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Updated Pre-feasibility Study for the Fenix Gold Project

Atacama, III Region, Chile

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1 EXECUTIVE SUMMARY

1.1 Introduction

This technical report, detailing the results of the Fenix Gold Project ("Fenix" or "the Project") Pre-feasibility Study (PFS), has been prepared and compiled by Mining Plus S.A.C ("Mining Plus") at the request of Rio2 Limited ("Rio2"), a publicly listed company trading on the Toronto Venture Exchange under the trading symbol RIO. 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").

The Project was formerly known as the Cerro Maricunga Project. Rio2 formally changed the name of the Project on September 17, 2018. The reason for the name change was to simplify reference to the project and differentiate it from others that use the word "Maricunga" in their project and/or company names.

This is an update to the 2014 PFS compiled by Atacama Pacific Gold Corporation ("Atacama") and presents a major change in strategy since that time.

The reasons for these changes are explained in this document.

All dollar denominations stated in this report are United States Dollars ($) unless specifically stated otherwise.

This PFS focuses on the development of the Fenix Gold Project on a throughput of 20,000 tpd. The primary reason Rio2 has elected to start at this rate of production is to allow for the trucking of water from Copiapo, which will expedite and simplify the approval and permitting process of the mine. By choosing the option of trucking water to the mine site, the Company has reduced the timeline to construction from five years to two years. Once the project is in production, the Company will focus on the logistics and timing of constructing the previously planned water pipeline from Copiapo (outlined in the 2014 PFS) which will sustain a mining rate of up to 80,000 tpd, four times of what is contemplated in this PFS.

Under the PFS mine plan, the Project will be able to produce for sixteen years with average annual production of 85,000 oz of gold for total Life of Mine ("LOM") production of 1.37 million oz. LOM All in Sustaining Cost ("AISC") is estimated at $997/oz. The Project demonstrates strong returns with an after-tax Net Present Value discounted at 5% ("NPV5") of $121 million and an after-tax Internal Rate of Return ("IRR") of 27.4% using the base case gold price of $1,300/oz ($241 million and 44.3% at $1,500/oz gold price). The Project is expected to generate average annual after-tax net operating cash flow of $15.1 million with cumulative LOM after-tax net cash flow of $222 million. At $1,500/oz gold, the Project would average more than $25 million in after-tax net operating cash flow annually and generate more than $422 in cumulative after-tax net cash flow over the 16-year mine life.

1.2 Property and Location

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

The Fenix Gold Project includes Exploration and Exploitation concessions that partially overlap, including overlapping areas; the surface area of the concessions is approximately 16,050 hectares.

There are no significant population centres or infrastructure in the immediate vicinity of the Project. Small indigenous communities raise crops and livestock in the valleys that drain the region, but there is no farming in the immediate vicinity of the Project.

Chile is an advanced country in terms of mining technology and infrastructure. Copiapo, the nearest major city to the Project is located approximately 140 km southwest by major sealed and unsealed roads. Copiapo city has an approximate population of 175,162 inhabitants in 2017 as per the United Nations Statistics Division (http://data.un.org).

Experienced mining and processing 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.

Rio2 is a precious metals exploration and development company with a portfolio of Chilean and Peruvian projects including the flagship 100%-owned Fenix Gold Project. Rio2 owns and controls the Project through its Chilean subsidiary, Fenix Gold Limitada. The project location is shown in Figure 1-1.

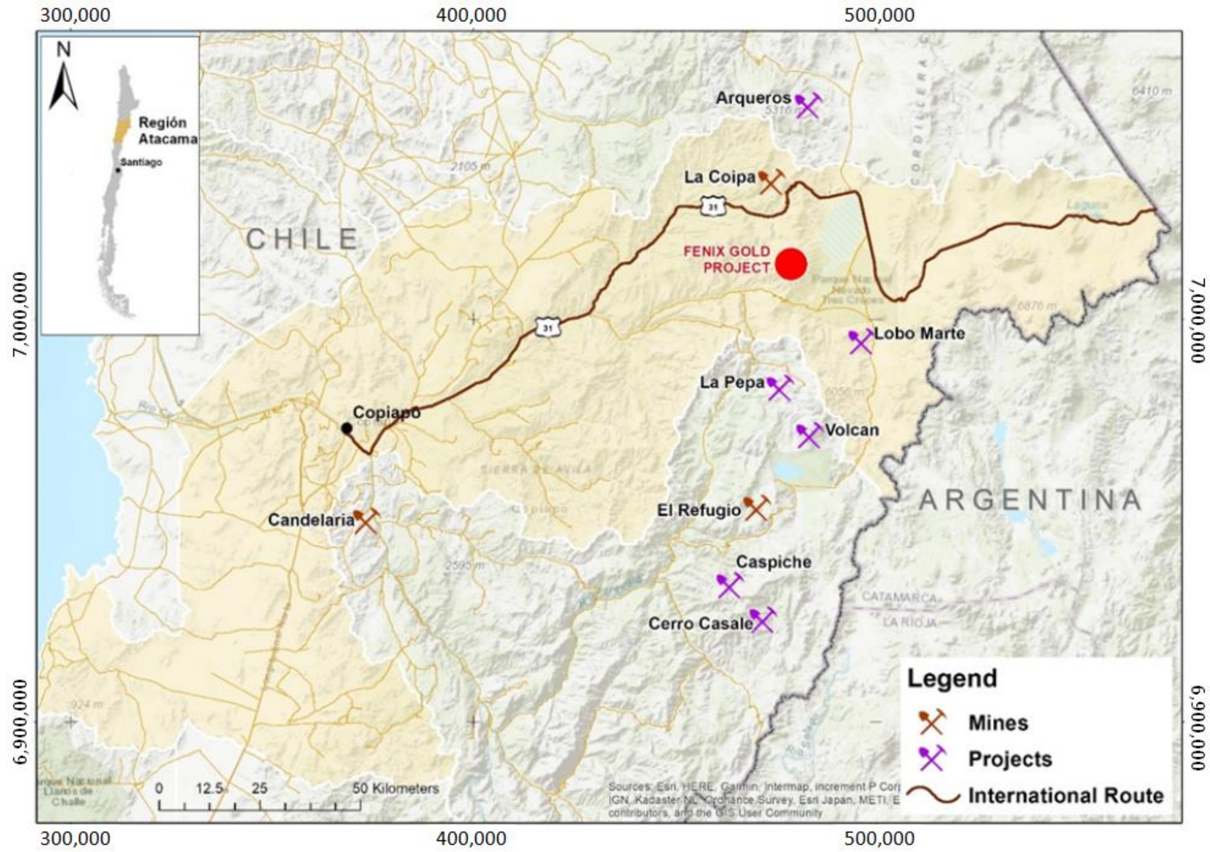

Figure 1-1: Location Map of the Fenix Gold Project

1.3 Geology and Mineralization

Surface mapping, trenching and drilling indicate that gold mineralization at Fenix is confined to a NW-SE trending corridor consisting of volcanic rocks that host large breccia complexes bounded by fault structures. The strike extension of mineralization is 2.5 km in NW-SE direction, up to 600 m in NE-SW direction and to depths of 600 m. The mineralization remains open at depth and to the east.

Three mineralized zones have been defined, based on gold distribution, in trenches, outcrops and drill holes. The zones are named Fenix North, Fenix Central and Fenix South.

Gold mineralization is disseminated and is most strongly associated with black banded quartz veinlets (BBV). The banding in BBV's is due to variable concentrations of tiny gas inclusions and very fine magnetite aligned parallel to the veinlet margins. Sulphides are rare in the deposit, typically accounting for <0.1 wt% and are primarily pyrite.

The deposit has been classified by Greg Corbett based on mineralogy and hydrothermal assemblage as Epithermal Low Sulphidation type deposit.

1.4 Exploration

The Project has been rigorously explored by trenching, mapping, geophysics and drilling over eight stages during the years 2008-2017 by Atacama. Rio2 completed a small infill drill program (7,066 m RC) and some check superficial mapping and sampling in 2018/19.

Exploration drilling consists of 91 diamond drill holes (DDH) totalling 31,047.21 m, and 293 reverse circulation (RC) drill holes totalling 84,500.55 m.

There has been very limited exploration of any kind outside the known mineralization footprint.

1.5 Mineral Resource Estimate

The Mineral Resource Estimate (MRE) has been updated to include a 39 hole (7,066 m) RC drill program completed in 2018/19, and surface channel sampling. For the first time, the updated MRE is based on a 3-D geological model.

The additional data, new geological model and revised modelling parameters have had no material effect on the combined Measured and Indicated resources when compared to the 2014 PFS. This suggests that the resource estimate is robust for bulk mining.

Inferred resources have increased markedly from the 2014 PFS. Inferred resources have been projected up to 150 m from the base of drilling, in line with ranges demonstrated in gold variograms.

Resources presented in Table 1-1 are constrained within a $1,500/ounce optimized open pit.

Table 1-1: Resource Statement for the Fenix Gold Project, 0.15 g/t Au Cut-off Grade

Resource Classification	Million Metric Tons	Grade Au g/t	Au Ounces (x1000)
Measured	122.4	0.41	1,630
Indicated	288.3	0.36	3,355
Total Measured + Indicated	**410.7**	**0.38**	**4,985**
Inferred	136.6	0.32	1,388

1. Mineral Resources reported is inclusive of mineral reserves.
2 Table 1-1 includes all Measured, Indicated, and Inferred Resources contained within the "Resource Pit", which represents the test for eventual extraction applied.
3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.
4. Mineral Resources are reported in accordance with Canadian Securities Administrators (CSA) National Instrument 43-101 (NI 43-101) and have been estimated in conformity with generally accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines.
5. Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

6. The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured mineral resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category.

1.6 Mineral Reserve

The Mineral Reserve estimate is shown in Table 1-2 and is effective as of August 15, 2019. Mineral Reserves reported as in-situ dry million tonnes, are based on a cut-off grade of 0.24 g/t Au and include 3% mining dilution and 97% mining recovery.

Table 1-2: Mineral Reserves (Mining Plus, 2019)

Reserve Category	Million Tonnes	Grade Au g/t	Contained Ounces Au x1000	Recoverable Ounces Au x1000
Proven	53	0.52	866	650
Probable	63	0.47	962	722
Proven and Probable	**116**	**0.49**	**1,828**	**1,372**

The Mineral Reserve Statement contains the total minable reserve for the deposits described in Section 15.1. The Mineral Reserve passed an economic test conducted on the production schedule. The results of the economic analysis are shown in Section 22.

1.7 Mineral Processing and Recovery Methods

Mineral Processing

High-grade ore will be crushed to a P80 size of 4 inches via a single stage Gyratory crusher with lime dosing occurring before the crushed ore is fed to a stockpile. Crushed ore will re-handled and trucked from the crushed ore stockpile to the leach pad. Agglomeration of crushed ore is not required.

Low-grade ore will be mined and stockpiled for crushing and leaching in later years.

Processing operations will treat the solutions from the heap leach facility operating in a new ADR (adsorption, desorption and refining) plant capable of treating 20,000 tpd of ore to pad or 1,058 cubic meters per hour of pregnant solution to produce doré bars. The plant layout is designed to be upgradeable to 40,000 tpd and 80,000 tpd respectively.

Processing costs are estimated at $4.10 per tonne treated over the current life of mine, which includes water purchase, and water transport costs.

Recovery Methods

The leach pad area will be prepared and covered with an impermeable liner. Corrugated, perforated drainage piping will be laid on the liner for collection of the pregnant leach solution. A protective layer of finely crushed, permeable ore will be placed on top of the liner to prevent damage from the mobile equipment and during ore loading. The ore will be stacked on the pad in 10m lifts.

The heap leach pad is located 4 km from the pit, at an elevation of 4,376 m above sea level. The pad will be developed in four stages with a stacking volume for Stage 1 of 10.3 Mt; 30.6 Mt for Stage 2; 27.7 Mt for Stage 3 and 60.7 Mt for the final stage. The total pad capacity will be 129 Mt. The irrigation system will uniformly apply cyanide solution directly onto the levelled surface of the leach pile through a drip irrigation system, at an irrigation rate of 10 L/hm2 with an irrigation cycle of 90 days.

Two stockpiles have been planned for storing the low-grade ore between years 1 and 12 with a total required capacity of 25.7 Mt. From years 13 to 17 low-grade ore will be recovered from the stockpile and taken to the crusher.

The percolation rate through the heap will depend on the viscosity and specific gravity of the solution, the mineral void space, the percentage of fines, mineral affinity for the solution and air entrapment.

Once the heap is irrigated and the ore reaches the absorption moisture, the gold rich solution will drain to the lowest part of the pad and then into the pregnant leach solution (PLS) pond before being pumped to the ADR processing plant.

1.8 Infrastructure

The Fenix Gold Project requires significant infrastructure for the mining and processing. The infrastructure includes roads, power supply, water supply, workshops, warehouses, offices, laboratories, site establishment, camp and other facilities as shown in Figure 1-2.

Power

The power supply for the Project will be generated via diesel generators. Three generators, two in continuous operation and one on standby, will be installed in the power plant located in the ADR plant. There will also be two generators installed at the crusher, which will also supply power to the mine workshops.

Grid power is located within 25 km of the mine site and connection to the grid will be considered as the Fenix Gold Mine is expanded.

Water

The 20,000 tpd project requires a water supply of up to 24 l/s. The Fenix Gold Project has access to water via a contract signed with Aguas Chañar S.A. (“Aguas Chañar”), the major water supplier to the town of Copiapo, to supply up to 80 l/s of treated town wastewater from its Piedra Colgada treatment facility located to the north of Copiapo. The original plan, outlined in the 2014 PFS, was to build a pipeline with associated power line from the Aguas

Chañar facilities to Fenix Gold along the existing main road, international road CH31, from Copiapo to Argentina, which passes within 20 km of the Project. This plan is still being considered for the future expansion of the Project, discussions are ongoing with infrastructure companies who are interested and able to finance and build the pipeline, and other mining companies who may wish to share in the benefit of the pipeline project. The capital costs, operating costs and cost of water for the larger water solution are set out in the 2014 PFS.

The water for the 20,000 tpd project will be transported by 30 tonne capacity water tankers, loading from the Aguas Chañar facility and discharging to the process plant located at the Project, a distance of approximately 158 km.

Estimated water costs are $1.56 per tonne of ore processed for the first four years of production then decrease to $1.51 per tonne for the remaining life of the project. The water cost includes the purchase price and transportation of the water to site.

The Company is currently reviewing a number of additional water options involving permitted, unused water rights, which are closer to the planned mining operations with the objective of improving the economics of the water supply to the Project.

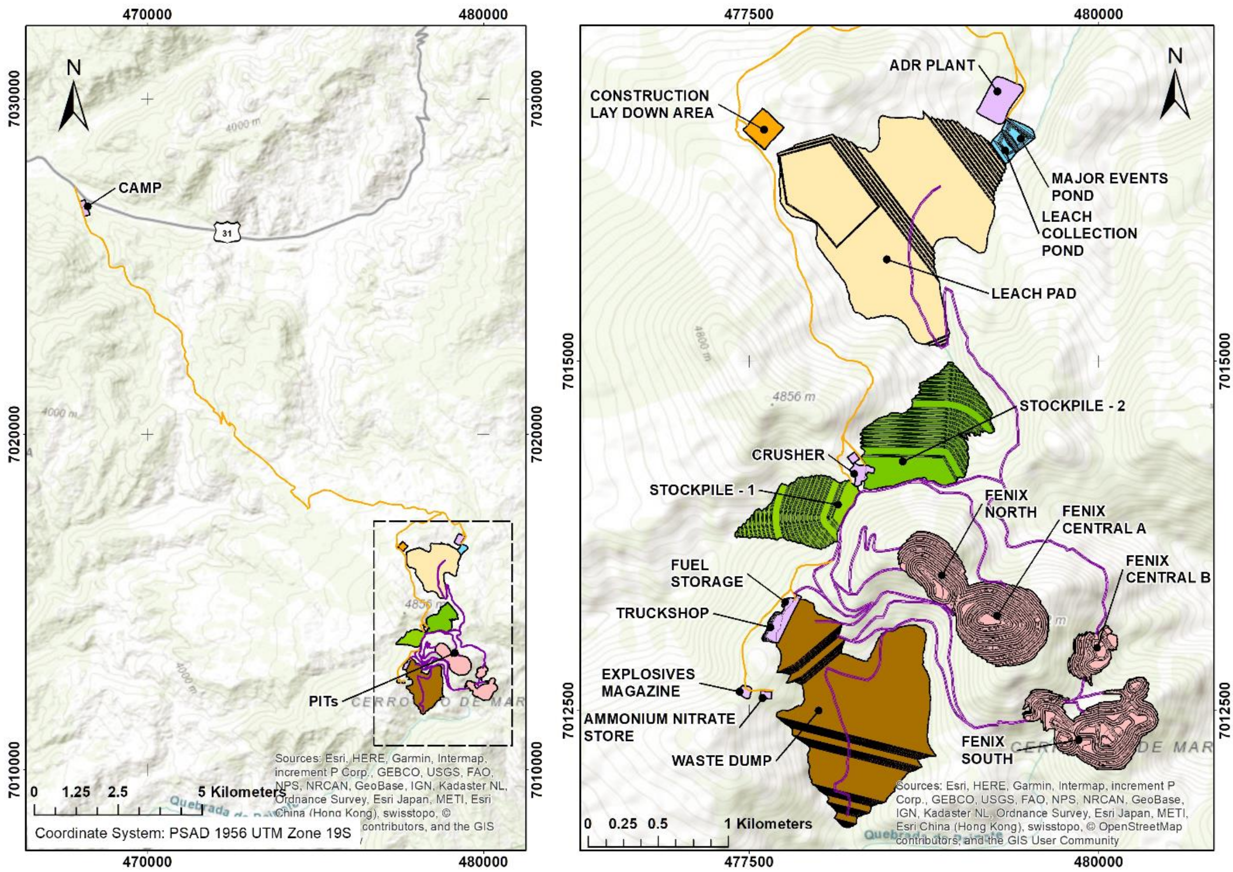

Figure 1-2: Key Infrastructure Facilities

1.9 Capital and Operating Cost Estimates

Capital and operating costs for the Fenix Gold Project were developed based the mine plan, production schedule, process plant design, and required infrastructure. The capital costs were estimated based on designs for the infrastructure, including, equipment, materials, labour, and services required for the construction and implementation of the various components. Operating costs were estimated for equipment, labour, materials, power, supplies, fuel, with supporting costs from consultants and potential suppliers to operate the mine and plant as designed.

The capital and operating cost estimates have been prepared by HLC Ingeniería y Construcción (HLC), Anddes Asociados (Anddes), STRACON and Rio2.

Capital and operating costs are presented in Table 1-3 and Table 1-4.

Table 1-3: Capital Cost Summary

Area	Capex $M	Sustaining $M	Total $M
Mining	8.58	0.85	9.43
Process Plant	35.37	16.27	51.64
Civil Construction	41.18	44.09	85.27
Contingency	14.23	13.81	28.04
Owner costs	11.84	4.57	16.41
Closure Costs		15.4	15.4
Total	**111.2**	**95**	**206.2**

Table 1-4: Summary of Operating Costs

Area	LOM Cost ($M)	US $/t ore
Mining	505.8	4.4
Processing	467.2	4.1
G&A	228.6	1.99
Off-site Overhead	41.5	0.36
Gold Sales, Insurance, Legal and Social	27.4	0.24
Royalty	1.3	0.01
Total	**1271.8**	**11.1**

1.10 Economic Analysis

The financial evaluation presents the determination of the net present value (NPV), payback period (time in years to recapture the initial capital investment), and the internal rate of return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production cost, and sales revenue. Revenues are based on the gold production.

The cash cost summary and the financial analysis results are presented in Table 1-5 and Table 1-6 respectively.

Table 1-5: Cash Cost Summary

Description	Million $	$/Oz Au*
Mining	505.8	368.8
Processing	467.2	340.6
G&A	228.6	166.7
Off-site Overhead	41.5	30.3
Gold Sales, Insurance, Legal and Social	27.4	20
Royalty	1.3	1
Total	**1271.8**	**927.4**

*$/Oz Gold recovered

Table 1-6: Financial Analysis Results

Million $	After Tax	Pre Tax
NPV @ 0%	222	305
NPV @ 5%	121	168
After-tax IRR	27.40%	31.90%
Payback Years	4.3	3

1.11 Conclusions

The following conclusions have been made:

- The Fenix Gold Project has a 16 year LOM and will produce 1.37M ounces of gold with strong economic returns:
  - LOM AISC of $997/Oz.
  - After-tax NPV5 of $121M using a base case gold price of $1300/Oz.
  - After-tax IRR of 27.4% using a base case gold price of $1300/Oz.
  - The project is expected to generate annual after-tax profits of $15.1M.
  - Cumulative LOM after-tax net cash flow of $222M.
- The use of trucked water in place of a piped water supply offers the following advantages when compared to the 2014 FS:
  - Reduced permitting time.
  - Reduced timeline to production.
  - Reduced CAPEX requirements.
- The mine design allows for a reconfiguration and upscaling of mine operations if a piped water supply becomes available.
- The identification of alternative water supplies closer to mine operations offers the potential to reduce operating costs and improve project economics.

- Connection to the Chilean power network (SIC) could potentially improve project economics.
- The plant is designed for easy upscaling from 20,000 tpd to 40,000 tpd and 80,000 tpd.
- Gold recovery of 75% is achievable with simple processing; ore crushed to a P80 size of 4 inches via a single stage Gyratory crusher with lime dosing prior to placement on the leach pad.

1.12 Recommendations

These are the recommendations for further work in order to advance to the next phase of developing the project and prepare for a full construction decision for the 20,000 tpd starter project Rio2.

Recommendations are estimated to cost $3.54M to complete (Table 1-7):

Table 1-7: Estimated cost to complete Recommendations

Item	Estimated Cost $M
Complete EIA including studies	1.20
Complete Mechanical and Electrical Engineering	1.00
Investigate Connection to SIC	0.02
Geotechnical Drilling and design	0.60
Condemnation Drilling	0.35
Optimise Mine Schedule	0.02
Model Mg Distribution	0.01
Column Leach Testing of P80 4"	0.15
Mineralogical Analysis of Head Samples	0.02
Trade-off Study Truck v Conveyor to move ore to stockpile	0.02
Production scale pilot tests of run of mine ore (ROM)	0.15
Total	**3.54**

1.12.1 General

- Complete environmental baseline studies and commence the Environmental Impact Study (EIA) presentation process.
- Complete the hydrological and geochemical study required for the EIA.
- Complete mechanical, electrical and geotechnical engineering for all the components of the project to the level adequate to apply for the relevant permits.
- Maintain and enhance relationships with relevant social and community groups throughout the EIA process.
- Plan for a phase of trial mining.

1.12.2 Engineering

- Continue to refine the civil engineering plans for the waste dump, process, heap leach and stockpile areas to level of construction ready.

1.12.3 Water

- Continue to review water supply options; new water sources offer the potential to provide time and cost savings and improve project economics, and the potential to expand the project.
- Continue discussions with Trends Industrial SA on their ENAPAC Project to build a desalination plant and a pipeline from the coast to partner mining projects.

1.12.4 Power Supply

- Investigate the potential to connect the Project to the Chilean power grid (SIC).

1.12.5 Mining

- Complete a geotechnical drilling program and study to confirm pit design parameters.
- Complete condemnation drilling in pad and waste dump footprints.
- Optimize mine planning and scheduling in order to improve costs.
- Optimization of waste dump, pad and stockpiling distances.
- Define the terms of the proposed mining alliance agreement.
- Source quotes for supply for diesel and explosives.

1.12.6 Mineral Processing

- Model Magnesium (Mg) distribution to understand lime consumption.
- Undertake a column leach test campaign on mineral crushed to a P80 size of 4” from Fenix North, Fenix Central and Fenix South to optimize gold recovery and reagent consumption in order to better define the metallurgical properties of each zone.
- It is recommended to carry out the mineralogical analysis on the remaining head samples from the KCA 2017 Fenix South leach tests to determine if chalcocite or other cyanide soluble copper minerals are present or if there are other causes for the higher refractory behaviour.
- For tests with the Fenix South material, copper extraction should be measured at the same frequency as the gold extraction to determine if there is any correlation between the two.
- Undertake geotechnical laboratory testing of leached ore samples taken after column tests are completed to better understand geotechnical properties such as shear strength and permeability.

- Mineralogical analyses to be carried out on the head samples at the start of the tests and the residues at the end of the tests.
- The Mine Plan currently shows trucking ore from the crusher to the leach pad. A trade off study for Trucking from Crusher stockpile vs Conveyor system to Pad needs to be completed. This trade off study should consider the re-handle of the future stockpile material to the Pad also.
- Production scale pilot tests of Run of Mine ("ROM") material for recoveries in the first year of mine production to determine the cost benefit of crushing vs ROM.
- Obtain formal process plant reagent quotes from suppliers.
- Continue to develop engineering solutions to manage the impact of the climatic conditions, specifically cold weather and high winds, on the operation of the leach pad and the ADR Plant.
- During the production scale pilot tests and future column tests quantify the as mined moisture content as a percentage of ROM and 4" crushed material. During these tests measure and capture the saturation percentage required for solution to percolate through the mineral, which will help confirm the water requirement for "wetting" mineral, also conduct tests on leached material to capture the residual moisture percentage retained in the mineral.
- Undertake evaporation measurements in the Pad location to confirm the evaporation rate that should be applied to the Leach Pad Water Balance.

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2 INTRODUCTION

2.1 Purpose of the Technical Report

This Technical Report has been prepared for Rio2 Limited ("Rio2"), a publicly listed mine development company listed on the TSX Venture Exchange under the trading symbol "RIO". The Technical Report is an updated Pre-feasibility Study (PFS) for the Fenix Gold Project (previously Cerro Maricunga Project) and 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").

The updated MRE for the Project is 5.0 million ounces (oz) of gold in the Measured and Indicated category and 1.4 million oz of gold in the Inferred category constrained within a $1,500 gold price pit shell. The mineral resource remains open at depth and along strike.

This PFS is strategically focused on an optimally configured mine plan which will facilitate the shortest possible timeline to construction/production, a lower initial capex, higher grades initially being mined, and a lower initial strip ratio as compared with the 2014 PFS. The PFS focuses on a low-cost heap leach gold mine with 1.83 million oz of gold reserves that will produce 1.37 million oz of gold.

The PFS contemplates mining ore at a rate of 20,000 tonnes per day ("tpd") with water for the project trucked from Copiapo. This compares with the ore mining rate of the 2014 PFS, which was a constant 80,000 tpd with water for the project being piped from Copiapo. To maximize cash flow, high-grade ore will be placed on the leach pad during the initial 13 years of production and low-grade ore will be stockpiled for leaching in the subsequent 3 years of production giving a total mine life of 16 years. Average annual gold production during the first 13 years will be 93,000 oz and 50,000 oz during the final 3 years of production as stockpiled ore is being crushed and leached.

With a large mineralized resource and potential for resources to grow through further drilling, there remains considerable opportunity to increase annual production and extend the mine life of the Fenix Gold Project. Timing to increase production will depend on transporting a greater volume of water via a pipeline, alternative water solutions closer to the project and changes to the gold price during the initial years of production.

2.2 Qualified Persons

The Qualified Persons (QP) responsible for each section of the Technical Report is given in Table 2-1.

Anthony Maycock (QP) and Andres Beluzan (QP) visited the Fenix Gold Project on April 22, 2019.

Raul Espinoza (QP) visited the Fenix Gold Project on May 14, 2019.

Denys Parra (QP) visited the Fenix Gold Project on December 4th, 2018.

Mario Rossi (QP) has not visited the Fenix Gold Project as it was not deemed necessary.

Table 2-1: Qualified Persons

Section #	Section Name	Qualified Person
1	Exec Summary	Raul Espinoza Mario Rossi Anthony Maycock Denys Parra Andres Beluzan
2	Introduction	Raul Espinoza
3	Reliance on Other Experts	Raul Espinoza
4	Property, Description and Location	Mario Rossi
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Mario Rossi
6	History	Mario Rossi
7	Geological Setting and Mineralization	Mario Rossi
8	Deposit Types	Mario Rossi
9	Exploration	Mario Rossi
10	Drilling	Mario Rossi
11	Sample Preparation, Analyses and Security	Mario Rossi
12	Data Verification	Andres Beluzan
13	Mineral Processing and Metallurgical Testing	Anthony Maycock
14	Mineral Resource Estimates	Andres Beluzan
15	Mineral Reserve Estimates	Raul Espinoza
16	Mining Methods	Raul Espinoza Denys Parra
17	Recovery Methods	Anthony Maycock
18	Project Infrastructure	Anthony Maycock Raul Espinoza
19	Market Studies and Contracts	Raul Espinoza
20	Environmental Studies, Permitting and Social or Community Impact	Raul Espinoza
21	Capital and Operating Costs	Anthony Maycock Raul Espinoza Denys Parra
22	Economic Analysis	Raul Espinoza
23	Adjacent Properties	Raul Espinoza
24	Other relevant data and information	Raul Espinoza
25	Interpretation and Conclusions	Raul Espinoza Mario Rossi Anthony Maycock Denys Parra Andres Beluzan
26	Recommendations	Raul Espinoza Mario Rossi Anthony Maycock Denys Parra Andres Beluzan
27	References	Raul Espinoza

2.3 Effective Dates

The effective date of this report is August 15, 2019.

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

Frequently Used Acronyms are listed in Table 2-2.

All currency is reported in United States Dollars ($).

All coordinates are reported are as UTM PSAD, Zone 19S.

Table 2-2: Technical Terms and Abbreviations

Abbreviation	Description
$	United States Dollar
$M	Millions of United States Dollars
%	Percentage
AAS	Atomic Absorption Spectrometry
ADR	Adsorption, Desorption, Recovery
Ag	Silver
AISC	All in Sustaining Costs
As	Arsenic
Au	Gold
BM	Block Model
CAPEX	Capital Expenditure
Cu	Copper
d	Day
DDH	Diamond Drill Hole
DIA	"Declaración de Impacto Ambiental" (Environmental Impact Statement)
ENAMI	"Empresa Nacional de Minería"
EW	Electro winning
FeOx	Iron Oxides (Collectively)
ft	Foot
g	Gram
ha	Hectare
Hg	Mercury
hr	Hour
HR	Host Rock
IRR	Internal Rate of Return
K/Ar	Potassium / Argon Geochronology
kg	Kilogram
Kg/bcm	Kilogram per bank cubic meter
km	Kilometer
koz	Thousands of ounces
ktpd	kilotonnes per day
l	Litres

Abbreviation	Description
lb	Pound
m	Meter
m^2	Square meter
m^3	Cubic meter
masl	Meters Above Sea Level
Max	Maximum Value
Mg	Magnesium
Min	Minimum Value
mm	Millimetre
MMB	*Maricunga Mineral Belt*
Moz	Million ounces
Mt	Million tonnes
Mt	Million Tonnes
Mtpa	Million tonnes per annum
Wh/t	Watt Hour per tonne of mineral
NPV	Net Present Value
NPV5	*Net Present Value discounted at 5%*
OK	Ordinary Kriging
OPEX	Operational Expenditure
oz	Ounce
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
PLS	Pregnant Leach Solution
ppb	Parts per billion
ppm	Parts per million
QA	Quality Assurance
QC	Quality Control
RC	Reverse-Circulation Drilling Method
ROM	Run of Mine
RQD	Rock-Quality Designation
SEIA	"Sistema de Evaluación de Impacto Ambiental"
SIC	Chilean Power Grid for the Central Zone
t	Tonne
t ppt	*Tonnes of precipitates*
tpd	Tonnes per day
wt%	Weight percentage
x1000	Multiple of 1000
yr	Year
Zn	Zinc

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QP´s) responsible for this report have relied upon information provided to them by the issuer (Rio2) concerning, legal, political, environmental, and tax matters relevant to this technical report.

4 PROPERTY, DESCRIPTION AND LOCATION

4.1 Location

The Fenix Gold Project (Project) is located in Chile’s III Region (Atacama) and in the Maricunga Mineral Belt (MMB), a well-known mining district with a history of mining and a gold endowment of over 70 million ounces. MMB hosts the La Coipa and Maricunga mines, and the Volcan, Caspiche, Lobo Marte and Cerro Casale deposits. The Atacama Region benefits from an experienced mining workforce, support from mining equipment suppliers and professional and technical consultants.

The Pan-American Highway and the provincial road network connect the Project to the Pacific Ocean ports at Antofagasta and Coquimbo. Chile’s central power grid passes within 25 km of the property.

The Project is approximately 117 km (straight-line) northeast of Copiapo City (III Region Capital) and is approximately 50 km west of Chile´s border with Argentina. The Project is located along the western flanks of the Chilean Andes at a mean elevation of approximately 4,200 m (Figure 4-1).

Copiapo is in the Atacama Desert and receives little annual rainfall (12 mm per year). The population of Copiapo as of 2017 was approximately 175,162 inhabitants. Copiapo has a diversified economy, but mining is the largest economic activity.

The Fenix Gold Project is centred at latitude 27°0'7.00"S and longitude 69°12'58.00"W; approximately 20 km south of Kinross Gold’s La Coipa Au-Ag mine (currently on standby), 60 km north of Kinross’s Maricunga Gold Mine (currently on residual leaching) and 40 km north of Hochschild’s Volcan Gold Project.

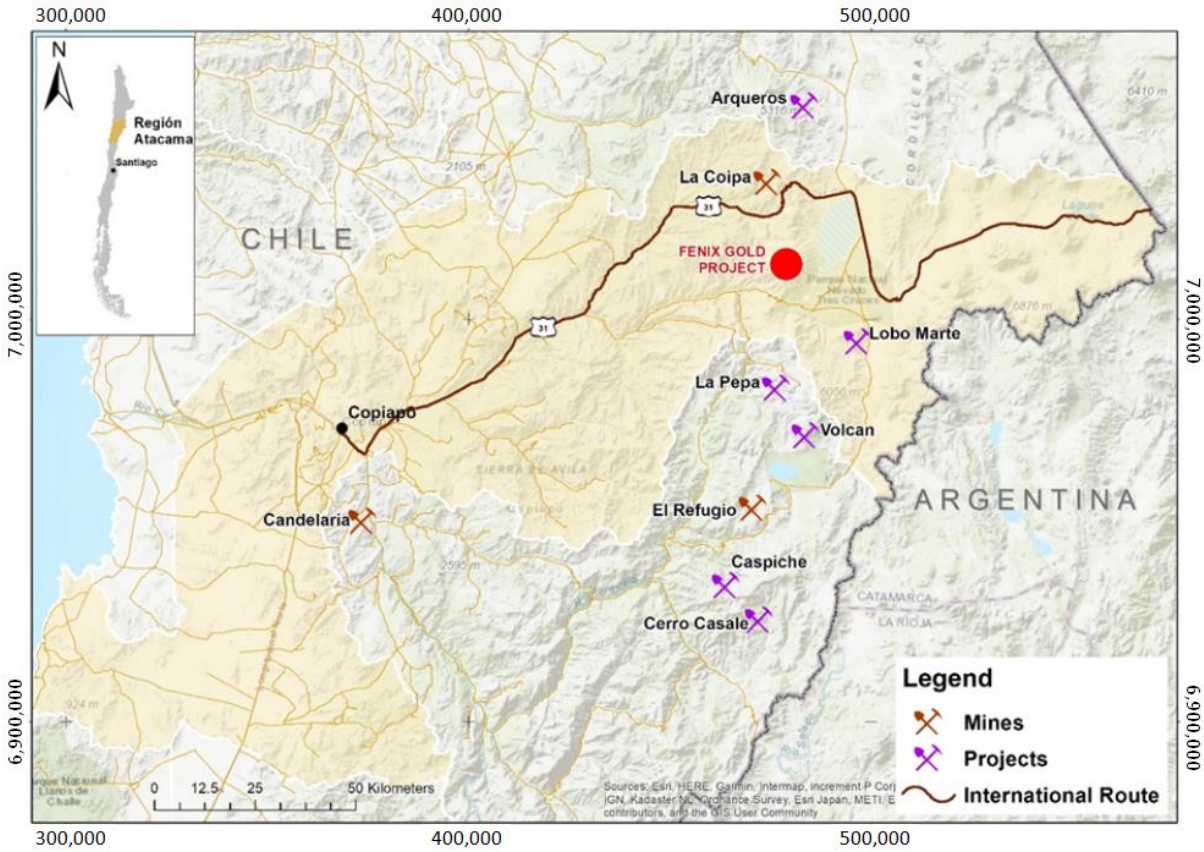


Figure 4-1: Fenix Gold Project Location

4.2 Land Tenure

Mining concessions in Chile are classified as Exploration or Exploitation concessions.

The Fenix Gold Project includes Exploration and Exploitation concessions that partially overlap, including overlapping areas; the surface area of the concessions is approximately 16,050 hectares. The Exploration and Exploitation concessions that form the Fenix Gold Project are summarized in Table 4-1, Table 4-2 and Figure 4-2.

Chile´s mining laws state that:

- Mining concessions can be held in perpetuity provided that the appropriate annual payments have been made.
- There is no requirement for a property to commence mining within a specified period.
- There is no requirement to reduce the size of a concession over time.

Annual payments to maintain the Project in good standing are up to date. The annual cost to maintain the Project concessions is approximately $78,000.

Table 4-1: Fenix Gold Project – Exploration Concessions

Concession	Type	Hectares
Maricunga II 3	Exploration	300
Maricunga II 4	Exploration	300
Maricunga II 5	Exploration	300
Maricunga II 6	Exploration	300
Maricunga II 7	Exploration	300
Maricunga II 8	Exploration	100
Maricunga II 9	Exploration	100
Maricunga II 10	Exploration	100
Maricunga II 11	Exploration	100
Maricunga II 12	Exploration	100
Maricunga II 13	Exploration	300
Maricunga II 22	Exploration	300
Maricunga II 23	Exploration	200
Maricunga II 24	Exploration	200
Maricunga II 25	Exploration	300
Maricunga II 26	Exploration	300
Maricunga II 27	Exploration	140
Maricunga II 28	Exploration	51
Maricunga II 30	Exploration	285
Maricunga II 31	Exploration	300
Maricunga II 32	Exploration	300
Maricunga II 33	Exploration	300
Maricunga II 34	Exploration	300
Maricunga II 35	Exploration	300
Maricunga II 36	Exploration	300
Maricunga II 37	Exploration	160
Maricunga II 39	Exploration	300
Maricunga II 40	Exploration	300
Maricunga II 41	Exploration	300
Maricunga II 42	Exploration	300
Maricunga II 43	Exploration	300
Maricunga II 44	Exploration	300
Maricunga II 45	Exploration	300
Maricunga II 46	Exploration	300
Maricunga II 47	Exploration	300
Maricunga II 48	Exploration	300
Maricunga II 49	Exploration	300
Maricunga II 50	Exploration	200
Mónica III 2	Exploration	200
Mónica III 3	Exploration	300
Mónica III 4	Exploration	200
Mónica III 5	Exploration	200
Mónica III 6	Exploration	160
Mónica III 7	Exploration	160

Concession	Type	Hectares
Mónica III 9	Exploration	300
Mónica III 10	Exploration	90
	Total	**11,146**

Table 4-2: Fenix Gold Project - Exploitation Concessions

Concession	Type	Hectares
Cerro Maricunga 1 1/17	Exploitation	170
Maricunga 14 1/10	Exploitation	100
Maricunga 15 1/10	Exploitation	100
Maricunga 16 1/10	Exploitation	100
Maricunga 17 1/10	Exploitation	100
Maricunga 18 1/10	Exploitation	100
Maricunga I 1, 1/60	Exploitation	279
Maricunga I 2, 1/60	Exploitation	300
Maricunga I 19, 1/60	Exploitation	290
Maricunga I 20, 1/60	Exploitation	300
Maricunga I 21, 1/60	Exploitation	300
Maricunga I 27, 1/60	Exploitation	60
Maricunga I 28, 1/60	Exploitation	30
Maricunga I 29, 1/60	Exploitation	178
Maricunga I 38, 1/60	Exploitation	297
Mary 4 1/30	Exploitation	300
Mary 5 1/20	Exploitation	200
Mary 6 1/30	Exploitation	300
Mary 7 1/20	Exploitation	200
Mary 8 1/30	Exploitation	300
Mary 9 1/20	Exploitation	200
Mary 10 1/30	Exploitation	300
Mary II 2 1/10	Exploitation	100
Mary II 3 1/10	Exploitation	100
Monica 1 1/40	Exploitation	200
	Total	**4,904**

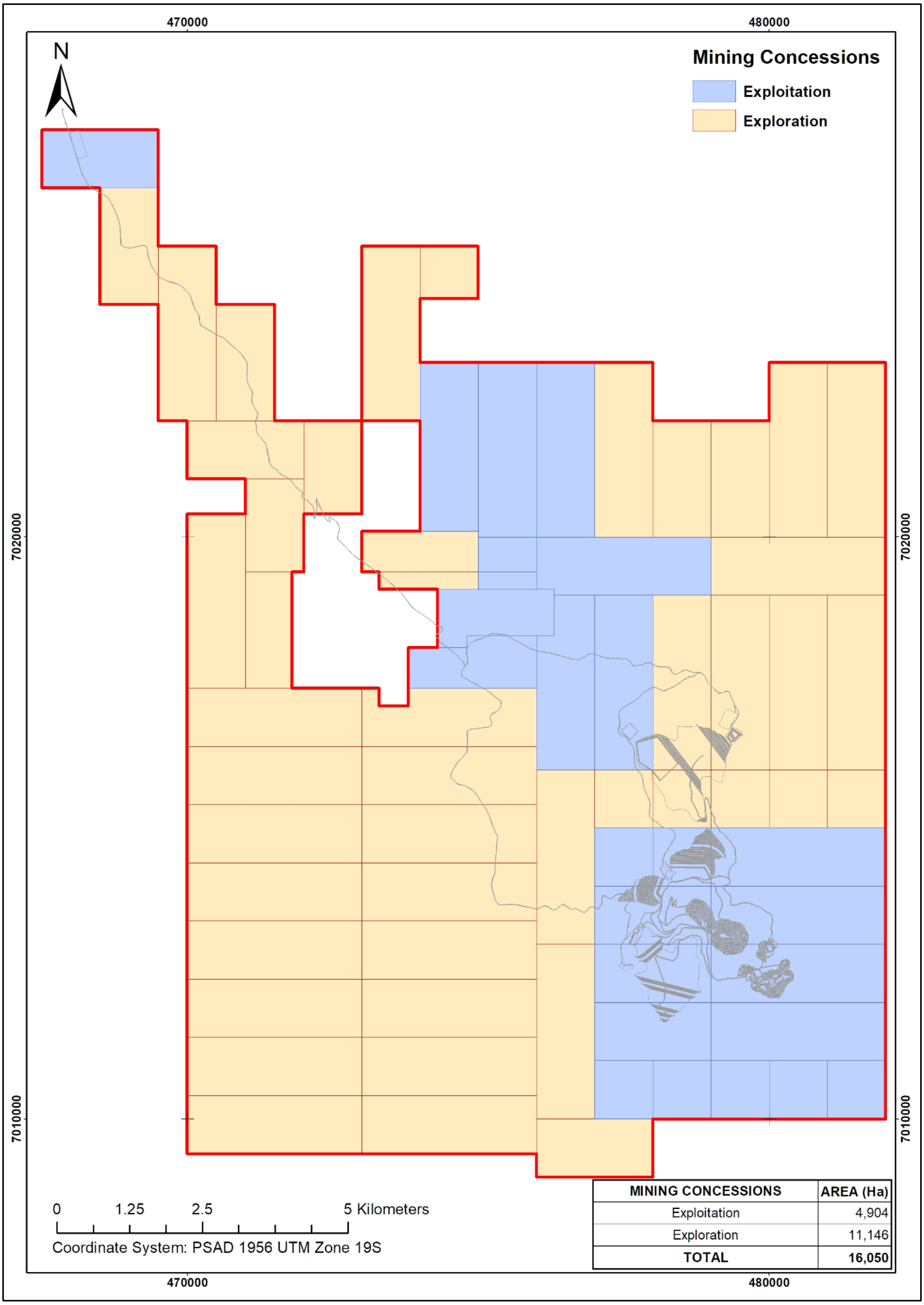


Figure 4-2: Fenix Gold Project Concession Map

4.3 Environmental Liabilities

There are no known environmental liabilities at the Project.

