truck shop, 12 years leasing, ARRIGONI
- Mine equipment, 5 years leasing, KOMATSU
- Water pipeline and electrical high voltage line, 13 years leasing, ELECNOR
- Crushing equipment, budgetary quotation, METSO
- Overland and in-plant conveyors, budgetary quotation, STM
- ADR plant, budgetary quotation, DENM
- Irrigation pumps, budgetary quotation, PGIC.
- Sodium cyanide supply, budgetary quotation, OXIQUIM.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Summary

The Cerro Maricunga project is the result of successful exploration undertaken by Atacama at the Maricunga belt since 2011. Atacama submitted the Environmental Impact Declaration (DIA) of “Cerro Maricunga Mining Prospecting” Project to the Environmental Impact Evaluation System (SEIA), which was favorably qualified through the Exempt Resolution No. 232/11 by the Environmental Evaluation Commission of the Atacama Region.

The environmental components included in the present analysis are, Geology, Geomorphology and Natural Risks, Water Quality, Hydrology and Hydrogeology, Edaphology, Air Quality and Meteorology, Noise and Vibrations, Flora and Vegetation, Fauna, Limnology, Human Environment, Archaeology, Landscape, Tourism and Facilities.

The study area was defined considering basin borders, water division lines and the location of the Project main facilities.

Information available, collected from bibliographic data and field campaigns conducted during 2013, is adequate to characterize all relevant environmental components of the area. However, in order to fulfill the criteria established by the regulations in effect since December 2013, the information should be completed with the seasonality on certain components and additional studies will have to be undertaken.

Table 20-1 contains a detail of the background information that must be generated in future stages, since it will be required during the environmental evaluation process.

Table 20-1 Required Environmental Studies

Component	Proposed Activity
Noise and vibration levels	Measure sound pressure levels (background noise) in several points of the area of influence of the Project, giving special emphasis on sensitive receivers. Conduct basal vibration measurements to obtain the speed and/or the vertical acceleration at the same noise point. Authorities are requesting noise propagation models from the sources to the sensitive receivers that will be defined.
Hidrology and Hidrogeology	Conduct an hydrological and hydrogeological characterization to prepare a conceptual and numeric model of water and hydrogeological systems existing in the area of interest.
Soil	Conduct an edaphological study with test pits to characterize the soil profile, texture, pedregosity, structure, and color under wet and dry conditions, reactivity to HCl, plasticity and stickiness.
Archaeological patrimony, historical, religious and paleontological	Conduct a field campaign to identify findings with patrimonial value in the area of direct intervention of the physical works of the project.
Fauna	Conduct a characterization campaign during spring time so as to complete measurements in all four seasons.
Protected Areas and Priority Sites	Conduct a detail analysis , considering the location and the management objectives of the units included in the National System of Government Protected Areas (SNASPE) of the Region. Characterize the areas declared Nature Sanctuary pursuant to the Law of National Monuments of the Region and the Priority Conservation Sites existing in the Region. Prepare a list of other priority areas defined by the Authority for the Region.
Landscape	Conduct a thorough characterization of the landscape quality of the sector.
Tourism	Characterize the tourism activity in the area of influence of the Project. It is also recommended collecting geo-referential data of the tourism offer in the area of study, both of the official and non-official areas of tourism attractive.
Human Environment	Strength the relationship with indigenous communities that live close to the Project area, and addressing their requirements at the proper dialogue instances in order to improve their perception of the project. Update the map of key stakeholders, which provides information of the institutions, individuals or communities with decision-making power in the zone.

20.2 Relevant environmental components for the Project

20.2.1 Air Quality

Air quality conditions in the area surrounding the Project site do not show latency or saturation levels, as no emitting sources exist in the area.

According to measurements done by Atacama between October 2011 and November 2012, the average concentration of breathable particulate material (MP) in the mine area is as follows:

- MP10 = 8 µg/m³N
- MP2.5 = 1.3 µg/m³N.

20.2.2 Noise Levels

No potentially sensitive noise receivers were detected around the project site, such as human groups developing activities.

Specific measurements must be conducted in the areas where the Project main activities will be developed as well as sound pressure level projections on sensitive receivers, whether these are human groups or fauna living in the area. Bird nesting sites may exist, mainly in the proximities of Maricunga Salt Deposit and Santa Rosa Lagoon.

20.2.3 Hydrology and Hydrogeology

No permanent superficial water flows were detected in the study area. In this sector, superficial water flows are associated with melting events or with pluviometric phenomena characterized mainly by Altiplano rains during summer time; and, therefore, they are specific phenomena.

It is relevant to evaluate the behavior of the aquifers in the basin or sub-basin in the project site in order to prevent damaging the water resources of the area.

According to hydrogeological information provided by Atacama, the project area is locally underlain by aquifers of little hydrogeological importance (in agreement with the drilling campaign), made up of rock or unconsolidated deposits, essentially devoid of underground water resources.

The results of underground water quality in the sector are subject to the lithological characteristics and the rock mineralogy in the area, which natural condition show high conductivity and therefore, a great amount of dissolved solids. It is worth outlining high concentrations of Aluminum, Cobalt, Iron and Manganese.

For further information, see sections 24.4.3 and 24.4.4.

20.2.4 Surface Water

According to the results obtained in the three campaigns conducted in the study area, superficial water, according to the values established in Chilean Norm No. 1,333 for aquatic life, shows good quality, although conductivity levels exceed the values established by the norms of reference. This situation becomes more evident in the sampling stations located in the Maricunga Salt Deposit, unlike the waters analyzed in the vicinity of Cerro Maricunga.

In biological terms, phytoplankton and phytobenthos were recorded throughout the study area, with a total of 27 and 35 species, respectively. In both cases, diatoms of the Basillariophyceae type were detected.

This is one of the most common groups in the vegetable plankton. Its morphological characteristics (Silica exoskeleton) turn it resistant to adverse environmental conditions.

Zooplancton low abundance and representativeness could be associated with physical (concentrations of conductivity, total dissolved solids and low oxygen concentration) and extreme morphological conditions of the sampling stations (low depth and flow, as well as the reduced size of puddles). Presence of birds (flamencos) was noticed in all stations; flamencos feed from the filtration of this type of organisms (zooplankton) and from vegetable plankton.

In connection with ichthyofauna (presence of fish) no species were detected in the vicinities of Cerro Maricunga or the Maricunga Salt Deposit. The above could be related to the fact that the water bodies of Cerro Maricunga are closed systems. Similarly, the extremely harsh physical conditions existing in the Salt Deposit (e.g. high concentrations of dissolved salts) could be another reason that explains the lack of fish.

20.2.5 Flora and Vegetation

According to the flora and vegetation characterization results in the study area, 65 species were detected of which 57 are native, 7 are endemic and 1 is non-native. In connection with the growth species, 29% are bushes, 68% are herbs and 3% are succulent plants.

One specie classified in the official listings was detected (Maihueniopsis glomerata, Rara, Bol N°47). However, according to the reference listings in the Region, the presence of 6 species under the conservation category, were noted, all of them classified as deficiently known (according to Squeo, 2008):

- Adesmia capitellata
- Catabrosa werdermannii
- Distichlis humilis
- Hordeum halophilum
- Phacelia pinnatifida
- Senecio haenkei.

In the area where the mining and process operations are planned, seven (7) different formations and 53 species were found.

Maricunga Salt Deposit area showed the most homogeneous environment with only three (3) different formations detected, due to the homogeneity of its topography and to the humidity conditions.

The only species under official conservation category found in the study area was recorded in the sector of Santa Rosa Lagoon. Such species will not be disturbed by the project, as no activities close to this sector will be developed.

20.2.6 Fauna

According to fauna characterization results, a total of 58 species recorded. The sector with the largest abundance of species was the Maricunga Salt Deposit with a total of 35 species, whereas the Project site (mine area) showed lower number with only 5 species identified.

In connection with the conservation category of the species recorded in the study area, 23 species are within any conservation category, two of them are classified as "Endangered" according to the Hunting Law Regulation (Supreme Decree No. 5/1998 of Minagri) that correspond to *Vicugna vicugna* and *Lagidium viscacia*. One specie, the *Lama guanicoe*, is under the "Vulnerable" category according to Supreme Decree No. 33/2012 of the MMA (Ministry of the Environment). Eight species detected are under the "Vulnerable" category according to the Hunting Law Regulation, as follows:

- Ctenomys fulvus
- Phoenicoparrus andinusl
- Chloephaga melanoptera
- Vultur gryphus
- Phoenicopterus chilensis
- Phoenicoparrus jamesi
- Larus serranus
- Fulica cornuta.

Of the species classified under any category, eight are classified as "Rare" pursuant to the Hunting Law Regulation (Supreme Decree No. 5/1998 of the Minagri) that correspond to:

- Liolaemus patriciaiturrae
- Liolaemus velosoi
- Liolaemus bisignatus
- Gallinago paraguaiae
- Liolaemus atacamensis
- Liolaemus isabelae
- Liolaemus rosenmanni
- Attagis gayi.

Three species are under the category "Least Concern" were identified: *Abrocoma cinerea* according to Supreme Decree No.19/2013 of the MMA and *Pseudalopex griseus* and *Pseudalopex culpaeus* according to Supreme Decree No.33/2012 of the MMA. Identification was made of a species classified under the category "Near Threatened" (NT) that corresponds to concolor Puma according to Supreme Decree No. 42/2012 of the MMA.

According to Supreme Decree 05/1998, two bird species, rufous-bellied seedsnipe (*Attagis gayi*) and the Andean flamenco (*Phoenicoparrus andinus*) are classified as "Rare" and "Vulnerable", respectively, and were detected in the Altiplano salt deposit wetlands.

<u>Fauna Monitoring Program</u>

Atacama has Biological Monitoring Programs in place in order to protect the species identified in the Project area; the Company conducts periodic inspections of the fauna and especially the guanacos and the vicunas of the zone.

20.2.7 Archaeological Patrimony

In the area around the projected location of the pit and waste dump, the presence of 2 sites of low patrimonial value was verified, corresponding to two stone wall fenced areas with evidence of historical occupation.

In the Project prospecting area, a total of eleven (11) archaeological sites were detected. In these sites, the presence of small flood meadow corresponding to a pre-hispanic site with historical reoccupation (meadow 1) should be noted. Given the absence of cultural material on the surface it was not possible to identify and assign a cultural period to three (3) stone wall structures and one (1) rocky shelter. However, their conservation status characterized by collapsed stone walls could reflect an older age as compared to structures with sub-current or more recent occupation.

20.2.8 Landscape and Tourism

The area surrounding the project shows a high landscape and tourism quality, and it is part of the main tourism circuits of the region.

Several attractive area from a tourism perspective, were noted, such as the P.N. Nevado Tres Cruces overlook, Portezuelo de Maricunga, Santa Rosa Lagoon overlook, Virgen de La Candelaria, Salar de Maricunga overlook. The overlooks offer a good view of the surrounding landscape.

In connection with well-known tourism resources in the area, it was recognized the Tourism Zone of Interest Salar de Maricunga – Volcán Ojos del Salado (Res. 662 10/07/2006), the Priority Tourism Area Ojos del Salado – Cordillera de Atacama, Nevado de Tres Cruces National Park; the tourism circuit La Puerta –Santa Rosa Lagoon –Nevado Tres Cruces National Park, the Cordilleran-Seaside Patrimonial Route: El Derrotero de Atacama Circuit, including Milestone 5: Portezuelo de Maricunga, Milestone 6: Santa Rosa Lagoon overlook, MIlestone 7: Virgen de la Candelaria and Milestone 8: Salar de Maricunga Overlook, both circuits run across road C- 601 and road 31 CH. In addition to the macro-section of Sendero de Chile Ramal Molle – Diagüita and the sections: Manflas – Salar de Maricunga and Salar de Maricunga – San Francisco passage. Also, this sector holds three poles of tourism attractive defined by SERNATUR: Santa Rosa Lagoon (international hierarchy), Maricunga Salt Deposit (regional hierarchy) and Nevado de Tres Cruces National Park (international hierarchy). The main trekking routes of Nevado de Tres Cruces National Park are: Sendero de Chile Santa Rosa Lagoon Stretch –Maricunga Salt Deposit, Sendero de Chile Negro Francisco Lagoon Stretch.

20.2.9 Human Environment

Pastures, flood meadows and areas with relatively plenty of potable water available are used by the local Colla communities for pasturing their livestock and, the Colla inhabitants have traditionally built their homes adjacent to these areas, such as the San Andrés Ravine's Codocedo Slope (road Ch 31).

Seasonal variations, droughts and other factors force Colla inhabitants to occupy a section of the ravine for specific periods and then move to another area. Sometimes they spend periods of the year in other parts of the region, working in mining activities and taking care of school age children.

According to Colla communities, the territory traditionally travelled and used starts in the ruins of Puquios town, at the bottom of the ravine, extending to the east, and going up by the Paipote and San Andrés ravines to get to the Puna highlands. In longitudinal terms, by the North this territory starts in Pedernales Salt Deposit and extends to Negro Francisco lagoon, by the South.

However, these borders are relative because, as is the case with other Andean peoples, Colla groups' occupation of the territorial space is discontinuous and scattered. The image of this space varies according to each community's perspective, as a reflection of its historical use (their grandparents' tradition) and of its current use (linked to ceremonial practices and the migrating habits of the residents).

In connection with the territorial distribution of the sites of cultural importance for the community, it is worth mentioning the little importance given by this community to Road 31-CH. It could be mentioned that most of them are grouped around the communication paths in the area, namely Roads C-341, C-601 and C-607.