Rio2 has contracted Mineria y Medio Ambiente Limitada (MYMA) to support the environmental permitting needs of Fenix Gold Project. MYMA has extensive experience elaborating environmental studies for the mining sector in Chile.

MYMA commenced an Environmental Baseline Study for the Project in November 2018, and have noted the following:

- All key environmental sustainability variables identified and analysed (potential environmental impacts) in this report can be fully addressed and there are measures in place to effectively manage them.
- A successful environmental permitting is closely linked to the availability of relevant (project design) and essential information (baseline studies and evaluation of environmental impacts).
- Rio2 should continue discussing 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. Although there are no indigenous communities in the area where the Project will be developed, it is fundamental to maintain good relationships with the neighbouring communities to enhance communications and to facilitate the environmental permitting of the project.

4.4 Permits Acquired

MYMA developed an Environmental Impact Statement (DIA by its Spanish acronym) – "Fenix Gold Drilling", submitted to the Environmental Assessment Service (SEA by its Spanish acronym) in April 2019, which considers the execution of 249 RC holes and 27 DDH holes, the latter of which include geometallurgical and geotechnical drilling.

4.5 Ownership, Royalties and Other Payments

The Fenix Gold Project is 100% owned by Fenix Gold Limitada, a subsidiary of Rio2, and is not subject to third party royalties, back-in rights or payments.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The Project is approximately 680 km north of Santiago, Chile´s capital city. Santiago and Copiapo, the city closest to the Fenix Gold Project, are connected via the National Road network and daily flights.

From Copiapo, the Project is accessed via paved highway, salt paved road, and a 20 km section of maintained single-track dirt road (Figure 5-1). The distance between Copiapo and the Project is approximately 140 km and takes approximately 2.5 hours to drive.

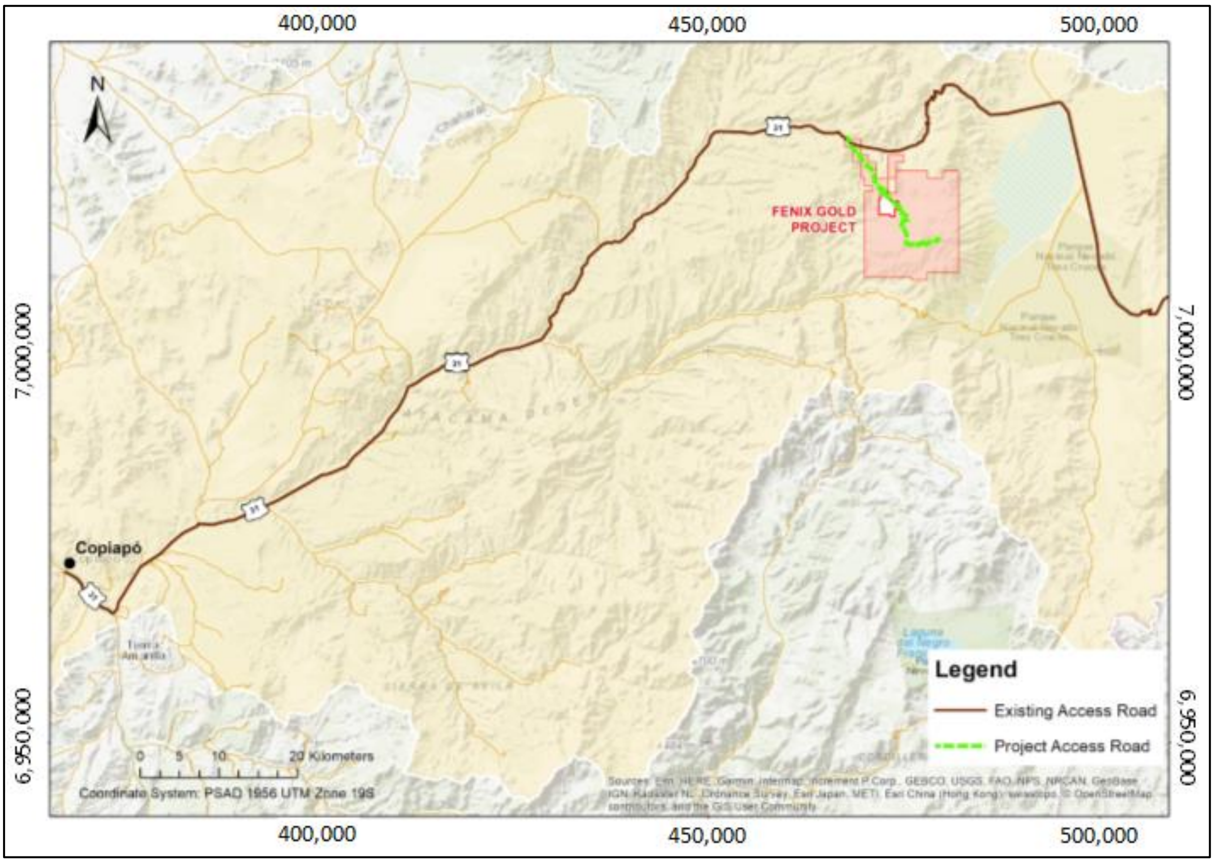


Figure 5-1: Project access from Copiapo

5.2 Climate

The Project is located on the western slopes of the Andes Cordillera in the high desert of the Atacama Region of Chile between 3800 and 5,000 masl.

The climate is extremely dry and annual precipitation totals approximately 12 mm falling largely as snow during the winter months (June to September). Short sporadic rainstorms can occur between January and May. Evaporation from surface varies between 1,500 to 2,000 mm per year resulting in the extremely arid conditions.

Average temperatures in the project area range between -30°C at night in winter to 20°C during the day in summer.

Flora is sparsely developed and fauna is limited to transient vicuñas.

5.3 Local Resources and Infrastructure

There are no significant population centres or infrastructure in the immediate vicinity of the Project. Small-scale arable and livestock farmers with indigenous heritage are present in the valleys that drain the highlands. Farming activity is not recorded in close proximity to the Fenix Gold Project.

Chile has an established mining industry and high-quality mining technology, infrastructure, supplies and professionals are available in country. Copiapo, an established regional mining support and logistics hub, has a population of approximately 175,162 and can supply a skilled and experienced mining and mineral processing workforce.

The Project is approximately 25 km from Chile´s national power grid Central Interconnected System (SIC). Onsite electrical generators are considered in the proposed mine plan but connection to SIC could provide a sufficient and reliable supply of electrical power for the proposed mining operations.

Surface water does not flow through the Project area and no underground water sources have been identified.

Rio2 has agreed to a 13-year contract with Aguas Chañar S.A. to supply treated industrial water at a rate of 80 l/s. Under the terms of the contract, Aguas Chañar S.A. would supply water from their water treatment facility at Copiapo, and Rio2 would truck water to site.

Rio2 has identified sites within the project area that, subject to relevant permitting and studies, could be used to establish infrastructure for an open-pit mining operation, including: heap leach pad, waste dump, low-grade stockpile and processing plant.

5.4 Physiography

The Project ranges between 3800 and 5000 masl with topography characterized by broad open areas with moderate relief, pronounced slopes and prominent ridges (Figure 5-2). These features reflect horst and graben tectonics and recent volcanism.

Figure 5-2: Looking south from planned leach dump Location towards Fenix North outcrop

6 HISTORY

6.1 Exploration History of the Project

6.1.1 Project Area Recognition

Private prospectors identified mineralization in the general area of the Project in the early 1980’s.

6.1.2 SBX

In December 2007, SBX, a private Chilean exploration company, constructed access roads and conducted trench sampling and mapping at 1:25k scale in the Project area. Classic “Maricunga Style” Black Banded Veinlets (BBV) were detected in the Cerro Maricunga intrusive breccia complex with gold grades, ranging from 0.2 g/t to 3 g/t, and SBX named the Project “Cerro Maricunga”.

Minera Newcrest Chile Ltda (MNCL) entered in to an agreement with SBX to evaluate the Project and took 325 surface samples that confirmed anomalous gold values over 2.5 km strike. Following their evaluation, MNCL choose to exit the option agreement with SBX.

In 2008, Gold Fields (GFC) entered into an agreement with SBX to evaluate the Project and conducted independent mapping, trenching, channel sampling, induced potential/resistivity and magnetic surveys. Following their work, GFC concluded that Cerro Maricunga had the potential to host a significant gold deposit, and that exploration drilling was warranted. However, GFC elected to discontinue their interest in the Project.

Between 2008 and early 2010, SBX privately funded an extensive program of surface sampling, trenching, geophysical surveys, metallurgical testing and an eight-hole maiden diamond drill hole program (Phase I - 2,142 m).

Phase 1 drill results were positive and in October 2010, SBX took the Cerro Maricunga Project public after listing on the Toronto Stock Exchange as Atacama Pacific Gold Corporation (“Atacama”).

6.1.3 Atacama Pacific Gold Corporation

In October 2010, Atacama commenced Phase II drilling at the Project and generated further positive results supporting the potential for significant oxide-gold deposit. By the end of April 2011, Atacama had drilled 33,438 m over a combined 90 DDH and RC holes.

Metallurgical test work conducted during 2011 indicated that oxide-gold mineralization at Cerro Maricunga was amenable to heap-leach processing. Eleven column tests and 36 bottle roll tests indicated gold recoveries in the range of 80% at a 19 to 25 mm crush. Column testing on material crushed to 50 mm indicated gold recovery of 78%.

A third phase of drilling (Phase III, 45,983 m) designed to define the extents of mineralization began in 2011. Trenching and metallurgical sampling continued in parallel with drilling.

Atacama funded a program of infill drilling and additional metallurgical testing (Phase 4, 26,335 m) that concluded in May 2013. Following the results of Phase 4 drilling, Atacama published a Pre-feasibility Study (PFS) for the Cerro Maricunga Project (PFS, 2014), outlining a large-scale open-pit oxide-gold heap-leach mine operation.

In 2017, Atacama commenced Phase V drilling that included three PQ diameter diamond drill holes for metallurgical testing to better define the primary crushing circuit.

6.1.4 Rio2 Limited

In July 2018, Rio2 and Atacama announced a business combination, and Rio2 took control of Cerro Maricunga Property (Press Release 1).

To differentiate the Property and to stop confusion surrounding the multiple use of the name “Maricunga” such as Maricunga Desert, Cerro Maricunga Project, and Maricunga (Refugio) Mine, Rio2 renamed the Project “Fenix Gold Project”.

Rio2 incorporated Fenix Gold Limitada (FGL), a Chilean company that is conducting the Property development and will be the mining operator.

Since taking control of the Project, Rio2 has completed the following:

- Phase VI drilling consisting of 7066 m over 39 RC drill holes within the resource area.
- Twelve trenches and took 729 channel samples (2 m length) over the resource area.
- Relogging 28,176 m of historical diamond drill core from 79 holes, and 21,184 m of RC chips from 59 holes.
- Rio2 engaged recognized Economic and Structural Geologist Dr. Greg Corbett to spend one week at the Project to investigate geological controls on mineralization.
- Developed the first 3-D geological model for the Project.
- Engaged environmental consultants MYMA to commence environmental baseline studies for a future EIA.

6.2 Resource Development History of the Project

6.2.1 Initial Resource Estimate 2011

Based on 25 DDH and 65 RC holes, Atacama reported the maiden Mineral Resource Estimate (“MRE”) (Press Release 2) for the Project, summarized in Table 6-1.

The MRE was an Ordinary Kriged model based on a 0.15 g/t Au grade shell to define the modelling boundary. The MRE considered a 0.3 g/t Au cut-off grade and was not constrained by a conceptual open pit optimization.

Table 6-1: Initial Resource Estimate, 2011

Cut-off	Indicated Category			Inferred Category		
	Million	Grade	Gold Ounces	Million	Grade	Gold Ounces
Au g/t	Tonnes	*Au g/t*	*x1000*	Tonnes	*Au g/t*	*x1000*
0.1	163.1	0.40	2,094	354.6	0.29	3,321
0.2	134.1	0.45	1,949	202.5	0.40	2,626
0.3	**92.8**	**0.54**	**1,616**	**116.7**	**0.52**	**1,949**
0.4	59.8	0.65	1,247	69.2	0.64	1,429
0.5	40.8	0.74	973	47.7	0.73	1,121
0.6	28.7	0.83	761	34.4	0.80	887
0.7	19.4	0.91	569	21.4	0.90	617
0.8	13.0	0.99	413	13.8	0.98	435

6.2.2 Resource Update 2012

An updated MRE (Press Release 3), summarized in Table 6-2, considering 63 DDH and 157 RC holes, was prepared for the Project in 2012. The updated MRE was prepared by NCL Consultores Limitada, Magri Consultores Limitada and NTK Consultores Limitada.

The updated MRE was based on Ordinary Kriging and was bound to a 0.15 g/t Au grade shell. The resource was quoted at 0.3 g/t Au cut-off grade and was not constrained by a conceptual open pit optimization.

The 2012 MRE includes Measured resources for the first time.

Table 6-2: Resource Update, 2012

Cut-off	Measured		Indicated		Measured & Indicated			Inferred		
Au g/t	Million Tonnes	Au g/t	Million Tonnes	Au g/t	Million Tonnes	Au g/t	Moz Au	Million Tonnes	Au g/t	Moz Au
0	66.627	0.41	202.619	0.4	269.246	0.4	3.464	271.613	0.33	2.908
0.1	66.576	0.41	202.567	0.4	269.143	0.4	3.464	271.275	0.33	2.907
0.2	60.411	0.44	187.526	0.41	247.937	0.42	3.344	226.338	0.36	2.654
0.3	**40.733**	**0.53**	**123.141**	**0.5**	**163.874**	**0.51**	**2.667**	**120.738**	**0.47**	**1.81**
0.4	24.535	0.64	71.241	0.61	95.776	0.62	1.912	57.832	0.6	1.118
0.5	15.14	0.77	42.778	0.72	57.919	0.74	1.37	32.286	0.73	0.754
0.6	9.935	0.88	26.324	0.84	36.259	0.85	0.99	19.737	0.84	0.535
0.7	6.758	1	16.449	0.95	23.208	0.96	0.719	12.845	0.95	0.392
0.8	4.56	1.12	10.503	1.07	15.063	1.08	0.524	8.134	1.06	0.278

6.2.3 PEA 2013

In 2013, NCL Consultores Limitada developed a Preliminary Economic Assessment (PEA) for the Project. The PEA developed a conceptual plan for an open-pit heap-leach operation based on the 2012 MRE considered 261 Mt @ 0.40 g/t Au for 3.4 Moz (Table 6-3) of mineralized material.

The PEA evaluated an owner operator model based on large-scale material movement and three stage crushing of mineralization prior to leaching. The PEA reported a positive outcome and indicated capital requirements of $515M and sustaining capital of $249M. Based on a gold price of $1450/oz, the after tax NPV (5%) for the Project economics were reported as $513M with an IRR of 26.6%, and a 3-year payback.

Table 6-3: Mining Inventory for PEA, 2013

Year	Mineralized Material			Waste	Total	Plant Feed				
	Million Tonnes	Au g/t	Contained Ounces Au x1000	Million Tonnes	Tonnes x1000	Million Tonnes	Au g/t	Contained Ounces Au x1000	Average Recovery %	Recovered Ounces Gold x1000
Pre-strip	6.62	0.38	81	4.78	11,400	-	-	-	-	-
Y1	22.57	0.44	319	27.87	50,450	29.20	0.43	400	79.60%	318
Y2	29.20	0.41	389	55.35	84,550	29.20	0.41	389	79.50%	309
Y3	29.20	0.41	384	55.35	84,550	29.20	0.41	384	79.50%	305
Y4	29.20	0.4	374	55.35	84,550	29.20	0.4	374	79.40%	297
Y5	29.20	0.35	330	55.35	84,550	29.20	0.35	330	79.00%	261
Y6	29.20	0.36	340	55.27	84,473	29.20	0.36	340	79.10%	269
Y7	29.20	0.38	357	49.93	79,129	29.20	0.38	357	79.20%	283
Y8	26.76	0.4	340	41.04	67,799	26.76	0.4	340	79.40%	270
Y9	20.69	0.47	309	18.61	39,300	20.69	0.47	309	80.00%	247
Y10	8.54	0.58	159	3.13	11,670	8.54	0.58	159	81.00%	129
Y11	0.74	0.66	16	0.12	860	0.74	0.66	16	81.70%	13
Total	**261,123**	**0.4**	**3,397**	**422,158**	**683,281**	**261,123**	**0.4**	**3,397**	**79.50%**	**2,700**

6.2.4 PFS 2014

In 2014, NCL Consultores Limitada and Magri Consultores Limitada produced the first Pre-feasibility Study (PFS) for the Project. The PFS considered an updated MRE based on 86 DDH and 234 RC holes (Table 6-4). Ordinary Kriging was used for modelling bound to a 0.15 g/t Au grade shell. The MRE was quoted at 0.15 g/t Au cut-off grade and was not constrained by a conceptual open pit optimization.

Table 6-4: Resource Update for PFS, 2014

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 x1000	Tonnes (Millions)	Grade (g/t Au)	Gold Ounces x1000
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
Total	**152.8**	**0.39**	**281**	**0.37**	**433.8**	**0.38**	**5,249**	**57.9**	**0.32**	**603**

The 2014 PFS reported a Mineral Reserve of 294 Mt @ 0.40 g/t Au for 3.7 Moz (Table 6-5).

Table 6-5: Mineral Reserve for PFS, 2014 at 0.15 g/t Au Cut Off

Category	Tonnes Millions	Au g/t	Gold Ounces x1000
Proven	126.9	0.39	1,603
Probable	167.6	0.4	2,140
Total Proven and Probable	**294.4**	**0.4**	**3,743**

The 2014 PFS presented a project that would require $399M CAPEX investment and $188M of sustaining CAPEX. The mine scenario considered an equipment-leasing owner operated mining model for a large-scale material movement and three stage ore crushing.

The base case Au price used for optimization was $1300/oz. The after tax NPV (5%) for the project was $409M and the after tax IRR was 25% with a 3-year payback.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geological Setting

Paleozoic to Triassic basement geology of the north to south trending Maricunga Mineral Belt (MMB) is intruded by of a series of Mesozoic-Cenozoic volcanic arcs and plutons related to the subduction of the Pacific tectonic plate under the South American plate. Volcanic deposits and flows limit the exposure of basement lithologies.

Subduction related tectonism has had a pronounced effect on structural trends in the MMB. Northwest to north-northeast orientated thrust faulting occurs with approximately perpendicular transform faulting.

Hydrothermal and mineralizing systems in the MMB often developed in the structural framework described above.

Volcanism and the development of caldera complexes in the MMB has been K/Ar dated between 24-13 million years. Caldera development evolved from andesitic to dacitic and advanced west to east:

- Early andesitic caldera complexes have been dated between 24-20 million years.
- Later dacitic volcanism is dated between 14 and 13 million years.

Hydrothermal alteration and precious metal mineralization centred on caldera complexes is associated with both early and later stages of volcanism.

Several significant mineralized hydrothermal systems are known in the MMB where over 70 million ounces of gold has been defined regionally in deposits such as: Marte-Lobo, La Pepa, La Coipa, El Volcan, Maricunga (previously Refugio), Aldebarán (previously Cerro Casale) (Figure 7-1).

Various mineral deposit styles are recognized in the MMB:

- High sulphidation epithermal with marked hydrothermal alteration (i.e. Ojo de Agua and El Volcan).
- Low sulphidation epithermal.
- Au-Cu porphyry (i.e. Marte & Lobo, Maricunga and Aldebaran).

Mineral deposits often represent multiple styles of mineralization often telescoped (superimposed) over one another.

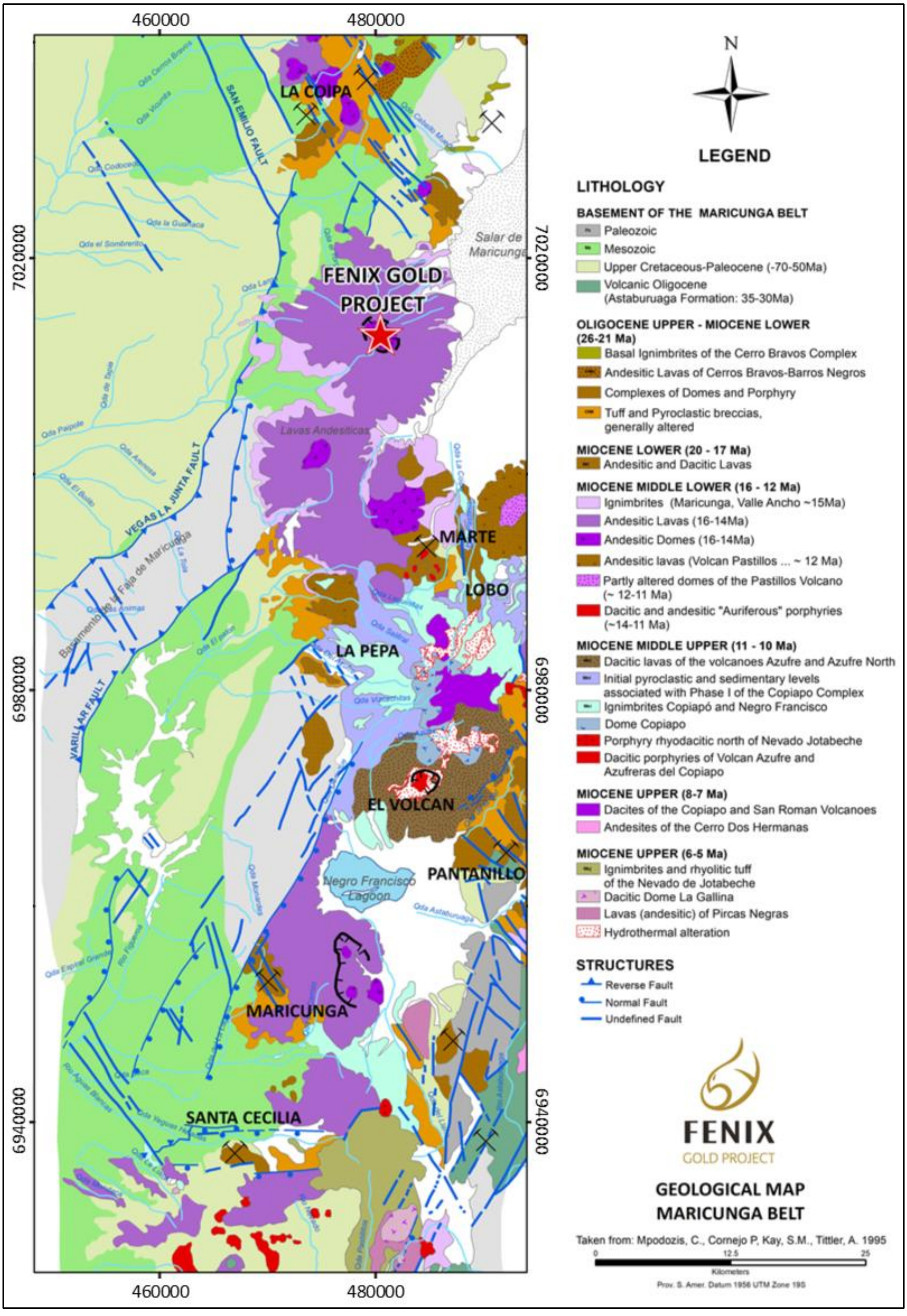


Figure 7-1: Regional Geology and Deposits location of the Maricunga Mineral Belt (Mpodozis et al 1995)

7.2 Local Geology

A sub-volcanic andesite dome intruded Triassic-Jurassic basement sediments (shale and limestone) of the El Mono Formation during the Miocene. Dacite doming and contemporaneous breccias complexes subsequently intruded the andesite. These intrusions and breccia complexes form the Cerro Maricunga strato-volcano that hosts mineralization at the Project.

During the formation of Cerro Maricunga, volcanic deposits and flows (Figure 7-2 and Figure 7-3), were deposited unconformably over the Triassic-Jurassic basement.

Mineralization at the Project is associated with sub-volcanic dacitic and andesitic intrusive domes and breccia complexes (phreatic, phreatomagmatic and magmatic) exposed in the core of the eroded Cerro Maricunga strato-volcano.

Mineralization has been defined over a 2.5 km northwest-southeast strike and up to 600 m across. Drilling has confirmed oxide-gold mineralization to a depth of 600 m below surface and the deposit remains open.

Mineralization at the Project has been divided in the Fenix North, Fenix Central and Fenix South areas, these areas host similar mineralization offset by late northeast trending faulting.

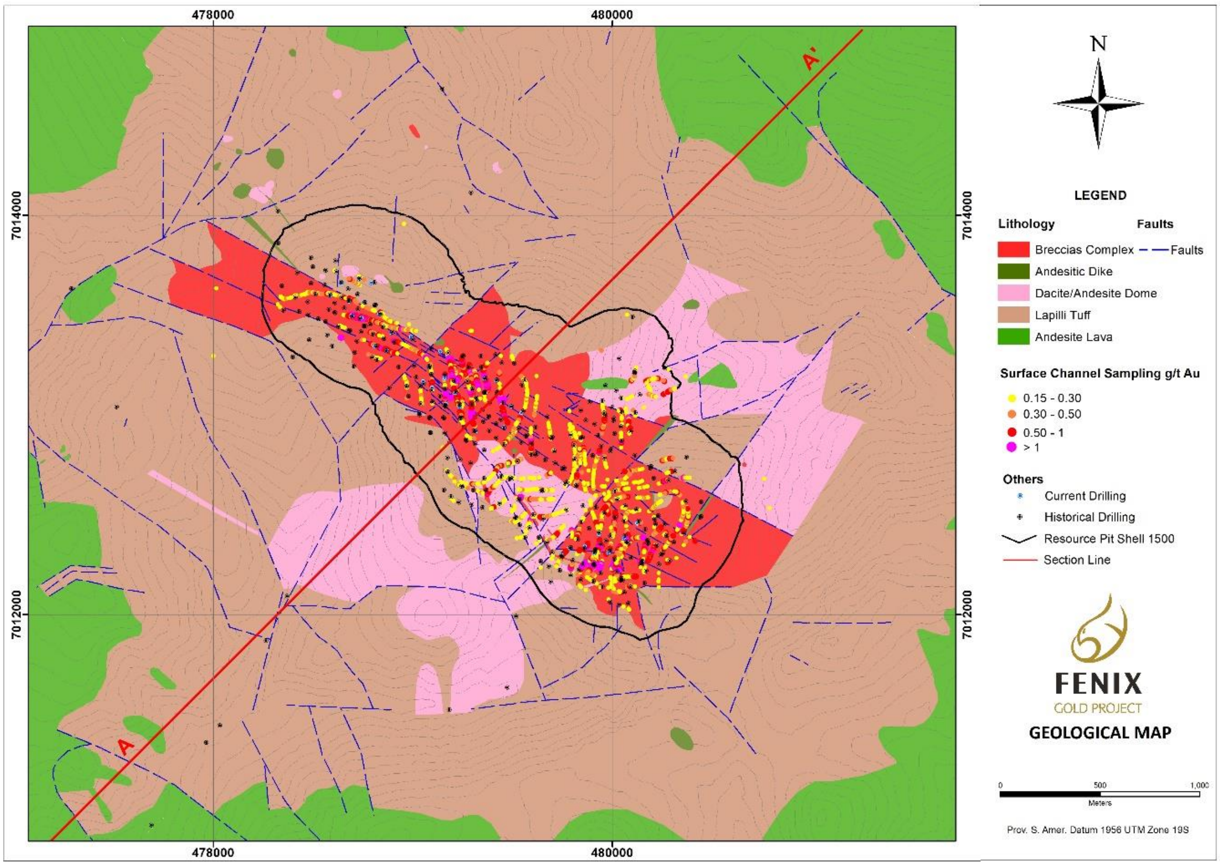


Figure 7-2: Geology of the Fenix Gold Project (Dietrich, 2010)

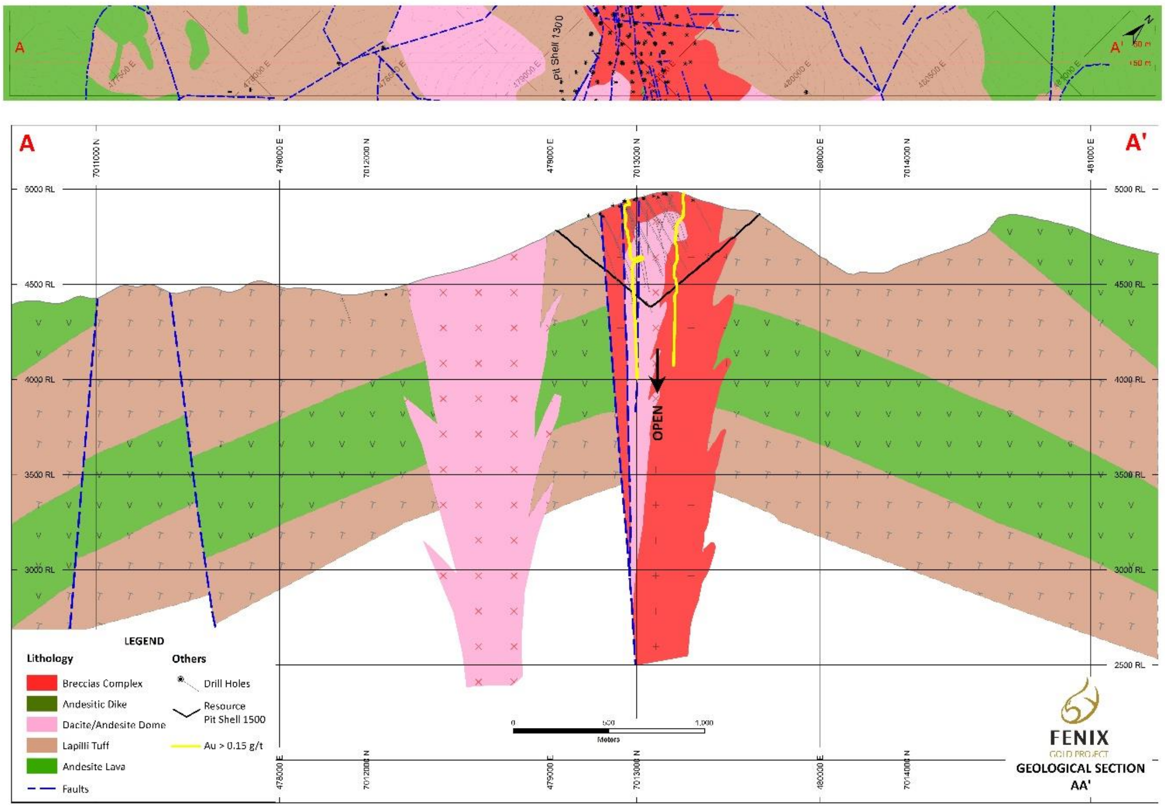


Figure 7-3: Cross Section of Fenix Gold Project Looking NW (Garay, 2019, based on Dietrich, 2010)

7.2.1 Lithology

Oxide-gold mineralization at the Project is hosted in a dacitic dome and breccia complex intruded through andesite (Figure 7-4). Volcanic deposits and flows are deposited over the flanks of the dome, including pyroclastic deposits, lapilli and crystal tuffs, dacite-andesite lava flows, tuffaceous arenite, volcano-clastic conglomerate and laharic deposits.

Multiple phases of intrusion and brecciation events are recognized at the Project, such as:

- Emplacement of andesite dome through basement sedimentary sequence.
- Intrusion of dacite flow dome complex and breccia development.

Away from the contact with dacite breccia complex, the andesite intrusion is fresh and massive. Elongate andesite breccia clasts at the contacts between the andesite and the crosscutting milled matrix breccias of the diatreme flow dome complex, are indicative of vertical emplacement of the milled breccias (Photo 1 in Figure 7-5).

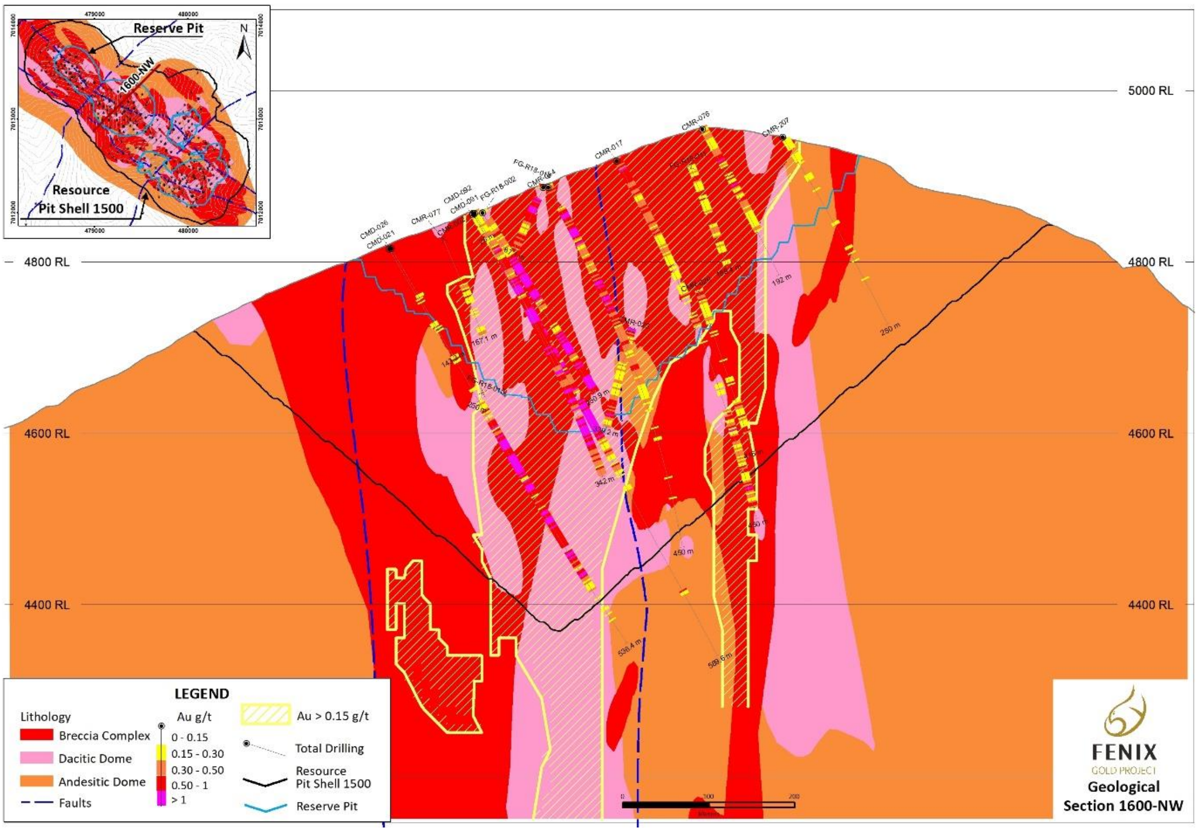


Figure 7-4: Cross section in the Central Part of Fenix Gold Deposit. Gold mineralization is hosted in the Breccia Complex and the Dacitic Dome.

Local embayed andesite clast margins are interpreted to result from erosion of the andesite by the milled matrix breccias.

Locally derived angular and fresh andesite clasts are recognized throughout more abraded and altered milled matrix breccias, which have undergone significant upward transport and emplacement (Photo 2 in Figure 7-5).

The breccia complex represents the main breccias hosted within the diatreme flow dome complex, developed by clast abrasion and alteration during the forceful upward emplacement of the diatreme breccia pipes driven by phreatomagmatic eruption of the rising dacite domes (Photo 3).

Consequently, breccias contain rounded dacite and vein clasts, which are matrix supported within a silicified rock flour matrix of comminuted clasts. The milled matrix breccias contained within the diatreme breccia cut through the previously emplaced high-level andesite domes and therefore host angular fresh andesite clasts, with local shingle-like forms indicative of collapse.

Re-brecciated clasts, and crosscutting tuffisite dykes, attest to the polyphasal character of the diatreme flow dome complex, and vein clasts are indicative of continued mineralization during this process.

Compositional variations of the dacite and associated breccias provide an order of emplacement within a paragenetic sequence, from early to late:

- The quartz eye dacite, characterized by only weak alteration and coarse-grained easily discernible quartz eyes, is best developed at the margins of the diatreme flow dome complex as the earliest intrusion and forms crowded milled matrix breccias.
- The silica-poor dacite is interpreted as the magmatic source associated with breccias, which cut the quartz eye dacite and associated intrusions with altered contacts, and are cut by the milled matrix breccias (below). It is compositionally transitional to an andesite as the two are difficult to distinguish and termed a daci-andesite in some earlier literature.
- The main milled matrix breccia is typically dominated by an oxidized (weathered) and silicified, rock flour matrix derived from the comminution (milling) of rock clasts which are dominated by a fine to medium grained dacite, although some portions contain abundant angular fresh andesite clasts. Despite the strong oxidation, disseminated magnetite and clasts of magnetite flooded breccias are common within this breccia.
- Tuffisite Dykes, cut all earlier breccias and intrusions, as a final phase of the main milled matrix breccias, and locally display a relationship to the sheeted, banded, quartz vein mineralization (photos 4, 5 & 6), no doubt by exploitation of the same structures by the tuffisite dykes and later quartz veins. The dacite intrusions responsible for breccia formation are interpreted as the source rocks for later Au mineralization.
- Orange to Red Fine Grained Matrix Breccias cut the earlier breccias as the last breccia event, which may feature the entry of oxygenated groundwater, which has oxidized the magnetite and any pyrite to provide iron oxide colours.

		
Photo 1. Brecciated margin of a fresh andesite with embayed clast rims, at the contact with a cross cutting, vertically emplaced, milled matrix breccia of the diatreme flow dome complex.	Photo 2. Elongate shingle-like andesite breccia clast within a milled matrix breccia cut by sheeted veins, DDH249, 309m, 0.75 g/t Au.	Photo 3. Rebrecciated breccia clasts in outcrop.

		
Photo 4. Early fresh angular andesite clasts and a vein clast within the later milled matrix breccia cut by a later tuffisite dyke, DDH249, 511m, 0.54 g/t Au.	Photo 5. Tuffisite dyke cuts silica-poor dacite, DDH249, 392.4m, 0.42 g/t Au.	Photo 6. Tuffisite dyke cuts earlier milled matrix breccia and is exploited by banded sheeted quartz veins, DDH249, 372.1m, 1.46 g/t Au.

Figure 7-5: Breccia Textures

7.2.2 Structure

Mineralization at the Fenix Gold Project straddles a regionally significant northwest trending structural zone, and mineralization is typically hosted in northwest trending structures.

Three structural systems have been defined at the Project; a northwest fault system, a tensional east-west system, and a late northeast fault system (Figure 7-6):

- The northwest fault system consists of three principal sub-parallel northwest striking faults which cross cut the northern portion of Fenix Central and Fenix North zones. Dips are vertical to sub-vertical. Strike slip movement partially controls the location of intrusions and mineralization.
- The East-West system is tensional from the NW system. Locally, the EW system is controlling the emplacement of the Black Banded Veins, which are associated with the gold mineralization.
- Post mineralization, sub-vertical northeast trending normal faulting that has divided mineralization into three blocks (Fenix North, Fenix Central and Fenix South).

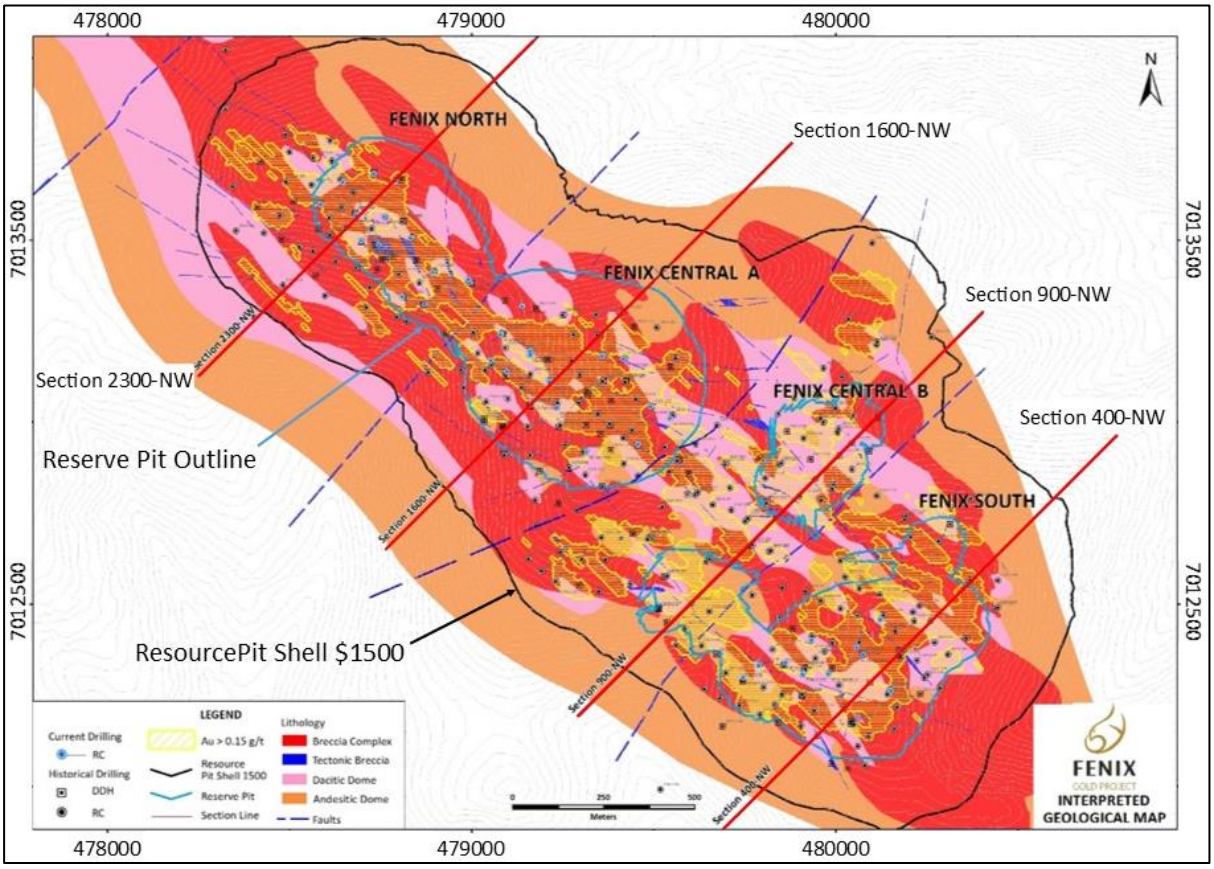


Figure 7-6: Summarized Structural Geology. The deposit is elongated to the NW, this system is cut by later NE trending system.

7.2.3 Alteration

Argillic alteration of the permeable volcanic breccias present between the inner diatreme flow dome complex and the outer rim of the volcano is assumed to have included original pyrite, which has weathered to provide the outer red stain (Photo 7 in Figure 7-7). This alteration pre-dated the emplacement of the diatreme flow dome complex breccia.

The early andesite domes are essentially fresh. These take the place of the post-mineral andesite dykes interpreted by earlier workers.