Human Environment Monitoring Program

Due to Colla's negative perception of mining activities it is important to build trust relationships taking into consideration the opinion of the communities interested, both indigenous communities and other local stakeholders. The communities are organized and aligned regarding the benefits they expect to achieve as a result of the development of mining projects in what they consider their own territory by ancestral right. However, a proper identification of indigenous groups and the construction of instances of dialogue will allow building confidence and reaching agreements that are beneficial both for the Project and the communities.

20.3 Protected Areas and Priority Sites

The following protected areas and priority sites have been defined within the Project site:

- Nevado Tres Cruces National Park. 77 fauna species, organized in 26 families and 49 genders have been identified. Of these 62 are birds, 11 are mammals and 4 are reptiles. In connection with flora, 65 species were found, which may be grouped in 23 families and 42 genders
- RAMSAR Site: Laguna del Negro Francisco and Laguna Santa Rosa Lake Facilities corresponds to two High Andes lagoons united by the Pantanillo- Ciénaga Redonda biological corridor. The site operates as an important regulator of the biotic and abiotic components that make up the ecological network of this Andean ecosystem, which supports species such as the Andean seagull

(Larus serranus) and the Vicuna (Vicugna vicugna) classified as Vulnerable pursuant to the Chilean law, and at least 1% of the total population of Parina grande (Phoenicoparrus andinus), Parina chica (Phoenicoparrus jamesi) and Tagua cornuda (Fulica cornuta).

20.4 Potential Emissions, Waste and Effluents Generated by the Project

The development of Project works will generate emissions, effluents and waste in all its stages.

20.4.1 Atmospheric Emissions

The Project will generate emission of breathable particulate and sedimentary material as a result of the typical activities, such as construction of roads, massive earthworks, construction of foundations for the different works, passage of vehicles and machinery by unpaved roads, transportation of personnel and materials, ore extraction and hauling, loading and unloading of trucks with minerals and waste, ore crushing.

20.4.2 Noise and Vibrations

During the construction stage, noise will be generated by heavy duty machinery and by the passage of mine trucks performing earthworks as a result of the pre-stripping activity at the mine and the haulage of material to the waste dump. However, noise and vibrations will be temporary and will come mainly from movable sources.

During the operation stage, noise will be the result of the passage of vehicles, mine equipment, blasting, loading and unloading activities, crushers, etc. These emissions are typical of the ore mining activity and will be confined to the industrial operations area. Among them, blasting is the most noise intensive activity; however, this is a short-term and specific activity which is normally conducted once or twice a day depending on the operation program defined.

20.4.3 Mine Waste

The mine waste (waste rock) generated by the project during construction and operation stages will be disposed at the waste dump.

Due to the nature of the process, static heap leaching pad, the waste generated will be managed on site.

20.4.4 Industrial Waste

Hazardous and non-hazardous industrial waste will be generated during construction and operation of the project.

20.4.5 Residential Waste

Solid residential waste will be generated in all Project stages, mainly resulting from the presence of people performing activities in the area, such as liquid industrial waste due to the use of sanitary services in the area.

20.5 Closure and Abandonment Stage

20.5.1 Closure Plan

An essential part of the Project is the development of a closure plan that outlines activities for decommissioning and mitigation of impacts during operation and closure. The preparation of a closure strategy prior to the development of the Project is an integral part of the closure design process. This approach to Project planning recognizes that mining represents a temporary use of land and that appropriate closure of the operation is in line with the sustainable use of available resources.

The Project closure plan will focus on safety, stabilization of the land surfaces, post mine utilization of facilities and structures and protection of the environment. Since the Project is located in an extreme arid, high altitude environment, re-vegetation is considered impractical and not conducive to the surrounding environment.

In Chile, there are clear and precise rules regarding the closure of mining facilities (Regulation on Mine Safety No. 72. section 5), which indicate the activities required to carry out the closure of a mining project. The following is a summary of the objectives of the Regulation, as well as the activities listed.

- Ensure that the remaining facilities will not affect human health or degrade the environment
- Ensure maintenance of physical stability and that the areas affected by mining activities are in stable condition at the closure of the project
- Ensure the maintenance of stability associated with chemicals in the long term, in order to reduce effects on biological diversity and to avoid endangering public health and safety
- Ensure environmental components, both surface and underground are not affected as a result of the closure.

The reclamation and closure activities will include removal of all buildings, power lines, pipe lines and process components, securing the pit and waste rock storage facilities, ensuring that the spent leach pad and tailings storage facility are chemically and structurally stabilized, and returning the area to its previous land use. To the extent possible, reclamation will be carried out concurrently with operations.

20.5.2 Post-closing Stage or Abandonment

After the closure, it is necessary to follow and monitor all environmental and physical variables, with the purpose of verifying the correct performance of the plan and if any contingent event were to happen, adopt the necessary corrective measures.

20.6 Summary of Main Environmental topics for the Project

The Project has design characteristics that will allow reducing potential impacts such as the extraction of water from the sector, as water will be supplied by third parties. This situation is highly favorable because no potential impacts will be generated on the aquifers of the sector from water supply, taking into consideration the ecosystem dynamics in the areas close to the Project such as Nevado Tres Cruces National Park, RAMSAR site Santa Rosa and Negro Francisco Lagoons and Maricunga Salt Deposit.

No significant environmental issues have been identified which could hamper or halt the development of a mining and associated heap leach processing facility at Cerro Maricunga. The use of cyanide in the leach pad and the containment of solutions have been identified as a concern that has been addressed in the current report.

To accommodate to the current SEIA regulation, it is necessary to conduct an ecosystem study of sensitive areas around the Project. This ecosystem study will allow assessing the environmental conditions of the area and their interaction with project activities, which will contribute to the design of future environmental management measures to protect these ecosystems, if necessary.

Since no population exist in the proximity of the project area that could be affected by the activities undertaken therein, there is no risk of having the zone be declared saturated or latent due to emissions of particulate material and gases. The same happens with noise emissions and the release of effluents or the generation of waste, whether it is residential, industrial or hazardous waste. However, the authorities are now requesting data that enable them to verify the effects or the absence thereof by means of emissions modeling. Consequently, the Atacama should conduct studies to assess the impacts.

During the next stage of the project, it is recommended that the Environmental Authorities and the neighboring communities be engaged to reinforce the relationship and to facilitate the communication during the environmental evaluation. It is fundamental to maintain good relationships with the neighboring communities to enhance communications and facilitate the environmental permitting. It should be noted that Atacama has an active community relations program and has established good relations with the local communities.

The mining operation will not cause an alteration in the lifestyle or the customs of the inhabitants surrounding or their dwellings; no cultural or anthropological changes are foreseen in the human groups indicated above. In turn, the project will generate jobs for the local work force.

21 CAPITAL AND OPERATING COST

21.1 Capital Costs

The capital and operating cost estimates for the PFS were developed in Chilean Pesos (CLP) and United States Dollars ($) according to source currency of costs. The exchange rate (CLP/$) used is 600. All costs for the PFS are estimated as of the Effective Date of this Technical Report. All cost projections are presented on a nominal dollar basis.

Table 21-1 summarizes the total capital costs estimate for the project. This estimate is based on information developed during the PFS.

The purpose of this estimate is to define the total cost of the project to verify its economic viability. The estimate has an accuracy of -5% to +20% with a 90% probability of occurrence.

Table 21-1 Capital Costs Summary

Area	Initial ($ Millions)	Sustaining ($ Millions)	Total ($ Millions)
Mining	42.4	6.3	48.7
Process Plant	252.4	152.5	404.9
Support Facilities	44.9	0.9	45.9
Owner Costs	12.7	0.0	12.7
Closure Costs	0.0	5.0	5.0
Contingency	46.5	23.0	69.5
Total	398.9	187.7	586.6

21.1.1 Mining

The estimated mine capital cost includes the following items:

- Mine pre-production development expense
- Main mine equipment monthly leasing payments during pre-production period
- Mine heavy vehicle work shop and warehouse monthly leasing payments during pre-production period
- Mine support equipment
- Mine equipment rebuild
- Other investment
- Dispatch system.

The main mine equipment fleet and heavy equipment work shop are considered to be acquired via leasing agreements. Capital costs estimate only include the monthly payments for both items during the pre-production period.

The mining estimate does not include fuel, lubricant and explosive storage facilities, considering that suppliers will provide storage for their items as part of their work contract.

Table 21-2 summarizes the mine capital costs by category for the initial and sustaining capital.

- The initial project period includes all funds spent prior to the operational start-up of the process plant
- The initial capital period includes mine pre-production development of $ 15.3 million for pre-stripping and ore stockpiling as shown in Table 21-2
- Mine equipment cost for pre-production (pre-stripping and ore stockpiling) is $ 27.1 million as shown in Table 21-2 which includes initial leasing payments, support equipment, other investment and dispatch
- Sustaining capital expenditures have been estimated to assure the continuity of the mining operation, considering equipment replacement and rebuild, totaling $ 6.3 million through year 13
- The increase in the equipment fleet to achieve the required material movement is reflected on the leasing payments.

Table 21-2 Mining Capital Costs Summary

Area	Initial ($ Millions)	Sustaining ($ Millions)	Total ($ Millions)
Mine development	15.3	0.0	15.3
Equipment leasing - 1st payment	6.5	0.0	6.5
Truck shop leasing - 1st payment	1.0	0.0	1.0
Support equipment	3.9	0.8	4.7
Equipment rebuild	0.0	3.8	3.8
Other investments	13.3	0.5	13.8
Dispatch	2.4	1.1	3.5
Total	42.4	6.3	48.7

The following should be noted:

- Base equipment prices are shown in constant 2nd quarter 2014 US dollars. It is assumed that payment for the equipment is made at the time of delivery
- The costs for the major equipment are based on quotes obtained by NCL for this project during second quarter 2014. The costs for minor equipment are based on NCL experience with recent similar projects
- Equipment costs shown include delivery to the site and assembly
- It is likely that final fleet price will be somewhat lower than the budget quotes used for this study, due to closing business discounts.

Table 21-3 shows mine capital expenditure by year.

Table 21-3 Mining Capital Costs by Year

Item	Y 00	Y 01	Y 02	Y 03	Y 04	Y 05	Y 06	Y 07	Y 08	Y 09	Y 10	Y 11	Y 12	Y 13	Total
	($ Millions)														
Mine development	15.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	15.3
Equipment leasing - 1st pay	6.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	6.5
Truck shop leasing - 1st pay	1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.0
Support equipment	3.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.5	0.2	0.0	0.0	0.0	0.0	4.7
Equipment rebuild	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.7	0.7	0.4	0.0	0.0	3.8
Other investments	13.3	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	13.8
Dispatch	2.4	0.5	0.4	0.1	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	3.5
Total	42.4	1.1	0.4	0.1	0.0	0.1	0.0	0.0	0.5	2.9	0.7	0.4	0.0	0.0	48.7

21.1.2 Process Plant and Support Facilities

Process plant and support facilities capital cost estimates are composed of the following:

- Direct cost of construction and assembly: Acquisitions of equipment supply, labour, auxiliary equipment for construction and building materials are considered
- Indirect project costs: Transportation and equipment insurance, general spare parts, vendor's representatives, detailed engineering, EPC, start up and owner costs are considered
- Contingency estimation based on Direct Cost plus Indirect Cost
- Sustaining costs: Required to maintain operations, may include capital spent on expansion or new infrastructure items.

Table 21-4 Process Plant Capital Costs Summary

Area	Initial ($ Millions)	Sustaining ($ Millions)	Total ($ Millions)
Primary Crushing	15.3	0.0	15.3
Overland Conveyor - Section 1	21.8	0.0	21.8
Coarse Stockpile	18.9	0.0	18.9
Secondary / Tertiary Crushing	38.6	0.0	38.6
Tertiary Screening	18.3	0.0	18.3
Overland Conveyor - Section 2	16.0	0.0	16.0
Fine Stockpile	14.5	0.0	14.5
Heap Leaching	83.7	152.5	236.2
ADR - EW	23.3	0.0	23.3
First Fill	2.0	0.0	2.0
Total	252.4	152.5	404.9

Table 21-5 Support Facilities Capital Costs Summary

Area	Initial ($ Millions)	Sustaining ($ Millions)	Total ($ Millions)
Plant Roads	1.9	0.0	1.9
Plant Offices	0.6	0.0	0.6
Support Vehicles	2.5	0.0	2.5
Laboratory	0.7	0.0	0.7
Operations Camp	1.7	0.0	1.7
Water Treatment Plant	0.2	0.0	0.2
Fire Water & Process Water Distribution	2.5	0.0	2.5
Reagents Plant	5.8	0.0	5.8
Power Distribution	19.0	0.0	19.0
Waste Dump Diversion Channel	0.0	0.9	0.9
Access Road	10.1	0.0	10.1
Total	44.9	0.9	45.9

Direct Costs

The main items considered to calculate the Direct Costs for the process plant and support facilities are as follows:

Earthworks

The site earthwork, and general improvements take-offs were based on the PFS level civil sketches. Material take-offs ("MTOs") were prepared using an industry standard software and spreadsheets. Quantities were reviewed by both the responsible civil engineers and estimators. In general, the areas requiring mass earthworks will be mass excavated and then back-filled with suitable fill. The estimate assumes that suitable fill material will be found on site and used for common and structural backfill. MTOs are net quantities.

Civil Works and Architecture

Civil works and architectural MTOs were prepared by Alquimia civil engineers from PFS level civil sketches. In general, this area includes all kind of covers for ponds and the pad, not industrial buildings, offices and laboratories enabling, sanitary facilities, etc. MTOs are net quantities.

Structural Steel

Structural steel take-offs were obtained from the preliminary structural steel drawings. The take-offs were developed by Alquimia structural engineers. The structural steel take-offs categorizes structural steel by the following weight classifications: Extra Heavy, Heavy, Medium and Light. Steel structures items such as grating, handrails, and stairs, are factorized from main structures quantities. MTOs are net quantities. It is assumed that all steel is sourced locally.

Doors, gates, and finishing for the buildings are factored. All other miscellaneous buildings are priced on a dollar per square metre basis using historical data.

Concretes

Concrete quantities were determined from material take-offs prepared by Alquimia engineers from PFS level sketches. MTOs are net quantities.

Piping, Pumps and Tanks

Piping material take-offs were based on PFS level layout sketches. MTOs are net quantities.

Mechanical Equipment

Mechanical Equipment has been calculated as described in Section 17, by Alquimia's process engineers and utilizing its own registered methodology. Table 21-1 shows quotation prices for all major mechanical process equipment. It is likely that final purchase prices will be somewhat lower than the budgetary quotes used for this study, due to closing business discounts.

Table 21-6 Budgetary Quotations for Process Mechanical Equipment

Proponent	Equipment	Unit Price ($ Millions)	Quantity	Total ($ Millions)
Metso	MK-II Primary Gyratory Crusher 62x75	4.2	1.0	4.2
Metso	MP1000 Cone Crusher	3.4	2.0	6.7
Metso	MP800 Cone Crusher	2.8	3.0	8.3
Denm	ADR Plant	13.0	1.0	13.0
STM	Overland Conveyor	4.2	3.0	12.5
Metso	Screen RF-3073-2	0.3	2.0	0.6
Metso	Screen RF-3061-2	0.3	3.0	0.9
PGIC	Centrifugal vertical pump CNP 350 VTC 1000	0.3	12.0	3.4
STM	Grasshopper conveyors	0.3	28.0	9.2
Total				58.7

Electrical and Instrumentation

Electrical MTOs were prepared by Alquimia electrical engineers based on PFS level single line diagrams. Material take–offs are net quantities.

MTOs for instruments and control valves were factored based on mechanical process equipment.

Table 21-7 Direct Costs by Construction Item

Commodity	Initial ($ Millions)	Sustained ($ Millions)	Total ($ Millions)
Earthworks	36.4	26.1	62.5
Civil / Arquitectural	13.1	16.2	29.3
Structural Steel	28.0	0.0	28.0
Concretes	33.8	0.8	34.5
Piping, Pumps and Tanks	27.4	103.1	130.5
Mechanical Equipment	99.3	5.2	104.5
Electrical & Instrumentation	15.8	0.0	15.8
Total	253.7	151.3	405.1

Growth factors were applied in order to cover omissions, rework, losses and others.

Construction equipment and material usage costs for the Project include fuel and maintenance. The costs for equipment operators are incorporated into the unit man-hour rate of the work performed. Equipment erection data were obtained from recent or on-going projects of similar process type and geography.

Estimated construction man-hours were obtained from recent projects in Chile. Site-specific conditions considered for productivity includes, but are not limited to altitude, weather, skills availability, camp distance and construction equipment usage. Wages – all in salaries - were obtained from Chilean construction contractors and historical data from Alquimia while crew composition was prepared based on Chilean practices for similar projects.

Pricing for the remaining equipment and material were either budgetary-quoted or estimated from in-house databases from recent similar projects.

<u>Indirect Costs</u>

Freight and Insurances

The freight cost includes inland freight, port handling, forwarding fee, ocean freight, duties and local freight. Cost for the contract is included in the indirect costs. The freight cost was factored by cost of material and process equipment supply both local and foreign as follows:

- 8% of the FOB cost for all imported equipment and materials
- 2% of the ex-factory cost for all national equipment and materials.

For all imported equipment and materials, 4% of the price was considered as internment rights and insurances.

Import Duties

Import duties were factored as 4% of the imported equipment supply cost.

Spare Parts

One year of spare parts were estimated as 5% of the mechanical equipment total supply cost.

Vendor Representatives

The vendor cost was factored as 2.5% of the cost of all major equipment supply.

Engineering, Procurement, and Construction (EPC)

The project execution will materialize under one or more EPC contracts.

- Engineering considers Basic, Detail and On-site engineering, including project management, engineering, cost engineering and scheduling, estimating support, accounting and construction contract planning
- Procurement includes purchasing from local Chilean as well as off-shore sources, including activities like purchasing enquiries, negotiating terms and conditions, placing purchase orders, providing logistics and traffic control, expediting equipment and materials and inspecting purchased equipment and materials
- Among the construction activities not included in the Direct Costs are: preparing the construction schedule, on-site material handling, on-site client communications, valuation and cash flow control, and closing.

These EPC activities cost has been defined as 10% of direct plus indirect costs.

Start-up and Pre-Commissioning

Start-up and pre-commissioning costs include craft labour support, field engineering, and supervision. Costs for these activities were defined as 0.5% of the total cost of equipment, construction and assembly of the plant.

First Fill

First fill consists of supplying chemicals and lubricants for the plant and process equipment, which are required for process plant start-up. The first fill cost was calculated considering two weeks of supply of the major reagents.

Table 21-8 Indirect Costs Summary

Area	Initial ($ Millions)	Sustaining ($ Millions)	Total ($ Millions)
Freight & Insurance	5.0	1.6	6.6
Import Duties	1.4	0.0	1.4
Spare Parts	5.0	0.2	5.2
Vendors Representatives	2.5	0.1	2.6
EPC	26.8	0.0	26.8
Start Up	1.1	0.0	1.1
First Fill	2.0	0.0	2.0
Total	43.6	1.9	45.6

21.1.3 *Leasing*

The project execution considers a leasing scheme contracts for mine equipment, truck shop, water supply and power supply. Table 21-9 shows leasing cost by year and contract.

Table 21-9 Leasing Cost by Year

Item	Y 01	Y 02	Y 03	Y 04	Y 05	Y 06	Y 07	Y 08	Y 09	Y 10	Y 11	Y 12	Y 13	Total Cost
	($ Millions)													
Mine Fleet	28.2	38.6	41.1	38.0	40.3	21.2	10.6	4.7	3.6	1.7	0.0	0.0	0.0	227.9
Truck Shop	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	1.4	0.0	27.4
Power Supply	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	65.0
Water Supply	17.5	17.5	17.5	17.5	17.5	17.5	17.5	17.5	14.9	13.5	12.7	11.3	10.0	202.4
Total	53.1	63.4	66.0	62.9	65.2	46.0	35.5	29.6	25.9	22.5	20.1	17.7	15.0	522.7

<u>Mine Equipment and Truck Shop</u>

Komatsu has provided a quotation for a $234.4 million lease to own arrangement for the major mining fleet equipment. The quote is for a lease period of 5 years for the drilling/loading/hauling fleet with a final payment in following year for the purchase of the fleet. The auxiliary equipment lease is for a period of 3 years with a payment in the fourth year for final purchase. First payment of $ 6.5 million is included in mining capital cost.

The construction of a truck maintenance shop will be undertaken by ARRIGONI under a $28.4 million leaseback over a period of 13 years. First payment of $ 1.0 million is included in the mining capital cost.

The total leasing payments cost during the production period (after the start of metal production) is $ 255.3 million. This amounts to 0.32 $/t of mined material and 0.87 $/t ore during this period.

<u>Water and Power Supply Infrastructure</u>

Atacama has been provided a quotation from ELECNOR, a multinational engineering and construction firm, for the construction and operation of the water supply infrastructure from Copiapo to Cerro Maricunga. The annual cost of the lease is $17.5 million for the first 8 years of full production and declines from year 9 as the water consumption lowers consistent with the lower ore throughput. The same firm has offered to construct and operate the electrical distribution system at a price of $5 million per year.

Unitary leasing costs are as follows:

- Power supply infrastructure: $ 0.22 per tonne processed
- Water supply infrastructure: $ 0.69 per tonne processed.

21.1.4 Accuracy

The cost estimates have been developed to a level sufficient to assess/evaluate the project concept and verify its overall potential economic viability. The capital cost estimate is considered to have a level of accuracy of - 5% and + 20% with a 90% probability of occurrence. This is based on the level of contingency applied, the confidence levels of the authors in their respective estimates, and an assessment comparing this estimate to standard accuracy levels for PFS estimates.

21.1.5 Contingencies

The contingency is an amount added to an estimate to allow for unforeseen events, conditions, or occurrences that experience indicates will likely happen during a project. The amount of contingency applied to the project was determined by experience from similar projects. The following events were excluded from contingency analysis:

- Scope change
- Substantial design change
- Force majeure events
- Acts of war
- Labour conflicts
- Change in execution plan
- Insurance deductibles
- Escalation
- Currency effects
- Change in mechanical soil conditions.

21.1.6 Owner Cost

The owner cost has been factored as 15% of the direct cost of the process plant and support facilities, and covers mainly the owner team for the construction management.

21.1.7 Estimate Exclusions

The following items are not included in the capital estimate:

- Costs incurred prior to commencement of any prefeasibility study
- All Owner's taxes, including any financial transaction tax, withholding tax, or value-added tax (VAT)
- Future foreign currency exchange rate fluctuations
- Interest and financing costs
- Escalation beyond first-quarter 2014

- Risk due to political upheaval, government policy changes, labor disputes, permitting delays, weather delays, or any other force major occurrences.

21.2 Operating Costs

Operating costs have been estimated for the operating areas of Mining, Process Plant, and G&A (General and Administrative). Costs were reported under subheadings related to the function of each of the areas identified.

Table 21-10 summarizes total operating cost by area, as well as average unit operating cost. Labor costs for mine and process plant consider only up to the level of Manager Level (Mine or Plant). All other senior positions are considered as administration costs.

The operating costs are considered to have accuracy of - 5% and + 20% with a 90% probability of occurrence, based on the assumptions listed in this section of the Report. All unitary operating costs are expressed in processed tonnes.

Table 21-10 Operating Costs Summary

Area	Total Cost ($ Millions)	Unit Cost ($/t*)
Mining	1,124.5	3.82
Process Plant	740.9	2.52
G&A	159.0	0.54
Total	2,024.4	6.88

*$/t: Cost per tonne of ore processed

21.2.1 Mining

Table 21-11 summarizes the mine operating costs and unitary costs per tonne of total mined material.

- The $ 15.3 million shown in Table 21-11 is the pre-production development cost shown in Table 21-2. This cost is considered capital cost and amounts to 1.39 $/t of mined material during this period
- The total mine operating cost during the production period (after the start of metal production) is $ 1,124 million. This amounts to 1.40 $/t of mined material and 3.82 $/t of ore during this period
- During peak commercial production in years 1 through 9, the operating costs range from a low of $ 67.4 million in year 1 to a high of $ 123.4 million in year 7.

The following factors are considered for the operating cost calculations:

- In country costs for consumable items such as diesel fuel, blasting agents, tires and spare parts
- Hourly labour rates and fringe benefits were used.

Table 21-11 Mining Operating Costs Summary

Item	Pre-Production*		Production	
	($ Millions)	($/t)	($ Millions)	($/t)
Loading	2.1	0.19	209.5	0.26
Hauling	4.2	0.38	452.7	0.57
Drilling	1.2	0.11	92.5	0.12
Blasting	2.8	0.26	158.0	0.20
Ancillary	2.0	0.18	108.2	0.14
Support	1.1	0.10	39.3	0.05
Eng. & Admin.	1.9	0.17	64.3	0.08
Total	15.3	1.39	1,124.5	1.40

(*) Included in Capital Costs

21.2.2 Process Plant

Process plant operating costs incorporate labour, energy, maintenance, contracts and reagents and supplies. These operating costs were adjusted for local labor rates and supply costs, while tracking recent experience for projects with similar equipment.

Table 21-12 presents process plant operating costs by area and Table 21-13 by expense item. Leasing costs are not included.

Table 21-12 Process Plant Operating Costs by Area

Area	Total Cost ($ Millions)	Unit Cost ($/t*)
Primary Crushing	63.7	0.22
Secondary / Tertiary Crushing	119.2	0.40
Heap Leaching	472.1	1.60
ADR - EW	85.9	0.29
Total	740.9	2.52

*$/t: Cost per tonne processed

Table 21-13 Yearly Process Plant Operating Costs by Item

Item	Y 01	Y 02	Y 03	Y 04	Y 05	Y 06	Y 07	Y 08	Y 09	Y 10	Y 11	Y 12	Y 13	Total	Unit Cost
	($ Millions)														($/t)
Labour	5.8	5.8	5.8	5.8	5.8	5.8	5.8	5.8	5.1	4.3	4.2	2.3	1.6	63.9	0.22
Energy	11.4	12.4	13.4	13.9	13.9	13.9	13.9	13.9	9.5	7.4	6.1	4.0	2.0	135.9	0.46
Water	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	1.5	1.1	0.9	0.6	0.3	21.7	0.07
Reagents	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	25.7	19.9	16.4	10.8	5.4	378.4	1.29
Maintenance	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	5.0	3.9	3.2	2.1	1.0	73.6	0.25
Others	6.4	6.5	6.6	6.7	6.7	6.7	6.7	6.7	4.7	3.7	3.1	2.0	1.0	67.4	0.23
Total	70.6	71.7	72.8	73.4	73.4	73.4	73.4	73.4	51.6	40.3	33.9	21.7	11.3	740.9	2.52

Including the leasing cost of power and water supply infrastructure to the operating cost, the cost breakdown results as shown in Figure 17-1.