The milled matrix breccias display silicification, with local smectite varying to illite alteration. The silicification is much stronger in the later main milled matrix breccias than the earlier breccias related to the quartz eye dacite. This style of alteration is not typical of milled matrix breccias formed by phreatomagmatic eruptions, which are typically characterized by strong illite alteration with abundant fine disseminated pyrite.

The silicification has rendered these milled matrix breccias more competent than is typical for these rocks and so therefore capable of hosting crosscutting mineralized quartz veins. These rocks display well developed supergene oxidation (weathering) to the limit of drill investigation in central Fenix at about 600 m below surface (deepest drilling), although much of the magnetite remains preserved.

Magnetite is common throughout the Fenix rocks as:

- The dacite intrusions host sufficient primary magnetite to provide a strongly magnetic character. Magnetite alteration is common at the contacts between intrusions and pre-existing milled breccias (photo 8).
- Milled matrix breccias contain locally abundant disseminated magnetite, clasts of magnetite (photo 9) and magnetite breccias (photo 10), magnetite altered intrusions (photo 11), and local magnetite flooding. While much of the magnetite is clearly clastic, having formed prior to brecciation, the latter style is indicative of syn-breccia magnetite alteration.

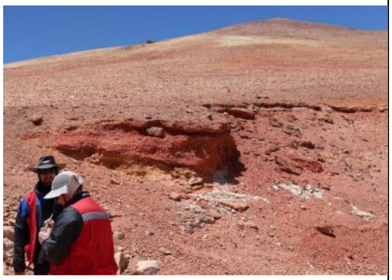


Photo 7. Lateritic weathering which has produced the pronounced red colour anomaly exposed in a road cutting.	Photo 8. Dacite cuts silica-poor dacite, DDH92, 366.2m, 0.27Au.	Photo 9. Fine grained milled matrix breccia with a high proportion of silica-magnetite clasts, DDH249, 257.6 m, 0.3 g/t Au.

	
Photo 10 Magnetite altered clasts within the milled matrix breccia, DDH249, 25.5 m, 0.35g/t Au.	Photo 11. Clast of quartz eye dacite with magnetite alteration, DDH249, 649m 0.21g/t Au.

Figure 7-7: Examples of Alteration Styles

A spectral study (Kerby, 2018) on field exposures and diamond core from 6 drill holes identified weak alteration of illite-smectite within 85% of samples as the principle hydrothermal alteration minerals, dominated by the Fe end-member nontronite with additional kaolinite ad gypsum (Figure 7-8). Kaolin appears to be best developed towards the outer margins of the mineralized zone.

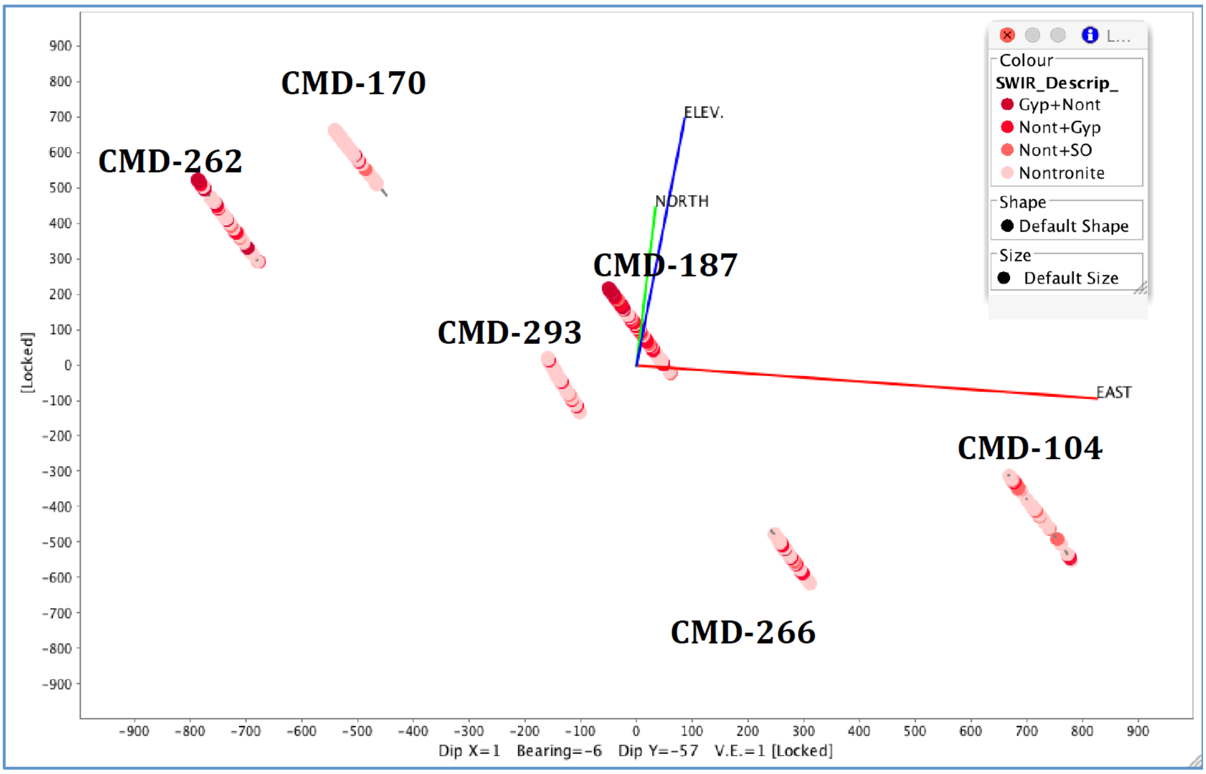


Figure 7-8: Terraspec® results of six drill holes that show the weak Fe end-member of nontronite.

7.2.4 Mineralization

Oxide-gold mineralization extends northwest over 2.5 km of strike and up to 600 m across strike. Drilling has traced oxide-gold mineralization to 600 m below surface and the resource remains open.

Microscope studies indicate that gold mineralization primarily occurs within black and grey banded veinlets (BBV and GBV) in the breccia complex and the dacitic dome, and secondarily within early chlorite-magnetite-quartz veinlets.

Gold mineralization may be encountered in phreatomagmatic breccia, surrounding hydrothermal breccia, in dacite porphyry and surrounding andesitic dikes and plugs.

Sheeted banded quartz veins vary from white to black/grey bands (photos 12, 13 & 14 in Figure 7-9). The white vein portions comprise bands of massive chalcedony and saccharoidal to fine crystalline quartz. Crystalline quartz is deposited from a cooling fluid and is not an indication of temperature of formation. Vein margins are sharp although wavy coliform dark bands are similar to some epithermal veins.

The dark bands display only local magnetic character, but the dark colour results from most abundant secondary inclusions (Lohmeier, 2017), in a manner similar to other sheeted banded veins in the Maricunga Belt (Muntean and Einaudi, 2000). In hand specimen, higher

Au grade mineralization is clearly associated with these veins, which may occur as submicroscopic free Au particles within the dark coloured inclusion-rich bands.

		
Photo 12. Milled matrix breccia rendered competent by silicification and cut by an open banded quartz vein, DDH92, 283.3m, 2.41 g/t Au.	Photo 13. Tuffisite dyke cuts earlier milled matrix breccia and is exploited by banded sheeted quartz veins, DDH249, 372.1m, 1.46 g/t Au.	Photo 14. Oxidized sheeted banded quartz veins cut milled matrix breccia, DDH249, 349.8m, 2.39g/t Au.

Figure 7-9: Examples of Mineralization Styles

7.2.5 Gold Form and Carriers

Metallurgical test work (AMTEL 2012) indicates Au occurs as native and submicroscopic forms. Native Au is fine grained (75-90% <10 µm of which 45-75& < 5 µm) and of a high fineness while the submicroscopic forms display a strong association with Fe oxide (FeOx) typically goethite and hematite, most pronounced in the near surface samples (Figure 7-10). Gold redeposited by supergene processes typically occurs as high fineness native Au in association with FeOx, especially if derived from a low-Ag low sulphidation quartz-sulphide Au + Cu deep epithermal source.

However, Au deposited by the mixing of rising ore fluids with oxygenated groundwater may also be associated with FeOx. In the metallurgical test work (AMTEL 2012), ore crushed to 11-13 mm yielded about 80% recovery in a cyanide leach. Submicroscopic Au accounted for 4-5% of cyanide leachable ore. In samples milled to 80-110 µm, cyanide leach recoveries rose to 85-90%, with 50-60% Au described as free.

Consequently, in the light of the deep oxidation recognized at Fenix, much of the fine-grained Au recognized in the metallurgical test work is interpreted to have been remobilized by supergene processes within the permeable milled matrix breccias.

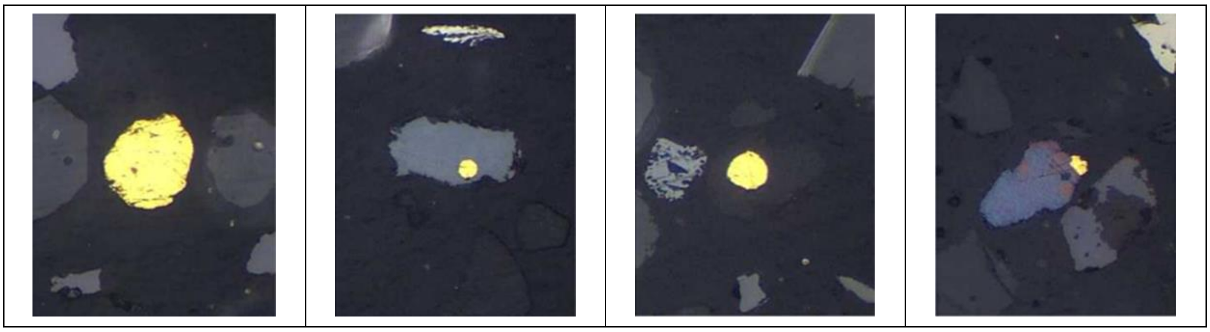


Figure 7-10: From Left to Right - Free gold; Gold grain enclosed in hematite; Gold grain enclosed in rock; and, Gold grain attached Cu sulphide (digenite and bornite)

Inspection of the drill core suggests Au grade rises with each additional intrusion / breccia / hydrothermal event. Rocks that display the full range of intrusion, phreatomagmatic and vein activity are likely to be best mineralized the following is also noted:

- Andesite is barren.
- The quartz eye dacite is expected to be barren or contain only very low Au <0.1 g/t Au (photo 15 in Figure 7-11).
- Medium grade Au (0.1 - 0.5 g/t) is recognized within the milled matrix breccias. Although inspection of the drill core suggests high Au grades occur in the magnetite-rich breccias (photo 16), there is no evidence in the petrological data of Lohmeier (2017) of magnetite hosting Au mineralization.
- High Au grades are associated with the sheeted banded quartz veins, which transect the milled matrix breccias, and locally occur as breccia clasts. Most high-grade Au mineralization is confined to the inclusion-rich dark bands and not the clean quartz (photos 18, 19 & 20).
- There is an event of typical low sulphidation quartz-sulphide Au + Cu mineralization (Corbett, 2019) discernible as heavily oxidized pyrite with quartz, with grades of up to 4 g/t Au, in DDH 104 (photos 16 & 17).
- The current interpretation is that much of the high fineness native and submicroscopic Au recognized at Fenix has been redeposited by supergene processes, facilitated by deep oxidation within the permeable milled matrix breccias. Much of that mineralization may have been derived from the weathering of low sulphidation quartz-sulphide Au + Cu mineralization including the sheeted banded quartz veins.

Photo 15. Crowded breccia dominated by the early quartz eye dacite clasts with low-grade Au, DDH249, 644m, 0.12g/t Au.

Photo 16. Oxidized pyrite veins cut a quartz vein, DDH104, 132.4m, 4.63 g/t Au.

Photo 17. Oxidized quartz-pyrite vein cuts black silica vein, DDH104, 132.5m, 4.63g/t Au.

Photo 18. Milled matrix breccia rendered competent by silicification and cut by an open banded quartz vein, DDH92, 283.3m, 2.41 g/t Au.

Photo 19. Tuffisite dyke cuts earlier milled matrix breccia and is exploited by banded sheeted quartz veins, DDH249, 372.1m, 1.46 g/t Au.

Photo 20. Oxidized sheeted banded quartz veins cut milled matrix breccia, DDH249, 349.8m, 2.39g/t Au.

Figure 7-11: Examples of texture related to Au Grade.

8 DEPOSIT TYPES

The Fenix Gold Project shares characteristics with other deposits in the Maricunga Mineral Belt (MMB).

Gold mineralization is considered to represent an intrusion-related, low-sulphidation, quartz-sulphide, Au and Cu deep epithermal system. These systems often host very fine (refractory) gold in sulphides.

Diatreme flow dome complexes vent to the surface and in all other cases host epithermal Au mineralization, locally cutting or telescoped over older porphyry manifestations.

Rather, the dominance of massive chalcedony in these veins is more typical of formation in an epithermal environment. Petrology work (Lohmeier, 2017) describes bornite replaced by chalcopyrite along with sphalerite (of unknown Fe:Zn ratio) and tetrahedrite as the sulphide minerals, typical of an intrusion-related fluid in a deep epithermal setting. Lohmeier (2017) also recognized a correlation between Au and Ag-Bi-Cu, typical of a low sulphidation deep epithermal geochemical signature.

While any possible association between Au and magnetite is unusual, the final event of quartz-pyrite mineralization is most certainly typical of the intrusion-related low sulphidation quartz-sulphide Au + Cu deep epithermal style.

Overprinting magmatic events include pre-mineral andesite domes, followed by emplacement of polyphasal dacite dome (including some transitional to andesite in composition) and associated diatreme (milled matrix) breccia events. The mineralization of low sulphidation Au is best developed in sheeted quartz veins, breccia clasts and very finely disseminated with a possible supergene component (Corbett, 2019) (Figure 8-1).

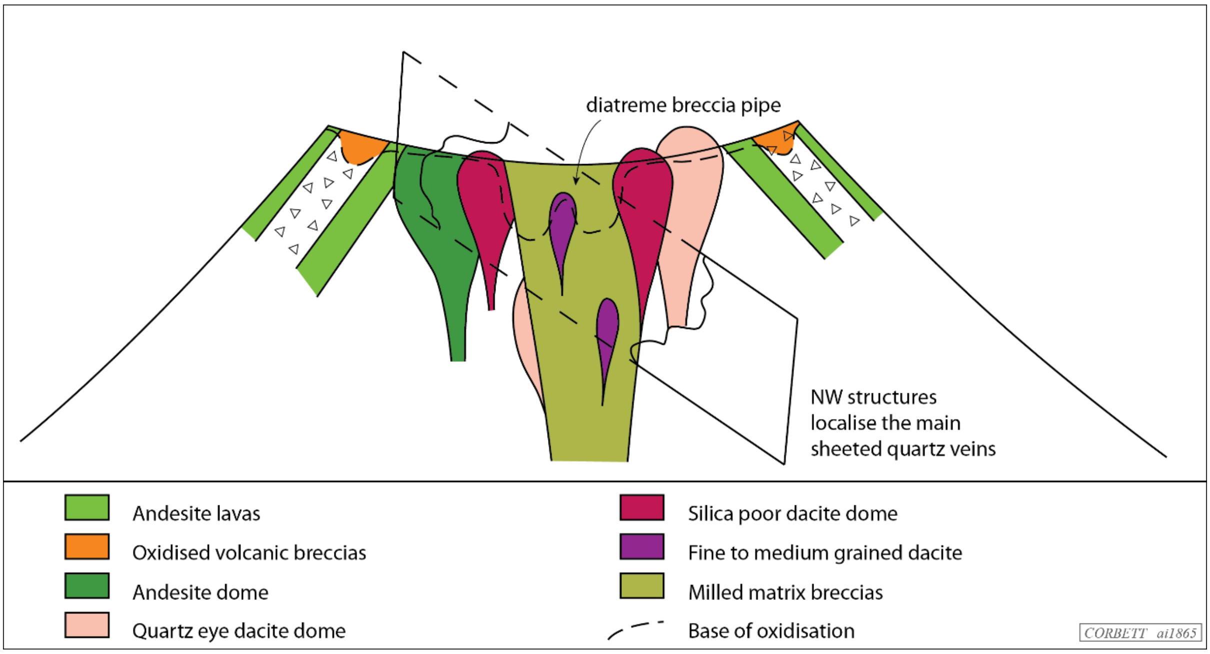


Figure 8-1: Cartoon to illustrate some of the relationships associated with the Fenix diatreme-dome complex developed within the core of the Volcan Ojo de Maricunga.

9 EXPLORATION

Systematic exploration in the area of the Project is first recorded in the 1980´s and has advanced through multiple stages.

Atacama (previously SBX), lead multiple phases of exploration at the Property, including:

- Surface mapping.
- Outcrop sampling.
- Trench sampling and mapping.
- Ground Geophysical studies (magnetics, resistivity and IP).

Atacama identified oxide-gold mineralization exposed at surface, mineralization was recognized to in Black Banded veins (BBV), typical of the Maricunga Belt, and in breccias.

Atacama defined anomalous mineralization extending over 2.5 km northwest strike, 600 m across strike.

Atacama attracted interest from Minera Newcrest Chile Ltda (MNCL) and later Gold Fields (GFC). GFC funded ground based magnetic and induced polarization surveys focused over areas of recognized mineralization only. Geophysical surveys confirmed the potential for mineralization to extend to depth.

Rio2 took control of the Project in 2018 and undertook a program of 12 trenches and 729 channel samples to verify previous work. All channel samples taken from the Project are shown in Figure 9-1.

Rio2 contracted Dr. Greg Corbett, world-renowned epithermal and structural geologist to visit the Fenix Gold Project to review mineralizing controls and to provide inputs to help develop the first 3-D geological for the Project.

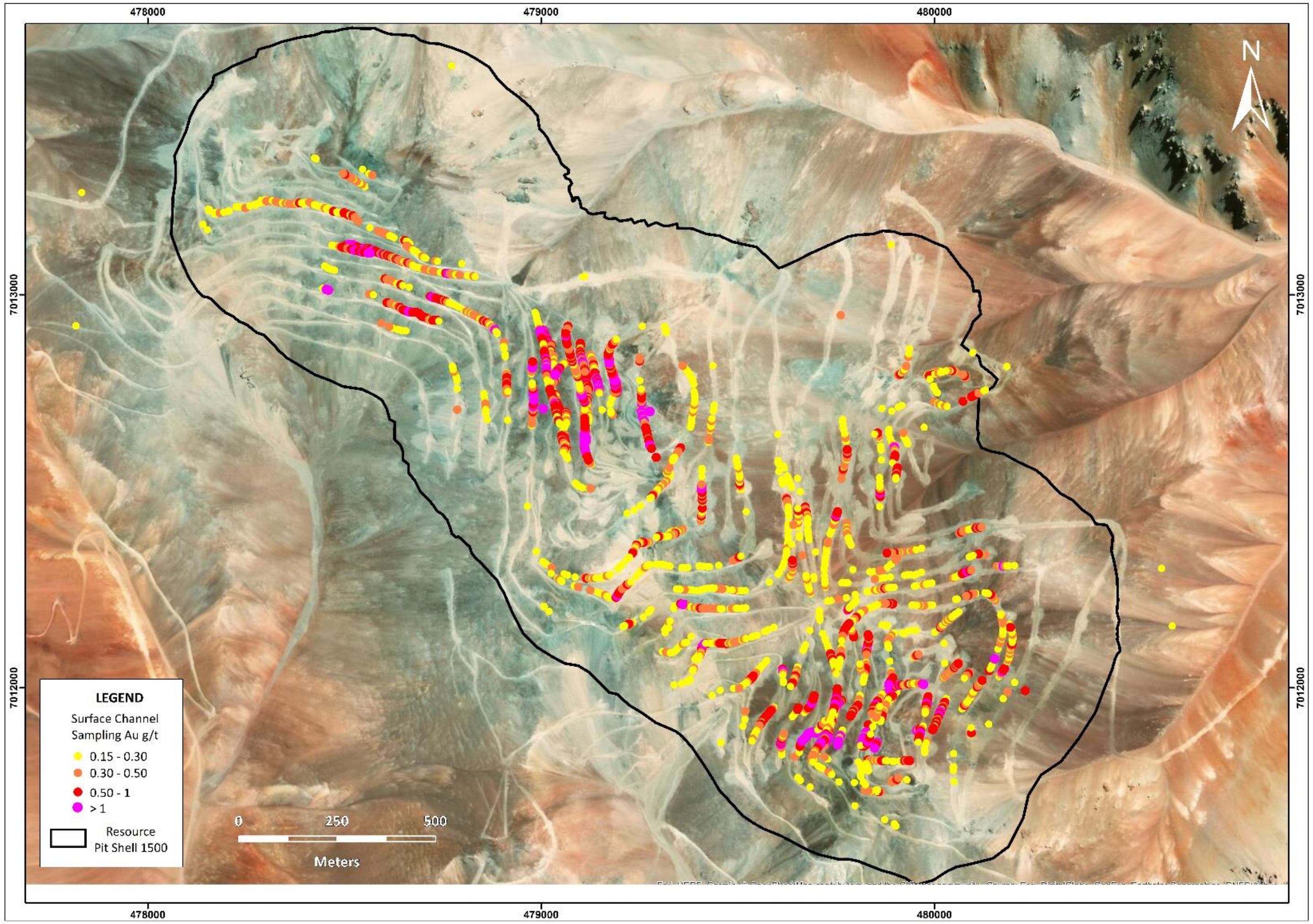


Figure 9-1: All Surface Channel Sampling

10 DRILLING

SBX, drilled eight diamond drill holes (Phase 1) in 2010 ahead of listing as Atacama on the Toronto Stock Exchange.

SBX and Atacama completed five drilling campaigns at the Property, totalling 108,481.76 m over 345 holes of combined RC and DDH. Since taking control of the Project in 2018, Rio2 has drilled and additional 7066 m over 39 RC holes (Table 10-1).

Three hundred and eighty four (384) drill holes and 115,547.76 m have been drilled at the Project; of the 115,547.76 m drilled, 112,611.21 m have been assayed, typically at 2 m intervals.

Exploration drilling consists of 91 DDH totalling 31,047.21 m, and 293 RC drill holes totalling 84,500.55 m. Other drilling includes condemnation, and metallurgical testing.

A total of 112,409 m drilled is within the modelled mineralized zones, accounting for 98% of total meters drilled into the deposit. The percentage of total meters within Fenix South, Fenix Central, and Fenix North are 20%, 56% and 23% respectively (Table 10-2).

Drilling has been aligned along 50 meter spaced northeast oriented sections, orientated approximately perpendicular to northwest trending mineralization. Drill hole locations and mineralized zones are shown in Figure 10-1.

Drill collars were surveyed using differential GPS and by conventional survey means. Downhole survey measurements were routinely taken out at either 3 m or 10 m downhole intervals by Comprobe (Santiago) and Wellfield Services Ltda (Antofagasta) using gyroscopes.

Eight of 320 holes used in the MRE do not have a collar survey and two of these eight drill holes do not have a downhole survey due to extreme ground collapses in the upper 30 m of the hole.

Details of non-surveyed drill holes are shown in Table 10-3.

On a meterage basis, approximately 2.1% of RC and 0.8% of DDH included in the resource block model have not been downhole surveyed.

Table 10-1: Number of holes and meters drilled per phase

Phase	I	II	II	III	III	IV	IV	V	VI
Year	2010		2011		2012		2013	2017	2018
N° Drill Holes	8	15	67	28	102	38	71	16	39
Reverse Circulation (m)	1,422	3134	21,446	6,350	25,272	6,181	12,670	960	7,066
Diamond Drilling (m)	720.25	2,152.80	4,727.95	3,446.05	10,915.26	2,838.95	4,645.95	1,600	0
Total (m)	**7,429.05**		**35,970.00**		**45206.76**		**17,315.95**	**2,560**	**7,066**

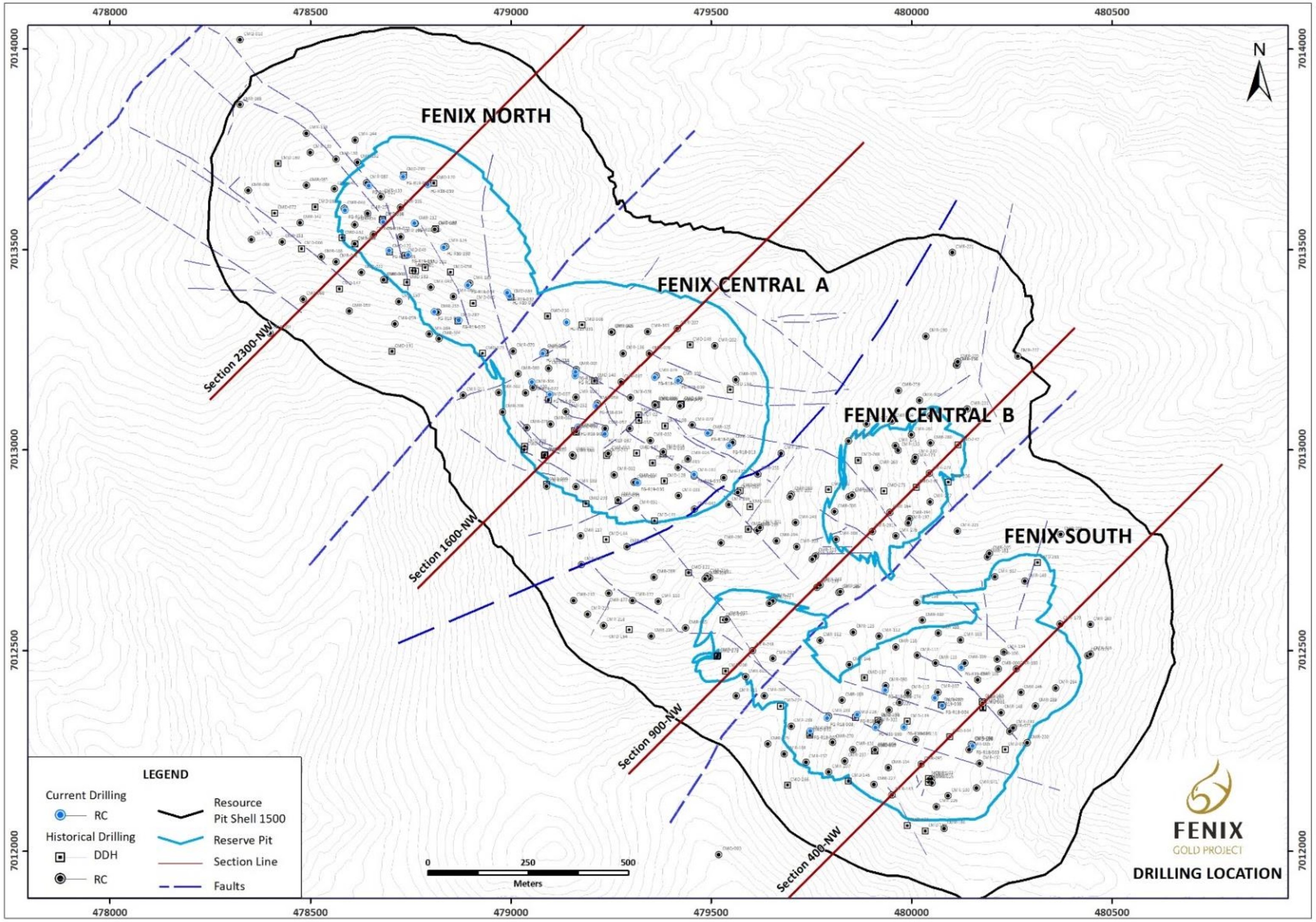


Figure 10-1: Drill Hole Location, Mineralized Zones and Pit Outlines

Table 10-2: Fenix Gold Project Drilling Phases – Meters Drilled & Meters Assayed

Zone	Drilling 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)
Fenix North	I	181.90	181.90	-	-	-	-	181.90	181.90	-
	II	2,013.40	2,013.40	-	6,734	6,734	-	8,747.40	8,747.40	-
	III	3,046.40	3,046.40	-	4,912	4,912	-	7,958.40	7,958.40	-
	IV	1,570	1,570	-	2,050	2,050	-	3,620	3,620	-
	I	450	-	450	-	-	-	450	-	450
	VI	-	-	-	2,060	2,052	8	2,060	2,052	8
	Sub-Total	**7,261.60**	**6,811.60**	**450**	**15,756**	**15,748**	**8**	**23,017.60**	**22,559.60**	**458**
Fenix Central	I	321.10	321.10	-	570	570	-	891.10	891.10	-
	II	2,568.10	2,568.10	-	16,500	16,484	16	19,068.10	19,052.10	16
	III	7,812.10	7,812.10	-	13,976	13,944	32	21,788.10	21,756.10	32
	IV	4,674.20	4,674.20	-	10,642	10,632	10	15,316.20	15,306.20	10
	I	600	-	600	-	-	-	600	-	600
	VI	-	-	-	3,338	3,336	2	3,338	3,336	2
	Sub-Total	**15,975.50**	**15,375.50**	**600**	**45,026**	**44,966**	**60**	**61,001.50**	**60,341.50**	**660**
Fenix South	I	217.40	217.40	-	852	852	-	1,069.40	1,069.40	-
	II	2,299.30	2,299.30	-	1,346	1,346	-	3,645.30	3,645.30	-
	III	3,502.82	3,502.80	-	12,734	12,728	6	16,236.82	16,230.82	6
	IV	940.70	940.70	-	5,580	5,562	18	6,520.70	6,502.70	18
	I	550	-	550	-	-	-	550	-	550
	VI	-	-	-	1,668	1,668	-	1,668	1,668	-
	Sub-Total	**7,510.12**	**6,960.10**	**550**	**22,180**	**22,156**	**24**	**29,690.12**	**29,116.10**	**574**
Outside the Resource	I	-	-	-	-	-	-	-	-	-
	II	-	-	-	-	-	-	-	-	-
	III	-	-	-	-	-	-	-	-	-
	IV	300	298	2	578.55	296	282.55	878.55	594	284.55
	I	-	-	-	960	-	960	960	-	960
	VI	-	-	-	-	-	-	-	-	-
	Sub-Total	**300**	**298**	**2**	**1,538.55**	**296**	**1,242.55**	**1,838.55**	**594**	**1,244.55**
All Zones	I	720.30	720.30	-	1,422	1,422	-	2,142.27	2,142.25	-
	II	6,880.80	6,880.80	-	24,580	24,564	16	31,460.75	31,445.75	16
	III	14,361.31	14,361.31	-	31,622	31,584	38	45,983.31	45,945.31	38
	IV	7,484.90	7,482.90	2	18,850.55	18,540	310.55	26,335.45	26,022.90	313.55
	I	1,600	-	1,600	960	-	960	2,560	-	2,560
	VI	-	-	-	7,066	7,056	10	7,066	7,056	10
	Sub-Total	**31,047.21**	**29,445.21**	**1,602**	**84,500.55**	**83,166**	**1,334.55**	**115,547.76**	**112,611.21**	**2,936.55**

Table 10-3: Non-surveyed Drill Holes Included in Resource Estimate

Zone	Non-surveyed drill holes	Non Surveyed (m)	Section	Drill hole Type
Fenix South	CMR-148	214	350 NW	RC
Fenix Central	CMR-018	444	1400 NW	RC
Total RC		**658**		**RC**
Fenix Central	CDM-010	165.4	1400 NW	DDH
Fenix Central	CMD-152	134.8	1400 NW	DDH
Fenix Central	CMD-091	26	1600 NW	DDH
Fenix Central	CMD-021	143.2	1600 NW	DDH
Fenix North	CMD-122	173.5	2150 NW	DDH
Fenix North	CMD-036	23	2300 NW	DDH
Total DDH		**665.7**		**DDH**

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

Atacama personnel were responsible for handling the diamond drill core and reverse circulation cuttings from drill site pending delivery to the sample preparation facility at Paipote on in the outskirts of Copiapo.

Rio2 personnel were responsible for handling their reverse circulation cuttings from drill site until delivery to the preparation facility at Paipote.

11.1.1 Reverse Circulation Drill Holes-RC

Atacama and Rio2 applied the same methodology to RC chip sampling. Atacama used the sample preparation facility managed by Geoanalitica in Copiapo. Rio2 used the sample preparation facility managed by ALS in Copiapo. The sample preparation methodology for both laboratories is:

- RC 2 m cuttings, weighing approximately 80 kg, were split at the drill site in a standard riffle splitter down to 25% of the sample weight (approximately 20 kg). Two 20 kg samples were bagged and put into pre-labelled plastic bags under the supervision and control of company personnel. In addition, a geological technician collected representative samples (dust and cuttings), for each 2 m interval, in properly marked and identified plastic chip trays, which were used for logging purposes.
- Field duplicate samples are inserted at a rate of approximately one per 20 samples. Once the holes are sampled, the samples are transported to the core shed located in Paipote.
- At the core yard, 7 kg bagged blanks and reference materials are inserted into the sample stream after each field duplicate sample and then these samples are sent to the relevant sample preparation facility. The sample preparation stream, as well as the QA/QC protocol is shown in Figure 11-1.

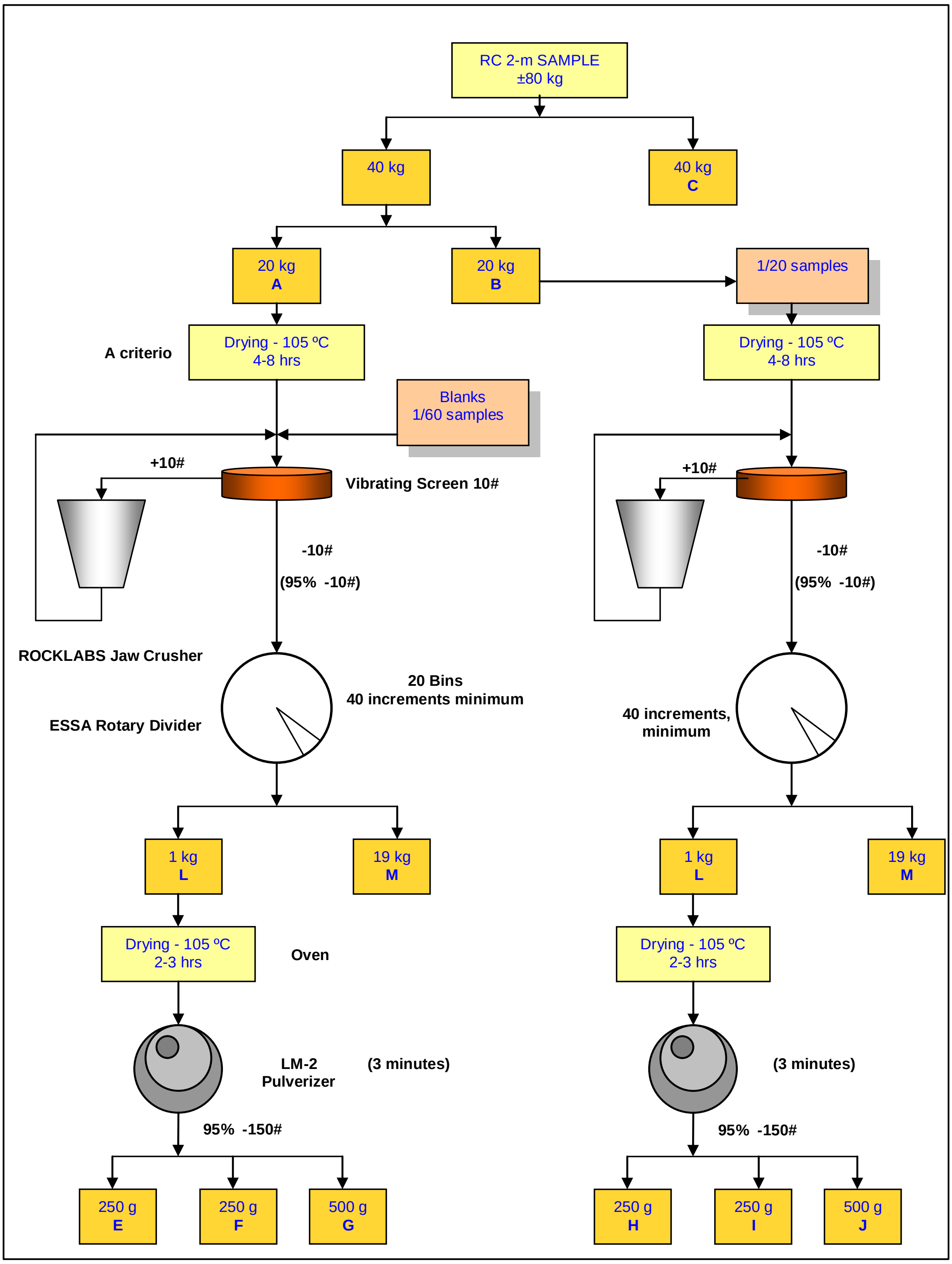


Figure 11-1: Sample Preparation Protocol-RC and QA/QC

11.1.2 Diamond Drill Holes – DDH

Rio2 has not produced any diamond drill core at the Project. Atacama applied the following methodology when sampling diamond drill core:

- Diamond drill core is boxed in aluminium 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 unique number.
- Core is quick logged by a senior geologist at the drill 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 core shed located in Paipote.
- 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 labelled, blanks are inserted, and then these samples are delivered to Geoanalitica for sample preparation together with the list of samples selected as duplicates. The sample preparation stream for diamond core is shown in Figure 11-2 and Figure 11-3.

Atacama used the Geoanalitica laboratory in Coquimbo because of the extensive and positive past experience that SBX has had with Geoanalitica in other projects. Geoanalitica is an ISO9000:2001 certified laboratory. A number of major mining companies including Barrick, Codelco and Antofagasta Minerals utilize Geoanalitica's services.

The following summarizes the sample preparation procedures used at the Geoanalitica Paipote sample preparation facility:

- The samples are coarse crushed to 95% passing 2mm.
- The material is then rotary split with 50% (~8 kg) of the sample being returned to Atacama for storage. The other 50% is rotary split to two – 1 kg samples and 1 6 kg - samples. The 6 kg sample is retained as a coarse duplicate and stored.
- One of the 1 kg samples is then dried and ground to 95% passing 0.1mm and an "original" 250 grams pulp is taken.
- The second 1 kg duplicate is likewise dried and ground (95% passing 0.1mm) and 3 splits are taken – 2 – 250 grams splits (duplicate coarse and duplicate pulp) to be assayed.
- The remaining 500 g split is stored.

Atacama collected the prepared pulps and inserted the duplicates, standards and blanks as part of the entire batch, utilizing a different sequential numbering system. The re-numbered pulps were then re-delivered to the sample preparation facility in Paipote, which then shipped the samples to the Geoanalitica laboratory in Coquimbo. At each stage of the process, Atacama utilized shipping slips, which were signed as appropriate by Geoanalitica and by Atacama.

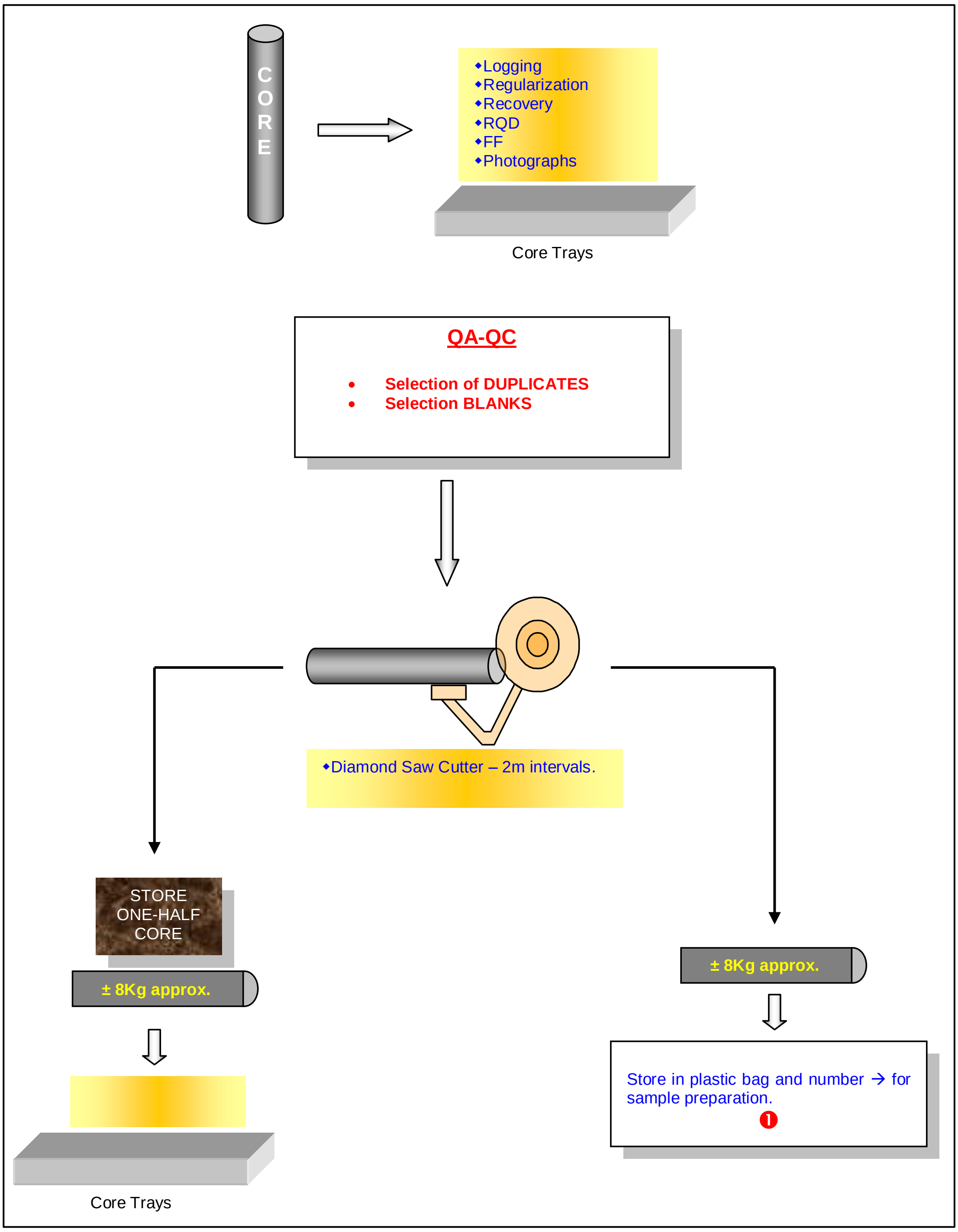


Figure 11-2: Sample Preparation Protocol – DDH and QA/QC

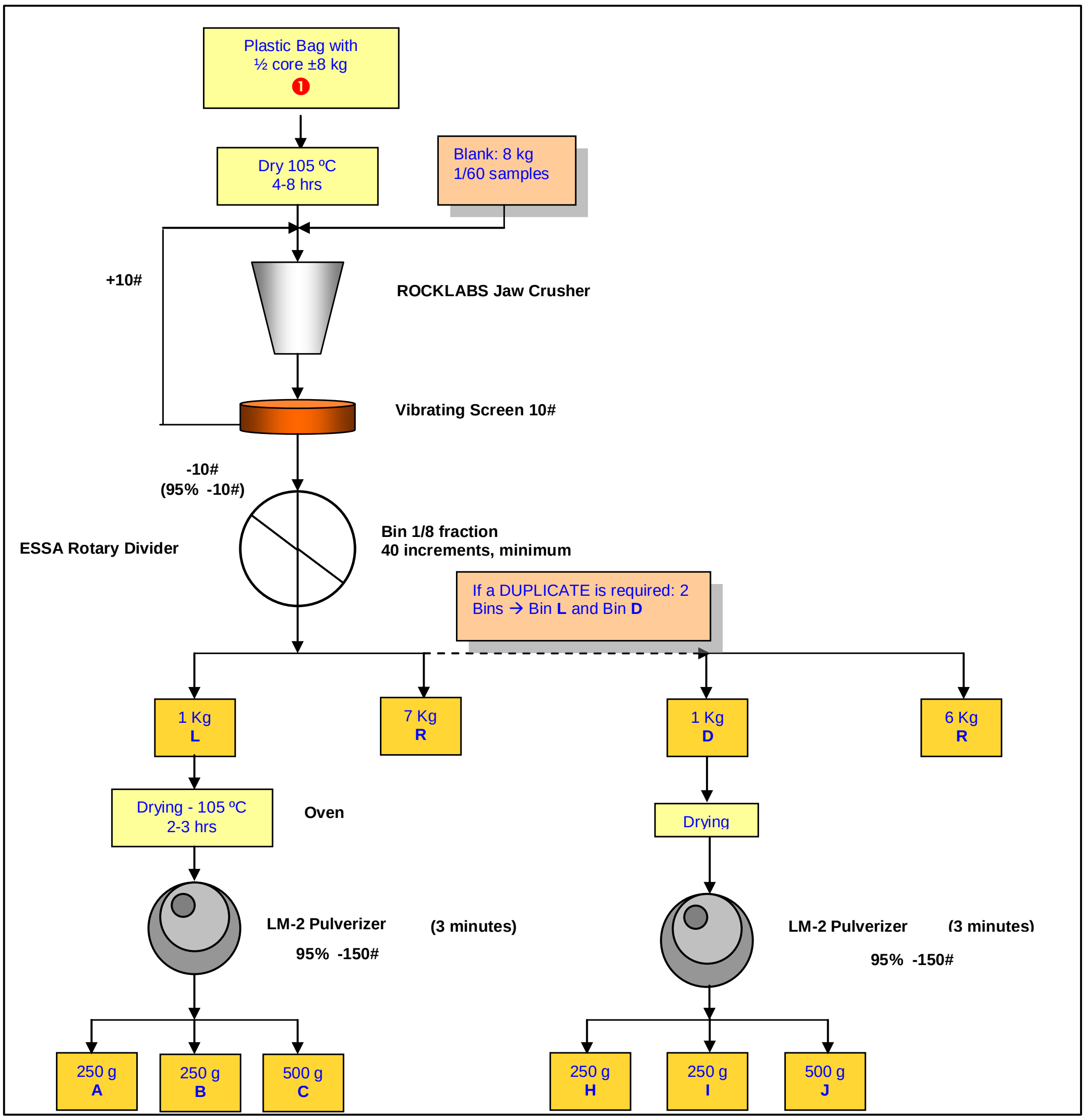


Figure 11-3: Sample Preparation Protocol

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

Finally, each hole contained the following quality control material:

RC Holes:

- Field duplicate (every 20 m) – Envelope H in Figure 11-1.
- Pulp duplicate (every 20 m) – Envelope I in Figure 11-1.
- Standard (every 20 m).
- Blank (every 60 m).