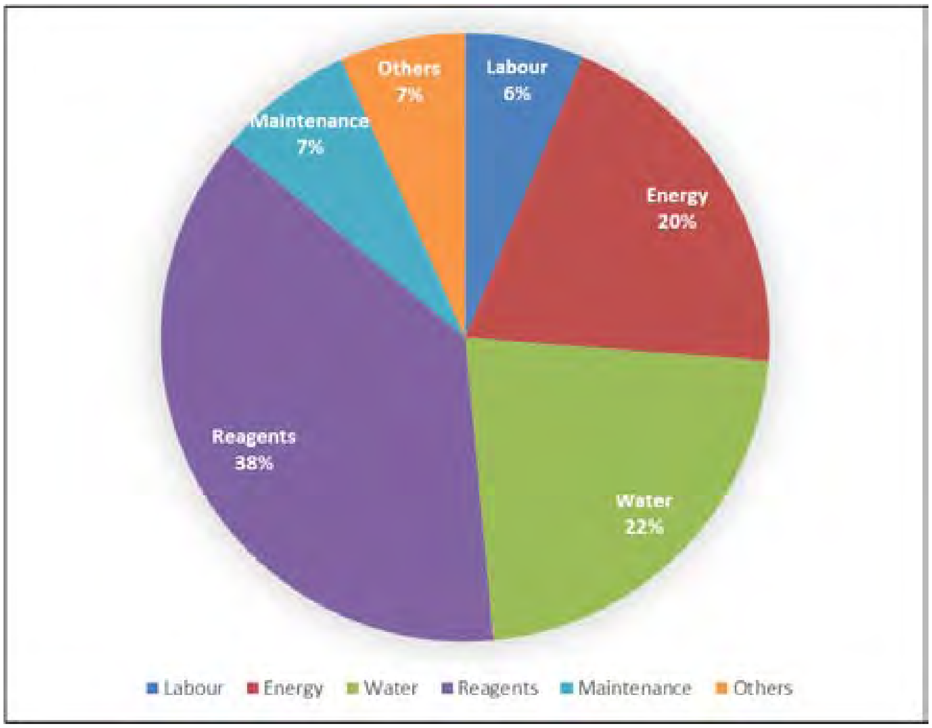

Figure 21-1 Process Plant Operating Cost Distribution by Item

Labour

Labour costs were estimated considering the organizational structure of operation, maintenance, supervision and administration. Table 21-14 shows the labor requirements considered for the process plant and the pay scale.

Table 21-14 Process Plant Labour Requirements and Salaries

Position	Quantity	Salary ($/y)
Plant Manger	1	200,000
Superintendent	3	80,000
Professional	7	60,000
Operator	70	40,000
Mantainer	12	40,000
Administrative	4	30,000

For all workers, excepting the General Manager – a replacement factor was considered which accounts for shifts and vacations. This factor is different for operators, maintainers, supervisors and administrative personnel and was obtained from similar projects benchmarking.

Energy

Global energy consumption was calculated considering the nominal power consumption of each mechanical equipment with its utilization, efficiency and demand factors.

The increase of 2,000 MW in the Chilean central power grid (SIC) and its connection with the power grid for the northern zone (SING) ensures energy availability for Cerro Maricunga project.

Regarding energy price, the interconnection is expected to draw prices dawn, thus by year 2018 prices of 9.0 $/MWh are forecasted for the zone. However, for the calculation of the operating cost, an energy price of 10.0 $/MWh, was used.

Reagents and Consumables

Consumptions considered for the Project were estimated by Alquimia using testwork data discussed in Section 13. Unit prices are based on referential quotations from suppliers. Table 21-15 shows the main reagents and consumable prices considered.

Table 21-15 Process Plant Reagent Consumption and Prices

Item	Consumption		Price	
	IP	Value	Unit	Value
Crusher Liner	g/t	45	$/t	2,000
Sodium Cyanide	g/t	234	$/t	2,650
Lime	g/t	2,700	$/t	178
Carbon	t/d	1.3	$/t	4,500
Borax	kg/ t ppt*	300	$/t	1,260
Sodium Nitrate	kg/ t ppt*	1,500	$/t	1,100
Silice	kg/ t ppt*	1,250	$/t	83
Sodium Hydroxide	g/t	10	$/t	830
Hydrochloric Acid (32%)	g/d	2.5	$/t	210
Energy	MWh/LoM	1,358,842	$/MWh	100
Water	m^3/d	7,877	$/$m^3$	0.75

*t ppt: Tonnes of precipitate to smelting

Maintenance

Maintenance costs were factorized as follows:

- A 5% of the equipment direct cost for crushing, material transportation and ADR
- A 20% of the cost associated to the power consumption of the pumps for the leaching area.

21.2.3 General and Administration Expenses

G&A expenses consider the administrative costs of the process plant, laboratories and warehouse. Senior administration and project overhead cost are also included here.

Table 21-16 G&A Operating Expenses

Item	Total Cost ($ Millions)	Unit Cost ($/t)
Operations Camp	80.5	0.27
Administration Staff	18.0	0.06
Central Laboratory	3.6	0.01
Communications	2.2	0.01
Security Items	0.9	0.00
Administration Vehicles	1.4	0.00
Capacitations	2.4	0.01
Community Relations	5.0	0.02
Sales and Marketing	7.6	0.03
Contracts/Logistic/Storages	2.6	0.01
Offices Maintenance	1.9	0.01
Expert Consultancy	6.4	0.02
Insurances	10.1	0.03
Link Office (Copiapó-Santiago)	1.2	0.00
Roads Maintenance	10.1	0.03
Computers and Softwares	5.0	0.02
Total	159.0	0.54

21.2.4 Cash Cost

Total cash costs include mine site operating costs (mining, processing, G&A, royalties and production taxes).

Total site cash cost is the sum of total cash costs, sustaining capital expenditures, capitalized and expensed exploration that is sustaining in nature and environmental reclamation/closure costs.

Table 21-17 Cash Cost Indicators

Item	Total Cost ($ Millions)	Unit Cost ($/oz)	Unit Cost ($/t)
Cash Cost	2,024	683.3	6.88
Transport + Insurance	14	4.6	0.05
Infrastructure Leasing	267	90.3	0.91
Mine Fleet Leasing	255	86.2	0.87
Total Cash Cost	2,561	864.3	8.70
Sustaining Capex	188	63.3	0.64
Total Site Cash Cost	2,748	927.6	9.33

22 ECONOMIC ANALYSIS

The economic analysis is based on the estimated Capex and Opex and revenue calculated thereof. The Capex and Opex were developed as noted in Section 21. This section presents the projects revenues and economic analysis.

The economic analysis has been calculated before and after taxes. As Alquimia is not a financial advisor, after-tax figures should be confirmed with a recognised tax expert. Sensitivities based on commodity price, metals recovery, operating cost and capital expenditures variation are highlighted in Section 22.6.

The PFS calculates an after-tax $NPV^{5\%}$ of $409.3 million and an IRR of 25.0% at a $1,350/oz gold price. The after tax payback period of capital is calculated to be 3 years.

Since the analysis is based on a cash flow estimate, it should be expected that actual economic results may vary from these results. The PFS has been completed to a level of accuracy of -5% and +20% with a 90% probability of occurrence.

22.1 Methodology Used

Alquimia has estimated the Project's net present value based on a discounted cash flow model, which uses mine production schedule, gold grades, estimated gold recoveries, and capital and operating costs as input, to calculate Project's NPV to the annual period prior to initial mining capital expenditure.

22.2 Financial Model Parameters

Chile is a politically stable country and the Cerro Maricunga Project has similar technical features as several other projects or operations in Chile. The major sources of financial uncertainties are metal price and cost escalation.

The exchange rate is not a direct input in the financial model since all the input costs are converted to United States Dollars ($). For fraction of costs received in Chilean pesos (CLP) Alquimia applied an exchange rate of 600 Chilean Pesos (CLP) per $ in the cost estimation.

Economic parameters used for the evaluation are shown in Table 22-1.

Table 22-1 Main Economic Parameters

Item	Unit	Value
Economics		
Au Price	$/oz	1,350
Discount Rate	%	5.0
Taxes & Royalties		
State Royalty	%	Variable
Income Tax Rate	%	20
Bullion Terms		
Payable Au	%	100.00
Transport & Insurances	$/t	0.046

22.3 Taxes and Royalties

22.3.1 Income Tax

The Chilean tax code applicable to mining companies and mining operations is complicated. Consequently the following is not a comprehensive review of all potential Chilean income tax requirements and is only a general summarized description of the Chilean income tax code applicable to mining companies.

The corporate income tax legislation provides for a system divided as follows:

First Category Tax

First Category Tax is due on income derived from commercial, industrial and agricultural activities; mining, fishing and other extractive activities; investment; and real estate. At the date of this Technical Report the corporate tax rate is 20% and affects all taxpayers which carry out these activities.

The new Chilean administration has recently approved a bill that changes the tax regime, which includes an increase in the first category income tax rate on corporate income over a four-year period, as follows: 21% in 2014, 22.5% in 2015, 24% in 2016 and 25% in 2017. These changes may affect the economic analyses presented in this Technical Report.

Additional Tax (Impuesto Adicional)

This tax operates as a withholding tax and affects, amongst others, Chilean-source income withdrawn or remitted abroad to non-residents or non-domiciled individuals, companies or other entities organised abroad with or without a permanent establishment in Chile in the form of branches, offices, agencies or representatives. Dividends paid to the shareholders not domiciled or resident in Chile are subject to a withholding tax on distribution at a rate of 35%.

However, if the distributed amounts had been subject to First Category tax, this is given as credit against the additional tax such that the total effective tax rate on profits distributed is 35%. The additional tax must be withheld by the corporation. The same tax procedure is applicable on remittances of profit to partners or profit withdrawn by individuals not domiciled or resident in Chile.

Depreciation

Depreciation on fixed assets, except for land, is tax deductible by the straight-line method based on the asset's useful life in accordance with the guidelines of the Internal Revenue Service (SII), computed on the asset restated value. A shorter lifespan has been set by the Internal Revenue Service to apply to fixed assets purchased after 2003. However, the taxpayer may opt for accelerated depreciation for new assets when acquired locally, or new or used assets when imported, with useful lives of over five years. For this purpose, the assets will be assigned useful lives equivalent to one-third of the normal, eliminating fractions of months. Taxpayers may discontinue the use of the accelerated method at any time but may not return again to the accelerated method. Publically available documents published by the SII include the straight-line and accelerated depreciation schedules for different asset categories.

No allowance is made for amortization of intangible assets such as goodwill, patents, trademarks, etc. Depletion is not tax deductible.

Dividends

Chilean companies receiving dividends from Chilean corporations are exempt from First Category tax. There is no distinction in Chile between personal dividends and inter-company dividends. A dividend in kind as such does not exist. Dividends are necessarily expressed in cash, notwithstanding the fact that the company may distribute certain assets corresponding in value to the dividend amount. Stock dividends in the form of bonus shares or increases in the par value of existing shares are not considered income for tax purposes.

Interest Deduction

Interest accrued or paid in the financial year is a deductible expense, provided that it has been incurred in connection with loans related to the business. The interest accrued on this expense is not tax deductible.

Losses

Tax losses incurred can be applied against future profits or carried back without any time limit. It is not possible to group profitable and non-profitable affiliates for tax purposes.

Foreign Sourced Income

Non-domiciled or non-resident corporations are only subject to income taxes on their Chilean-sourced income. If the domestic corporation receives amounts in excess of the book value of an investment when a foreign subsidiary is liquidated, these monies are considered income subject to regular taxes. From 2012, the income received or accrued from derivatives such as forwards, futures, swaps and options, by persons or entities without domicile or residence in the country, is not affected by income tax, except those arising from derivatives that are settled by physical delivery of shares or rights in companies incorporated in Chile.

Interest payments to financial institutions not domiciled in Chile are subject to an additional withholding tax of 4%. There is no withholding tax on interest paid between Chilean companies but the loan has to be made at a commercial rate of interest.

22.3.2 *Mining Royalty Tax*

All mining properties are subject to statutory obligations to the Chilean Government in the form of a Mining Royalty Tax or "Impuesto Específico a la Minería" in Spanish (IEM). This tax was introduced in 2006 and amended in 2010, and is applied against the collective operating (mining) profits of all the operating units. The tax rate is calculated on a step scale on the basis of fine copper equivalent sales:

- 0 to 12,000 t copper equivalent: No tax applied
- 12,001 to 50,000 t copper equivalent: 0.5% to 4.5% of the Mining Operating Income according to the scale presented in Table 22-2
- Greater than 50,000 t copper equivalent: A different scale applies that starts at 5% of the Mining Operating Income for Mining Operating Margins of less than 35% up to 34.5% for Mining Operating Margins in excess of 85%. This scale is shown in Table 22-3.

Table 22-2 Mining Royalty Tax Scale for Mining Exploitation Under 50,000 t of Equivalent Cu

Cu Eq (t)		Marginal tax
From	To	%
0	12,000	0.0%
12,001	15,000	0.5%
15,001	20,000	1.0%
20,001	25,000	1.5%
25,001	30,000	2.0%
30,001	35,000	2.5%
35,001	40,000	3.0%
40,001	50,000	4.5%

Table 22-3 Mining Royalty Tax Scale for Mining Exploitation Over 50,000 t of Equivalent Cu

Operating Profit (%)		Marginal Tax
From	To	%
0	35	5.0%
35	40	5.4%
40	45	5.7%
45	50	6.3%
50	55	6.9%
55	60	7.6%
60	65	8.4%

The Mining Operating Income on which this tax is applied is determined following certain specific rules. Certain expenses such as losses from past periods, accelerated depreciation of fixed assets, etc. are not allowed for this purpose.

The Mining Operating Margin is determined as a ratio of the Mining Operating Income to the mining operational revenues.

22.4 Production and Revenue

Table 22-4 presents total payable gold production and revenues associated and Table 22-5 presents annual plant feed and gold production.

Table 22-4 Production Summary

Item	Unit	Value
Mined Mineral	Mt	811
Au	Moz	2.96
Revenues	$ Millions	4,000

Table 22-5 Annual Production

Period	Plant Feed (kt)	Au Produced (koz)
Y00	0	0
Y01	29,200	285,367
Y02	29,200	314,050
Y03	29,200	303,135
Y04	29,200	275,087
Y05	29,200	277,998
Y06	29,200	269,650
Y07	29,200	256,804
Y08	29,200	263,225
Y09	20,025	219,956
Y10	15,480	165,436
Y11	12,778	155,108
Y12	8,374	118,995
Y13	4,174	57,819
Total	294,431	2,962,630

22.5 Economic Analysis

Based on the projections resulting from the financial model, NPV, IRR and payback periods are shown, before and after tax in Table 22-6.

Table 22-6 Summary of Economic Evaluation Results

Indicator	Unit	Pre-tax	After-tax
NPV @ 5%	$ Millions	521	409
IRR	%	28,6	25.0
Payback Period	years	2.76	3.00

Table 22-7 presents annual cash flow before and after tax and Figure 22-1 shows cumulative cash flows calculated before and after tax.

Table 22-7 Annual Cash Flow

Period	Pre-tax ($ Millions)	After-tax ($ Millions)
Y00	-399	-399
Y01	-237	-240
Y02	-96	-117
Y03	30	6
Y04	122	96
Y05	188	144
Y06	285	223
Y07	353	274
Y08	465	364
Y09	570	450
Y10	649	514
Y11	741	590
Y12	827	660
Y13	852	685
Total	4,349	3,252

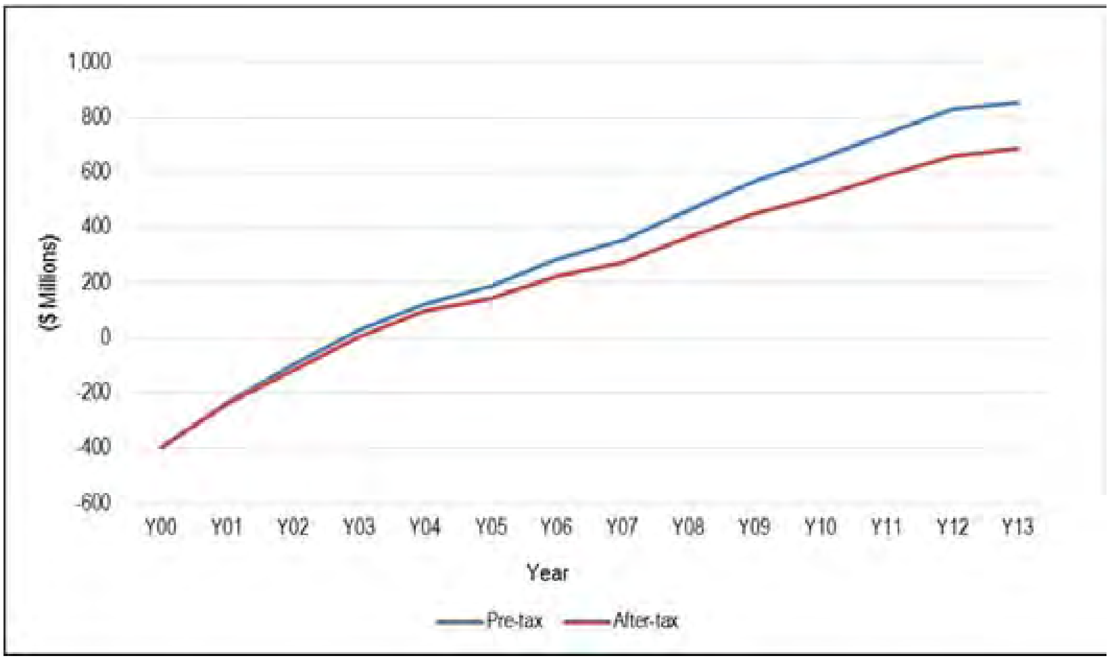

Figure 22-1 Cumulative Cash Flows

22.6 Sensitivity Analysis

NPV sensitivity analyses have been performed for changes in market gold price, changes in recovery rates, changes in capital and operating costs, and changes to discount rate. Sensitivity analyses were performed on base rate, 20% taxation only.

Changes in gold prices are expressed in increments of $50 per ounce for gold. Operating and capital costs are expressed in 5% increments of negative and positive deviation from the Base Case values. For discount rate, sensitivity has been made by increments of 1% between 0% and 10%.

Table 22-8 Gold Price Sensitivity

Item	Unit	Pre-tax	After-tax (20%)
Au Price: 1,200 US$/oz			
NPV @ 5%	$ Millions	199	152
IRR	%	14.5%	12.8%
Payback Period	Years	5.5	5.7
Au Price: 1,350 US$/oz			
NPV @ 5%	$ Millions	521	409
IRR	%	28.6%	25.0%
Payback Period	Years	2.8	3.0
Au Price: 1,600 US$/oz			
NPV @ 5%	$ Millions	1,058	822
IRR	%	50.2%	42.7%
Payback Period	Years	1.8	2.0

Table 22-9 Discount Rate Sensitivity (Au price: 1,350 $/Oz)

Item	Pre-tax ($ Millions)	After-tax ($ Millions)
NPV@5%	521.2	409.3
NPV8%	386.8	297.0
NPV10%	315.6	237.4

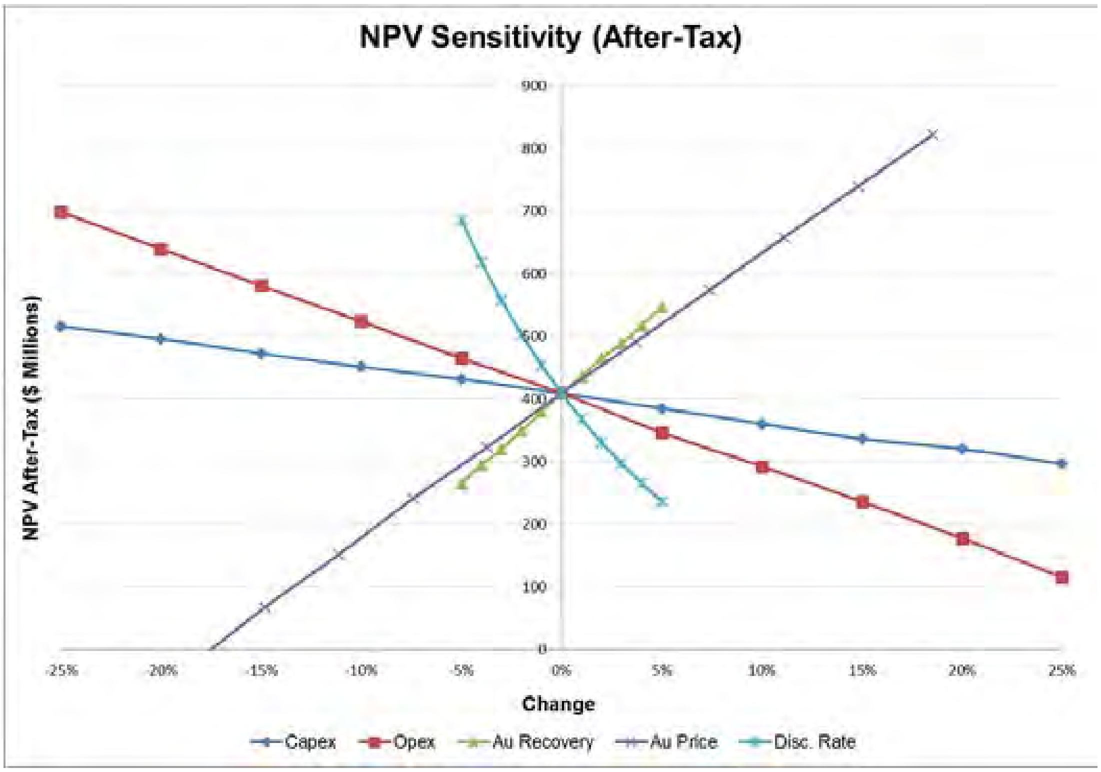

Figure 22-2 After-tax NPV Sensitivity

NPV is most sensitive to changes in discount rate and gold price. The effects on NPV of the variation of every sensitivity variable are presented as follows:

- For every point of recovery, NPV5 increases or decreases approximately $28 million or 7%. The breakeven point is at a 14% less of gold recovery
- For every 50 $/oz change in the gold price, NPV5 increases or decreases approximately $85 million or 21%. The breakeven point is at a gold price of 1,112 $/oz
- For every 1% of Capex variation, NPV5 increases or decreases approximately $22 million or 5%. The breakeven point is at a 93% more capital expenses
- For every 1% of Opex variation, NPV5 increases or decreases approximately $58 million or 14%. The breakeven point is at a 35% more operational expenses.

22.6.1 Combined Sensitivity Scenarios

Different scenarios and combined sensitivities have been performed in order to have a full and comprehensive understanding of the impact of changes of the most important variables in the economics of the project.

For this effect, $1,303/oz, the average gold price from June 20th 2014 to August 19th 2014, has been defined as the current price.

Table 22-10 presents the following sensitivities:

- Case 1: Current gold price and no changes in other variables
- Case 2: Current gold price and 10% variation in total Capex and Opex
- Case 3: Current gold price, 10% variation in total Capex and Opex and 8% discount rate
- Case 4: Income tax rate of 25%, base case gold price ($1,350/oz) and no changes in other variables.

Table 22-10 Combined Sensitivity Scenarios

Item	Unit	Pre-tax	After-tax
Case 1			
NPV @ 5%	$ Millions	420	323
IRR	%	24.3%	21.0%
Payback Period	Years	3.2	3.4
Case 2			
NPV @ 5%	$ Millions	222	170
IRR	%	14.7%	13.0%
Payback Period	Years	5.4	5.6
Case 3			
NPV @ 5%	$ Millions	131	91
IRR	%	14.7%	13.0%
Payback Period	Years	5.4	5.6
Case 4			
NPV @ 5%	$ Millions	521	385
IRR	%	28.6%	24.3%
Payback Period	Years	2.8	3.0

It is important to note that even in adverse conditions such as Case 3, the NPV of the project is still $ 91 million.

23 ADJACENT PROPERTIES

Figure 23-1 depicts the location of properties/projects which are adjacent to the Cerro Maricunga Project. The more significant properties are as follows:

Table 23-1 Main Adjacent Properties to Cerro Maricunga

Property	Status	Ownership
La Coipa	Mine	Kinross Gold (100%)
Maricunga (Refugio)	Mine	Kinross Gold (100%)
Marte-Lobo	Project	Kinross Gold (100%)
Can Can	Mine (Closed)	Kinross Gold (100%)
La Pepa	Project	Yamana Gold (100%)
Volcan	Project	Hochschild Mining (100%)
Caspiche	Project	Caspiche Exeter Resource (100%)
Cerro Casale	Project	Barrick (75%) , Kinross Gold (25%)
Caserones	Project	Pan Pacific Copper Co., Ltd. (77.37%), Mitsui & Co., Ltd. (22.63%)

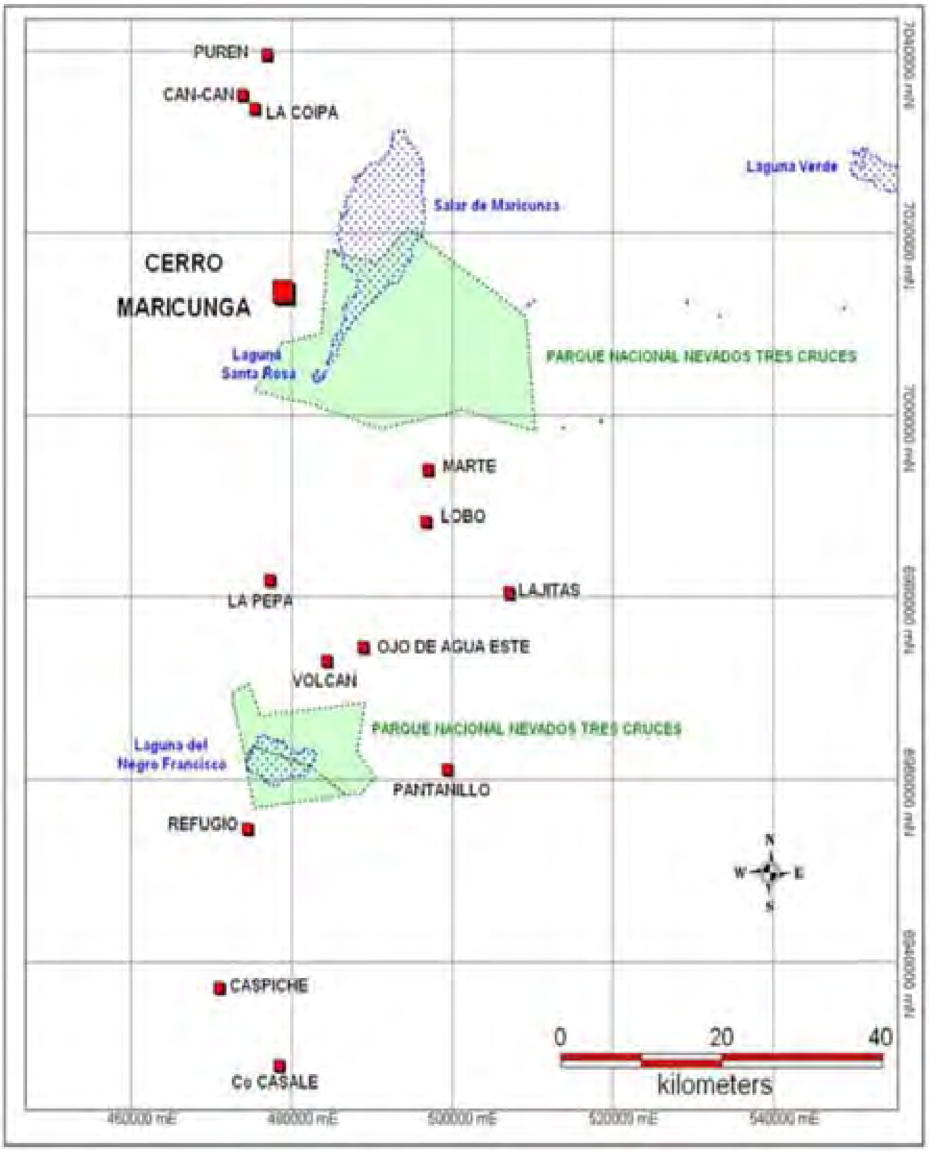

Figure 23-1 Significant Project in the Maricunga Mineral Belt

24 OTHER RELEVANT DATA

24.1 Mine Geotechnical

Slope configuration, shown in Table 24-1, for the Cerro Maricunga open pit plan is based upon open pit mining experience in the area and geotechnical information.