Diamond Drill Holes:

- Coarse (-10#) duplicates (every 20 m) - Envelope H in Figure 11-3.
- Pulp duplicates (every 20 m) – Envelope I in Figure 11-3.
- Standards (every 20 m).
- Blanks (every 60 m).

11.2 Analyses

Analyses were completed in three different laboratories. The laboratories used were:

- Geoanalitica-Coquimbo in 2010 for the eight hole IPO drilling.
- Actlabs-Coquimbo for all drilling after completion of the IPO drilling from December 2010 until late 2017.
- ALS-Lima for all 2018/19 RC drilling.

Analytical Methods used by Geoanalitica and Actlabs are the same, consisting of:

- 50 grams of material is subjected to a standard 50 gram fire assay; typically with an AAS finish is used, however if the resulting values are greater than 3 g/t Au then the reported result will be obtained using a gravimetric finish; the lower detection limit for Au is 5 ppb.
- Copper and molybdenum are analysed for utilizing a 4-acid digestion and an Atomic Absorption finish with a lower detection limit of 3 ppm.

Analytical Methods used by ALS for the 2018/18 drill program are slightly different:

- 50 grams of material is subjected to a standard 50 gram fire assay; typically with an AAS finish is used, however if the resulting values are greater than 5 g/t Au then the reported result will be obtained using a gravimetric finish; the lower detection limit for Au is 5 ppb.
- A 35 multi element suite is analysed for utilizing ICP-AES methods with variable lower detection limits.

11.3 Security

Atacama and Rio2 are conscientious about their sample preparation, security and storage procedures, and therefore maintain a tight control on all sample collection, transportation, processing and storage. At no time is an officer, director or associate of the issuer involved in any aspect related to the sample collection, sample preparation or the shipping of samples to the laboratory.

Prior to shipping the RC samples to Copiapo, an Atacama/Rio2 geologist at the drill site prepared a shipping slip, which detailed the samples that were being transported to Copiapo.

As the samples were being unloaded on arrival at the storage/processing facility in Copiapo, the samples were compared against the original shipping slip, which was then signed, approved, and filed.

The drill core, after being pre-logged and marked for splitting at the campsite, was shipped to the Atacama storage facility in Copiapo where it was prepared (cut and bagged) for transmittal to the Geoanalitica preparation facility at Paipote; the shipping procedures adopted here were as for the RC samples.

At Copiapo, the core was temporarily stored on racks prior to being split, and the cuttings and drill core coarse laboratory rejects samples are stored under cover in appropriately identified (by drill hole number) piles for possible future use, e.g. check sampling or metallurgical testing. Pulps are likewise stored by drill number in easily retrievable boxes at the storage facility.

Core trays, cutting boxes, pulps and coarse rejects are orderly and safely stored in Atacama's logging and storing facility in Paipote. Coarse rejects are 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 and Rio2 during the 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: Fenix Gold Project Database Quality Assessment and Quality Control

Phase Year	I 2010	II 2011	III 2012	IV 2013	Total 2010-2013
N° Drill Holes	8	82	130	100	320
Meters Assayed	2,142.4	31,421.7	45,903.5	25,398.6	104,866.2
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 (%)	**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	0	0.01	0.01	0	0
Maximum	2.79	2.99	4.5	3.39	3.39	3.33
Mean	0.24	0.24	0.36	0.35	0.27	0.27
Standard Deviation	0.3	0.3	0.46	0.44	0.35	0.35
T Test	-0.64		0.6		-0.76	
Mean Relative Error	13.93		7.52		10.71	
Bias (%)	-0.34		0.6		-0.23	
Correlation	0.99		0.99		1	
Intercept	0		0.02		0	
Slope	0.99		0.95		1	
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 modelling 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	0.1	0.1	0.04	0.06
Maximum	2.79	2.99	4.5	3.39	3.39	3.33
Mean	0.36	0.36	0.48	0.48	0.4	0.4
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	
Slope	0.99		0.94		1	
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.

Andres Baluzan (QP) considers that the 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
Pulp Duplicates	90% Data have Rel Diff < 10%	86	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.

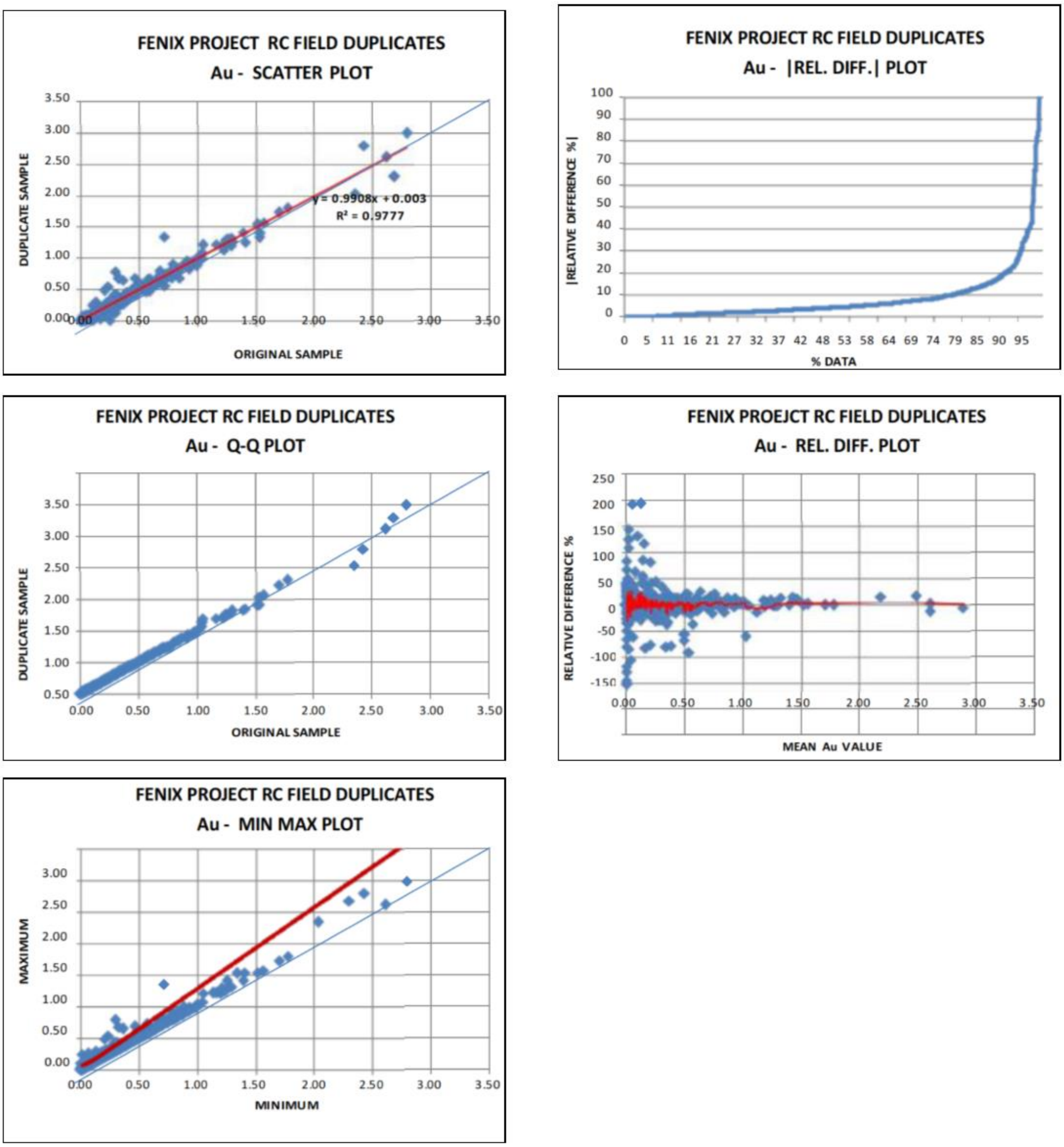


Figure 12-1: Results for all RC Field Duplicates – Au

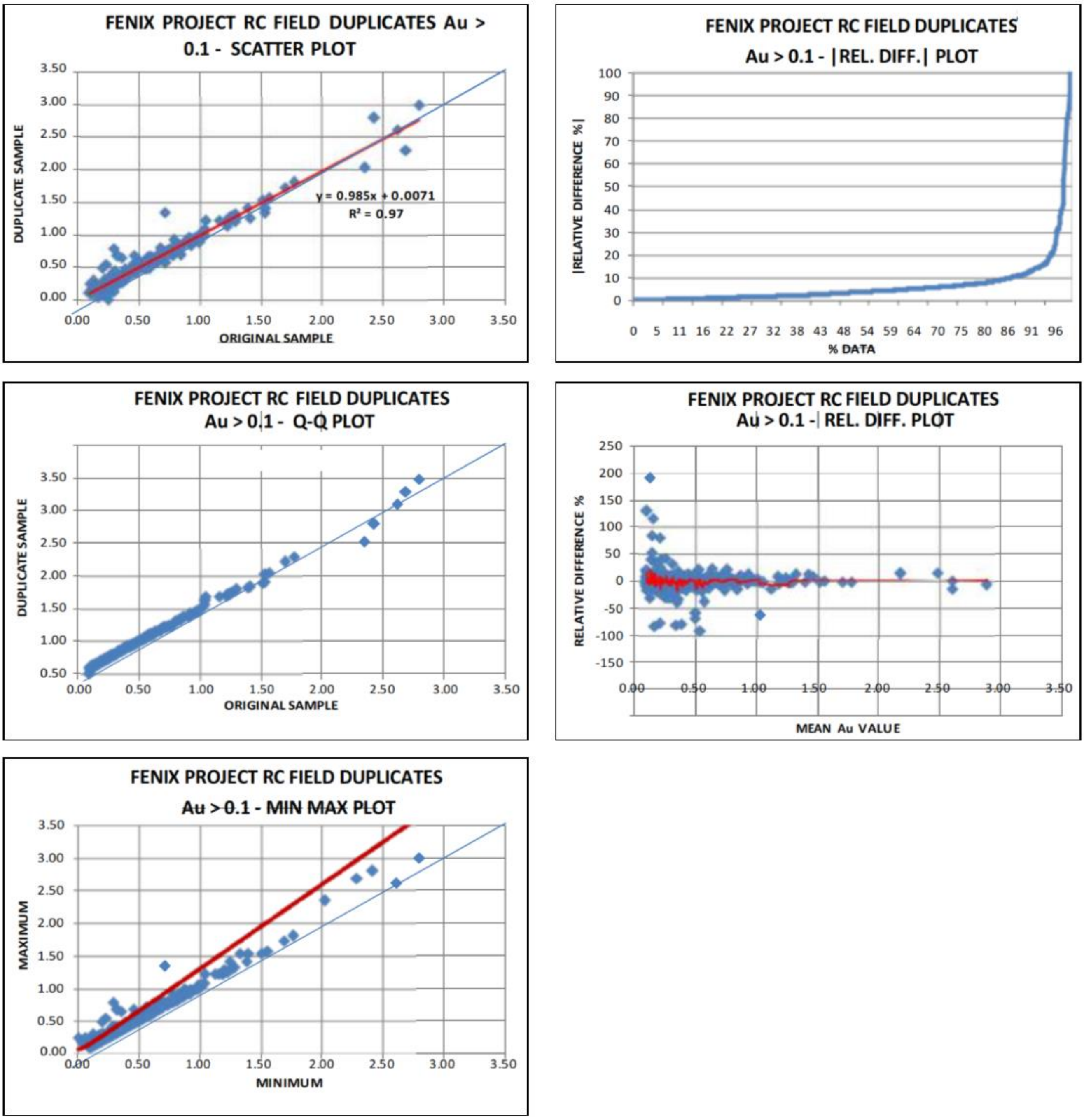


Figure 12-2: Results for the RC Field Duplicates - Au ≥ 0.1 ppm

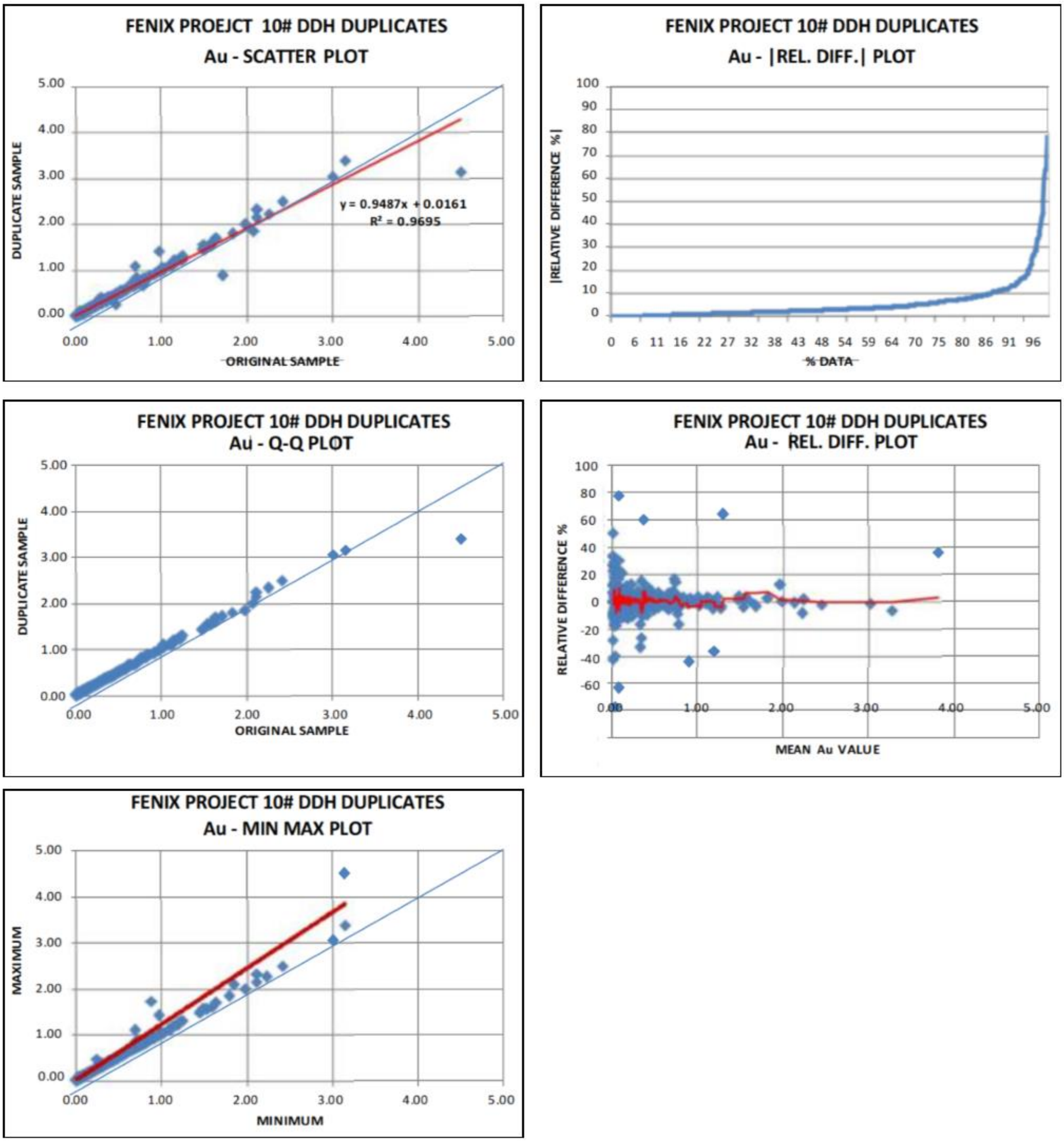


Figure 12-3: Results for DDH Coarse Duplicates – Au

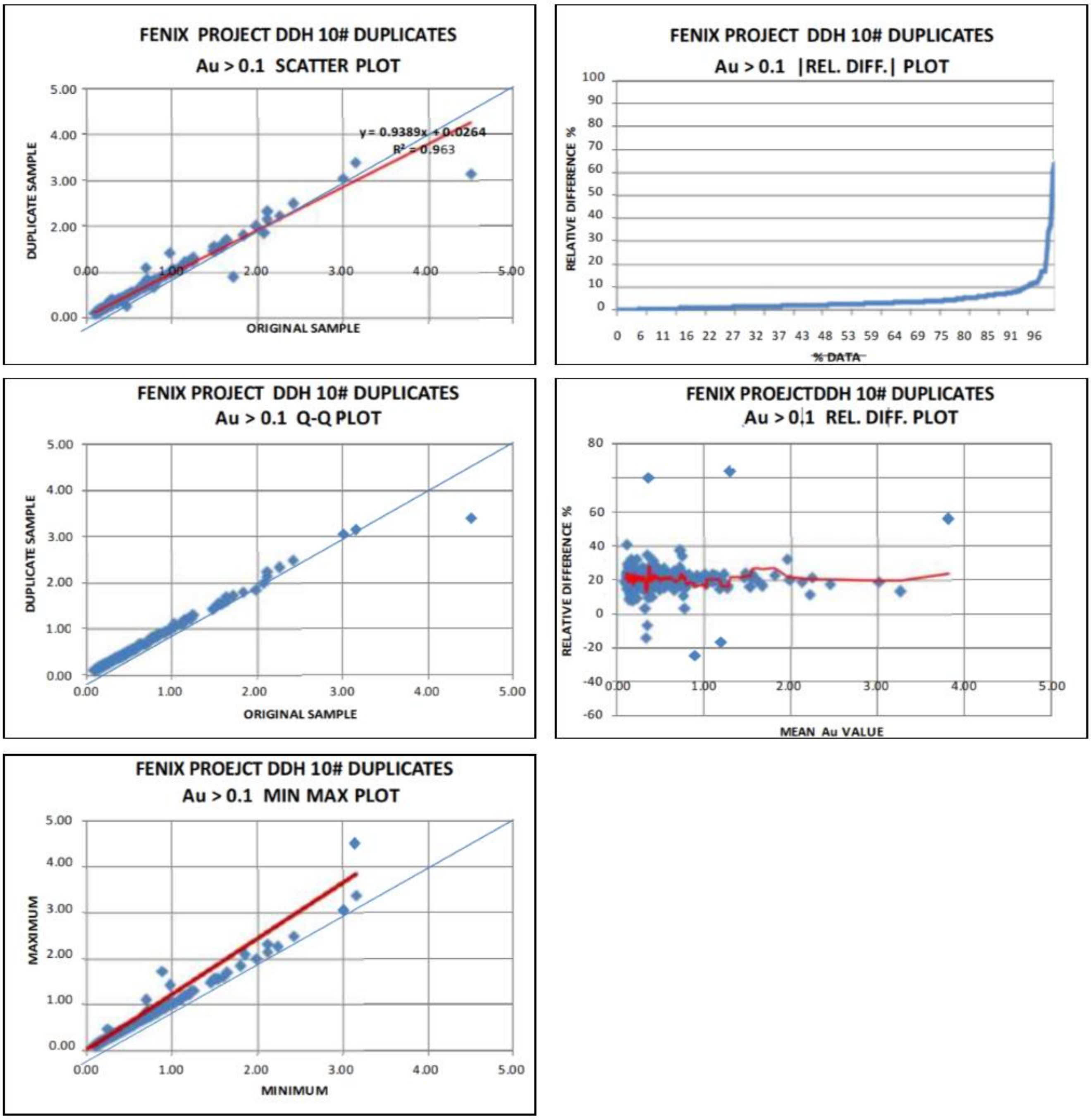


Figure 12-4: Results for the DDH Coarse Duplicates - Au ≥ 0.1 ppm

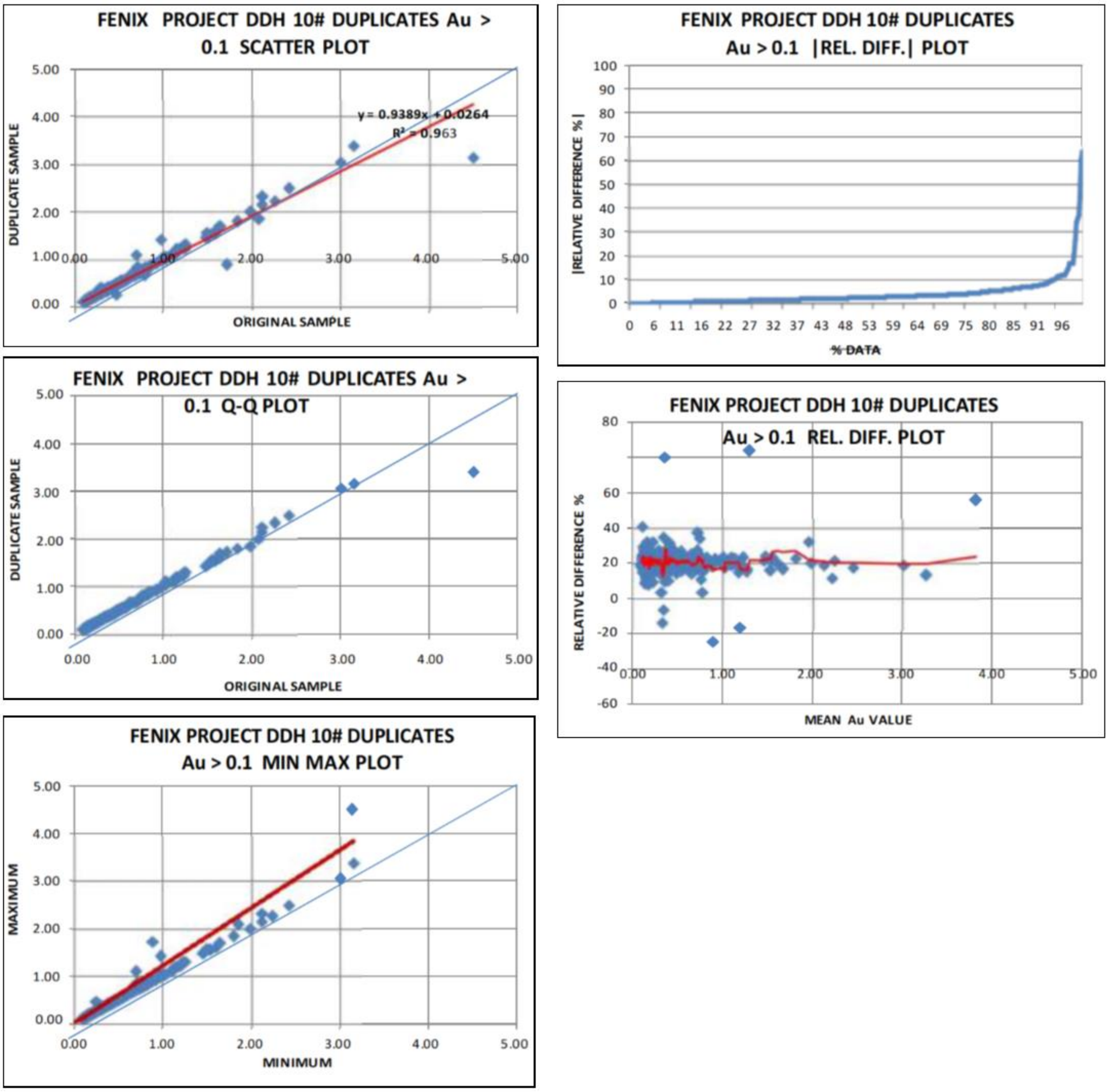


Figure 12-5: Results for all Pulp Duplicates – Au

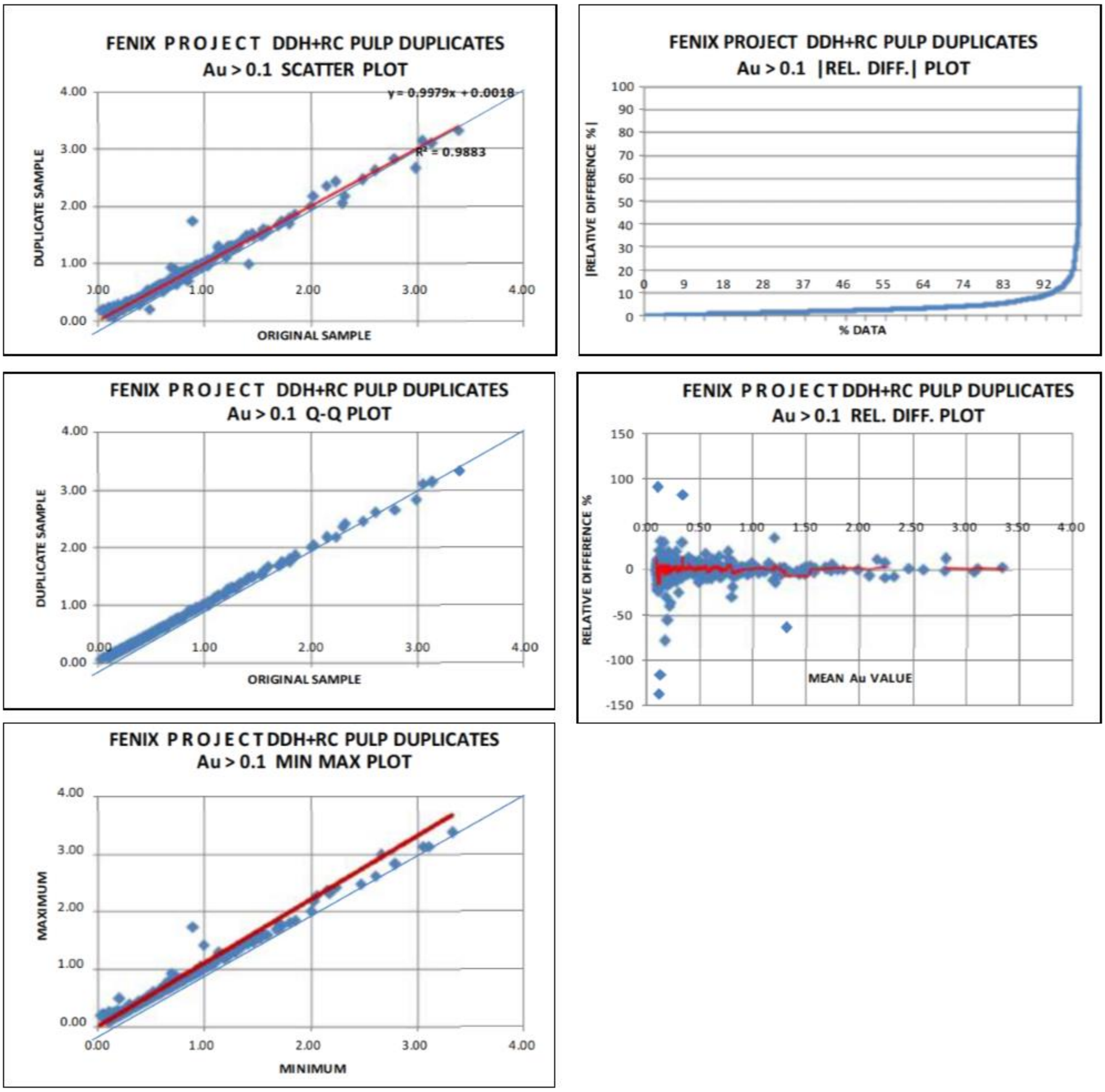


Figure 12-6: Results for the Pulp Duplicates - Au ≥ 0.1 ppm

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 from Geostats Pty Ltd 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). One thousand nine hundred and fifty seven (1,957) standards were assayed for Au. Details are shown in Table 12-5.

Table 12-5: Summary of Geostats Pty Ltd Standards Used for the Fenix Gold 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 (ppm)	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	0.49	0.021	-1.01	11	3.1
G 907 - 2	0.89	341	0.898	0.021	0.899	2	0.6
G 907 - 7	1.541	335	1.501	0.02	-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 behaviour of consistent negative bias in the two 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.

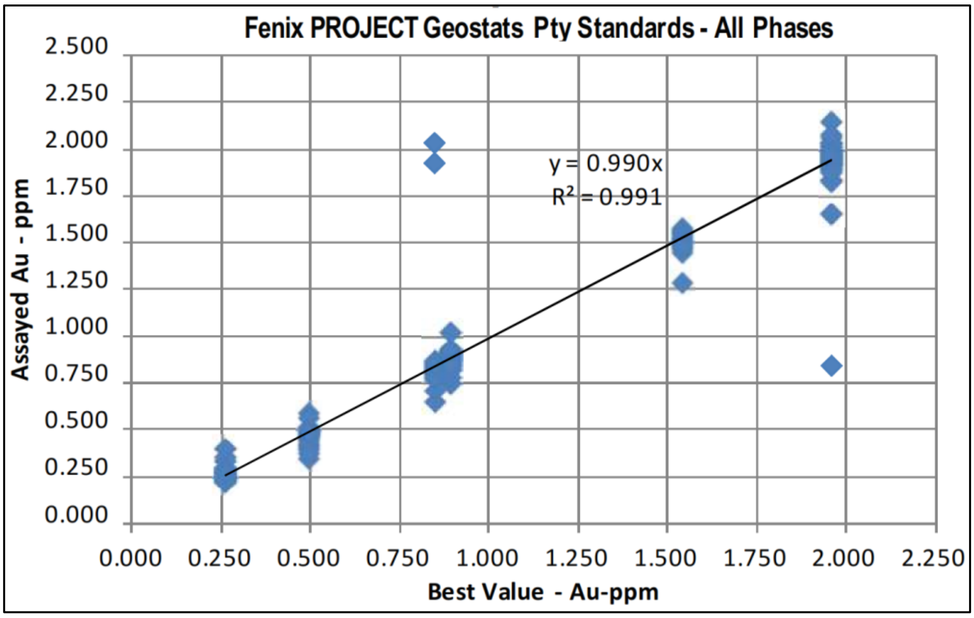


Figure 12-7: Results for all Standards

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.

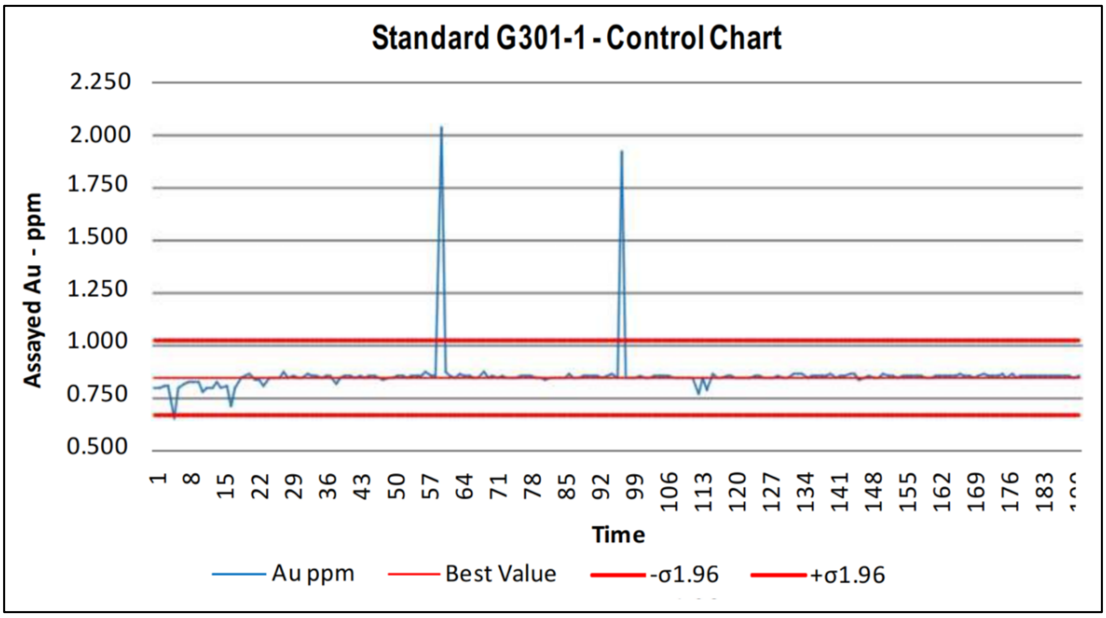


Figure 12-8: Control Chart for Standard G301-1

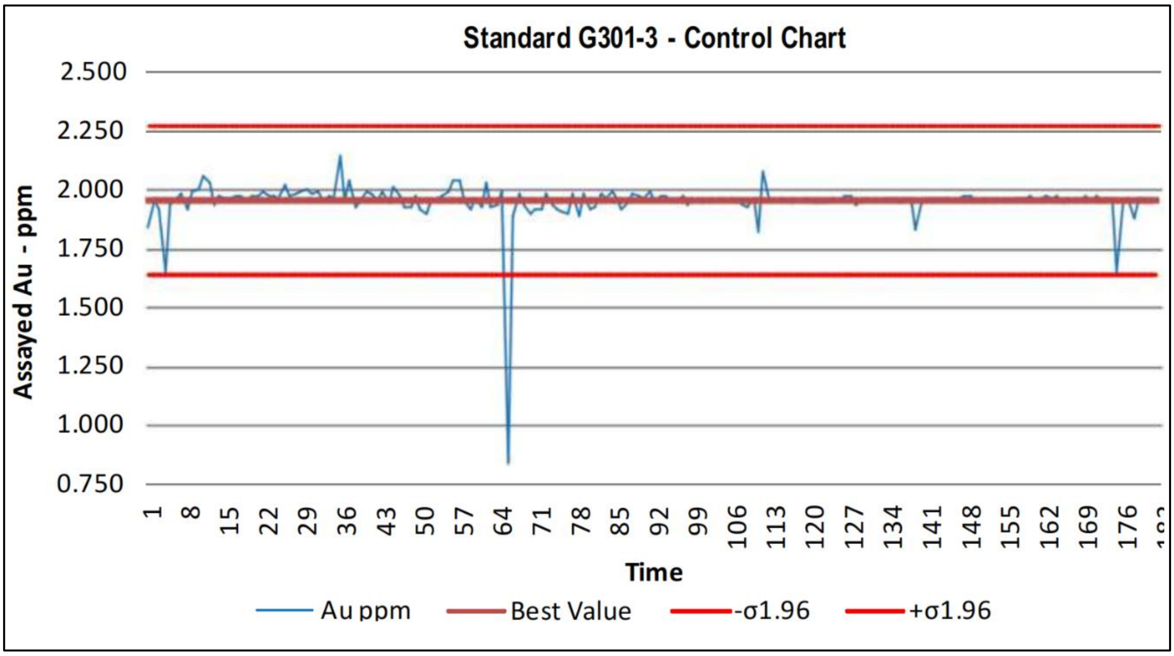


Figure 12-9: Control Chart for Standard G301-3

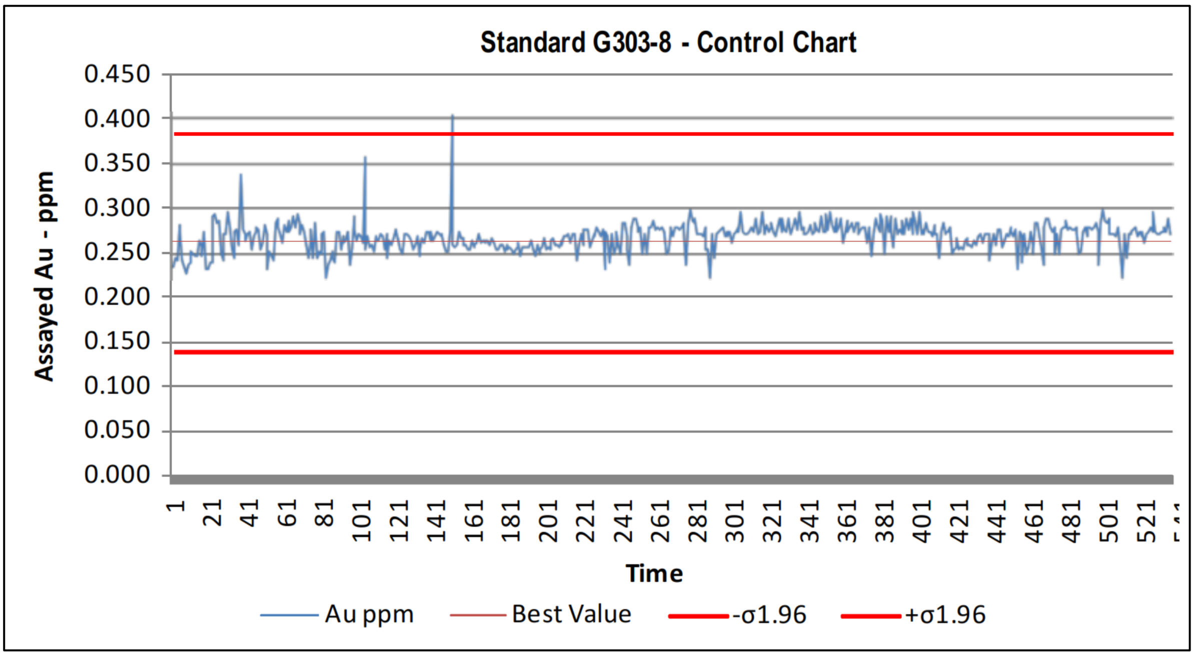


Figure 12-10: Control Chart for Standard G303-8

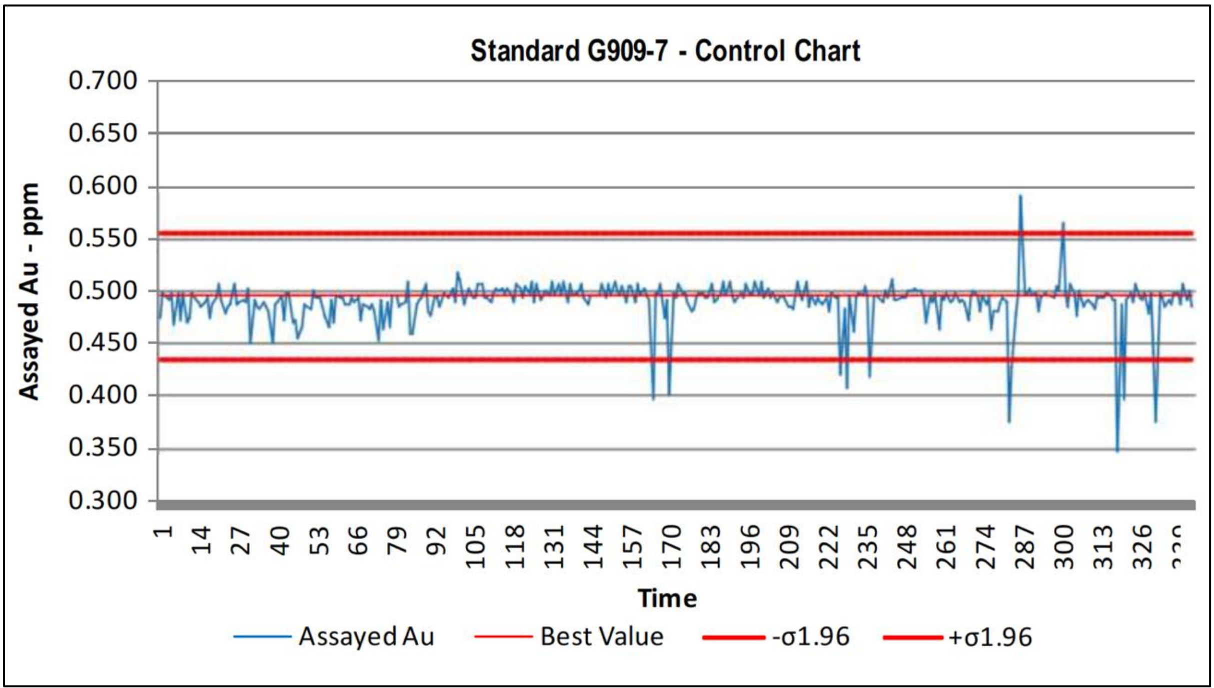


Figure 12-11: Control Chart for Standard G909-7

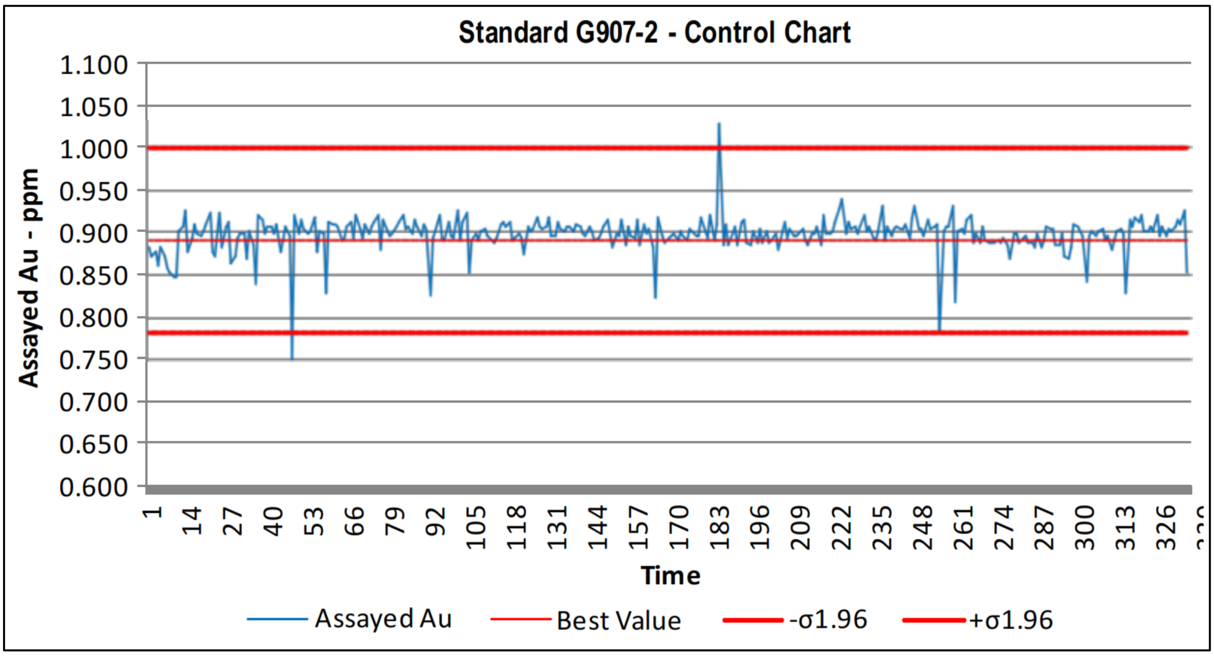


Figure 12-12: Control Chart for Standard G907-2

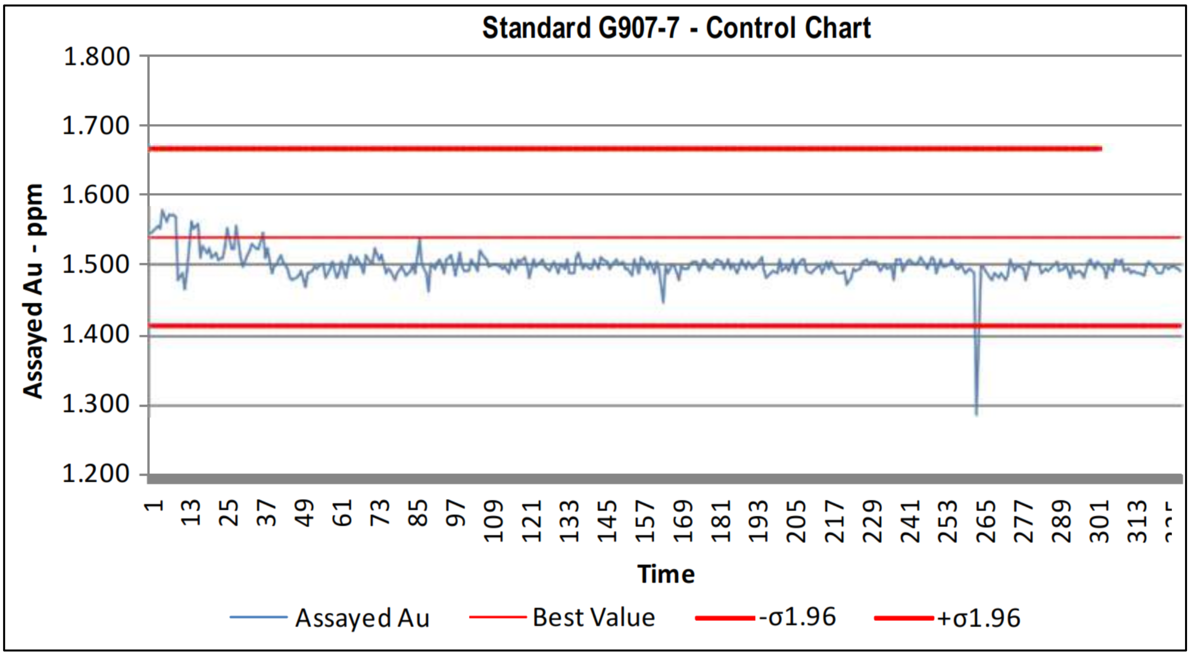


Figure 12-13: Control Chart for Standard G907-7

Control charts and Table 12-6 show that some samples lay beyond the two standard deviation upper and lower limits. The “out of bounds” percentage should be at most 5%. This condition was achieved 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 modelling 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 12-14 shows a sequential Au assay plot for blank samples inserted during the 2010 – 2013 drilling campaigns.

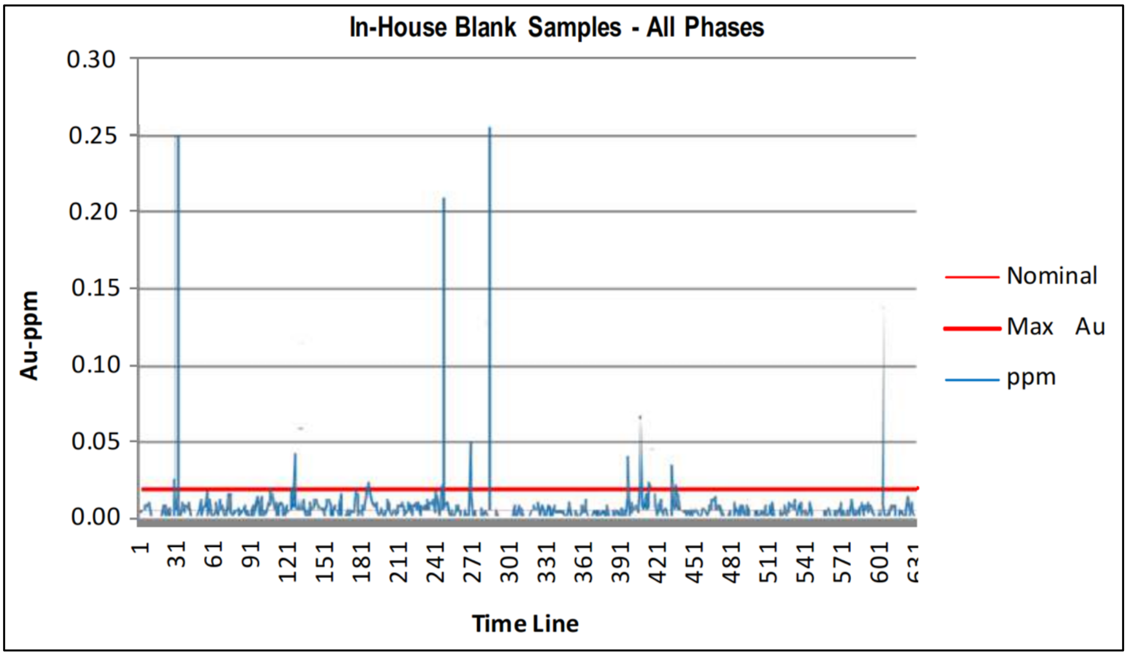


Figure 12-14: Time Sequenced Au Values – In House Blanks

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

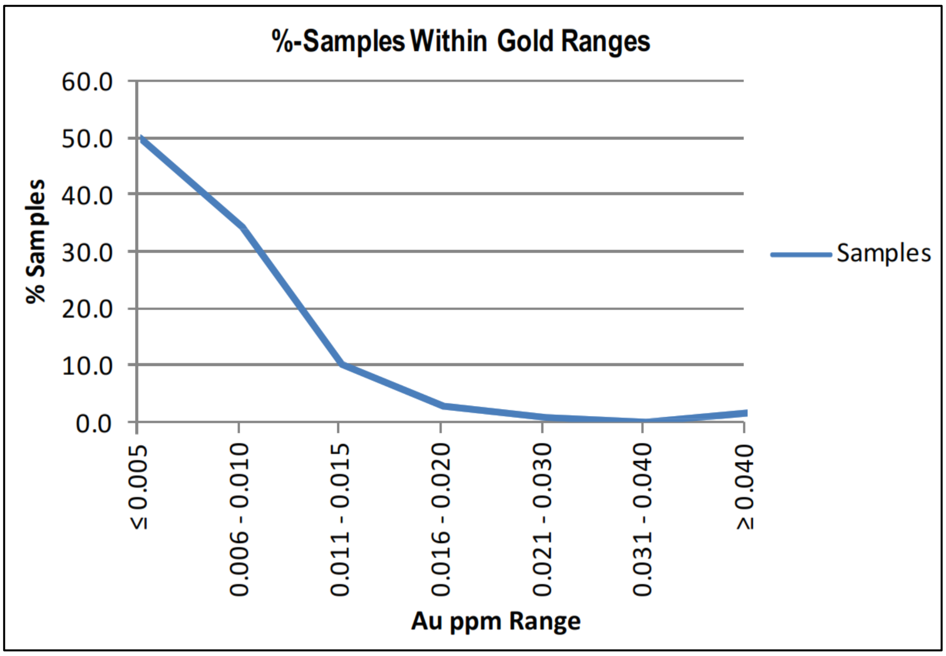


Figure 12-15: Frequency Plot - % Samples within Gold Ranges

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 are probably related to mislabelled samples (ore standards 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 Ltd, 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.

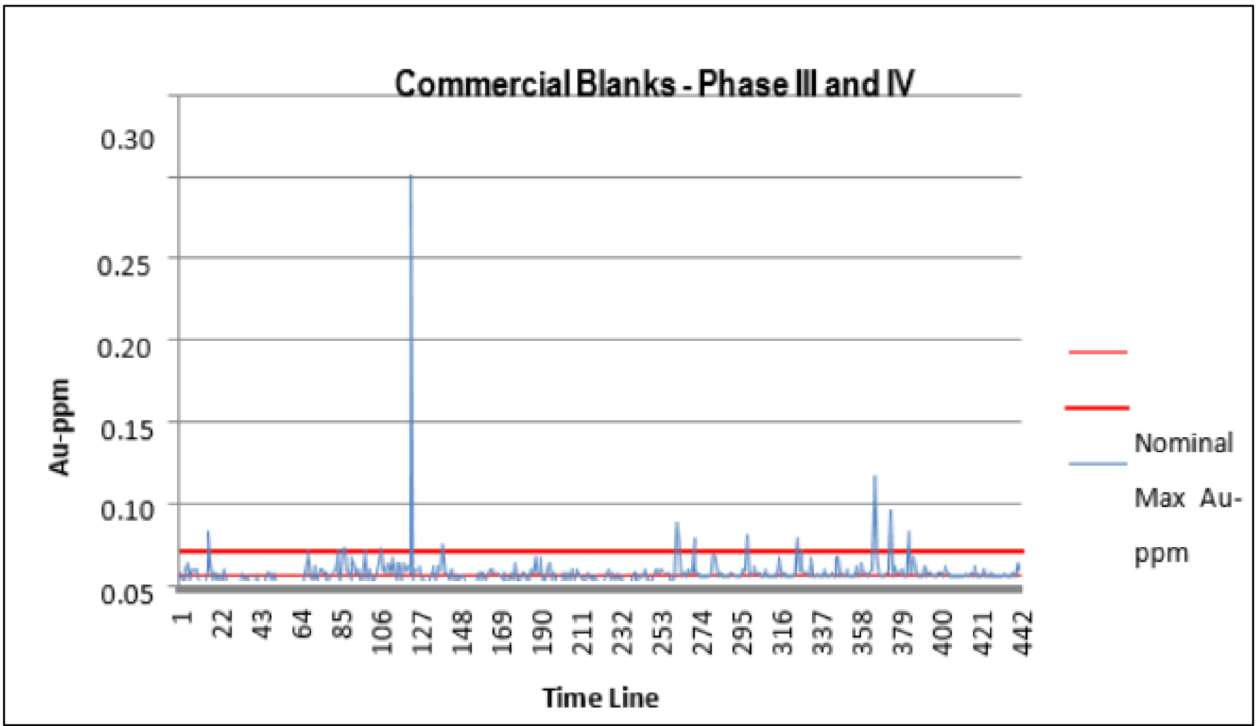


Figure 12-16: Geostats Pty Ltd Blank Certified Material - Gold Values

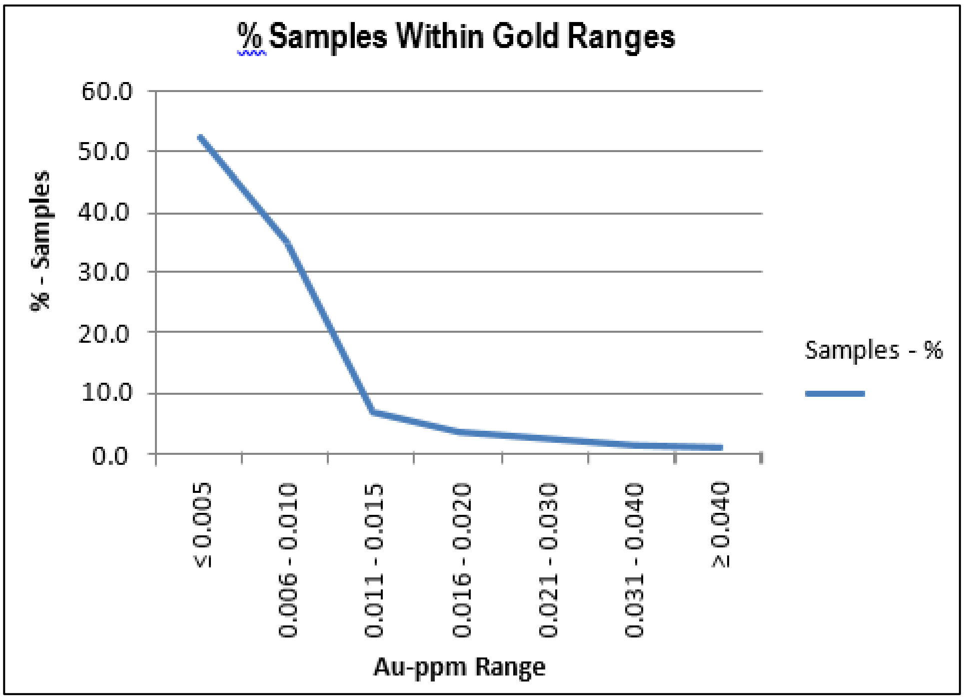


Figure 12-17: Geostats Pty Ltd Blank Certified Material - Frequency Plot

Table 12-8: Geostats Pty Ltd 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 for Au 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

A twin drill hole study was completed in the 2014 PFS. Eleven twin (DDH) holes were completed and these holes were compared to the corresponding nearest RC drill results. Twin samples were allowed to be within 10m of each other for evaluation, so these are not strict twin holes and therefore local results should be viewed accordingly. A summary of the study is presented below. The findings were that there was no bias introduced in the RC drilling and that it was valid to use this data in the resource estimate.

Eleven sets of twin holes were completed between 2011 and 2013. Two were completed in the Fenix North Domain, six were completed the Fenix Central Domain and three were completed in the Fenix South Domain. Identification and length of each hole, as well a section locations are listed in Table 12-9.

Table 12-9: List of Twinned Holes -Fenix Gold Project

Twin hole Set	DDH Hole-ID	DDH Length	RC Hole-ID	RC Length	Section	Zone
1	CM D004	181.85	CM R209	450	2300	Fenix North
2	CM D198	80.35	CM R089	350	2200	Fenix North
3	CM D010	165.35	CM R018	444	1400	Fenix Central
4	CM D092	589.60	CM R002	342	1600	Fenix Central
5	CM D093	531.00	CM R041	348	1400	Fenix Central
6	CM D096	351.85	CM R030	374	1500	Fenix Central
7	CM D099	700.00	CM R067	450	1550	Fenix Central
8	CM D178	107.85	CM R045	312	1400	Fenix Central
9	CM D192	320.00	CM R097	200	400	Fenix South
10	CM D193	180.00	CM R129	400	550	Fenix South
11	CM D196	250.02	CM R098	250	350	Fenix South

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:

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

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.

Variograms were generated for DDH and RC twin holes. Similar nugget effect and variogram shape suggested that a twin hole comparison was likely to be valid.

1.1.1 *Statistics and Graphs for all Twin Holes*

The analyses carried out showed variable local behaviour. Biases, for each drill hole, varied from +8.86% (DDH higher than RC) to -55.03% (RC higher than DDH).

The average gold grades for RC (0.545 g/t) and DDH (0.542 g/t) are very close.

Figure 12-18 shows 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 had a sizeable dispersion as shown by the correlation coefficient (0.602) and the pair-wise mean relative error of 50.54%

The Q-Q plot 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.

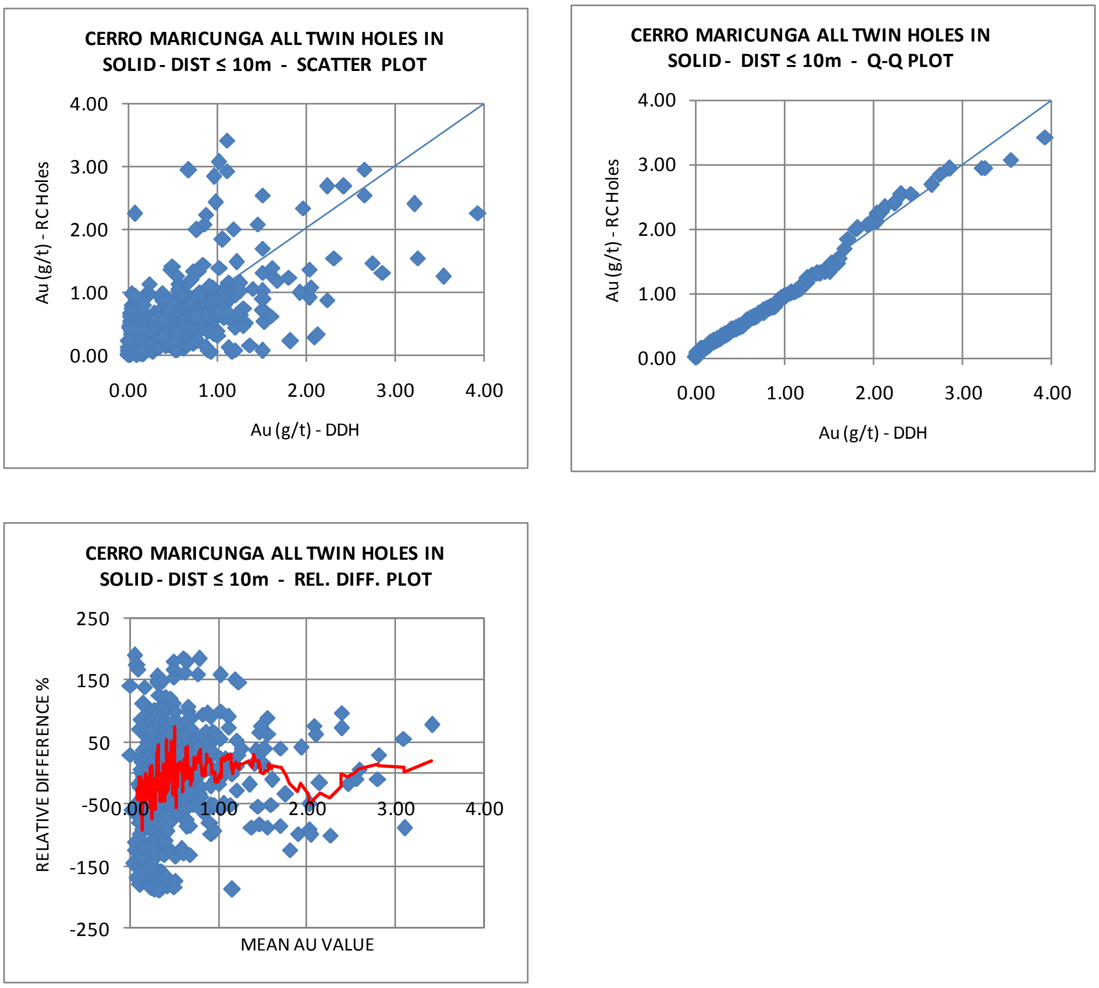


Figure 12-18: Statistical Plots for DDH & RC Twin Holes

12.7 Conclusions, 2013 QA/QC Campaign

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 Fenix meets acceptability criteria and therefore the exploration data used complies with required confidence for resource modelling and estimation.

12.8 QA/QC Report for the 2018-2019 Drilling Campaign

12.8.1 Introduction

In November 2018, Rio2 began an RC drill hole campaign with the aim of confirming previous data campaigns.

Rio2's 2018-2019 campaign included 39 RC drill holes and 12 surface trenches, all of which are fully executed. The drill holes were carried out by EXPLOMIN, while Rio2 staff completed trenches. Both RC drill holes and trenches have been sampled with a 2-meter support.

During this campaign, 4953 samples were sent to the laboratory, corresponding to approximately 16% of control samples. In addition, about 5% of control samples have been sent to a secondary laboratory (external check-up duplicates). The results of these controls are presented below.

12.8.2 Data Analysed

This report includes the review of 79 batches prepared in the ALS laboratory in Copiapo and Antofagasta, and analysed in ALS Lima, corresponding to 4,953 samples processed between November 2018 and February 2019. These batches include 3,528 drill holes samples and 729 trench samples, plus 696 control samples (Table 12-10). These samples represent a total of 7,066 meters of drill holes (10 m without recovery), and 1,462 meters of trenches. Duplicates of pulp for external check were sent to Actlabs La Serena.

Table 12-10: QA/QC Samples Insertion Rate, November 2018 – March 2019

QA/QC Sample Type	RC Samples Count	Trench Samples Count	Count	Rate of Insertion
BLK-FINE	67	15	82	1.93%
BLK-COARSE	66	15	81	1.90%
STD	208	46	254	5.97%
DUP-FIELD	109	25	134	3.15%
DUP-COARSE	62	10	72	1.69%
DUP-PULP	60	13	73	1.72%
DUP-CHECK	172	44	216	5.07%
TOTAL	**572**	**124**	**696**	**21.43%**

12.8.3 QA/QC Results

12.8.3.1 Granulometric Control

Granulometric checks were carried out for on approximately 12% of samples sent to the preparation laboratory. The 4% of the crushed material was under 2 mm (#10), and 8% for the pulverized material under 75 microns. The results of these controls are good (Figure 12-19). In case of non-compliance with the minimum required through material (70% under 2 mm and 85% of material under 75 microns), the process of crushing or pulverized is repeated, according to ALS procedures.

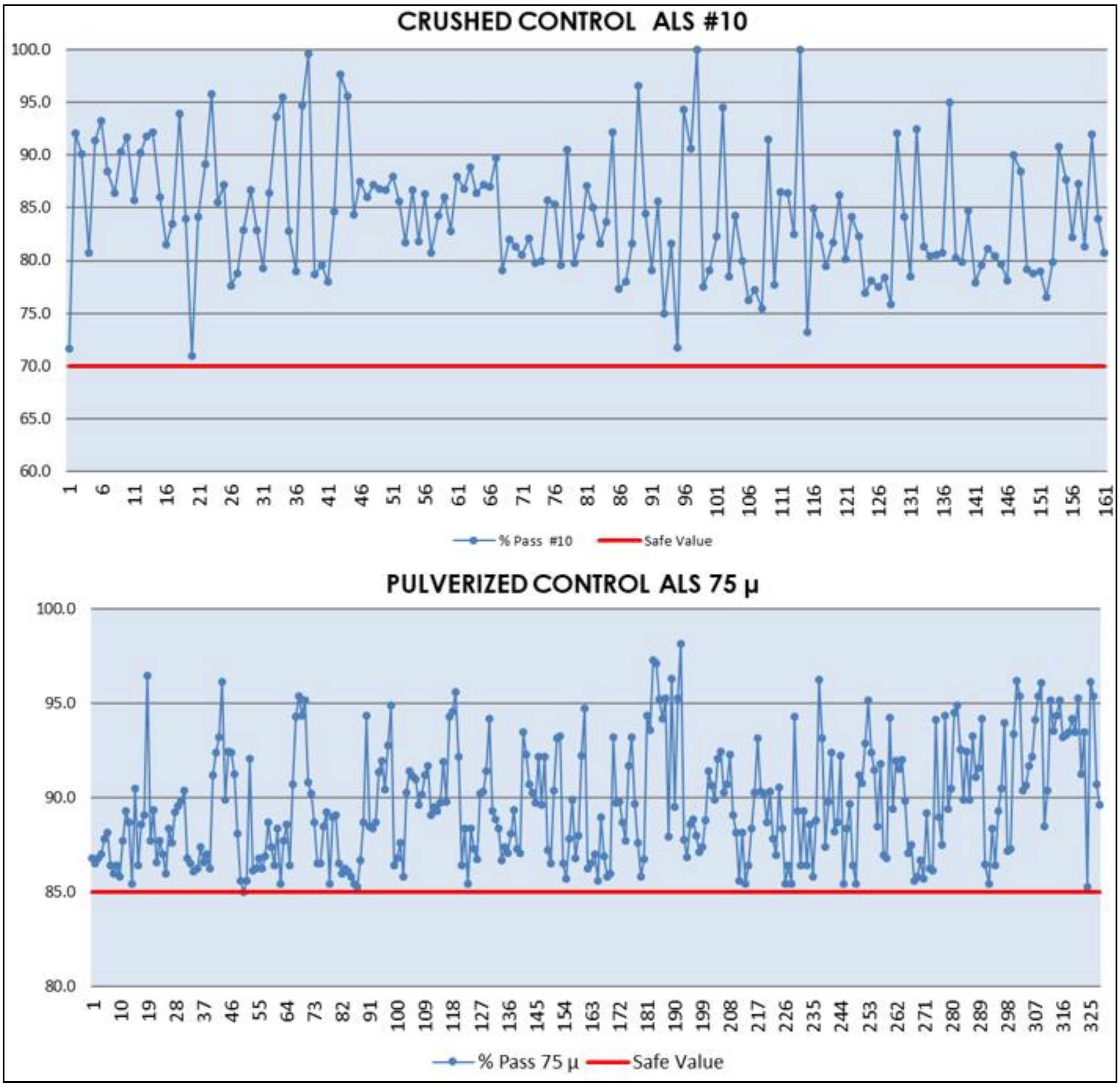


Figure 12-19: Granulometric control graphs for crushing and pulverized stages

12.8.3.2 Fine Pulp Blanks (BLK-FINE)

Eighty-two Fine Pulp Blanks Samples were reviewed. The blanks used are summarized in Table 12-11, both of which have certified Au values lower than the lower detection limit of the scanning method used. Blanks versus Precedent Sample charts were prepared for Au (Figure 12-20), and blanks in time (Figure 12-21). No contamination events were detected during the analysis (Table 12-12). The only irregular situation occurs between two samples, 1000048 and 1000050, originally sent as standard and fine blanks respectively, but with exchanged results. It is not possible to identify whether the error occurred during the preparation of the respective batch or in the laboratory. The database is corrected by assigning the corresponding reference material type.

Table 12-11: Fine Blank Types (certificated), used in campaign 2018-2019

BLK-FINE	ORIGIN	Au ppb	No.
CO18186443-F	ALS	< 1	32
GLG912-2	Geostats Pty Ltd	2.54	50

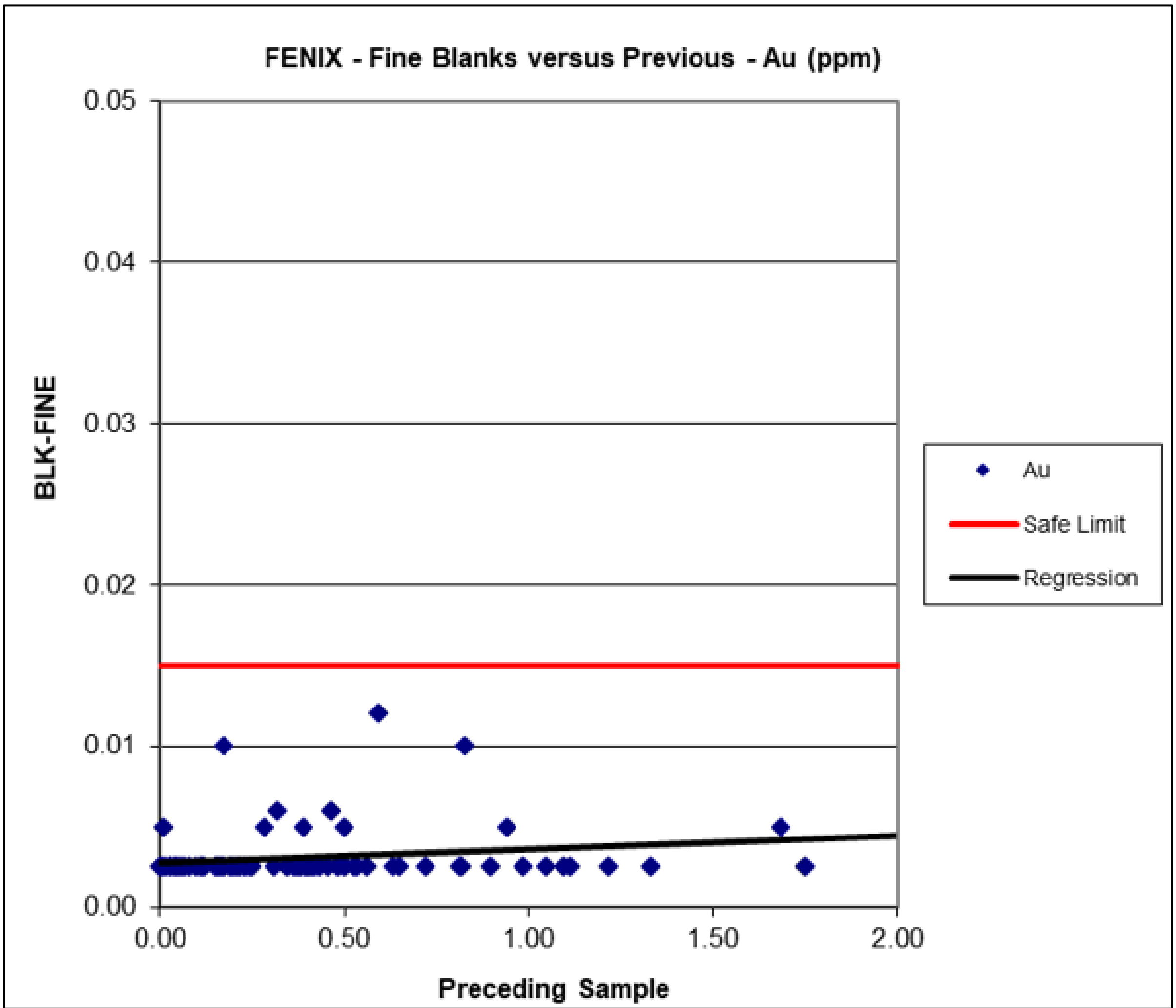


Figure 12-20: Fine Blanks vs Precedent Samples Graph

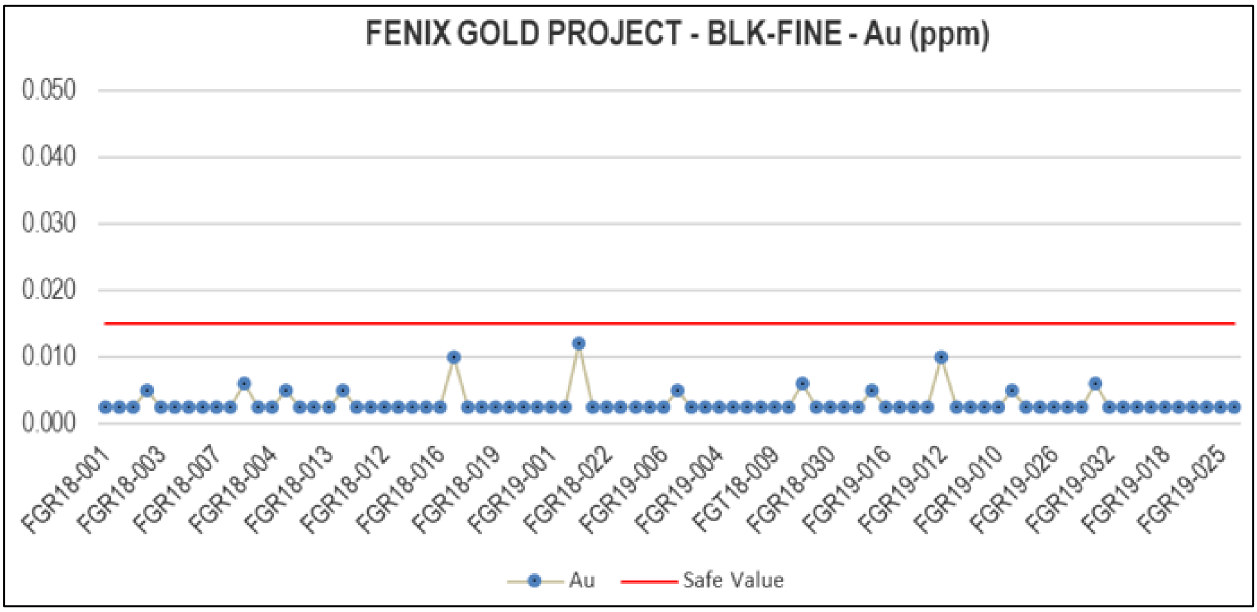


Figure 12-21: Fine Blank Graphs in time periods

Table 12-12: Fine Blank Analysis Summary

Fine Blanks Summary								
	No. BLF	Unit	Max Previous	Max Blank	Detection Limits	Max Ratio	Polluted Samples	Pollution Rate
Au	82	ppm	2.04	0.012	0.005	2.4	0	0.00%

12.8.3.3 COARSE BLANKS (BLK-COARSE)

In total, 81 Coarse Blanks were analysed. The blanks used are summarized in Table 12-13, and most of them have certified Au values lower than the lower detection limit of the analysis method used during the campaign. Coarse Blanks are certified by INTEM, despite having a certified value less than the detection limit used for certification (10 ppb), in practice have values less than the ALS detection limit (5 ppb), so they are considered appropriate to use. Coarse Blanks, for the most part, have been inserted after Fine Blanks, so that they can isolate and identify contamination during sample preparation.

Blanks versus Precedent Sample charts were prepared for Au (Figure 12-22), and blanks for time periods (Figure 12-23). A contamination event is observed in the preparation (Table 12-14), for batch FGR19-021, after a high-grade analysis. It is communicated to the laboratory and associated procedures are requested to be improved. The contamination rate for Coarse Blanks is 1.2%, which is within what is acceptable.

Table 12-13: Quantity and types of certified coarse blanks used in campaign 2018-2019

BLK-COARSE	ORIGIN	Au ppb	No.
CO18186443-C	ALS	< 1	27
IN-BMG-175	INTEM	< 10	15
IN-BMG-176	INTEM	< 10	19
OREAS-C26c	OREAS	< 2	11
OREAS-C27c	OREAS	< 2	9

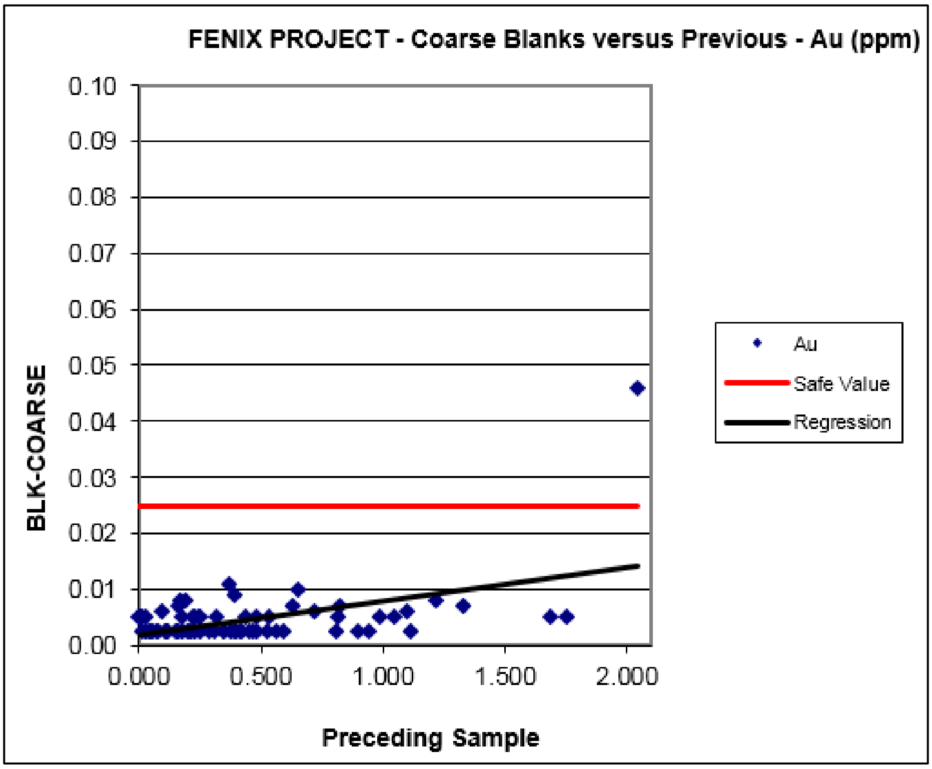


Figure 12-22: Coarse Blanks vs Precedent Samples

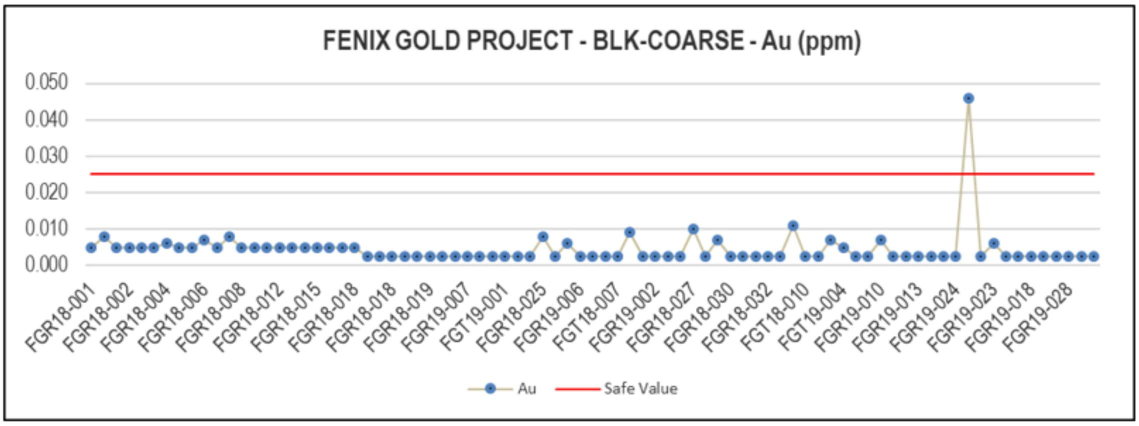


Figure 12-23: Coarse Blanks Graph on time

Table 12-14: Coarse Blanks Summary Analysis

Coarse Blank Summary								
Element	No. BLK	Unit	Max Previous	Max Blank	Detection Limits	Max Ratio	Polluted Samples	Pollution Rate
Au	81	ppm	2.04	0.046	0.005	9.2	1	1.20%

12.8.3.4 Standards (STD)

Two hundred and fifty four standard samples were used, representing an insertion rate of 6%. All standards are certified by Geostats Pty Ltd and correspond to the same standards used in previous campaigns. "Shewhart" (or process-behaviour) graphs were prepared for each type of standard (Figure 12-24 to Figure 12-27). The bias for each standard is calculated by removing out-of-control samples (Table 12-15). The laboratory was asked to re-analyse those out-of-control and some neighbouring samples, not identifying systematic errors.

However, for batch FGR19-015, the G303-8 standard failed again, giving a low-as-expected value. A new batch will be prepared with these samples to confirm the results.

Table 12-15 summarized types of standards used, quantity of each of them, number of out-of-control samples, certified values, mean, and associated biases. For bias calculation, the mean is considered of each standard with a precision of two decimal places (precision of the certified value). The biases obtained are measurable, considered good and within what is acceptable. Finally, in Figure 12-28, linear regression graph is shown for all standards used. The global bias Sg is 1%, calculated from the expression Sg-m-1, where m corresponds to the slope of the line of linear regression.