Table 24-1 Mine Slope Configuration

Bench - Shoulder Slope				Inter-Ramp Slope		Overall Slope	
hb (m)	**αB (deg)**	Q (m)	B (m)	**αr (deg)**	hr (m)	Br (m)	Hg (m)
20	75	5.4	9.5	53.4	160	35	400

Where:

hb	: Bench height	αr	: Bench face angle
Q	: Rupture	B	: Shoulder
αr	: Inter-ramp angle (toe-to toe)	hr	: Maximum inter-ramp height
Br	: Ramp width	Hg	: Maximum overall height

The overall angle ranges between 45º and 48º, depending on the position of ramp accesses.

The conceptual estimate for the Project's geometry is based on the following mining efficiency, geology and on-site geotechnical information:

- Perimeters and altitude of deposit
- Bench height: 10 m high benches or double (20 m)
- Geotechnical data for 84 diamond drill holes (RQD, FF attributes, and core photographs)
- Tele-viewer data for 17 drill holes (strike and dip discontinuity)
- Lithological zoning of major bodies: breccia, dacite-andesite porphyry, and volcanic dacite
- Structures: Major Faults: Saturno, Júpiter, Neptuno, Luna, Venus and Plutón. All structures are vertical to sub-vertical with predominant NW and N-NW orientation.

Geo-mechanical analysis that backs up the proposed open pit design is as follows:

24.1.1 Zoning of Structural Patterns:

Strikes and dips of minor fault structures detected by Tele-viewer were analyzed in conjuction with ltihological zoning. Results indicated that there is no correlation between faulting and lithology. Nevertheless, tele-viewer data allowed the definition of three structural domains; Saturno, Neptuno and Plutón as shown in Figure 24-1.

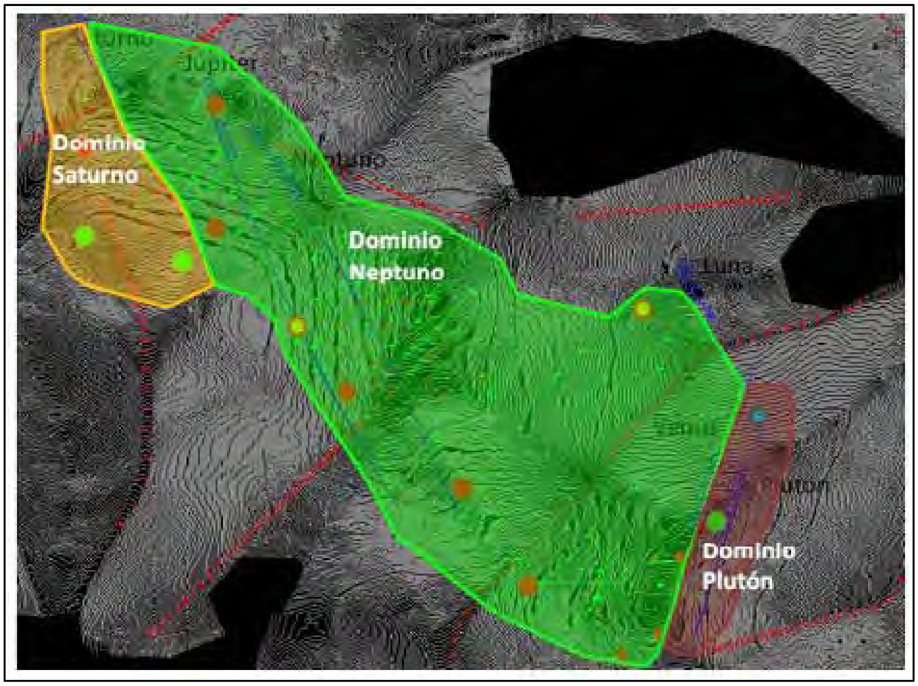

Figure 24-1 Structural Domains at Maricunga – Conceptual Stage

Minor fault records along sub-vertical dips indicated that these domains are maintained at depth.

24.1.2 Rock Quality Zoning - RQD, GSI, RMR

A 3-D model, constrained within the open pit project was generated based on available RQD data. This information is useful to derive in-situ rock properties for ongoing stages of the project.

Additionally, a GSI classification was prepared based on available core registered as photographic records. The statistics referring to the attributes of the lithological breccia (BX) and porphyry (DAP) bodies are shown in Figure 24-2 and Figure 24-3.

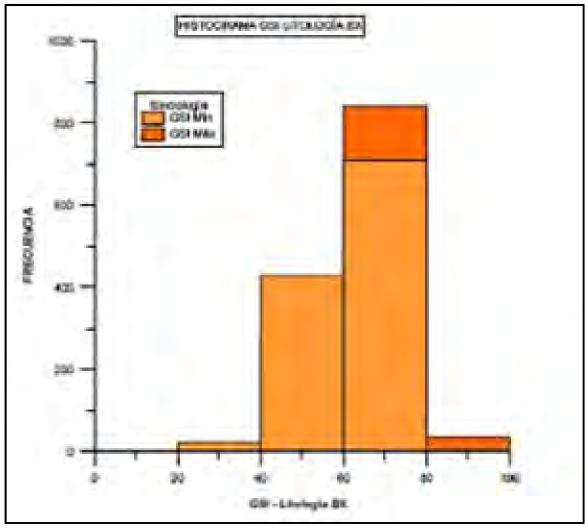

Figure 24-2   GSI Statistics on Breccia Lithology

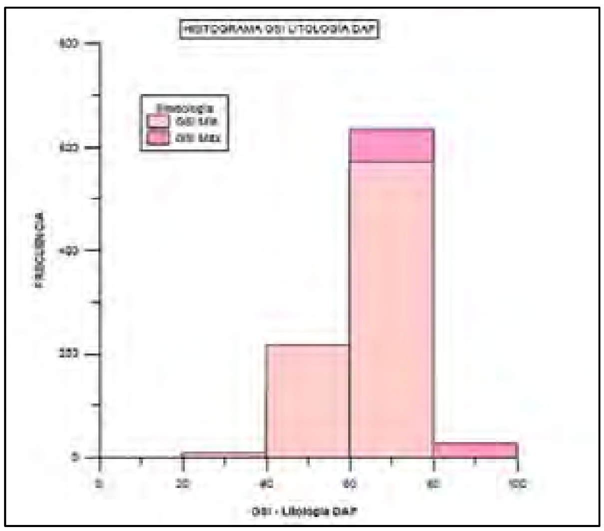

Figure 24-3   GSI Statistics on Porphyry Lithology

The following relevant GSI parameters were derived based on the empirical correlation; GSI = RMR – 5. These are as follows:

- The prevailing GSI is in the 60 - 80 range for both rock types (breccia and porphyry)
- GSI over 60 is observed in more than 50% of total rock
- GSI under 40 is observed in 3 to 5% of total rock.

The Project's slope stability analysis was conducted using the information shown above, as well as the following parameters:

<u>Empirical Review of the Proposed Design</u>

Regarding run-out safety, the Project includes a 9.5 m safety shoulder (B).

According to the Ritchie's empirical criterion, individual benches with an initial height (H) of 10 m require shoulders (B) for spillage containment equivalent to B = 4.6 + (0.2) H.

In our case, this shoulder is equivalent to 6.6 m and, by doubling the bench to a height of 20 m, based on Ritchie's criterion, the same shoulder requires a total height of 8.6 m. These numbers meet established requirements and suggest that the project is conservative in matters relating to Run-Out Safety.

Regarding safety on over-break at individual benches, it was considered an individual bench angle of 75º. The review of blasting practices (no pre-stripping) associated to porphyry and breccia rocks of similar qualities in open pit sites show the statistical pattern depicted in Figure 24-4.

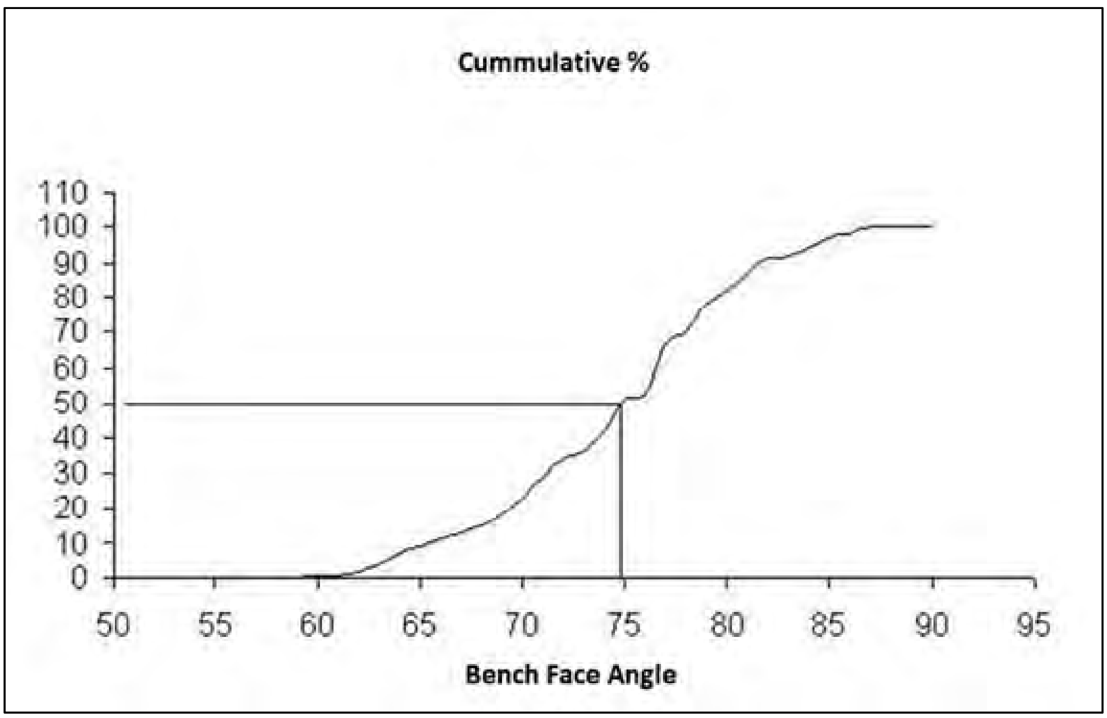

Figure 24-4 Bench Face Angle Distributions

This graph shows that the suggested design will not meet the 75° angle in 50% of the individual benches; therefore pre-splitting is recommended.

A maximum inter-ramp slope height of 160 m was considered. The largest portions of rock have a GSI above 60, hence a RMR greater than 55, and a MRMR of more than 50. Assuming a Safety Factor of 1.2, the Empirical ABACUS shown in Figure 24-5, indicates that an inter-ramp height of only 160 m and a slope of 53.4º, requires and MRMR of 45 rocks to meet the safety status.

Please note that the marginal status under these safety conditions is accepted based on the following existing conditions:

- MRMR is greater than 50 for most of the in-situ rock. As shown below, the computer-simulated analytical studies recommend it
- Slope design, based on 7 exploitation stages; indicate that the above mentioned inter-ramp height will be required occasionally due to the incorporation of a main access ramp.