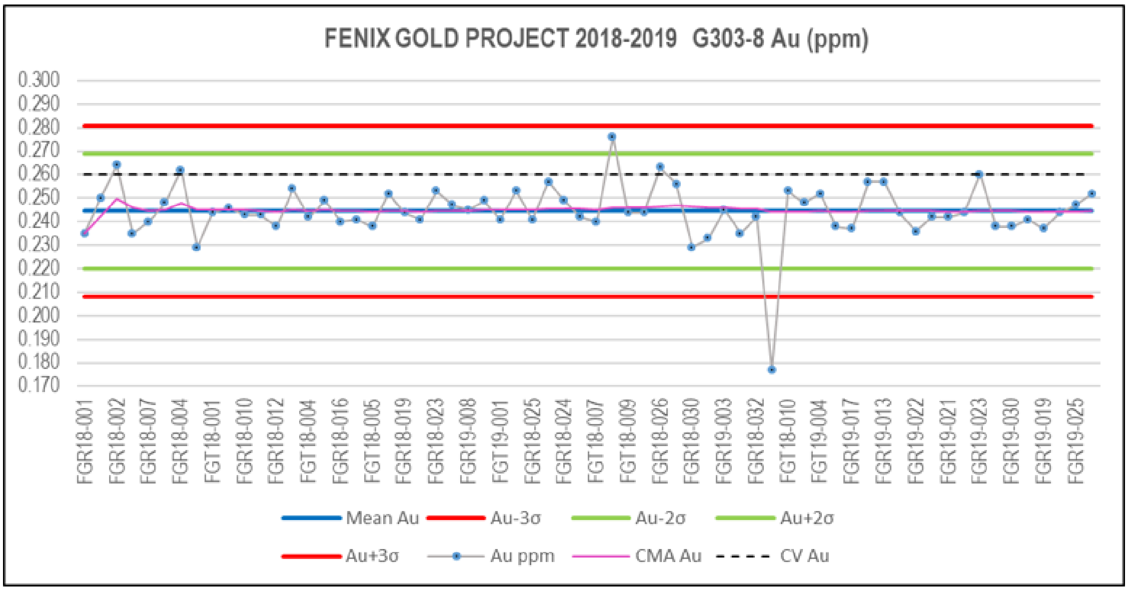


Figure 12-24: Graphic for Standard G303-8

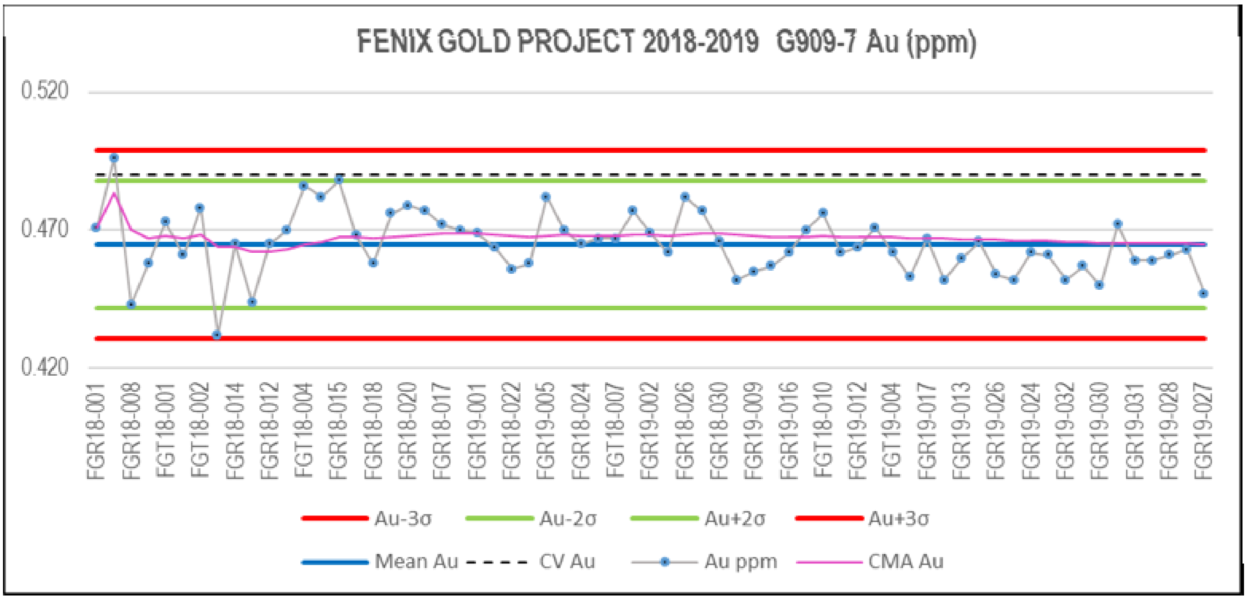


Figure 12-25: Graphic for Standard G909-7

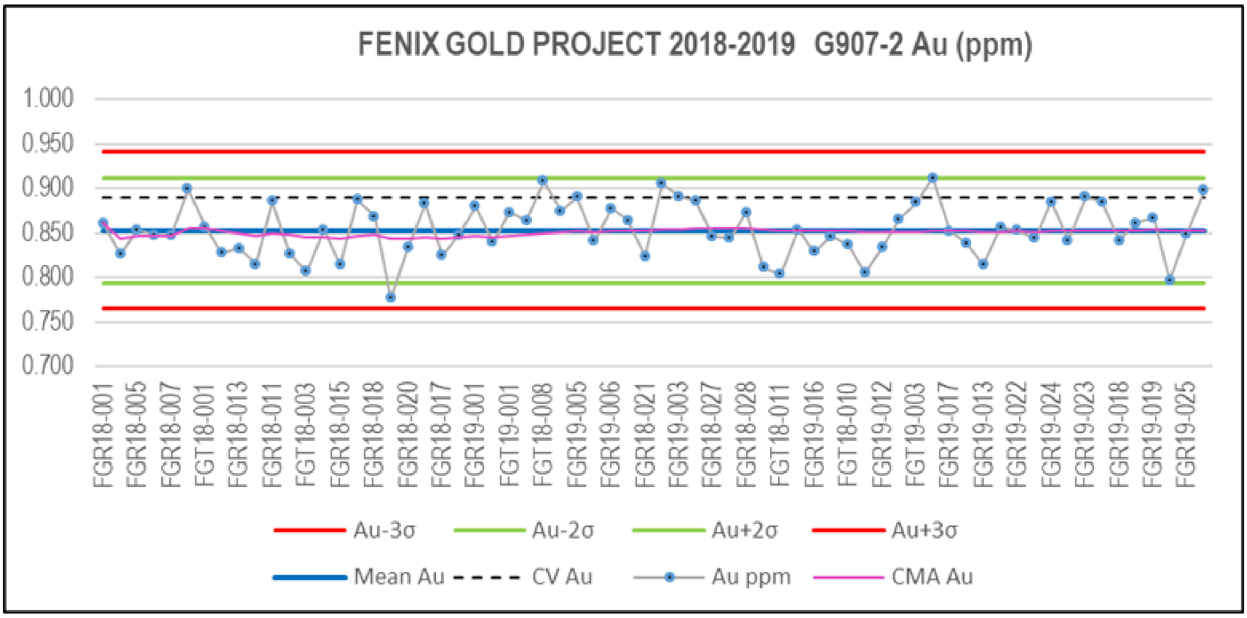


Figure 12-26: Graphic for Standard G907-2

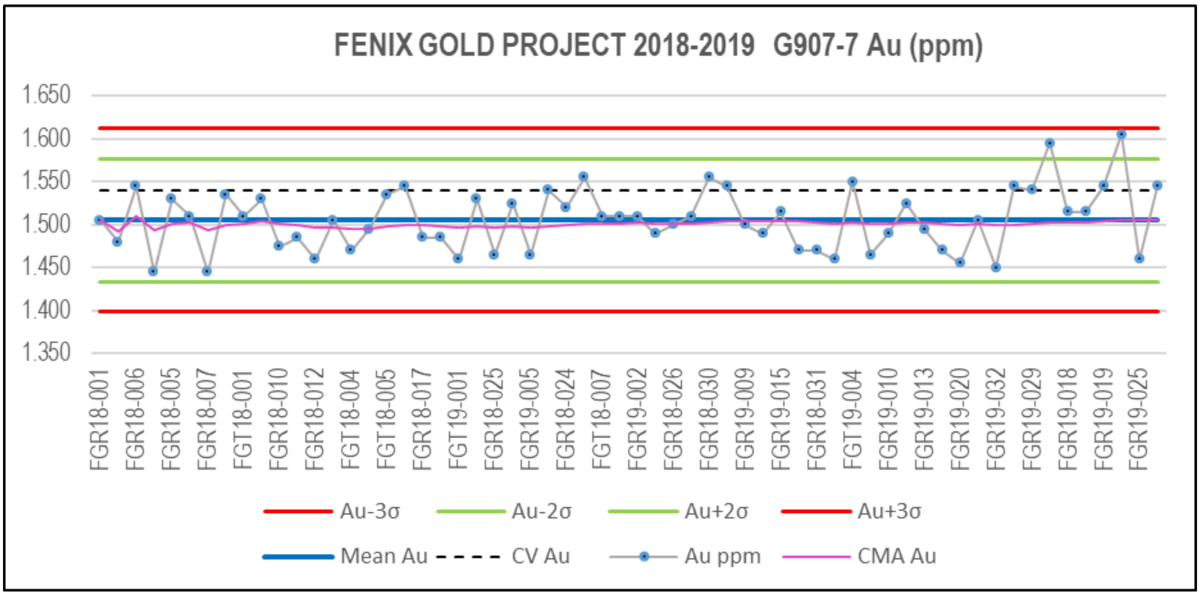


Figure 12-27: Graphic for Standard G907-7

Table 12-15: Review of CRM performance. CV= Certified Value, MCF= Outliers, SO= Excluding Outliers

STD	Element	n	CV	Mean (SO)	Bias (%)	MFC	MFC (%)
G303-8	Au	64	0.26	0.245	-3.8	2	3.13
G909-7	Au	65	0.49	0.465	-4.1	0	0
G907-2	Au	66	0.89	0.853	-4.5	0	0
G907-7	Au	60	1.54	1.506	-1.9	0	0

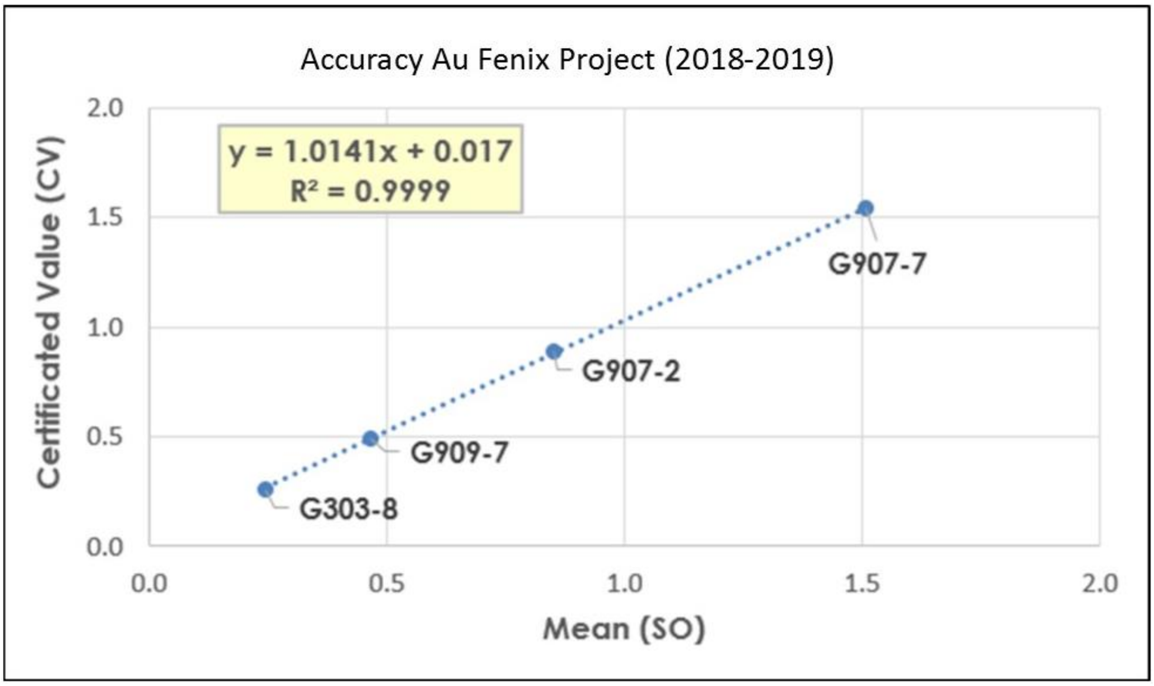


Figure 12-28: Linear Regression for Global Bias Calculation

12.8.3.5 External Laboratory Check

From the pulp rejections of the samples analysed during this campaign, 216 duplicates were selected for an external check, which were sent to the Actlabs laboratory in La Serena. In addition, 14 standards, 11 duplicates and 8 fine blanks (15% of control samples) were included, totalling 249 samples to be analysed.

This check is intended to complement the evaluation of the accuracy of the main laboratory (ALS). Granulometric control of 10% of the pulps sent (Figure 12-29), the results of which are mostly favourable, and the homogenization of the samples prior to analysis, following the same procedure and method of analysis of the main lab.

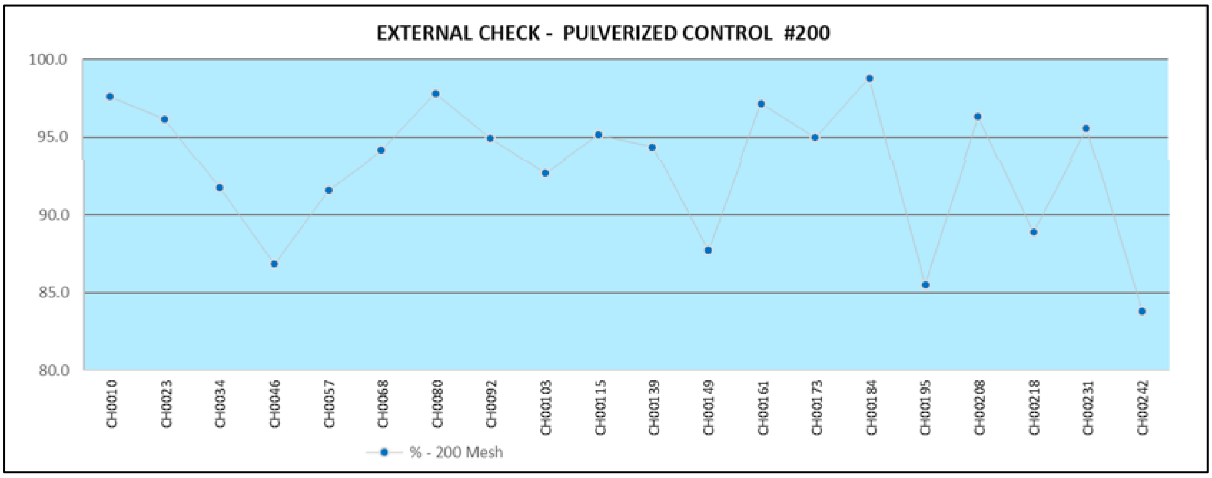


Figure 12-29: Granulometric graph control for external check

Preliminary results, including control samples, showed a systematic error during the analysis. The analyses for the trenches in question were repeated. Following the investigation, Actlabs reports that the problem is caused by "a deviation in the method volume dispensers, which were changed for standard reanalysis".

12.8.3.6 Duplicates

Table 12-16 summarizes the results of the duplicate analysis by type, which were evaluated by Au through the hyperbolic method. It is considered that a duplicate failed when the absolute value of the relative error, relative to its original, is greater than 30% for field duplicates (or twins), 20% for coarse duplicates, or 10% for pulp duplicates, making adjustments for samples with values close to the detection limit typical of the method used.

Table 12-16: Summary analysis for hyperbolic method

	Fields Duplicates			Twin Duplicates			Coarse Rejected Duplicates			Pulps Duplicates		
Element	Total	Fail	Error Rate (%)	Total	Fail	Error Rate (%)	Total	Fail	Error Rate (%)	Total	Fail	Error Rate (%)
Au	109	0	0	25	4	16	72	3	4.2	73	3	4.1

Field Duplicates and twins (DUP-FIELD)

The field duplicate, for RC drill hole samples, corresponds to a sample extracted from the rejection of the first division of the drilled material, following the same procedure of the original sample. For trenches, the twin duplicate (Twin Sample) corresponds to a parallel trench, immediately next to or under the original trench. In both cases, the sampling support is two meters.

Although both sample types are identified in the database as DUP-FIELD, the sampling procedure is different, so they are analysed separately.

The results of 109 field duplicates and 25 twin duplicates were reviewed, representing a total insertion rate of 3.15%. Max-Min graphics were prepared for Au (Figure 12-30 and Figure 12-31). For field duplicates, there are no failed samples. For twin duplicates the error rate is 16%, which is above the acceptable maximum, however, it must be considered that the number of trench duplicates is small and that 3 of the failed samples belong to trenches 7 and 8, whose orientation is approximately parallel to one of the main directions, of the veins that host most of the Au within the site, which may explain the bias between samples.

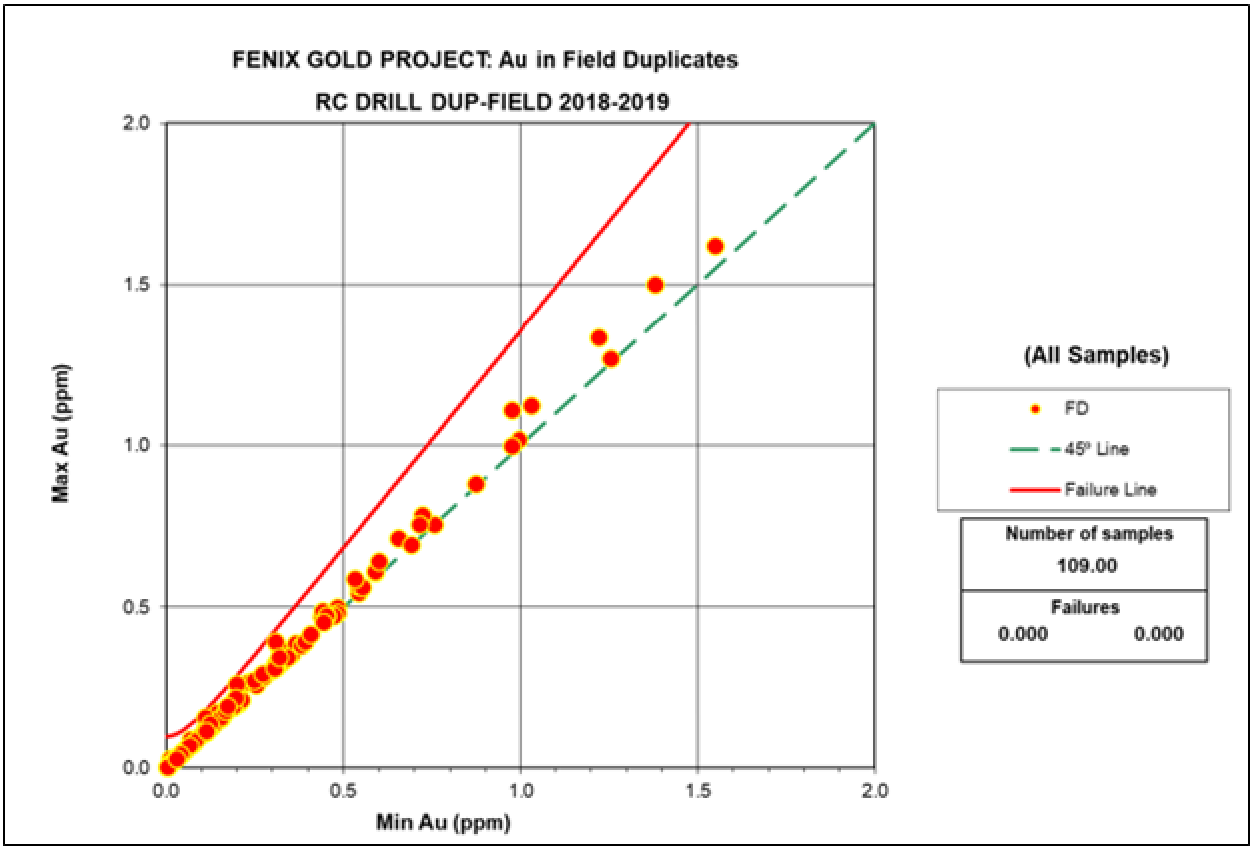


Figure 12-30: Graph Max-Min (Hyperbolic methodology), for field duplicates (DUP-FIELD) RC drill holes

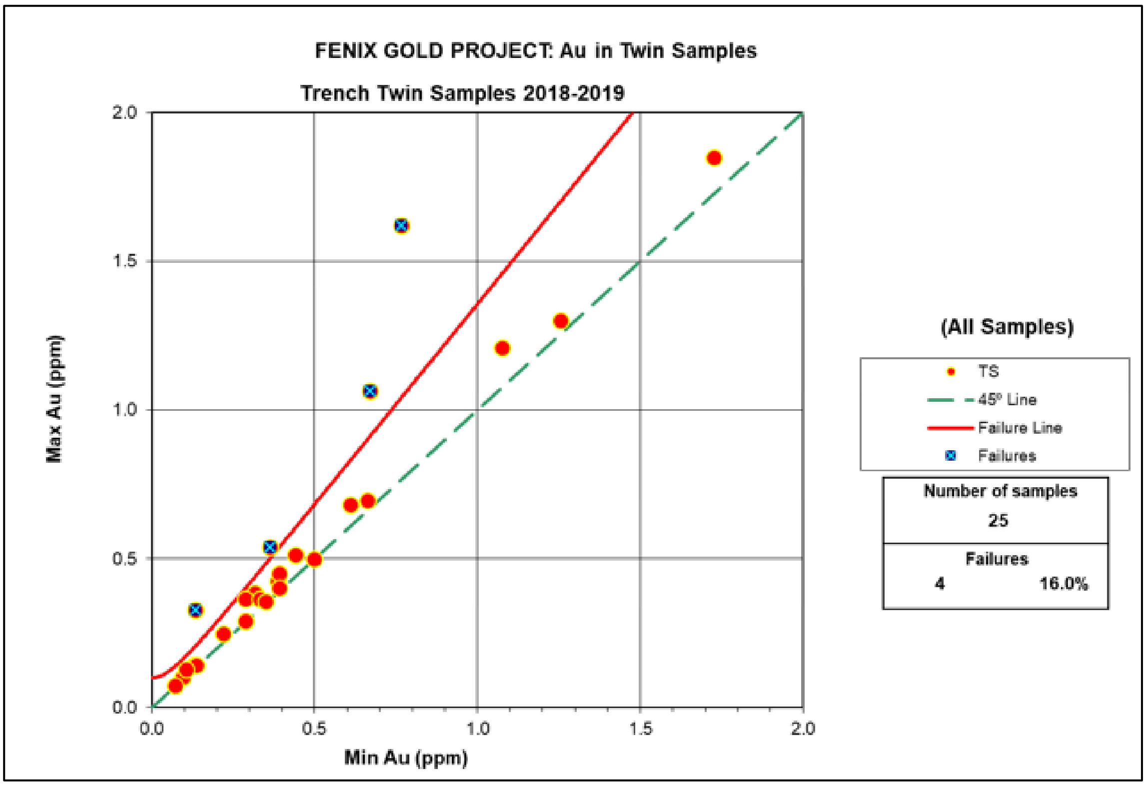


Figure 12-31: Graph Max-Min (Hyperbolic methodology), for twin Duplicates (Twin Samples), trench

Coarse Reject Duplicates (DUP-COARSE)

Seventy-two (72) pulp duplicates are reviewed, representing an insertion rate of 1.69%. Max-Min chart is prepared for Au (Figure 12-32). The error rate is 4.2%, which is considered within acceptable limits.

Pulp Duplicates (DUP-PULP)

Seventy-three (73) pulp duplicates are reviewed, representing an insertion rate of 1.72%. Max-Min chart is prepared for Au (Figure 12-33). The error rate is 4.2%, which is considered within acceptable limits.

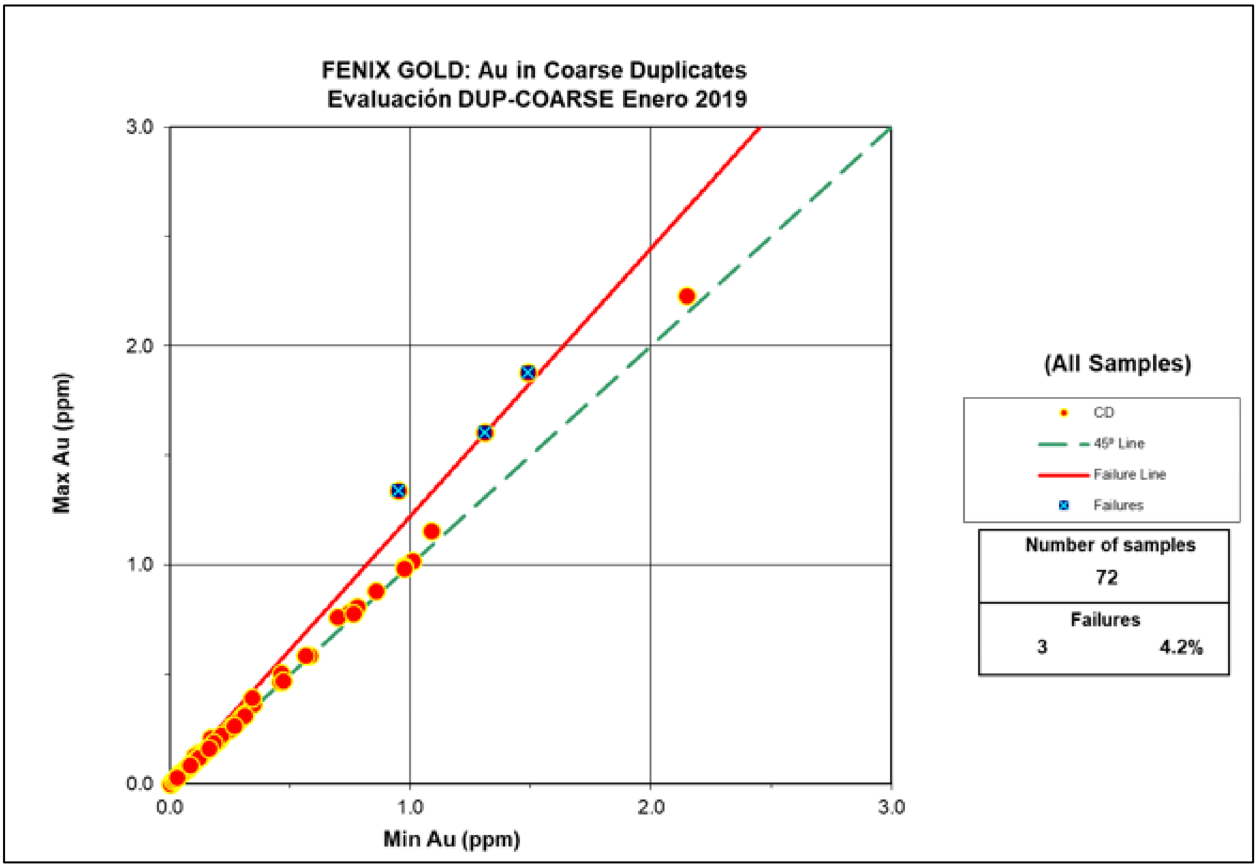


Figure 12-32: Graph Max-Min for Duplicates coarse reject

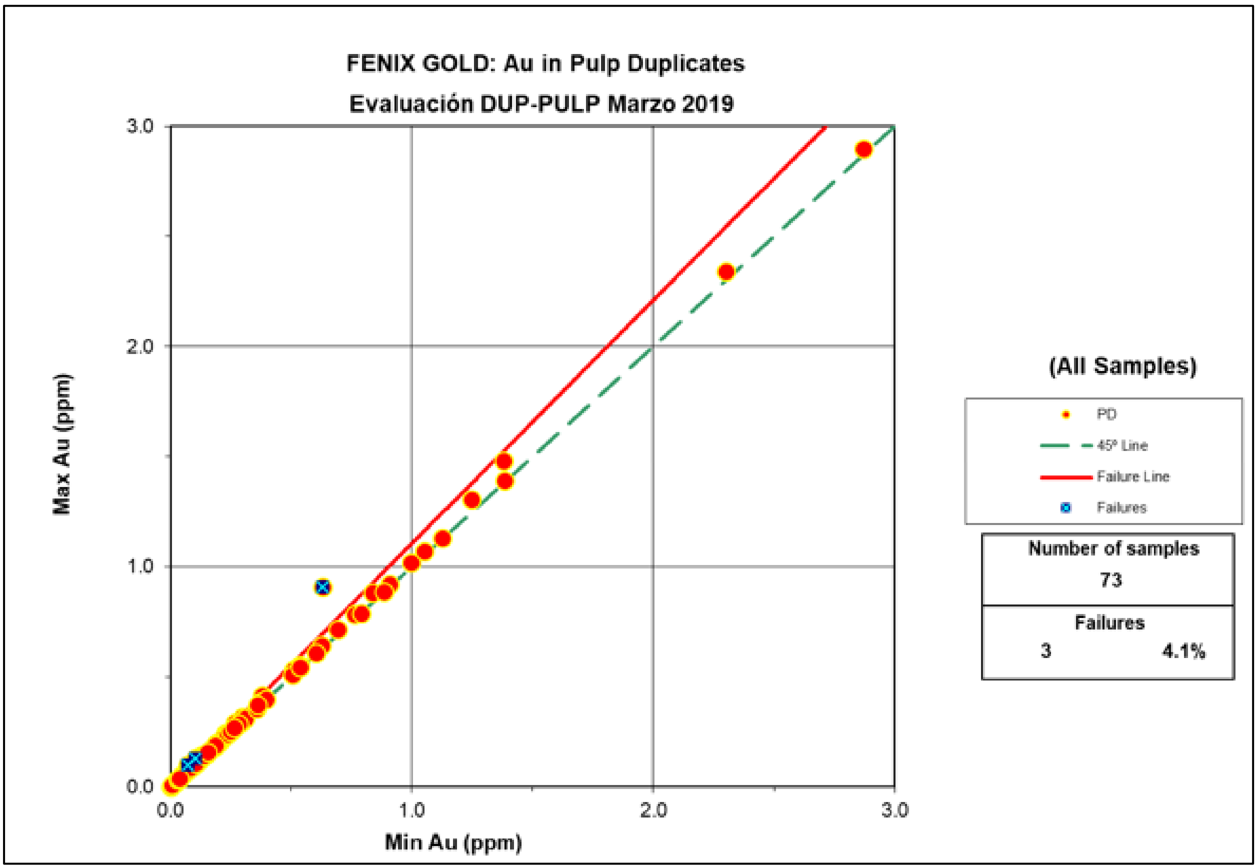


Figure 12-33: Graph Max-Min for pulp Duplicates

12.8.4 Conclusions, 2018-2019 QA/QC Campaign

Because of the QA/QC review during the period November 2018 – May 2019, it is concluded:

- The contamination rate during sample preparation is within acceptable ranges.
- No contamination events are detected during the analytical process.
- Analytical accuracy for Au is within acceptable limits.
- Analytical biases for Au are negative, implying that the ALS Laboratory underestimates grades, within acceptable ranges.
- The overall bias is 1%, which is within what is acceptable.
- Duplicate data indicate that sampling, sub-sampling, and analytical variance are within acceptable limits.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Over the period 2008 to 2017, several metallurgical test work campaigns have been carried out on composites of mineralized material from various zones in the Project. The sequence of campaigns is shown in Figure 13-1. The laboratories involved were AMTEL (Advanced Mineral Technology Laboratory Ltd), London, Ontario, Canada; Kappes, Cassiday & Associates (KCA), Reno, Nevada, USA; and Plenge, Lima, Peru.

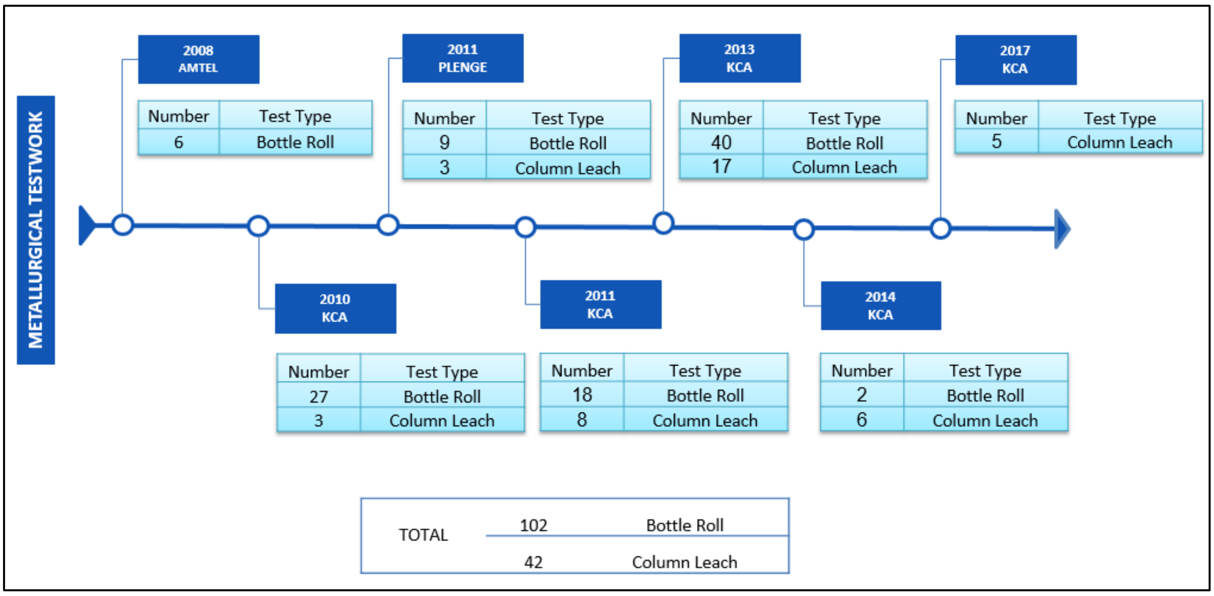


Figure 13-1: Test work Campaigns from 2008 to 2017

There were multiple campaigns from 2008 to 2014 to study Au and Ag extraction under different leaching conditions and material feed size distributions to provide the basis for the design parameters for an industrial heap leach circuit.

The leaching studies covered bottle roll tests and column tests, which were conducted on material crushed to a wide range of particle sizes, including fine sizes for the bottle roll tests and coarser sizes for the column tests. In parallel, leaching tests were carried out to determine the optimum pH and cyanide conditions and lime and cement consumptions.

Crushing tests were carried to determine the crushing work index and abrasion index. High pressure grinding roll (HPGR) crushing technology was also tested.

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 work reported in the 2014 Technical Report is summarized in paragraphs 13.2 to 13.4.

In 2017, a test work campaign was carried out by KCA for Atacama. The purpose of this work was to test column leaching with material crushed to P80 sizes of 100 mm and 75 mm. This work is summarized in paragraph 13.5. Overall conclusions are provided in paragraph 13.6 and recommendations in paragraph 13.7.

13.2 Characterization of Composite Samples (October 2014 PFS Report)

Table 13-1 shows the gold and silver assays in the composite samples that were processed in the test work. Rio2 has renamed the zones of the Fenix deposit as follows: Crux (CX) is now Fenix South; Lynx (LX) is now Fenix North; and, Phoenix (PX) is now Fenix Central (A and B). To avoid confusion with all other sections in this report, the new zone names have been used in this report.

Table 13-1: Composite Sample Identification

Zone	AMTEL SAMPLE NAME	KCA avg. Head assays Au g/t	KCA head assays Ag g/t	KCA head assays Cu g/t
Fenix South	Composite 1	1.08	0.4	376
Fenix South & Central	Composite 2	0.78	0.4	247
Fenix Central	Composite 4	0.28	-	129
Fenix Central	Composite 5	0.49	-	260
Fenix Central	Composite 6	0.55	-	195
All Zones Combined	Composite 7	0.23	-	139
Fenix South	Composite Crux 0.25	0.25	0.88	124
Fenix South	Composite Crux 0.45	0.44	1.41	148
Fenix North	Composite Lynx 0.25	0.24	0.73	139
Fenix North	Composite Lynx 0.45	0.47	0.82	203
Fenix Central	Composite Phoenix 0.25	0.24	1.06	133
Fenix Central	Composite Phoenix 0.45	0.45	0.95	284

The spatial location of the individual composites is shown in Figure 13-2.

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

13.2.1 Bond Work Indices (BWi) and Abrasion Indices (Ai)

The BWi values varied from 9.7 kWh/t to 11.3 kWh/t indicating medium hardness. The Ai values varied from 0.0669 to 0.1251 indicating relatively low abrasiveness.

13.2.2 Mineralogical Characterization

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

The analyses were carried out with conventionally crushed (P80 of 11-13mm) and milled (P80 of 80-100 µm) sub-samples of the composites mentioned above.

Summaries of the two deportment studies are detailed in Table 13-2 and Table 13-3.

Table 13-2: Gold Deportment in Crushed Ore

AMTEL SAMPLE NAME	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	74	13	13	63	80

Note: BRT = bottle roll tests, CLT = column leach tests

Table 13-3: Gold Deportment in Milled Ore

AMTEL SAMPLE NAME	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

Conclusions from the data shown in these tables are:

- Free and/or attached gold grains range from 74% to 81% in crushed ore and 78% to 90% in milled ore.
- Refractory gold occurs as very fine-grained gold contained within microcrystalline quartz.

Gold deportment studies were also carried out on 16 column leach test residues. The results indicated that gold in the residues occurs in very small amounts of water soluble gold (<1%). The ratios of exposed-enclosed-refractory gold were 1:4:5.

13.2.2 Spatial Distribution of Individual Samples

Composites were generated with mineralized material obtained from each mineralized zone of the deposit (Fenix South, Fenix Central and Fenix North). Location and mean grade of each composite are depicted in the longitudinal section shown in Figure 13-2.

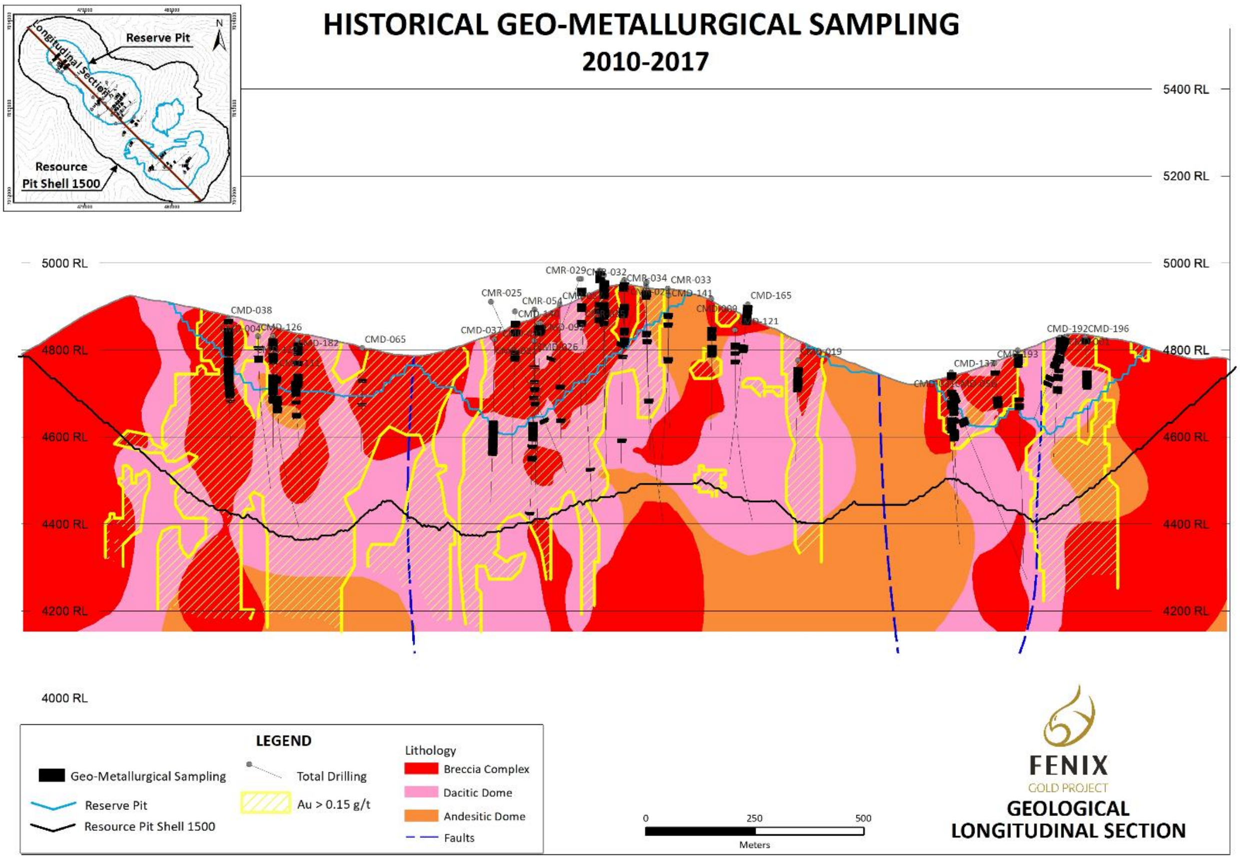


Figure 13-2: Location of Composites within each Mineralized Zone

13.3 Cyanidation Tests

13.3.1 Column Percolation Tests

Column percolation leach tests were conducted on 12 composite samples representative of the deposit. The primary objective of these tests was to determine gold recovery for crushing sizes between P80 19 mm and P80 100 mm. The leach parameters and results are shown in Table 13-4 and Table 13-5.

Table 13-4: Summary of Selected Column Percolation Leach Test Parameters

AMTEL SAMPLE NAME	Target P_{80}/P_{100} crush size (mm)	Column diameter (mm)	Initial charge height/weight (m)	Initial NaCN concentration gpl NaCN	Maintained NaCN concentration gpl NaCN	Maintained pH value	Cement dosage (kg/t)
Composite 1	19	127	1.66	1	0.5	9 to 11	1
Composite 2	19	127	1.59	1	0.5	9 to 11	1
Composite 2	9.5	127	1.67	1	0.5	9 to 11	1
Composite 4	19	152	1.55	1	0.5	9 to 11	1
Composite 4	19	152	1.58	1	0.5	9 to 11	1
Composite 5	19	152	1.52	1	0.5	9 to 11	1
Composite 5	19	152	1.58	1	0.5	9 to 11	1
Composite 6	100	445	3.01	1	0.5	9 to 11	1
Composite 6	50	292	2.5	1	0.5	9 to 11	1
Composite 6	19	152	1.68	1	0.5	9 to 11	1
Composite 7	19	152	1.6	1	0.5	9 to 11	1
Comp. Crux 0.25	25 (P_{100})	150	50 kg	1	0.5	9 to 11	0
Comp. Crux 0.45	25 (P_{100})	150	50 kg	1	0.5	9 to 11	0
Comp. Lynx 0.25	25 (P_{100})	150	50 kg	1	0.5	9 to 11	0
Comp. Lynx 0.45	25 (P_{100})	150	50 kg	1	0.5	9 to 11	0
Comp. Phoenix 0.25	25 (P_{100})	150	50 kg	1	0.5	9 to 11	0
Comp. Phoenix 0.45	25 (P_{100})	150	50 kg	1	0.5	9 to 11	0

Table 13-5: Summary of Column Percolation Leach Results

AMTEL SAMPLE NAME	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	1.08	1.13	GAC	89	57	1.03	3.08
Composite 2	19		0.76	GAC	79	57	1.06	3.07
Composite 2	9.5	0.78	0.79	GAC	80	57	1.19	3.06
Composite 4	19		0.31	GAC	80	87	0.82	2.51
Composite 4	19	0.28	0.31	GAC	82	87	0.52	2.51
Composite 5	19		0.53	GAC	86	87	0.74	2.01
Composite 5	19	0.49	0.5	GAC	84	87	0.97	2.03
Composite 6	100		0.58	GAC	77	87	0.09	6.61
Composite 6	50	0.55	0.54	GAC	78	87	0.1	6.66
Composite 6	19		0.58	GAC	80	87	0.44	6.53
Composite 7	19	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	5.1
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

Note: GAC = granular activated carbon; SA = solution assays

Conclusions from the column leach tests were:

- Gold extraction for composite 1, which has the highest grade, was 89% at P80 19 mm, after 57 days of leach from the column leach.
- Gold extractions for composite 2 only increased from 79% to 80% when the crush size was decreased from 19 mm to 9.5 mm.
- The average gold extraction for composite 4 was 81% at P80 19 mm, after 87 days of leach.
- The average gold extraction for composite 5 was 85% at P80 19 mm, after 87 days of leach.
- Gold extraction for composite 6 was 80% at P80 19 mm, after 87 days of leach.
- Gold extraction for composite 7 was 78% at P80 19 mm, after 82 days of leach.
- Gold extraction for composite 6 decreased from 80% to 78% and to 77% when the crush size was increased from 19 mm to 50 mm and to 100 mm, respectively.
- Gold extractions for the last six composites in Table 13.5 ranged between 78% and 82%, at P100 25 mm after 113 days of leach. There was no apparent relationship between head grade and gold extraction.

13.3.2 Additional column percolation tests

Additional tests were carried out to study the effects of crush size, agglomeration with cement and high pressure grinding roll (HPGR) crushing. The conclusions were:

- Coarsening the crush size P80 from 21 mm to 139 mm in the surface trench composite resulted in a 7% loss in gold recovery.
- Agglomeration with 12.5 kg/t cement had a detrimental effect on gold recovery.
- The effects of blending and agglomerating with 1 kg/t of cement on gold recovery were inconclusive.
- HPGR crushing showed only a small increase in gold recovery.

13.3.3 Bottle Roll Tests (BRT)

It was determined that bottle roll tests did not provide useful information for predicting column leach performance.

13.3.4 General Conclusions from the Cyanidation Tests

Effect of grade in the gold extraction

Figure 13-3 shows the relationship between gold head grade and gold extraction for column tests conducted on material crushed to a P80 of 19 mm. This includes tests where cement was used for agglomeration. Over the head grade range 0.22 g/t to 1.13 g/t the extraction varied from 78% to 89%.

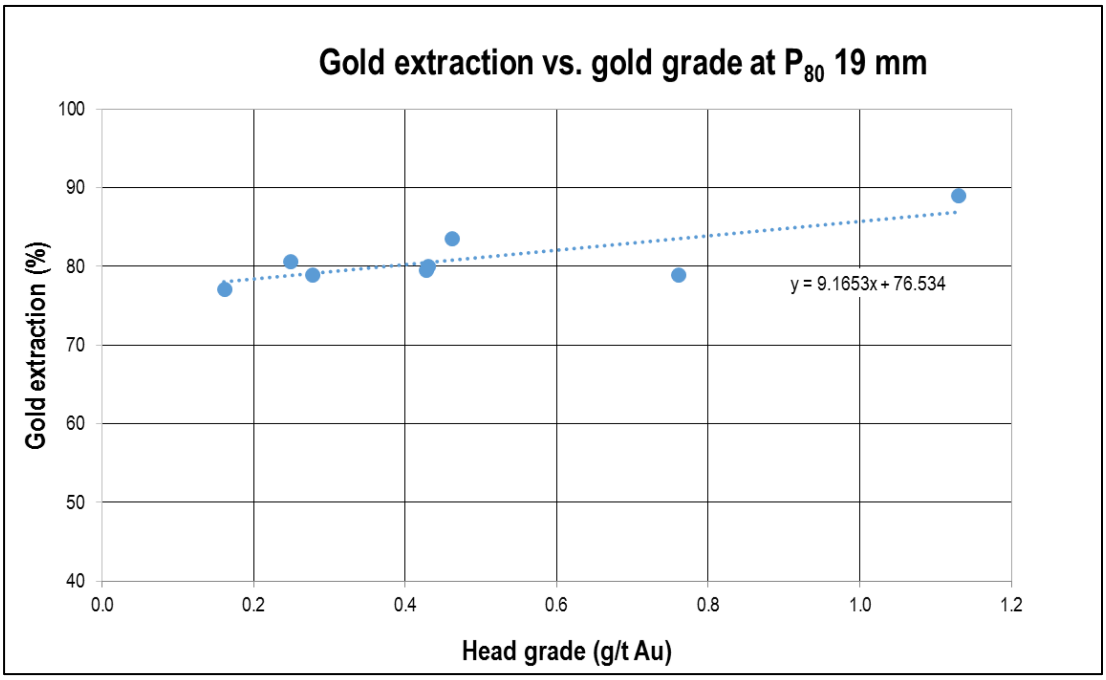


Figure 13-3: Gold Extraction v/s Gold Grade

<u>Effect of particle size in gold extraction</u>

Based on the column percolation leach test results, the effect of the crush size on gold extraction was quite low; the extraction obtained for composite two at P80 9.5 mm was similar to that obtained at P80 19 mm. Recoveries for different crush sizes in composite six varied only 3% as shown below:

- P80 - 100 mm, recovery 77%.
- P80 - 50 mm, recovery 78%.
- P80 - 19 mm, recovery 80%.