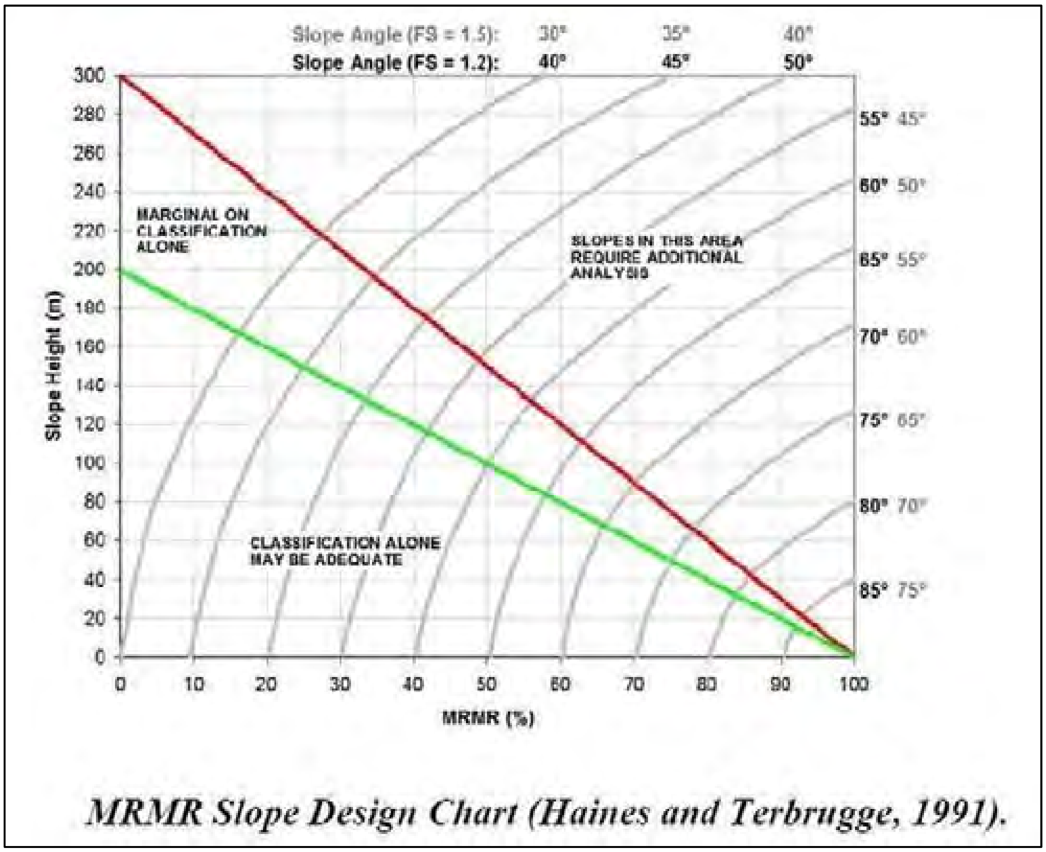

Figure 24-5 Empirical Formulation MRMR / Slope Height / F.S.

<u>Analytical Review of the Proposed Project</u>

The proposed design was evaluated in 7 stages leading to a final pit; for all stages, the analytical assessment was conducted based on the same methodological arguments. These are as follow:

The structural pattern observed in Cerro Maricunga (shown in Figure 24-1 was superimposed onto the mine perimeters and individual benches in each stage. This allowed detecting the occurrence of rock volumes bounded by structural faults, referred to as "wedges". Consistent with this methodology, if the wedge outcrops using the proposed slope, an unstable condition is assumed when the assumed friction, equivalent to 30°, is surpassed by the gravitational component in the direction of the outcrop for that wedge. This preliminary review of stability leads to a debatable Safety Factor (FS); but in this case, FS = 1.2 has been considered as sufficient for the stability of the slopes.

The results thus obtained indicate that stability conditions determined by sliding wedges, are met in the Saturno structural domain. These conditions are not met in the Neptuno and Plutón structural domains which is consistent with the arguments given above in the sense that pre-splitting is required to ensure individual bench angles of 75°.

Limit of Stability Balance Test: There is no sufficient information to characterize the in-situ rocks; in this case, a Limit of Stability Balance Test was conducted on the geometrically identifiable slopes with 15 representative profiles for the 7 stages planned in this study.

The analysis was conducted by using SLIDE 6.0, sold by Rocsciense, assuming in-situ rock parameters and extreme seismic conditions equivalent to horizontal 0.2 g. Notwithstanding the fact that the results thus obtained can be speculative, it is worth noting that in all cases assessed both the Static and Dynamic Safety Factors were higher than the 1.2 limit established for inter-ramp and overall slopes.

24.2 Waste Dump

The waste dumps will contain material resulting from truck transportation and dumping of waste. Design variables for the construction of the dump are a natural dumping angle of 37º, height of the sub-slopes or layers and the decoupling shoulders, plus the total material pile height. The design is summarized below:

- Maximum dump height: 170 m
- Maximum inter-ramp or layer height: 60 m
- Overall angle: up to 29°
- Inter-ramp or layer angle: up to 37°
- Decoupling shoulder: from 30 m.

The waste dumps will contain material resulting from truck transportation and dumping of waste. Design variables for the construction of the dump are a natural dumping angle of 37º, height of the sub-slopes or layers and the decoupling shoulders, plus the total material pile height. The design is summarized below.

In practical terms, the gravitational and operational traffic confinement, results only in an increase in the inter-granular friction numbers that control the stability of these materials. In other words, this is favourable for the stability of the dump.

Regarding dynamic stresses, following the traditional, state-of-the-art formulations and considering the minimum acceptable Safety Factor to be FS = 1.1, the conclusion is that the design proposed sufficiently meets this requirement.

Additionally, given the deformable nature of these materials, it has been necessary to establish the maximum particle displacement (of the materials) that could happen in the event of a large earthquake. This was achieved by means of empirical displacement graphs that are standardized based on acceleration n (assuming a maximum acceleration = 0.5 g for materials in inclined planes). In our case, the representative profile for the dump is shown in Figure 24-6 and has a run-off angle of approximately 20º.

Figure 24-6 Longitudinal Profile of Ojos de Maricunga Ravine Axis

This topographical condition exposed to a maximum earthquake will result in slope deformations between 12 and 72 cm, which are considered insufficient to cause a pervasive dump collapse.

24.3 Project Implementation

Atacama needs to complete an environmental impact study, and detailed engineering in order to meet the required technical studies for the start-up of the Cerro Maricunga Project.

Construction program and pre-stripping activities can commence once the project is approved by authorities. The construction of the project will be executed considering EPC, EP and construction contracts. Atacama will be leading the construction.

The construction phase of the Cerro Maricunga Project is estimated to last 18 months, beginning in August 2016. The climatic conditions and altitude are of consideration, with major foundation work to be completed during the early summer months.

Key milestones are as follow:

- Feasibility Study: November 2014 – June 2015
- Environmental Impact Study: November 2014 – June 2016
- Detailing Engineering and Procurement: July 2015 – August 2016
- Bidding and Evaluation of Construction Contracts: June 2016 – September 2016
- Construction: August 2016 – January 2018
- Start-up and Commissioning: January 2018 – May 2018.

The main EPC contracts are: Earth movements, Concretes and Structures & equipment assembly. Construction of the electric and fresh water supply will be executed through an EPC plus a leasing type of contract. The construction works will be developed in parallel to the plant construction and should not be within the critical path.

Critical Path

The critical path is related to development of more advanced engineering stages and the project construction. However, it has to be considered that environmental approval is an important limitation for beginning the construction of the project, and hence, if it takes longer than expected the critical path will be completion of this study.

The program of the project is shown in Figure 24-7. The critical path is highlighted in red. As shown in the program, purchase order for critical mine and plant equipment will be placed during the feasibility study period, in order to have them in site for the beginning of the pre-stripping and early work

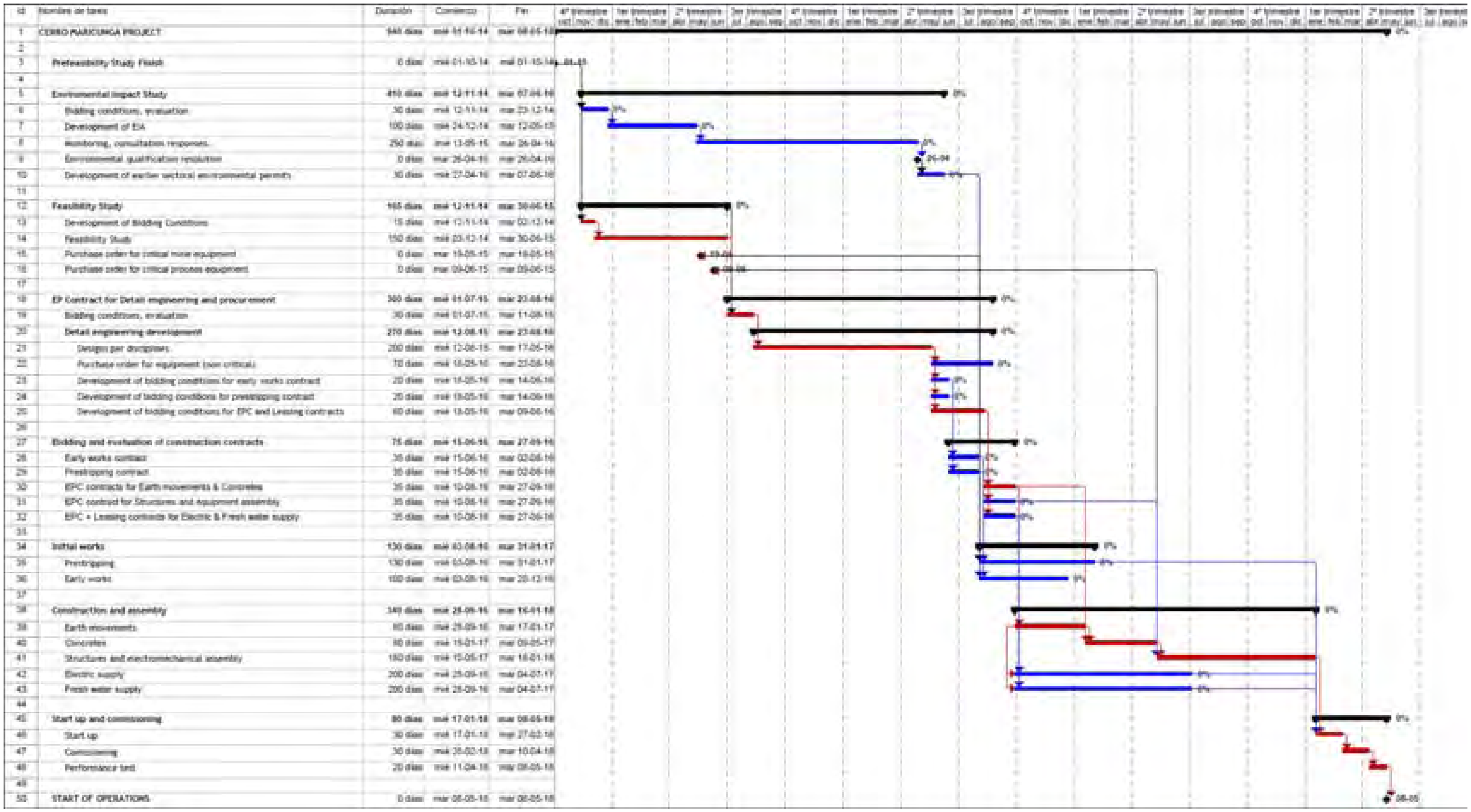

Figure 24-7 Project implementation program

24.4 Hydrological, Hydrogeological and Hydrochemical

This section describes the hydrological, hydrogeological and hydrochemical information available at the location of the Cerro Maricunga project. Additionally, several recommendations in terms of additional work on surface water management, identifying potential effects and future studies are needed to further develop the Water Resources Base Line.

24.4.1 Study Area

For purposes of the descriptions of the components, two levels of analysis were considered:

A regional study area is comprised of the sub-basins of the Quebrada Paipote (BNA Cod: 0344, 6,689 km^2) and Salar de Maricunga (BNA Cod: 0304, 3,005 km^2), which are in turn embedded into the Copiapo river Basin (BNA Cod: 034, 18,703 km^2) and the endorheic basins between Frontier and Pacific Rim (BNA Cod: 030, 15,618 km^2), respectively.

A local study area was also defined and consists of the following sub-sub-basins: San Andrés Creek (Cod BNA: 03440), Paipote Creek in San Andrés Creek (Cod BNA: 03441) and Campo de Piedra Pómez and Lamas River (BNA Cod: 03041), within which the following micro-watersheds associated with the location of the project works are distinguished:

- Quebrada del Toro (QT, 52 km^2)
- Quebrada de La Pelada (QLP, 29 km^2)
- Quebrada Larga (QL, 148 km^2)
- Quebrada Paipote Alta (QPA, 182 km^2)
- Quebrada Cerro Maricunga (QCM, 10 km^2).

The location of the micro-watersheds, regarding the project facilities, is shown in Figure 24-8. It is important to mention that part of the Project, specifically a portion of the pit is within the Regional Priority Site for Biodiversity Conservation Nevado Tres Cruces, recognized in the Regional Biodiversity Strategy Atacama (2010 - 2017). In percentage terms, the portion of the pit that will be in the priority site is only a 0.05% of that area, and besides, it should be considered that the Project will not locate any of its facilities in nearby areas relevant for the conservation of biodiversity.

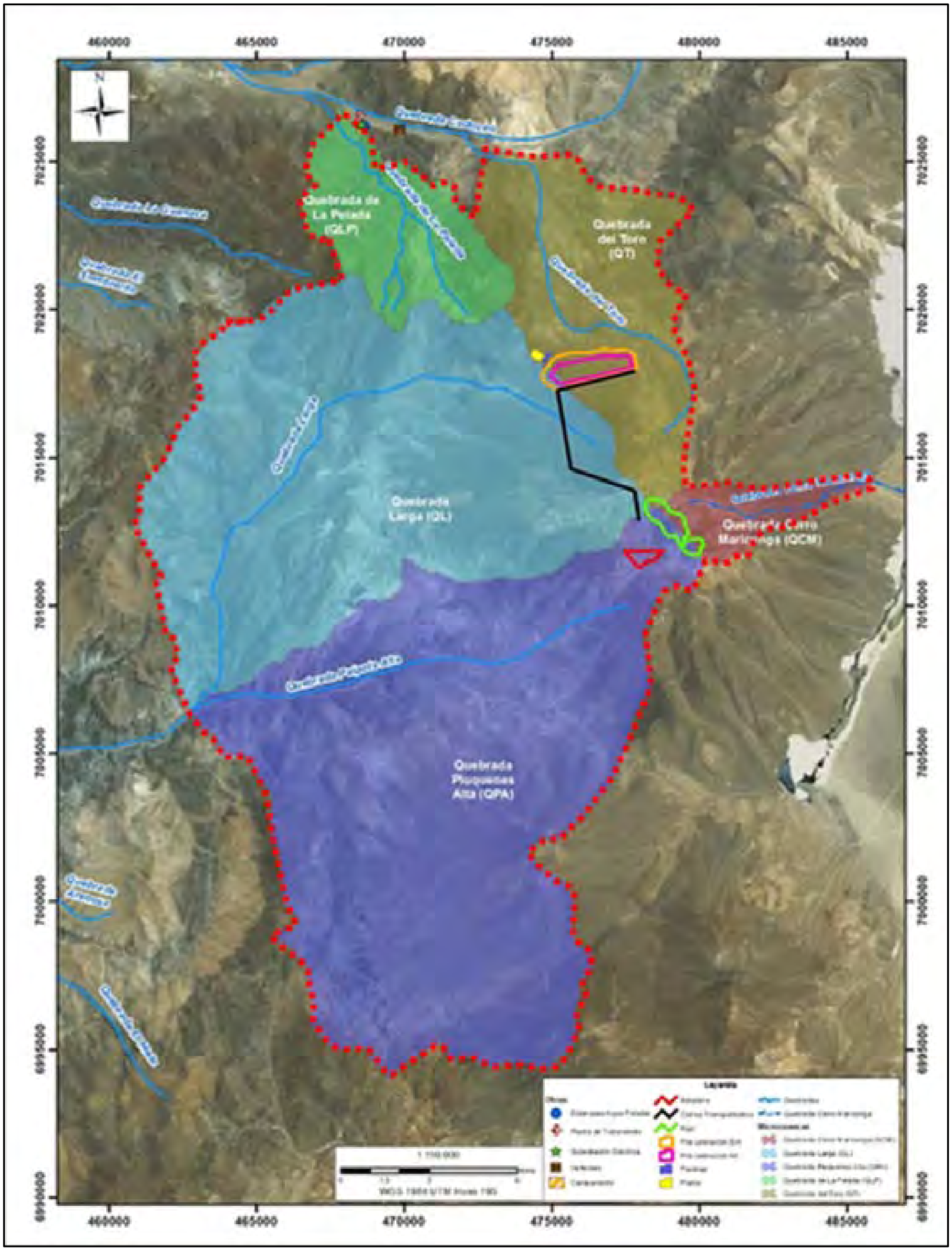

Figure 24-8 Micro-Watershed Level Local Study Area and Project Area of Influence

24.4.2 Hydrology

The project site presents no permanent or seasonal surface runoffs. Only eventual runoffs, associated with particular rainfall events, are observed.

24.4.3 Hydrogeology

At the local level, the collected information allows to distinguish the following hydrogeological units:

- Aquifer of High Hydrogeological Importance in Unconsolidated Deposits
- Aquifer of Medium to Low Hydrogeological Importance in Unconsolidated Deposits
- Aquifer of Medium to Low Hydrogeological Importance in Fractured Rocks
- Aquifer of Low Hydrogeological Importance in Unconsolidated Deposits to Semi Consolidated and Fractured Rocks
- Aquiclude and Acuifuge of Low to Zero Hydrogeological Importance in Rocks.

In terms of groundwater levels in the study area, information from 5 monitoring wells and several geotechnical drilling is available. Only on three of these monitoring wells have been registers of phreatic level.

Annual recharge study have been conducted for two scenarios, first assuming that in the micro-basins 80% corresponds to Rock and 20% to Sediment, and second assuming 90% Rock and 10% Sediment. The results obtained for the first scenario are presented on Table 24-2.

A conceptual water balance was prepared based on the preliminary baseline information regarding surface and groundwater components available.

Table 24-2 Preliminary Surface Water Balance at an Annual Scale (80% Rock – 20% Sediment)

Item	QT	QLP	QL	QPA	QCM
	INPUTS (l/s)				
Precipitation	164.9	59.8	328.5	606	34.9
Total Inputs	164.9	59.8	328.5	606	34.9
	OUTPUTS (l/s)				
Recharge	2.3	0.8	4.5	8.3	3.2
Evapotranspiration	162.6	59	324	597.7	31.7
Rainfall	0	0	0	0	0
Total Outputs	164.9	59.8	328.5	606	34.9

24.4.4 Water Quality

To characterize water quality, local data reports for the seven (7) campaigns conducted between March 2011 and October 2012, were used.

In general terms, there is a clear difference between the water quality of the Salar de Maricunga sector and the ones taken towards the west within the Paipote Creek area, more specifically in the Paipote Upper Creeks, del Toro and La Pelada.

Among the subsectors, the waters of High Paipote Creek are quite different from the El Toro and La Pelada Creeks, at least in relation to EC, SO_4 and Cl, suggesting that its origin is not necessarily the same.

In the sector of the Salar de Maricunga, differences are also observed but are directly associated with the location of the points around the salar itself.

24.4.5 Influence Area for the Project

<u>Hydrology</u>

Figure 24-8 shows the area of influence for the hydrological component of the Project, marked in a red dotted line. The area of influence is directly associated with the reduction of the micro-watershed area in which the project is located (with eventual runoffs).

<u>Hydrogeology</u>

At this stage and according to the available information, potential effects of the Project are not recognized in the hydrogeological component, which implies the absence of an area of influence.

<u>Water Quality</u>

The area of influence for the water quality component is similar to the one defined for the hydrology component, presented in Figure 24-8. This area of influence is associated with potential seepage from the Project facilities, which could affect local aquifer systems that may exist in the lower parts of the micro-watershed.

25 INTERPRETATIONS AND CONCLUSION

The Cerro Maricunga project is located in the Atacama Region (III Region, Chile) within the well-known Maricunga Mineral Belt. The region boasts a well-trained mining workforce, support from experienced and well established mining equipment suppliers, and first class professional and technical consultants. Chile has a long-standing mining culture.

No fatal flaws have been identified during the course of the pre-feasibility study for the Cerro Maricunga Project. Water availability is the most common constrain for mining projects in the area and Cerro Maricunga Project has secured a supply.

The planned increase of 2,000 MW in the Chilean central power grid (SIC) and its connection with the power grid for the northern zone (SING) ensures energy availability for project. It should be noted that the power requirement for the Cerro Maricunga are comparatively low.

The pre-feasibility study for this oxide gold deposit results indicate that, at a gold price of 1,350 $/oz, Cerro Maricunga project is an attractive and robust project, which warrants continued development to full feasibility level.

The following opportunities have been identified for further evaluation during next stage:

- Metallurgical testwork suggests that high gold recoveries could be achieved at 50 mm of P_{80}, leading to eliminate the tertiary crushing stage and resulting in some important capital and operating costs savings
- Revisit and re-evaluate main quotations, money costs and prices, regarding the slow-down on the mining investment super-cycle
- Explore synergies from sharing infrastructure and supplies with nearby projects
- Evaluate possible governmental stimulus for investment and project development, due to the slow-down of the Chilean economy.

There is no significant foreseeable risk that would impact the reliability or confidence of the mineral resources or reserves; however the following risks have been identified for further evaluation during next stage:

- Chilean government is addressing a new tax regulation and its impact on the project economics needs to be addressed and evaluated
- Project development critical path runs through the environmental permitting process, thus baseline studies need to be comprehensive to ensure permitting approval schedule is kept to minimum.

25.1 Mineral Resource

- Atacama's geology team, led by Mr. Alonso Cepeda, discovered a unique gold deposit in the Maricunga Belt. Mineralogical characteristics observed throughout this deposit are outstanding, since approximately 92% of gold is exposed and potentially available for leaching

- Total measured and indicated resource,at a 0.15 g/t Au cut-off grade, amount to 433.8 million tonnes grading 0.38 g/t Au containing 5.249 million ounces of gold
- Total inferred resources at Cerro Maricunga, at a 0.15 g/t Au cut-off amount to 57.9 million tonnes grading 0.32 g/t Au equivalent to 0.603 million ounces.

25.2 Mineral Reserves

- Proven and Probable Mineral Reserves total 294.4 Mt grading 0.40 g/t Au
- Mineral Reserves were calculated by constraining Measured and Indicated Resources within an open pit mine featuring a constant throughput of 80 ktpd
- Operational cut-off grades of 0.18 g/t Au during the first three years and 0.15 g/t Au from year four onward were used in order to improve the grade of the plant feed during production
- Mineral Reserve estimated could be most affected by changes in metallurgical recoveries and operating costs
- Gold price, even though the most important factor for revenue calculation, has a lower impact on the Mineral Reserve estimation because the selected Lerchs-Grossman shell used as the guide for practical mine design was obtained using the discounted method and the mine plan considers operational cut-offs higher than the internal cut-off.

25.3 Mining Studies

- Pit designs are based on optimized LG shells at a revenue factor of 1.0 with overall slope angles of 40° and 41°
- Seven pit phases are planned up to final pit configuration
- Six month pre-production period and 11 Mt of total material needs to be removed in order to expose sufficient ore to start commercial production in year 1
- Mining operation considers 42 m^3 hydraulic excavators and 290 tonnes trucks.

25.4 Metallurgical Information and Process Design

- A gold extraction of at least 80% is expected for ore with Au grade of 0.40 g/t and at P_{80} 19 mm crush size
- The overall recovery of gold in the processing plant, including Electro-winning and refining, is 79.2%
- Crushed ore contains three forms of gold; (1): exposed CN-able gold and hence easily recoverable, (2): enclosed CN-able gold and not particularly sensitive to crush size, and (3): refractory gold, which accounts for less than 10% of total gold
- Mineralogical characterization findings were contrasted with the metallurgical tests results, confirming that crush size does not have an important impact in gold extraction and that most of the extractable gold is recoverable quite fast. These findings allow considering new possibilities for crushing sizes and residence times, which could have a positive impact in CAPEX and OPEX, but with low impact in gold extraction.

25.5 Environmental

- Cerro Maricunga mining project is feasible from an environmental sustainability viewpoint

- All key environmental sustainability variables identified and analysed (potential environmental impacts) can be fully addressed and there are measures in place to effectively manage them
- It is necessary to develop the baseline studies and evaluation of environmental impacts due to the presence of relevant environmental elements in areas close to the Project site
- It is necessary to discuss the Project with the Environmental Authorities and the neighbouring communities to reinforce the relationship and to facilitate the communication during the environmental evaluation of the project
- The mining operation will not cause a significant alteration in the lifestyle or the customs of the inhabitants or their dwellings.

25.6 Pre-feasibility Study Results

Opex and Capex used for the project represents those expected for a project of this type exhibiting average characteristics of ore abrasiveness and hardness; grades and rock type characterizations as indicated in the geological sections. Costs are supported by budgetary quotations for all major items (operating and capital) and were benchmarked against other operations. The estimate has an accuracy of -5% to +20% with a 90% probability of occurrence. Table 25-1 and Table 25-2 presents a summary of capital and operating costs for the project respectively.

Table 25-1 Capital Costs Summary

Area	Initial ($ Millions)	Sustaining ($ Millions)	Total ($ Millions)
Mining	42.4	6.3	48.7
Process Plant	252.4	152.5	404.9
Support Facilities	44.9	0.9	45.9
Owner Costs	12.7	0.0	12.7
Closure Costs	0.0	5.0	5.0
Contingency	46.5	23.0	69.5
Total	398.9	187.7	586.6

Table 25-2 Operating Costs Summary

Area	Total Cost ($ Millions)	Unit Cost $/t*
Mining	1,124.5	3.82
Process Plant	740.9	2.52
G&A	159.0	0.54
Total	2,024.4	6.88

*$/t: Cost per tonne of ore processed

The financial analysis indicated that the proposed development reflected net positive cash flow and internal rate of return which could support the progression to the next stage of feasibility study.

Table 25-3 presents a summary of the economic evaluation results for the Base Case.

Table 25-3 Economic Evaluation Results Summary (1,350 $/Oz Au)

Indicator	Unit	Pre-tax	After-tax
NPV @ 5%	$ Millions	521	409
IRR	%	28,6	25.0
Payback Period	years	2.76	3.00

25.7 Risk and Uncertainties

The projects economic viability is dependent on factors that include, among others, the results of regulatory and permitting processes; future prices of gold; possible variations in grade or recovery rates; failure of equipment or processes to operate as anticipated; labour disputes and other risks of the mining industry; the results of further economic and technical studies, delays in obtaining governmental approvals or delays in obtaining financing.

26 RECOMENDATIONS

In order to continue to the next phase of developing a full feasibility study, Alquimia foresees the following activities.

26.1 General

- Develop environmental baselines studies in order to begin the environmental impact study presentation process
- Complete current geotechnical studies in mine, plant and heap leach areas.

26.2 Mining

- Mine fleet optimization studies and mine scheduling may be further developed in order to improve equipment matching and plant scheduling
- Investigate the use of contractors for the open pit mining developments.

26.3 Process

- Improve the geo-metallurgical model with current and new metallurgical testwork data
- Further metallurgical testwork to confirm viability of crushing to 50 mm
- Process optimization study based on new metallurgical testwork.

26.4 Proposed Budget for Next Phase

In order to develop the project to the next phase of Feasibility Study, Alquimia estimates that the budget presented in Table 26-1 is required.

Table 26-1 Proposed Budget

Task	($ Millions)
Complete field geotechnical and laboratory investigations to feasibility level	0.80
Metallurgical Testwork	0.50
Process Optimization Study	0.10
Mine design to feasibility level	0.80
Process plant and infrastructure design to feasibility level	1.00
Environmental Baseline Studies	0.50
Tramitation of the Environmental Impact Study	1.00
Permits and Concession Payments	1.00
Head office costs	2.00
Total	7.7

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