<u>Effect of Leaching Time on Gold Extraction</u>

The effect of leaching time on gold extraction can be seen in the kinetic curves in Figure 13-4 to Figure 13-6.

For composite 1 leaching was complete after 57 days at a gold extraction of 89%. For composite 2 extraction was complete after 45 days at the 9.5 mm and 19 mm crush sizes although leaching was slightly faster at 9.5 mm.

For composite 4 leaching was complete after 50 days with a gold extraction of 81%. Composite 5 was still leaching slowly after 87 days at about 85% extraction. Composite 6 completed leaching in 87 days at a crush size of 19 mm and a gold extraction 80%. At a coarser size, leaching was continuing slowly at an extraction of 77%. Composite seven completed leaching 60 days at a gold extraction of 77%.

All samples completed leaching in 123 days at gold extractions between 76% and 82%.

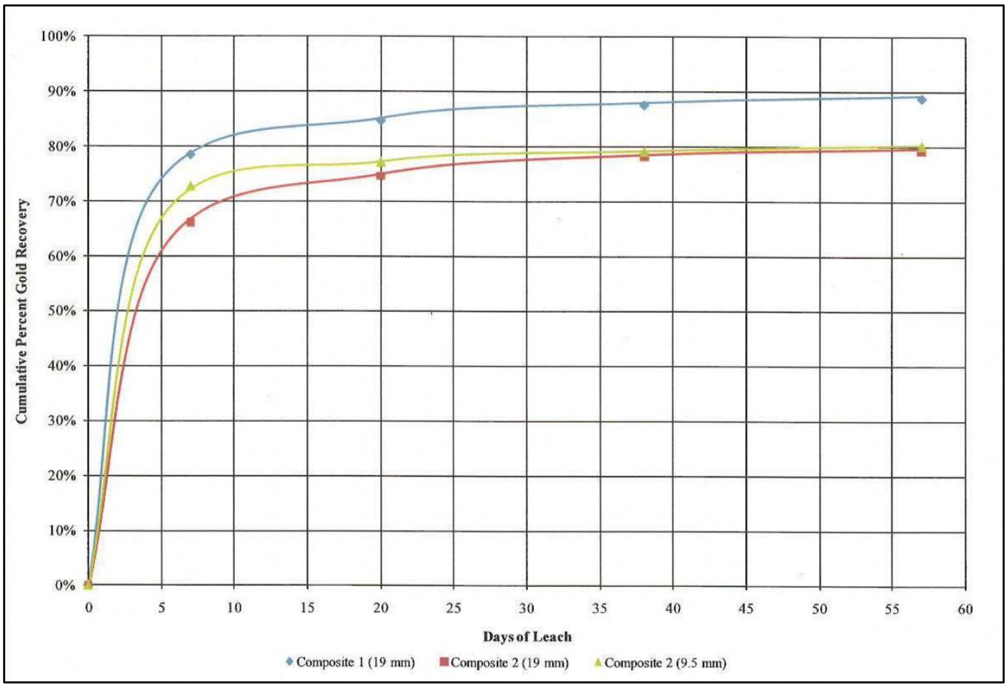


Figure 13-4: Kinetics Curves Composites 1 and 2

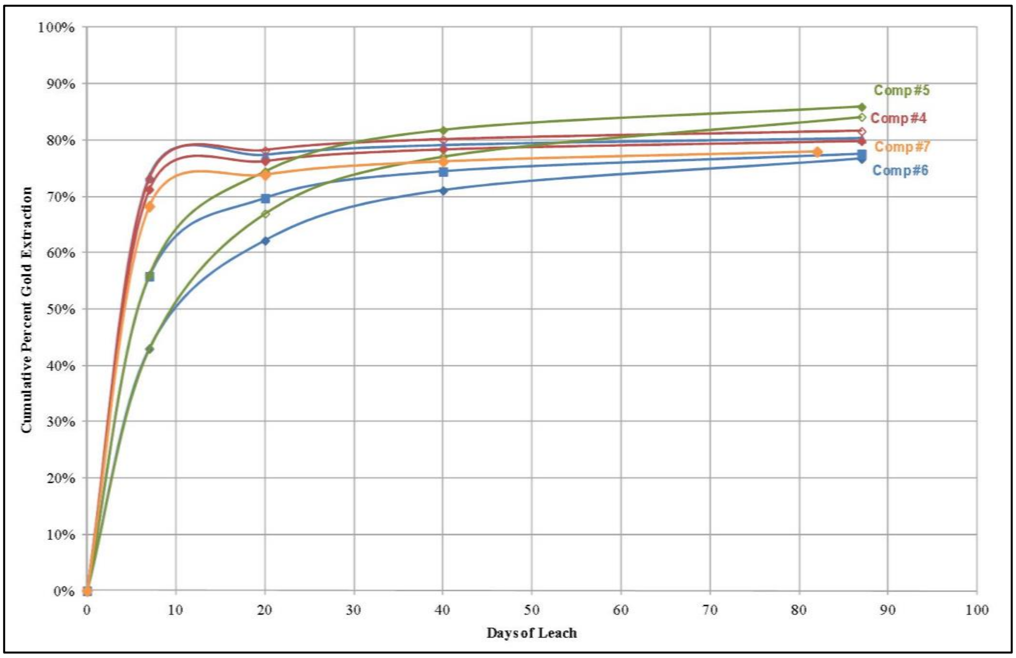


Figure 13-5: Kinetics Curves Composites 4, 5, 6 and 7

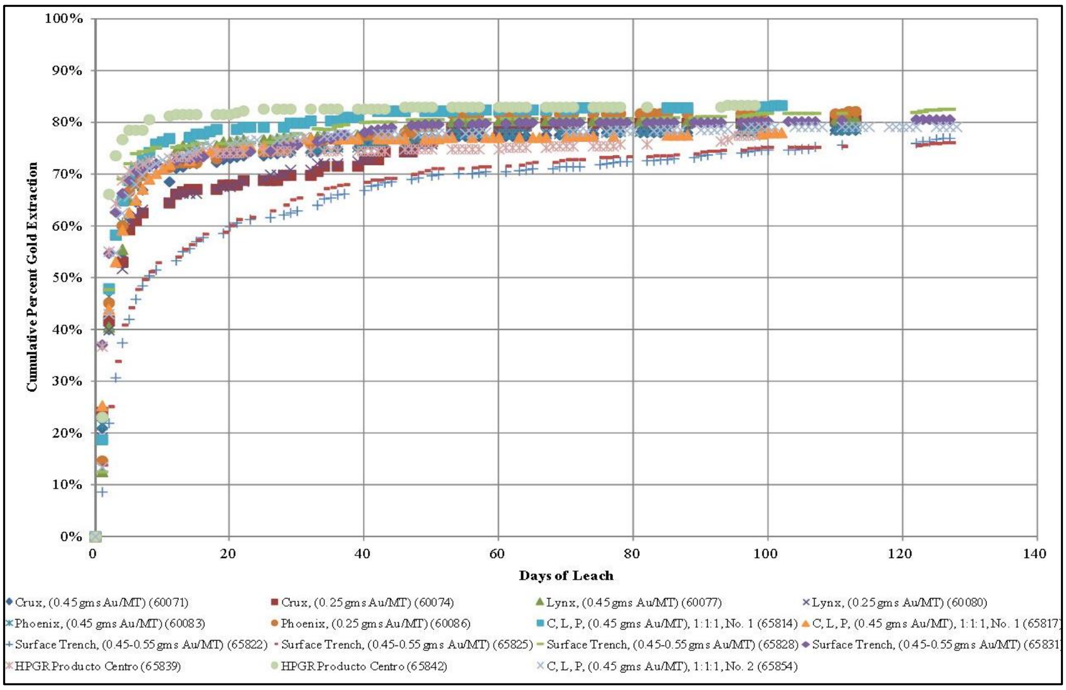


Figure 13-6: Kinetics Curves for all composites

13.3.4.1 Cyanide and lime consumption

The NaCN consumption varied from 0.03 kg/t to 1.03 kg/t with an average of approximately 0.75 kg/t. The lime consumption varied from 1.5 kg/t to 6.53 kg/t with an average of 4 kg/t.

Refractory Behaviour of Copper

The average head grade of copper in the twelve composites was 198 g/t. Copper extraction was measured in all composites and the findings were:

- Copper extraction in column leach tests on composites 1 and 2 was very low, ranging from 1.0% to 2.6%.
- Copper extraction on composites 4, 5, 6 and 7 was also very low, with values under 2%.
- Copper extraction on composites of Fenix South, Fenix North and Fenix Central was also low at 1.5%.

The low copper extractions were consistent with the mineralogical finding that chalcopyrite was the main copper mineral. Chalcopyrite does not leach significantly in sodium cyanide solution.

An approximate copper extraction of 2% was expected in an industrial cyanidation process, which means that a SART process will not be required.

13.4 Conclusions (2008 to 2014 work)

The following conclusions were drawn:

- Gold extractions of 80% can be achieved for material with 0.40 g/t Au at P80 19 mm crush size.
- A gold extraction versus gold head grade relationship was developed for material crushed to a P80 of 19 mm over the head grade range 0.22 g/t 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 cyanidable gold and hence easily recoverable; (2) enclosed cyanidable gold, not particularly sensitive to crush size; (3) refractory gold, which accounts for less than 10% of total gold.
- The mineralogical characterization findings suggested that crush size may not significantly influence gold extraction. This was confirmed in the test work and it was recommended that coarser crush sizes should be considered.

13.5 Test Work at Kappes, Cassiday and Associates (KCA, 2017)

The purpose of this work was to study gold extraction at coarse size fractions and potentially reduce the crushing requirement in an industrial plant.

13.5.1 Samples Dispatched to KCA

The samples were taken from three vertical 85 mm diameter holes drilled in the Fenix South (250 m long drill hole), Fenix Central (300m long drill hole) and Fenix North (150 m long hole) deposits. Descriptions and sample head assays dispatched to KCA are shown in Table 13-6.

Table 13-6: Sample Descriptions and Head Assays

KCA Sample No.	Description	Average Assay Au g/t	Average Assay Ag g/t
78601 A	Fenix South (Upper ½ of the hole)	0.344	0.45
78602 A	Fenix South (Lower ½ of the hole)	0.422	0.41
78603 A	Fenix North (Upper ½ of the hole)	0.483	0.25
78604 A	Fenix North (Lower ½ of the hole)	0.906	0.41
78605 A	Fenix Central (Upper ½ of the hole)	0.195	0.21
78606 A	Fenix Central (Lower ½ of the hole)	0.838	0.41
	Blended Samples		
78607 A	50% Fenix North Upper; 50% Fenix Central Upper	0.427	0.25
78608 A	50% Fenix North Lower; 50% Fenix Central Lower	0.803	0.41
78609 A	20% Fenix South Upper, 20% Fenix South Upper, 20% Fenix North Upper, 20% Fenix North Upper, 20% Fenix Central Lower	0.562	0.41

13.5.2 Sample Preparation

Sample preparation was conducted to provide material for head analyses, magnesium soak tests and column leach tests. In addition, composites were produced from the samples. A typical photograph of a sample after crushing to minus 150 mm is shown in Figure 13-7.

Figure 13-7: Fenix South Lower Sample Crushed to Minus 150 mm

The rock is generally competent, and the lack of fine material is notable (less than 1.5% in all samples).

Sample blending and separation for head sample characterization and preparation for column leaching tests was conducted according to KCA's standard procedures and industry standards.

13.5.3 Sample Characterization

Head sample characterization was carried on the samples prepared for column leaching shown in Table 13-6.

Gold and Silver Analyses

Head analyses for gold and silver were conducted on the sample material. A portion of the head material was crushed to 100% passing 1.70 mm. From the blended minus 1.70 mm material, quintuplet 500 gm splits were ring and puck pulverized to a target size of 80% passing 0.075 mm. Gold content was determined using standard fire assay methods with flame atomic absorption spectrophotometric (FAAS) finish. Silver content was determined using wet chemistry methods (4-acid digestion) with FAAS finish. The average gold assays for each sample are shown in Table 13-7; the silver assays were all between 0.21 g/t and 0.41 g/t.

Carbon and Sulphur Analyses

Head analyses for carbon and sulphur were conducted utilizing a LECO CS 230 unit. In addition to total carbon and sulphur analyses, speciation for organic and inorganic carbon and speciation for sulphide and sulphate sulphur were conducted. The Fenix South Upper sample had the highest total carbon and organic carbon assays at 0.18% and 0.11%, respectively. The corresponding assays for Fenix South Lower were 0.18% and 0.08%. For all other samples, the total carbon assays varied from 0.09% to 0.16% with organic carbon in the range of 0.03% to 0.07%.

The total sulphur assays ranged from 0.14% to 0.35%; sulphide sulphur assays were all 0.01% or less.

Mercury and Copper Analyses

Head analyses for mercury were conducted utilizing cold vapour/atomic absorption methods. Total copper analyses were conducted utilizing inductively coupled argon plasma–optical emission spectrophotometer (ICAP-OES) as well as by FAAS methods. The mercury assays varied from 0.02 mg/kg to 0.03 mg/kg in all samples.

The total copper assays varied from 194 ppm to 356 ppm. The Fenix North and Fenix Central the cyanide soluble copper was 7% to 11% respectively of the total copper present. For Fenix South samples had the highest cyanide soluble copper content at 60 ppm, which was 22% of the total copper present in Fenix South Upper and 29% for Fenix South Lower.

Multi-element Analysis

Semi-quantitative analyses were conducted by means of an ICAP-OES for a series of individual elements and whole rock constituents (lithium metaborate fusion/ICAP). Notable metal analyses were barium, 629 mg/kg to 703 mg/kg; manganese, 425 mg/kg to 952 mg/kg; and strontium, 781 mg/kg to 856 mg/kg. It is recommended that the radioactivity of the mineralized material is checked, and the strontium level is monitored in the leach solution to ensure there is no build-up of radioactivity.

The whole rock analysis showed the main constituent was SiO2 at 61% to 66% followed by Al2O3 at 15% to 16%, CaO at 4% to 4.8%, Fe2O3 at 3.5% to 4.4% and MgO at 1.7% to 3.9%.

Cyanide Soluble Analyses

Samples of 15 g were pulverized and leached in a cyanide solution for 24 hours. The results are shown in Table 13-7. The Fenix South Upper and Fenix South Lower samples show the highest copper extractions; Fenix Central Upper has the lowest gold extraction but also the lowest head grade. Gold extractions ranged from 74% to 91%; silver extractions ranged from 19% to 37%.

Table 13-7: Cyanide Shake Test Results

Sample	Head Assay Au g/t	Au extraction %	Ag extraction %	Cu extraction mg/kg	Cu mg/L
Fenix South Upper	0.344	80	37	60.32	30.3
Fenix South Lower	0.422	88	34	60.72	30.5
Fenix North Upper	0.438	89	34	16.48	8.5
Fenix North Lower	0.906	91	29	39.42	19.7
Fenix Central Upper	0.195	74	19	9.29	4.6
Fenix Central Lower	0.838	87	27	20.34	10.2
50% Fenix North Upper; 50% Fenix Central Upper	0.427	83	34	15.24	7.6
50% Fenix North Lower; 50% Fenix Central Lower	0.803	90	30	29.64	14.9
20% Fenix South Upper, 20% Fenix South Upper, 20% Fenix North Upper, 20% Fenix North Upper, 20% Fenix Central Lower	0.562	88	28	44.54	22.3

Hydrated Lime Requirement Tests

For each sample tested, 10 kg of material was crushed to minus 1.7 mm and mixed with water and hydrated lime in a 25 L carboy. The carboy was rolled for 72 hours and the pH checked at frequent intervals; lime was added to maintain the pH above 9. The lime consumptions were:

- Fenix South Upper, 7.5 kg/t.
- Fenix South Lower, 5.5 kg/t.
- 50/50 Fenix North Upper/Fenix Central Upper, 5.5 kg/t.
- 50/50 Fenix North Lower/Fenix Central Lower, 5.5 kg/t.

Water Soluble Magnesium

For each sample tested, 500 g was crushed to minus 1.7 mm, placed in 3.5 L bottle with 500 ml of tap water and rolled for 24 hours. A 50 ml sample of clear solution was assayed for magnesium. The percentages of water soluble magnesium were:

- Fenix South Upper 5%.
- Fenix South Lower 7%.
- 50/50 Fenix North Upper/Fenix Central Upper 7%.
- 50/50 Fenix North Lower/Fenix Central Lower 5%.
- 20% Fenix South Upper, 20% Fenix South Lower, 20% Fenix North Upper, 20% Fenix North Lower, 20% Fenix Central Lower 6%.

Magnesium Soak Tests

Samples were crushed to a P80 of 75 mm; 20 kg of each sample was soaked for 1 hour and washed in a 40 L drum. Water at an initial pH of 10.5 was circulated with a pump for 120 hours. Lime was added as required to maintain the pH. The total lime additions were:

- Fenix South Upper, (2 tests), 0.295 kg/t and 0.315 kg/t.
- Fenix South Lower, (2 tests), 0.296 kg/t and 0.302 kg/t.

13.5.4 Column Leach Test Work

Column leach tests were conducted utilizing material crushed to 100% passing 150 mm or 75 mm. During testing, the material was leached for 123 days with a sodium cyanide solution. The leach parameters are shown in Table 13-8.

Table 13-8: Column Leach Test Parameters

Sample	Crush Size mm	Calc.p80 Size mm	Column Diameter m	Initial Charge Height m	Charge Weight Kg
Fenix South Upper	150	99.5	0.381	3.997	561.40
Fenix South Lower	150	95.8	0.381	4.270	604.04
50% Fenix North Upper; 50% Fenix Central Upper	150	91.5	0.381	3.458	512.50
50% Fenix North Lower; 50% Fenix Central Lower	150	95.7	0.381	3.708	556.48
20% Fenix South Upper, 20% Fenix South Lower, 20% Fenix North Upper, 20% Fenix North Lower, 20% Fenix Central Lower (“20% Blended Sample”)	75	50.1	0.298	3.918	356.28

Gold extractions ranged from 53% to 77% based on calculated head grades, which ranged from 0.383 g/t to 0.898 g/t. The sodium cyanide consumptions ranged from 0.48 kg/t to 1.39 kg/t. The material utilized in leaching was blended with 5.56 kg/t to 7.98 kg/t hydrated lime. The metal extractions and chemical consumptions are shown in Table 13-9 and Table 13-10.

Table 13-9: Summary of Gold Extraction and Chemical Consumption

Description	Calculated Head Au g/t	Extracted Au g/t	Calculated Tail p80 Size mm	Consumption NaCN kg/t	Addition Hydrated Lime kg/t
Fenix South Upper	0.388	0.223	99.5	0.48	7.98
Fenix South Lower	0.456	0.224	95.8	0.97	5.56
50% Fenix North Upper; 50% Fenix Central Upper	0.383	0.298	91.5	1.39	6.56
50% Fenix North Lower; 50% Fenix Central Lower	0.898	0.694	95.7	1.01	6.04
20% Blended Sample	0.609	0.455	50.1	0.85	6.06

Table 13-10: Summary of Silver Extraction and Chemical Consumption

Description	Calculated Head Au g/t	Extracted Au g/t	Calculated Tail p80 Size mm	Consumption NaCN kg/t	Addition Hydrated Lime kg/t
Fenix South Upper	0.32	0.1	99.5	0.48	7.98
Fenix South Lower	0.34	0.07	95.8	0.97	5.56
50% Fenix North Upper; 50% Fenix Central Upper	0.22	0.02	91.5	1.39	6.56
50% Fenix North Lower; 50% Fenix Central Lower	0.33	0.04	95.7	1.01	6.04
20% Blended Sample	0.31	0.08	50.1	0.85	6.06

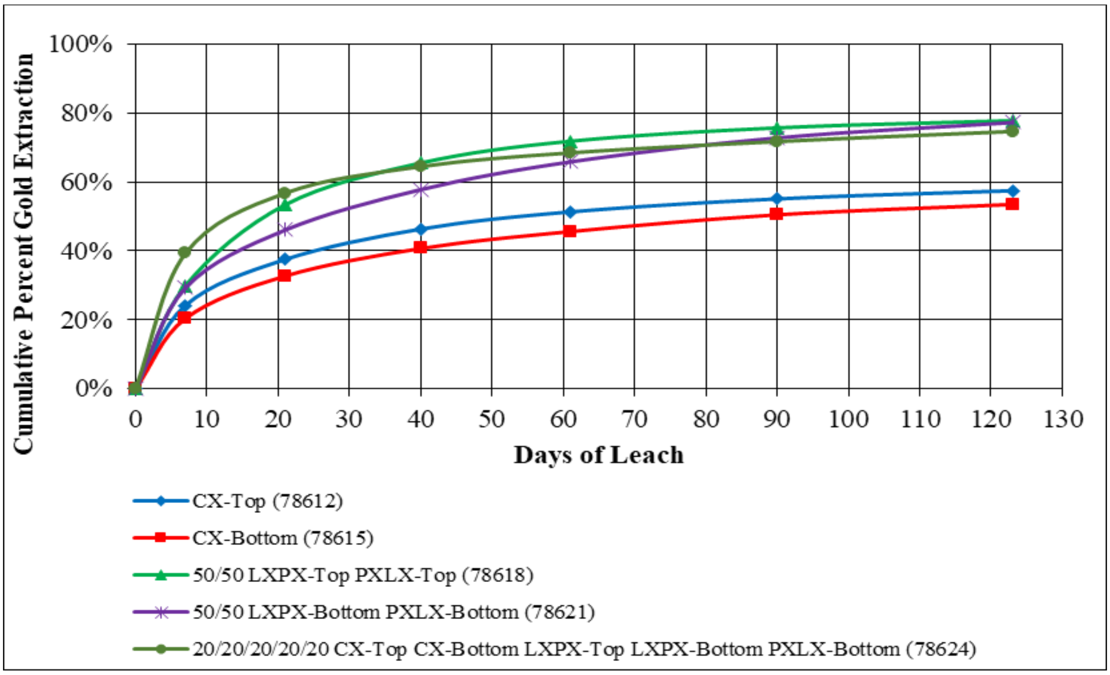


Figure 13-8: Gold Extraction versus Days of Leach

In all cases, the gold extraction reached close to its maximum after 123 days of leaching. The significantly lower gold extractions obtained from the Fenix South Upper and Fenix South Lower material were not evident in the previous test work. A notable difference is that copper extraction appears to be higher than has been reported previously where extractions were around 1.5%. The shake test results in Table 13-7 and the copper solution assays for the column tests in Table 13-11 shows that copper extraction from the Fenix South Upper and Fenix South Lower samples is higher than from the other samples.

A possibility is that chalcocite or other cyanide soluble copper minerals were present in the Fenix South samples, however, no mineralogical analysis was reported for this test work campaign to confirm this.

Table 13-11: Copper Concentration in Column Leach Solutions

Description	Low Copper mg/L	High Copper mg/L
Fenix South Upper	0.58	4.41
Fenix South Lower	2.28	16.50
50% Fenix North Upper; 50% Fenix Central Upper	0.49	2.58
50% Fenix North Lower; 50% Fenix Central Lower	0.56	6.58
20% Blended Sample	2.46	14.80

13.6 Summary of Column Test Results (2010 to 2017)

Results of column leach testing results between 2010 and 2017 are summarised in Table 13-12.

Table 13-12: Column Leach Tests from 2010 to 2017

Year	Report	F80 / F100 (mm)	Calculated Head Au g/t	Gold Recovery %
2010	2010 Maricunga Project Composite #1	19	1.13	89
		19	0.76	79
		9.5	0.79	80
2011	2011MAR02_CT_01	19	0.31	80
		19	0.31	82
		19	0.53	86
		19	0.5	84
		100	0.58	77
		50	0.54	78
		19	0.58	80
		19	0.22	78
	2011 PLENGE	19	0.87	76.4
		19	1.04	83
		19	0.87	82.2
2013	2013 KCA0120_03MAR03_6	25	0.476	80
		25	0.241	80
		25	0.487	82
		25	0.244	80
		25	0.468	80
		25	0.238	82
		25	0.491	85
		25	0.452	80
		25	0.502	80
		25	0.495	78
		25	0.534	84
		25	0.531	82
		25	0.342	80
		25	0.349	85
		25	0.502	81
		100	0.376	76
		100	0.403	70
2014	2014 KCA0130184_MAR06_02	100	0.329	72
		19	0.304	71
		9.5	0.44	84
		9.5	0.57	85
		9.5	0.603	85
		9.5	0.459	84
2017	2017 KCA0170013_MAR07_03	100	0.388	57
		100	0.456	53
		100	0.383	78
		100	0.898	77
		50	0.609	75

The gold recoveries are all in the range 70% to 89% except for the recoveries for the Fenix South Upper and Fenix South Lower samples discussed in section 13.5.4 and shown in Table 13-9. Figure 13-9 shows the gold and silver recoveries by P80 crush size.

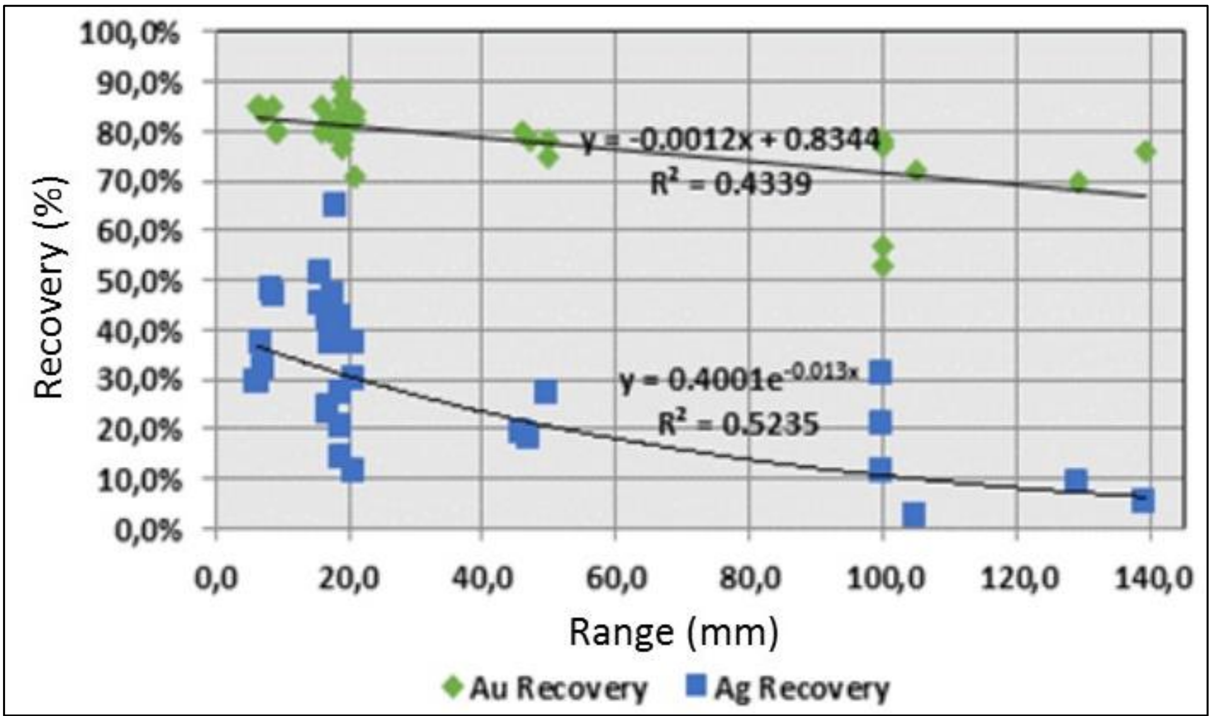


Figure 13-9: Gold and Silver Extraction versus P80 Crush Size

Figure 13-9 shows there is a weak relationship between metal recovery and P80 size. Over the P80 range 20 mm to 140 mm gold recovery falls approximately 7%.

There is almost no correlation between gold recovery and gold head grade.

13.7 Conclusions

- The results show an average gold recovery of 75% can be obtained in column leaching with material crushed to a P100 of 150 mm with a P80 of 4”. The range of gold recoveries is 72% to 78%. The QP confirms that the test results support an average gold recovery of 75%. Silver recovery is approximately 10%. Recoveries 2% to 3% lower would be expected in an industrial scale heap leach with ore crushed to this size range.
- The gold recoveries obtained from the Fenix South Upper and Fenix South Lower samples in the KCA 2017 work gave gold recoveries of 57% and 53% respectively. Previous test work showed no refractory behaviour with this material. However, the Fenix South samples for the 2017 work showed a higher level of cyanide soluble copper compared to samples from other zones.

- The 20% blended sample achieved 75% recovery. On a simple weighted average basis from the individual sample results, allowing for the finer crush size (P80 2”), a recovery of about 70% would have been expected. This indicates that blending the Fenix South material has provided a recovery improvement.
- The sodium cyanide consumption at a P80 4” crush size was in the range 0.5 kg/t to 1.4 kg/t; experience shows that industrial scale consumption is approximately 30% of laboratory consumption. A consumption of 0.4 kg/t is recommended for the operating cost estimate.
- The lime consumption at a P80 crush size was in the range 6 kg/t to 8 kg/t and when benchmarked to industrial scale operations, this would be reduced to 2 kg/t. The high lime consumption is likely due to the presence of water soluble magnesium and the formation of magnesium sulphate hexahydrate. The consumption of 4 kg/t is recommended for an operating cost estimate.

13.8 Recommendations

In order to gain more specific information regarding the metallurgical characteristics of the deposit the following future works are recommended:

- Model Magnesium (Mg) distribution to understand lime consumption.
- Undertake a column leach test campaign on mineral crushed to a P80 size of 4” from Fenix North, Fenix Central and Fenix South to optimize gold recovery and reagent consumption. The estimated cost of the campaign for metallurgical testing is $ 150,000.
    - Two tests per zone.
    - A test containing a blend of the material from the three zones to confirm previous results.
- Mineralogical analyses should be carried out on the head samples at the start of the tests and the residues at the end of the tests.
- It is recommended to carry out the mineralogical analysis on the remaining head samples from the KCA 2017 Fenix South leach tests to determine if chalcocite or other cyanide soluble copper minerals are present or if there are other causes for the higher refractory behaviour.
- For tests with the Fenix South material, copper extraction should be measured at the same frequency as the gold extraction to determine if there is any correlation between the two.
- During the production scale pilot tests and future column tests quantify the as mined moisture content as a percentage of ROM and 4” crushed material. During these tests measure and capture the saturation percentage required for solution to percolate through the mineral, which will help confirm the water requirement for “wetting”

mineral, also conduct tests on leached material to capture the residual moisture percentage retained in the mineral.

- Undertake evaporation measurements in the Pad location to confirm the evaporation rate that should be applied to the Leach Pad Water Balance.

14 MINERAL RESOURCE ESTIMATES

14.1 Modelling Procedure

14.1.1 Data Used

The resource model was created using the following data:

- Surface maps containing lithological units, structures and trenches with assays.
- Geological descriptions (logging) of 91 diamond drill holes totalling 30,533 m of core.
- Lithological descriptions (quick logging) of 291 reverse circulation holes totalling 84,101 m of RC cuttings.
- Assay data from 56,307 two meters samples of drill core and RC chips.
- Lithological descriptions of 5 trenches from the 2013 campaign (266 m) and 12 trenches from the 2018-2019 campaign (1,458 m). Totals are 17 trenches with 1,724 m of logging.
- Assay data from 131 two meter trench samples (2013 campaign), and 729 two-meter trench samples (2018-2019 campaign). Totals are 860 two-meter trench samples.

Andres Beluzan (QP) notes that the data used in the 2019 Mineral Resource Estimate (MRE) includes 39 RC drill holes drilled since the 2014 MRE. Additional trench samples determined by Andres Beluzan to be of good quality have also been included.

Andres Beluzan (QP) has reviewed the QA/QC procedures and performance for these samples and consider them acceptable.

As discussed in Section 12, another difference from the prior model is the change in the main laboratory used for Au assays, which for the 2018-2019 campaign is ALS; in addition, the samples were also analysed using ICP, and mercury was analysed using cold vapour method.

14.1.2 Geological Interpretation

The Fenix 2019 Resource Model is the first model for this project to make use and take advantage of a geologic interpretation and modelling. In the past, only grade shells were used to constrain grade interpolation. Now a combination of lithological modelling with consideration of prevalent main structures and main orientations, in addition to grade shells, were used to define estimation domains.

The estimation domains are thus based on lithological, structural and grade controls. Gold mineralization appears mainly within black banded veins that are hosted within the complex breccia, hydrothermal breccias and in contact with dacite domes.

The main structures mapped in the zone are trending NW and these structures are controlling most of the mineralization in the three zones. East-west tensional faults are controlling locally the emplacement of the black banded veins and Au mineralization. The NE trending faults are late and are segmenting the deposit into three zones, the Fenix North, Fenix Central (A and B), and Fenix South.

The gold estimation domains were finalized using a 0.15 g/t Au cut-off and within the lithological/mineralized domains described above. A total of 48 cross sections (SW-NE orientation, on 50 meters spacing and using an influence of +/- 25m) were interpreted, and the resulting three-dimensional model was obtained using the Leapfrog© software package. Figure 14-1 shows the geological map and estimation domains.

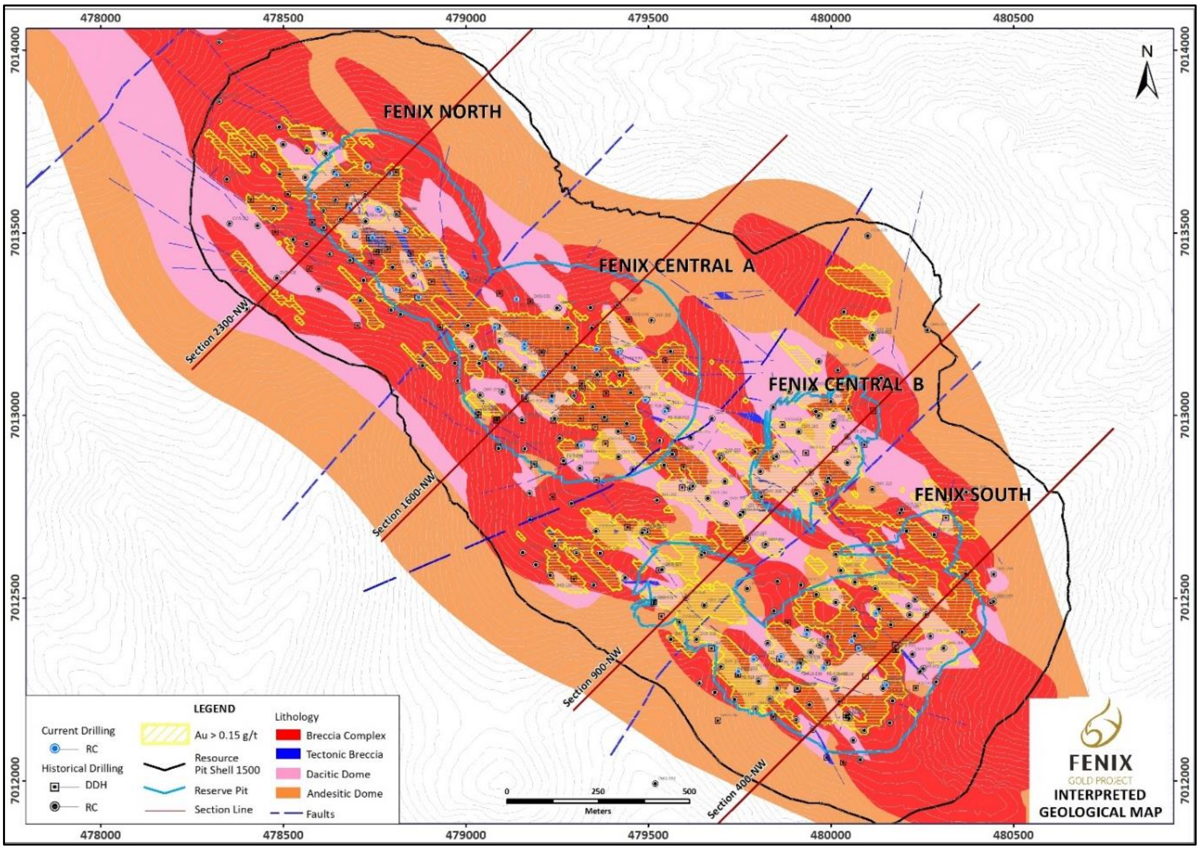


Figure 14-1: Geological Map and Estimation Domains with resource pit outline

14.1.3 Definition of estimation domains

The five estimation domains defined for the Project are based on major fault systems, which have displaced the mineralized east-west structures. All domains are grade shells using a 0.15 g/t Au cut-off, and are:

- Fenix North (FN).
- Fenix Central A (FCA).

- Fenix Central B (FCB).
- Fenix South (FS).
- Host Rock (HR).

Figure 14-2 is a three-dimensional representation of the Fenix model and domains.

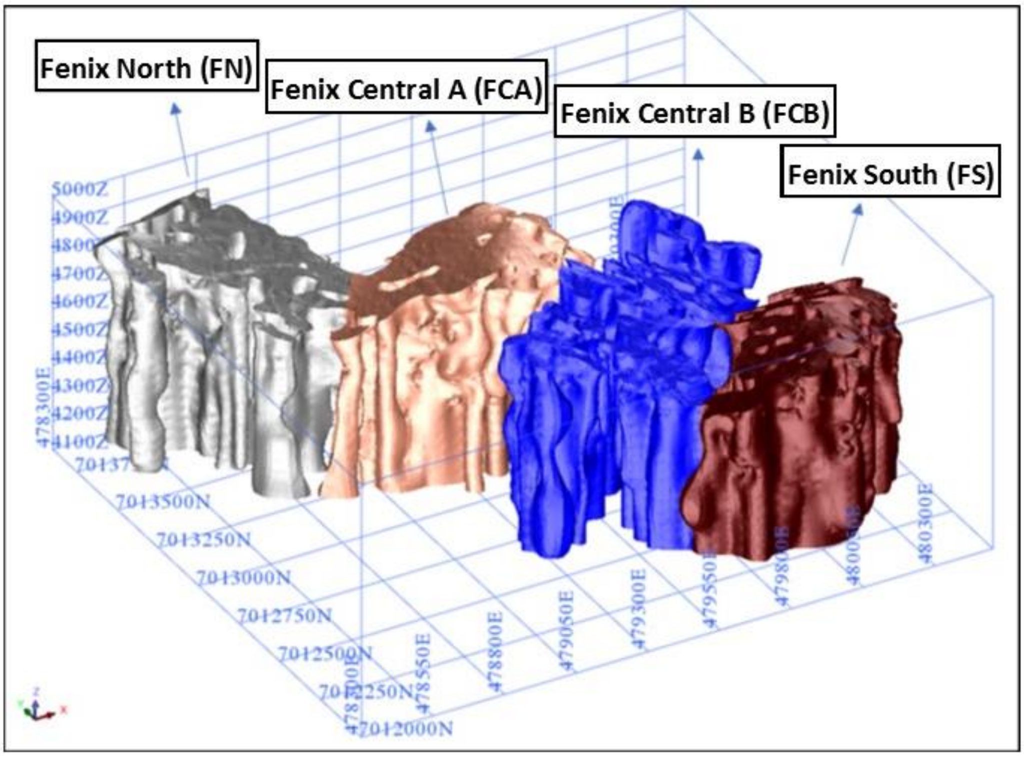


Figure 14-2: 3-D View of Fenix Mineralized Zones (HR encompasses the domains shown)

14.2 Database

The drill hole database used for the Fenix Gold Project Mineral Resource estimate contains 91 diamond drill holes (30,533 m) and 291 RC holes (84,101 m).

The drill hole database includes the data tables summarized in Table 14-1.

Table 14-1: Data Tables - Drill Hole Database

Table	Variable			
Collar	X - Easting	Y - Northing	Z - Elevation	
Survey	From (m)	To (m)	Azimuth (°)	Dip (°)
Assay	From (m)	To (m)	Au (g/t)	
In Situ Density	From (m)	To (m)	SG (g/cc)	

Table 14-17 lists the drill hole collars contained in the drill hole database.

14.2.1 Other Elements

Other potentially deleterious elements of interest are Arsenic (As), Mercury (Hg), and Magnesium (Mg).

Early drill phases did not routinely include ICP analysis of a wide range of elements. Thirty-nine drill holes completed during the 2018-2019 campaign (Phase 6) have been analysed for a suite of 36 elements, including As, Hg, and Mg.

Basic statistics on As and Hg values show that these elements are not likely to be consequential for the Fenix Gold Project. Specifically:

- There are 1,063 samples available for Hg. The average Hg grade is 0.012 ppm; the largest Hg value found is 0.359 ppm.
- There are 3,528 samples with As values. The average As grade is 7.67 ppm, while the maximum As value sampled to date is 212 ppm.

In the opinion of Andres Beluzan (QP), no As or Hg values reach levels that could be potentially considered deleterious and of interest and thus no model for these elements is deemed necessary at this point.

With respect to Mg, 3,528 samples show that overall average grades are around 1%, which are more or less constant when the same samples are analysed by lithology. The element appears to be prevalent across the deposit, and be part of the rock-forming minerals, at a more-or-less relatively uniform concentration, Figure 14-3. At this stage of the project, it is not deemed necessary to build a Mg model, but this will be considered at the next stage of the Project.

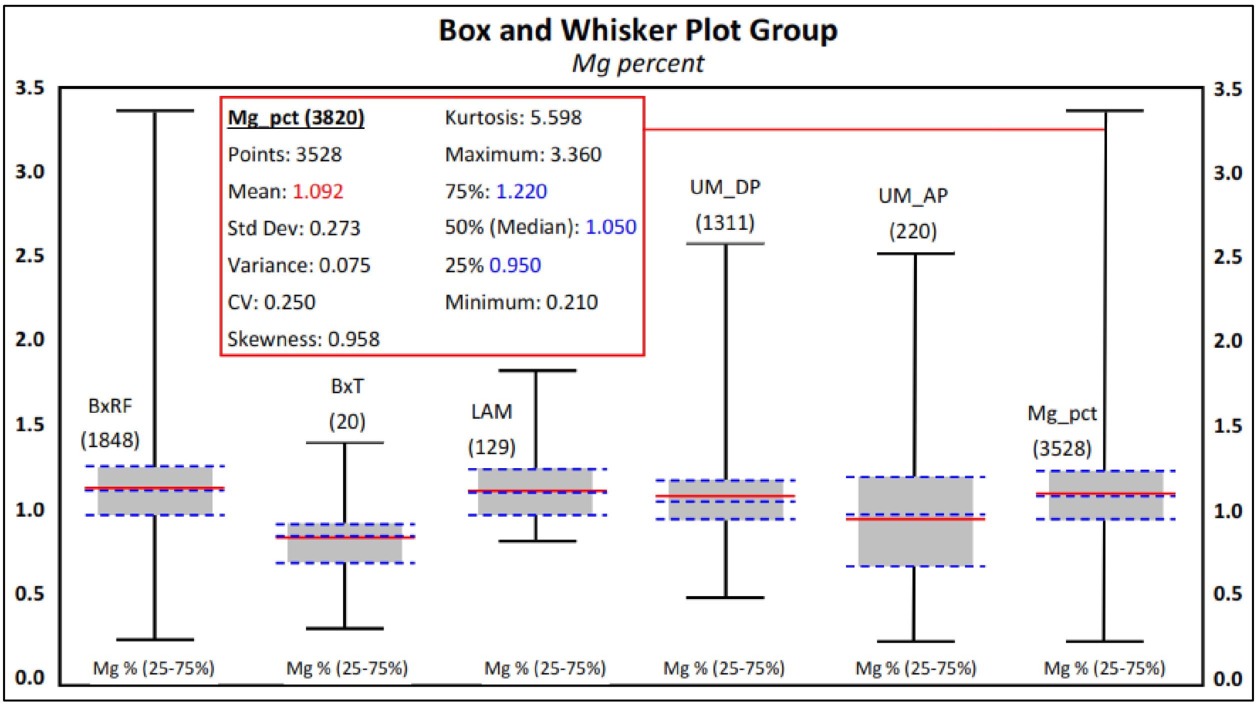


Figure 14-3: Plot of Mg by lithology, 2m composites

14.3 Compositing, Statistics and Outliers

Statistical analyses were performed for Au samples and included reviews of the number of samples, total length, minimum, maximum mean value, standard deviation, and CV.

Since sampling was carried out consistently at 2.0 m intervals, coordinates were assigned to the centre of individual samples via a function available in the Vulcan mining software, which preserved the original sample length.

There are 65 samples (out of 56,307) in which the sample length differed from 2.0 m.

Three trench campaigns were completed and classified by drilled year. The first one called, historical campaign, was completed from 2010 to 2011. There was no QA/QC performed for these data, so it will not be considered in resource estimation. The second one was completed by Atacama in 2013 and the third one by Rio2 in 2018-2019. For these last two, QA/QC procedures were in place. After checking the quality of the data and comparing it to the drill hole data, it was decided that these last two data sets would be used for resource estimation.

Table 14-2 shows basic sample statistics for each mineralized envelope and for the samples lying outside the mineralized envelopes (Host Rock). A set of histograms, cumulative probability plots, drift analysis box plots, and contact plots by domains were obtained in order to validate the domains and define outliers handling and other grade estimation strategies.

Table 14-2: Basic Sample Statistics by Domain

Domains	Data Type	Total Data	Min	Max	Mean	Desv Est	Coef Var
Host Rock	Drill hole	31,282	0.002	2.93	0.07	0.07	0.96
	Trench Fenix Gold	75	0.025	0.37	0.12	0.08	0.67
	Trench Atacama	25	0.03	0.28	0.1	0.05	0.52
Total		**31,382**	**0.002**	**2.93**	**0.07**	**0.07**	**0.96**
Fenix North	Drill hole	5,033	0.007	6.94	0.45	0.48	1.06
	Trench Fenix Gold	121	0.091	2.49	0.54	0.41	0.75
Total		**5,154**	**0.007**	**6.94**	**0.45**	**0.48**	**1.05**
Fenix Central A	Drill hole	8,971	0.005	3.68	0.43	0.36	0.83
	Trench Fenix Gold	405	0.057	4.9	0.52	0.5	0.97
	Trench Atacama	22	0.123	1.66	0.61	0.49	0.81
Total		**9,398**	**0.005**	**4.9**	**0.44**	**0.37**	**0.84**
Fenix Central B	Drill hole	4,926	0.005	4.65	0.3	0.22	0.73
Total		**4,926**	**0.005**	**4.65**	**0.3**	**0.22**	**0.73**
Fenix South	Drill hole	6,102	0.005	5.18	0.44	0.42	0.95
	Trench Fenix Gold	128	0.05	2.56	0.54	0.52	0.97
	Trench Atacama	84	0.081	1.55	0.3	0.2	0.66
Total		**6,314**	**0.005**	**5.18**	**0.44**	**0.42**	**0.95**
Total all domains		**57,174**	**0.002**	**6.94**	**0.23**	**0.31**	**1.39**

14.3.1 Fenix North (FN)

The 2 m composite data population for gold of FN is 5,154, presents a mean of 0.45 g/t Au, a standard deviation of 0.476 and a coefficient of variation of 1.047. The minimum and maximum values are 0.005 g/t Au and 6.94 g/t Au, respectively.

Figure 14-4 shows the histogram of gold grades. The composites present a classical gold grade distribution shape, similar to a log normal distribution.

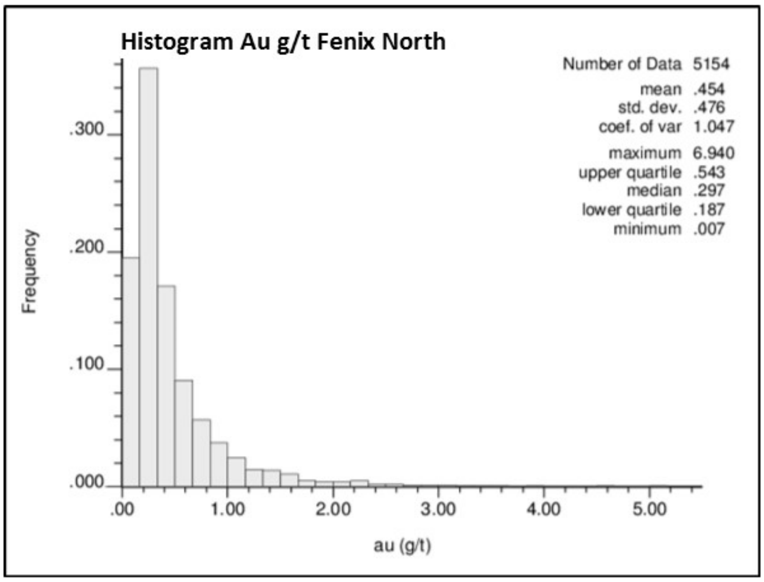


Figure 14-4: Histogram of gold composite grades of Fenix North

Figure 14-5 shows the gold probability plot of the composites. This curve shows that there is a population break for lower grades (below 0.15 g/t) with almost 8% of the total population and there are no outliers. This break is related to the use of a 0.15 g/t grade shell to define the domain, and indicates that there is still a significant portion of the distribution with grades below 0.15 g/t. This in turn indicates no need to add further dilution to the resource estimate.

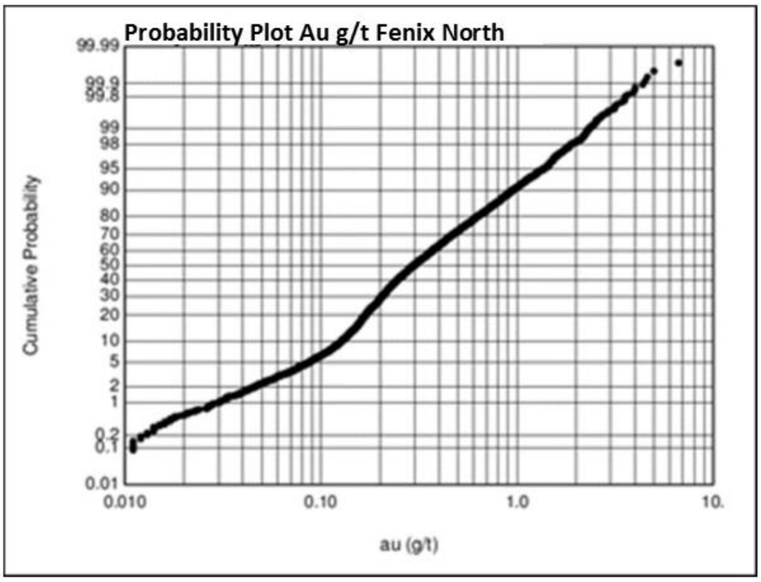


Figure 14-5: Probability Plot of Gold Composites, Fenix North

The trend analysis performed shows that gold grades do not show a tendency to rise or fall depending on the coordinates (Figure 14-6). The blue line on the graph represents the behaviour of grades along the North-South, East-West and Elevation, and grey dots represent the number of composites in each 20 m-wide swath.

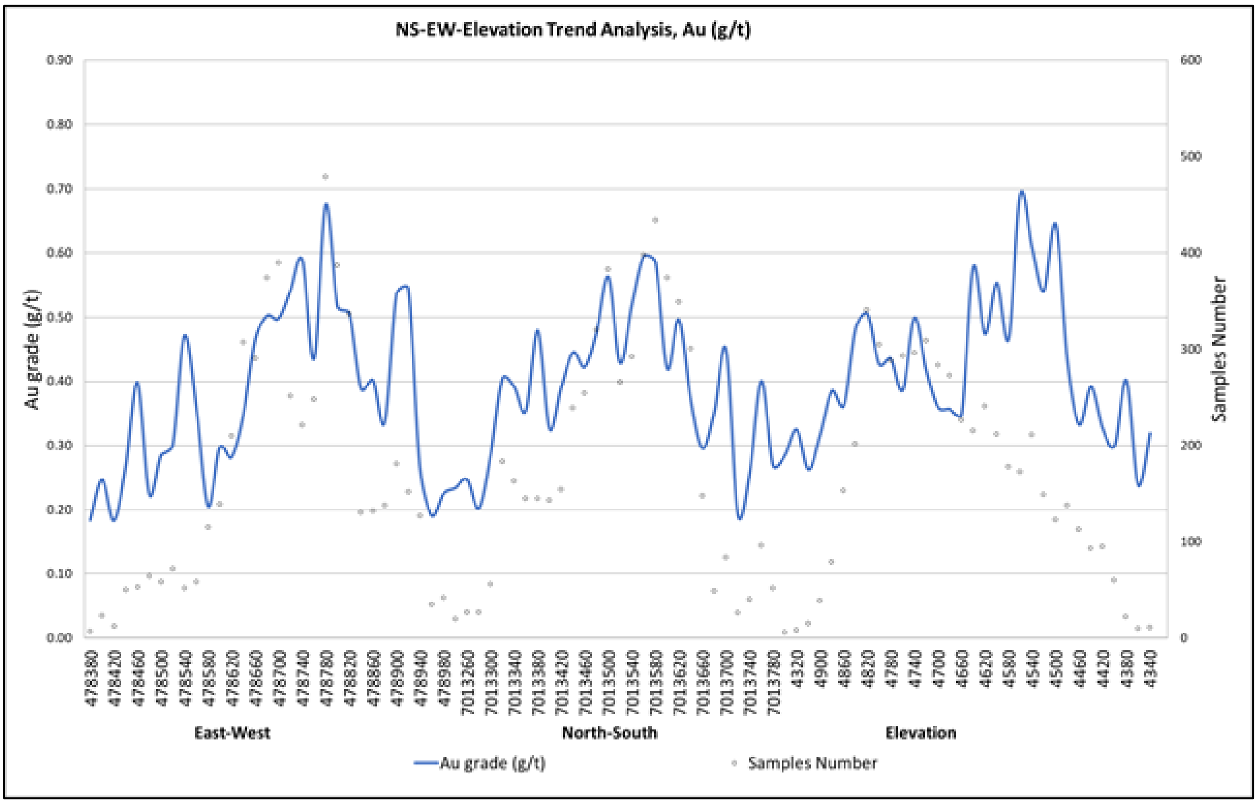


Figure 14-6: Drift Analysis, Fenix North

14.3.2 Fenix Central A (FCA)

The 2 m composites for gold of FCA is 9,398, presenting a mean of 0.44 g/t Au, a standard deviation of 0.368 and a coefficient of variation of 0.842. The minimum and maximum values are 0.005 g/t Au and 4.9 g/t Au respectively. Figure 14-7 shows the gold composites histogram. The composites present a classical gold grade distribution shape, similar to a log normal distribution.

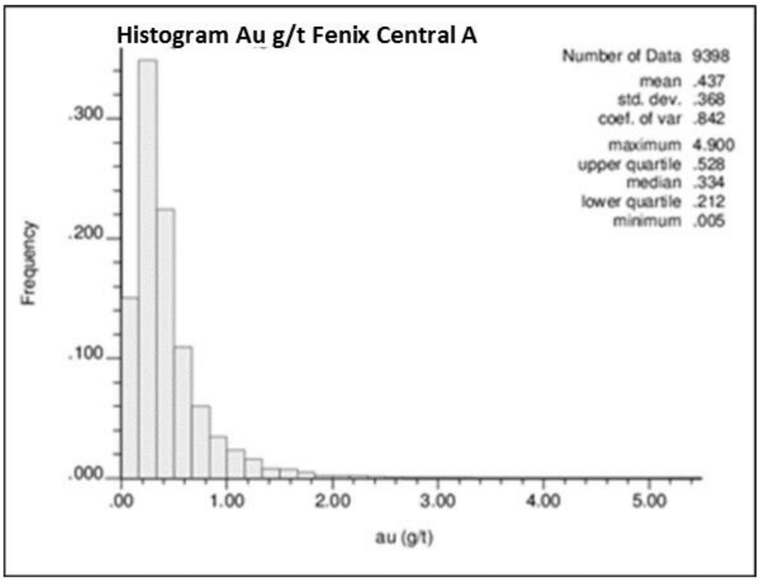


Figure 14-7: Composites gold histogram of Fenix Central A

Figure 14-8 shows the gold probability plot of the samples. This curve shows that there is a break for lower grades (below 0.15 g/t Au), comprising almost 10% of the total population, and there are no outlier values to consider. This break is related to the use of a 0.15 g/t grade shell to define the domain, and indicates that there is still a significant portion of the distribution with grades below 0.15 g/t. This in turn indicates no need to add further dilution to the resource estimate.

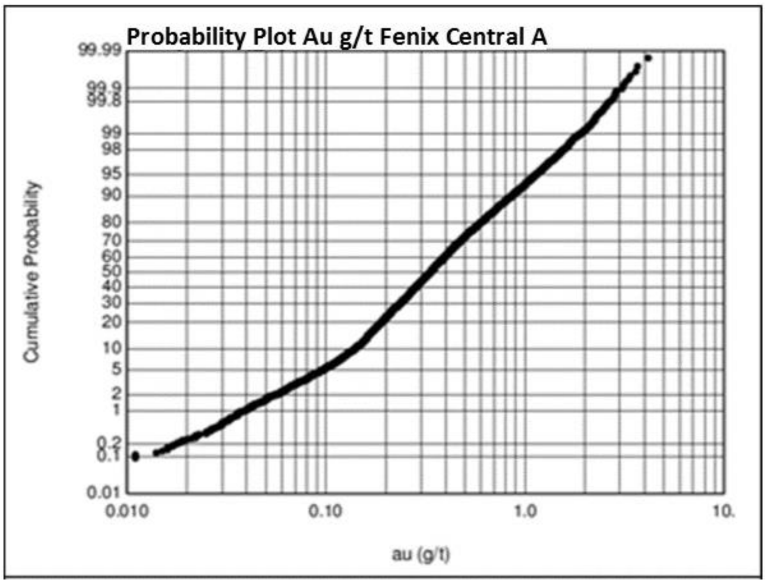


Figure 14-8: Composites gold probability plot of Fenix Central A

The trend analysis performed shows that gold grades do not show a tendency to rise or fall depending on North-South and East-West directions (Figure 14-9). Grades decrease below elevation 4,400 m.

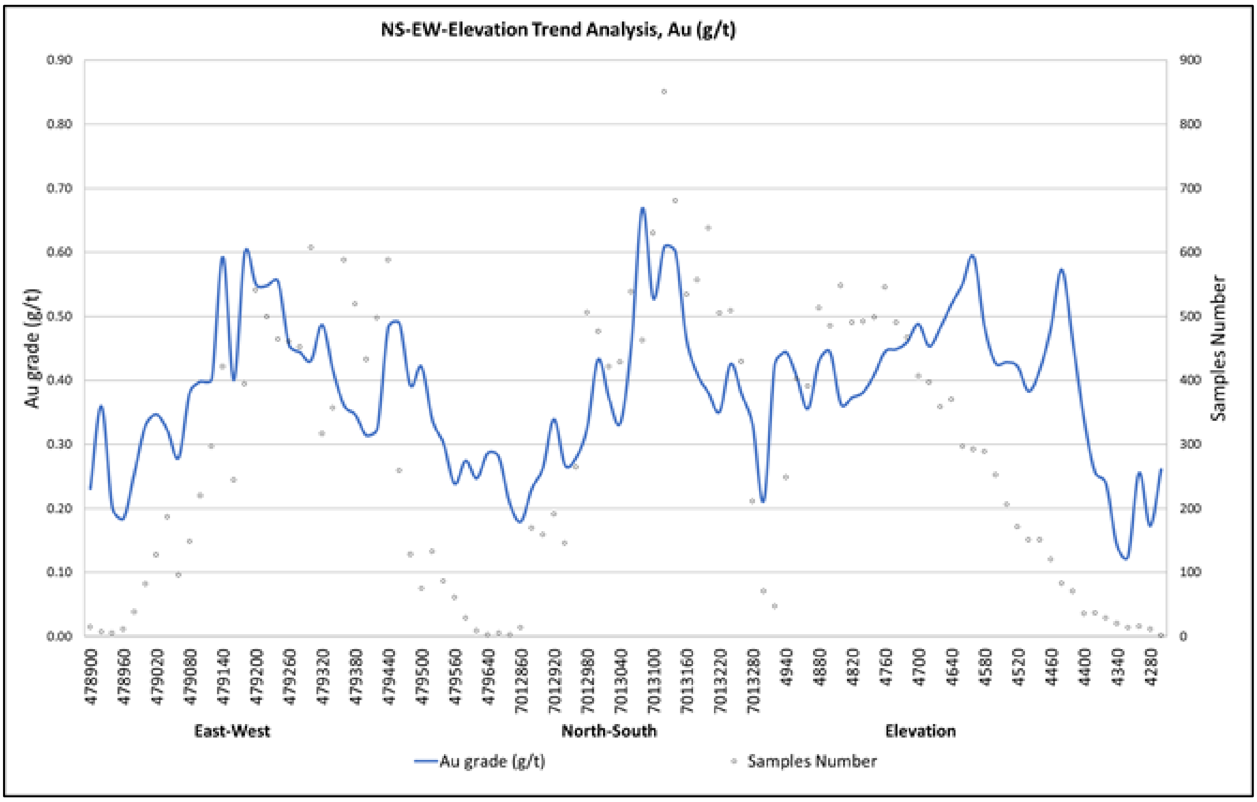


Figure 14-9: Drift Analysis, Fenix Central A

14.3.3 Fenix Central B (FCB)

The 2 m composite data population for gold of FCB is 4,926, presenting a mean of 0.295 g/t Au, a standard deviation of 0.217 and a coefficient of variation of 0.735. The minimum and maximum values are 0.005 g/t Au and 4.65 g/t Au respectively. Figure 14-10 to Figure 14-16 show gold composite histograms. The composites present a classical gold grade distribution shape, similar to a log normal distribution.

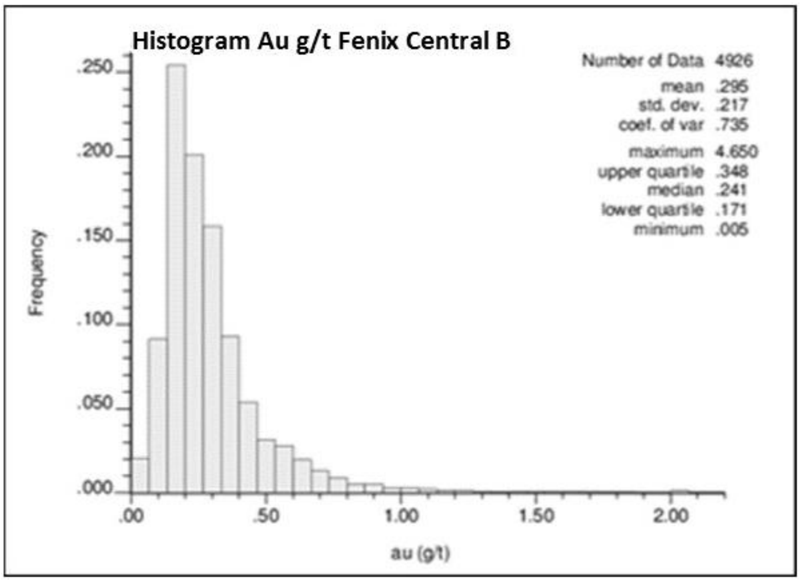


Figure 14-10: Composites gold histogram of Fenix Central B

Figure 14-11 shows the gold probability plot of the composites. This curve shows that there is a break for lower grades (approximately below 0.15 g/t Au), comprising almost 10% of the total sample population. This break is related to the use of a 0.15 g/t Au grade shell to define the domain, and indicates that there is still a significant portion of the distribution with grades below 0.15 g/t. This in turn indicates no need to add further dilution to the resource estimate.

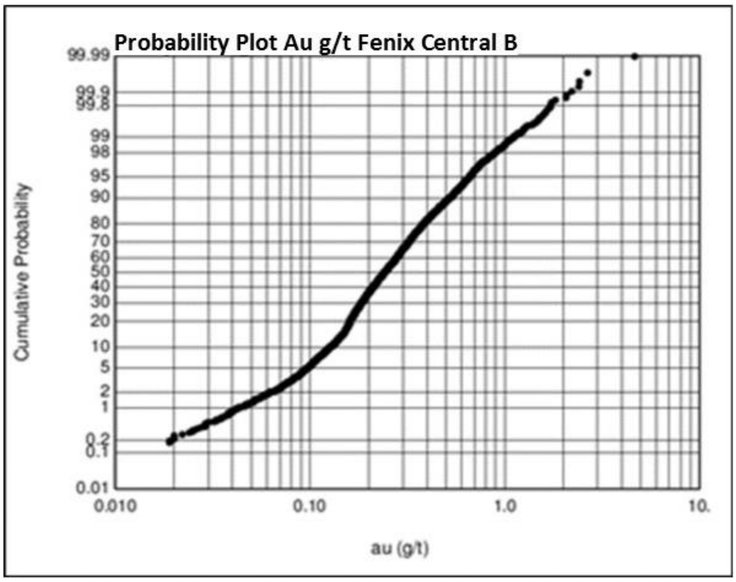


Figure 14-11: Composites gold probability plot of Fenix Central B

The trend analysis performed shows that gold grades do not show a tendency to rise or fall depending on the coordinates (Figure 14-12).

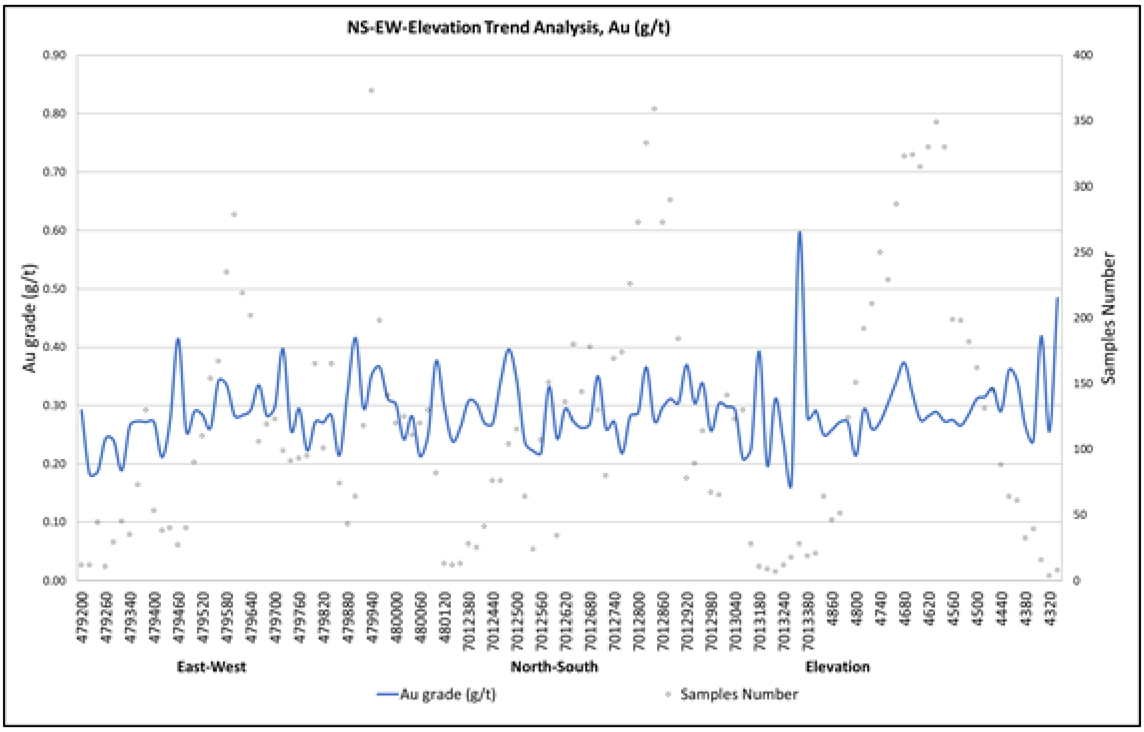


Figure 14-12: Drift Analysis, Fenix Central B

14.3.4 Fenix South (FS)

The 2 m composites data population for gold of FS is 6,314, presenting a mean of 0.439 g/t Au, a standard deviation of 0.418 and a coefficient of variation of 0.952. The minimum and maximum values are 0.005 g/t Au and 5.18 g/t Au respectively. Figure 14-13 shows the gold composites histogram. The samples present a classical gold grade distribution shape, similar to a log normal distribution.

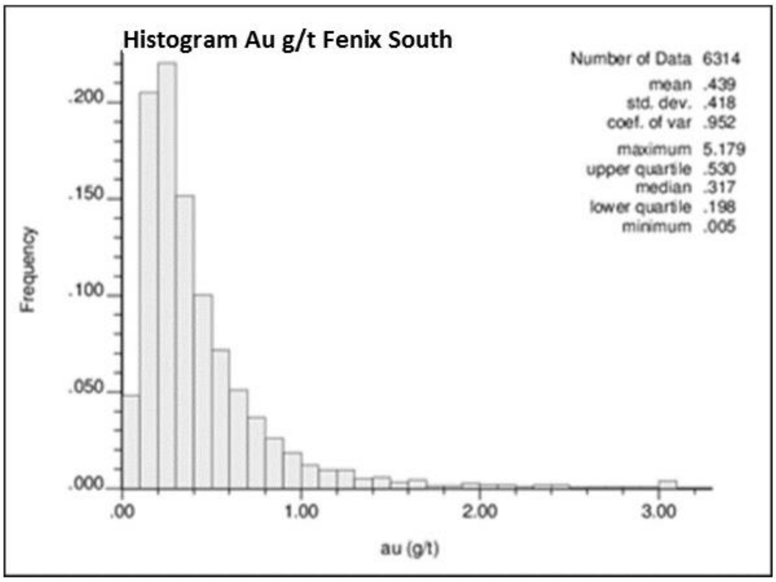


Figure 14-13: Composites gold histogram of Fenix South

Figure 14-14 shows the gold probability plot of the composites. This curve shows that there is a break for lower grades (below 0.15 g/t Au), comprising almost 10% of the total population. Again, this break is related to the use of a 0.15 g/t Au grade shell to define the domain, and indicates that there is still a significant portion of the distribution with grades below 0.15 g/t. This in turn indicates no need to add further dilution to the domain's database.

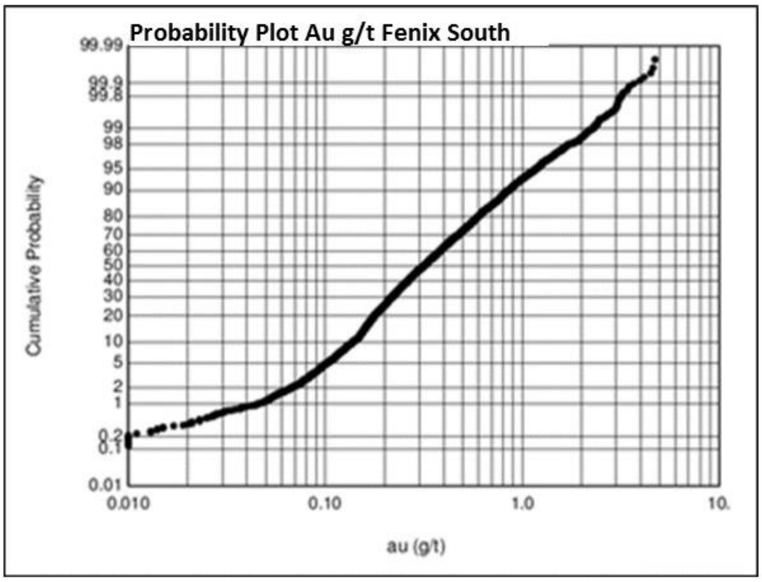


Figure 14-14: Composites gold probability plot of Fenix South

The trend analysis performed shows that gold grades do not show a tendency to rise or fall depending on the East-West or North-South coordinates (Figure 14-15). Gold grade decrease below 250 m from surface.

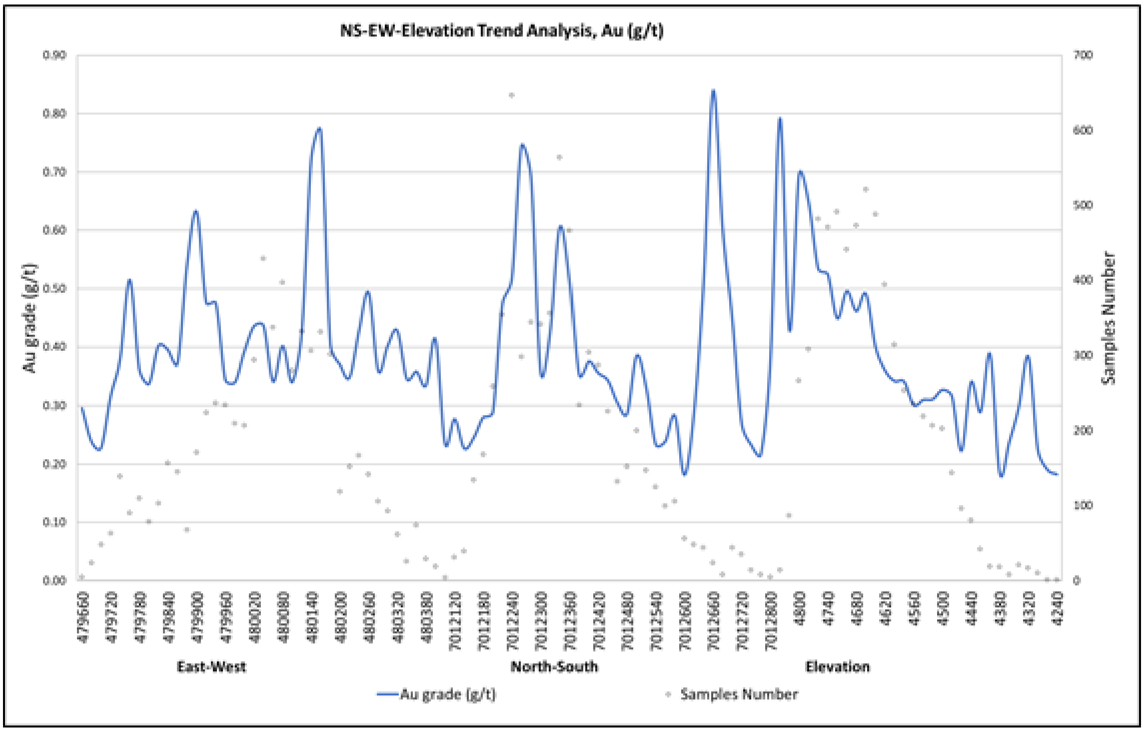


Figure 14-15: Drift Analysis, Fenix South

14.4 Contact Analysis

Hard estimation boundaries were used for all domains since there is a very abrupt change in gold grades between the mineralization lying inside the mineralized envelopes and the data lying outside. Figure 14-16 shows contact profiles between each mineralized envelope and the samples lying outside.

Contact profiles between Fenix North, Central A, Central B and South are not shown since there are no mineralized samples between these estimation domains due to faulting.

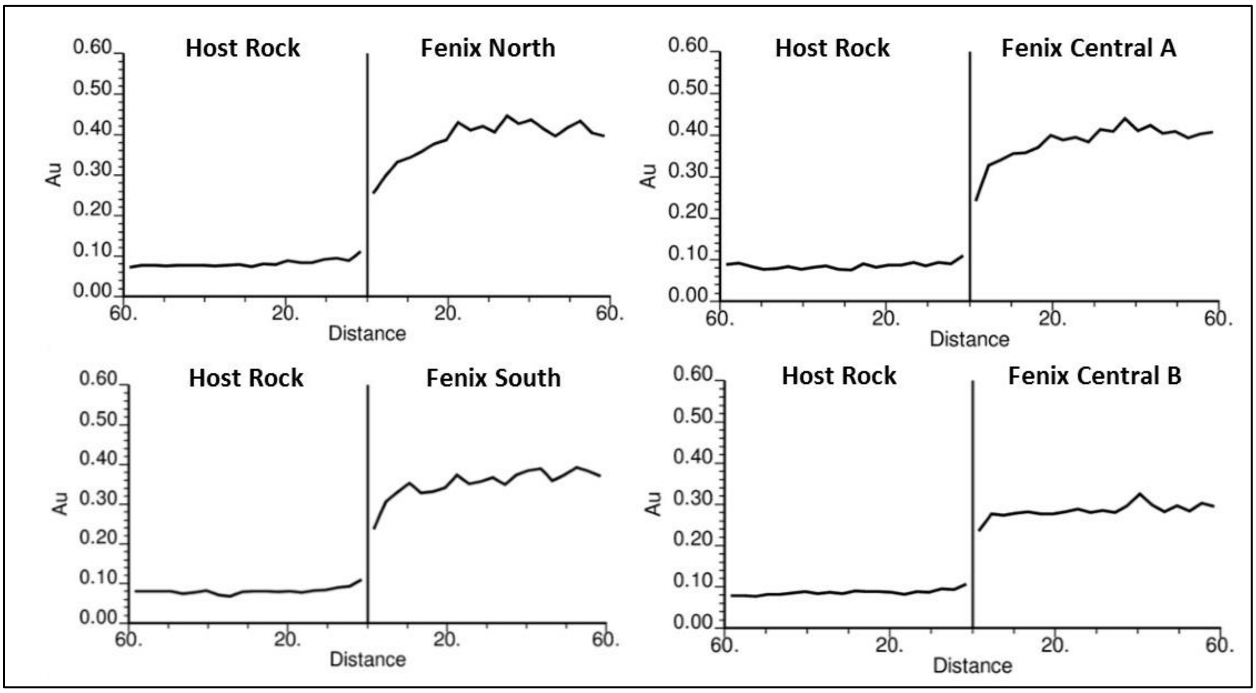


Figure 14-16: Contact Plots, Host Rock and Fenix North, Fenix Central A, Fenix Central Band, Fenix South

14.5 Variography

Experimental variograms were obtained for each domain. Anisotropic analysis shows a geometric anisotropy, meaning that the correlation is stronger in a specific direction. In geometrical anisotropy, the range changes but not the sill.

Snowden Supervisor© software was used to develop variographic analysis for gold grades. The rotation of the variogram model resulting from this analysis is consistent with the Vulcan 3D software format.

The variograms have been normalized by the variance of the data. For the calculation of the experimental variograms, a horizontal and vertical bandwidth of 30 m was used, with 15° angular tolerance and lags of 35 to 40 m. Downhole variograms were also obtained to help model the nugget effect for each domain.

Variogram maps (with color-coded variance) for each domain, were calculated in the three main planes and are shown in Figure 14-17 to Figure 14-20. Table 14-3 shows the final variogram models used for estimation.

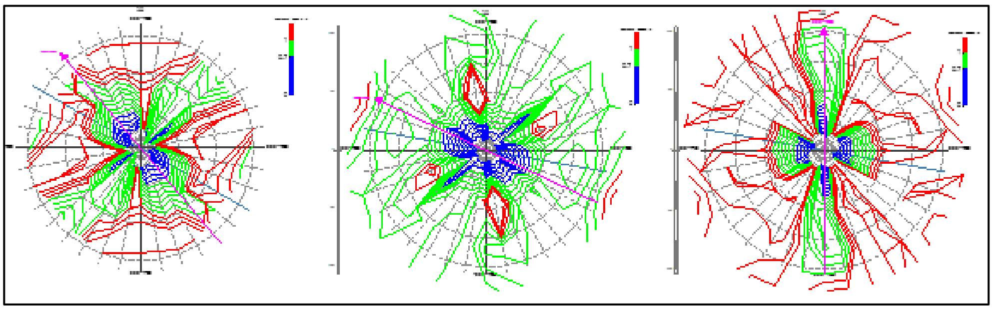


Figure 14-17: Variogram map, Fenix North

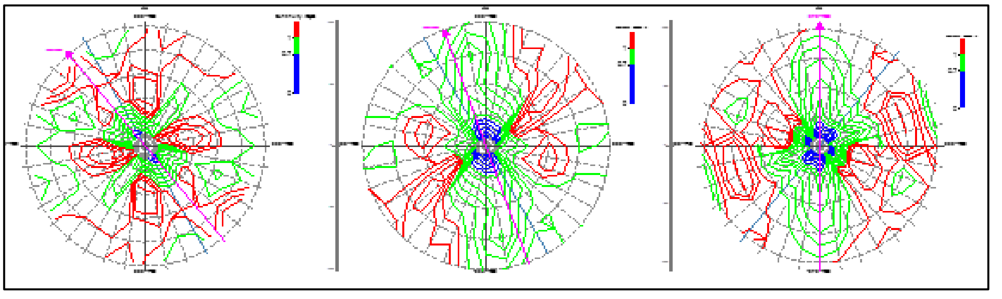


Figure 14-18: Variogram map, Fenix Central A

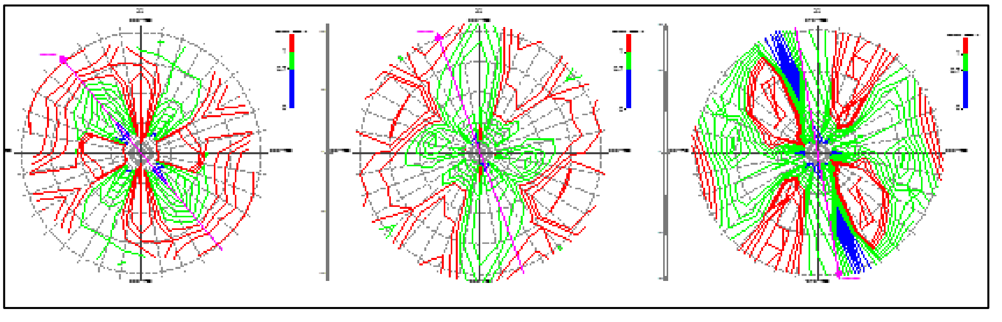


Figure 14-19: Variogram map, Fenix Central B

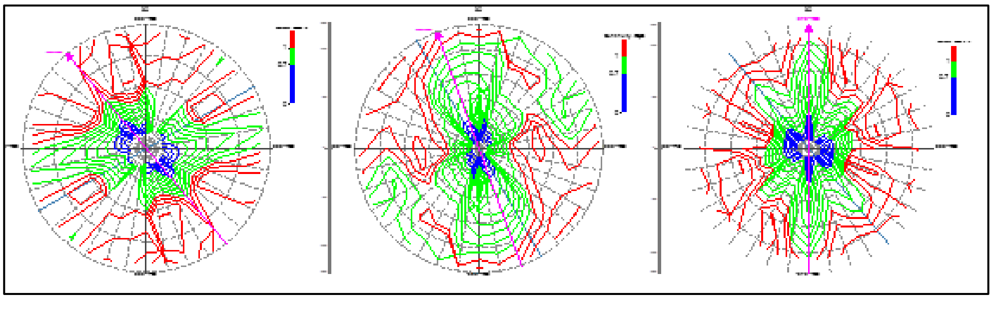


Figure 14-20: Variogram map, Fenix South

Table 14-3: Modelling and Plotting Parameters, Separately by domain

Domain	Nugget	Structure	Type	Sill	Search Angles			Range		
					Bearing	Plunge	Dip	Major	Semi	Minor
F North	0.17	1	Spherical	0.46	50	25	180	86	32	15
		2	Spherical	0.37	50	25	180	136	81	75
F Central A	0.17	1	Spherical	0.38	50	70	180	19	18	16
		2	Spherical	0.44	50	70	180	128	64	52
F Central B	0.22	1	Spherical	0.4	257.3	-67.7	-154.5	23	25	13
		2	Spherical	0.44	257.3	-67.7	-154.5	105	79	72
F South	0.22	1	Spherical	0.35	50	70	180	25	36	17
		2	Spherical	0.42	50	70	180	174	76	53

Figure 14-21 to Figure 14-24 show for each domain the directional as well as fitted models for the downhole variogram and for the three principal directions. Weak geometrical anisotropies are apparent for Fenix North, Central A, Central B and Fenix South.

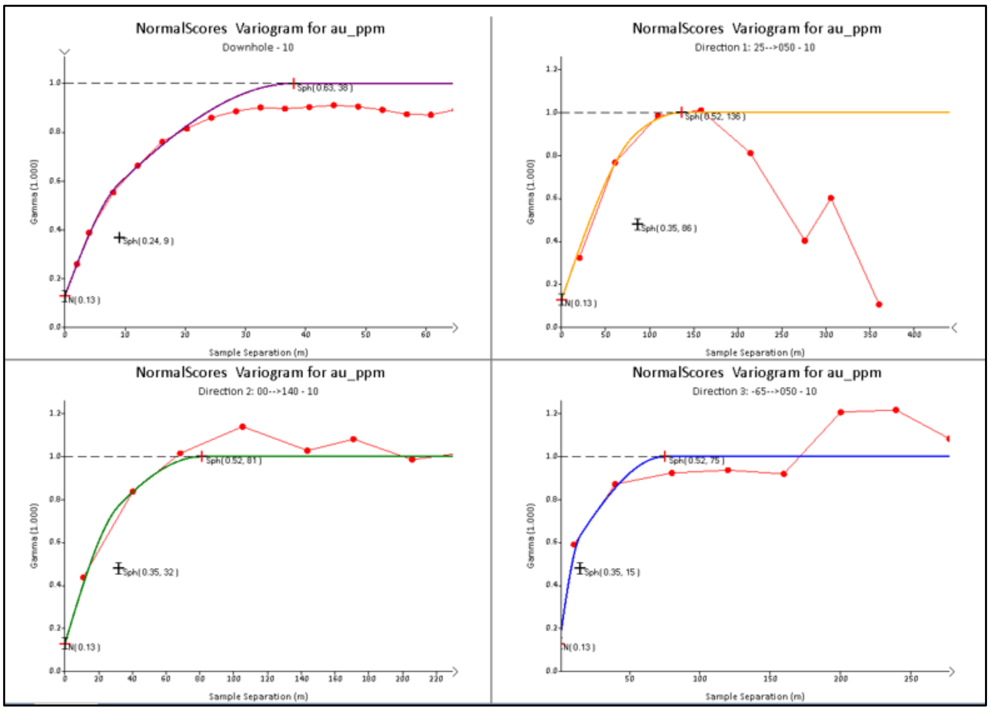


Figure 14-21: Directional Variogram Model, Fenix North

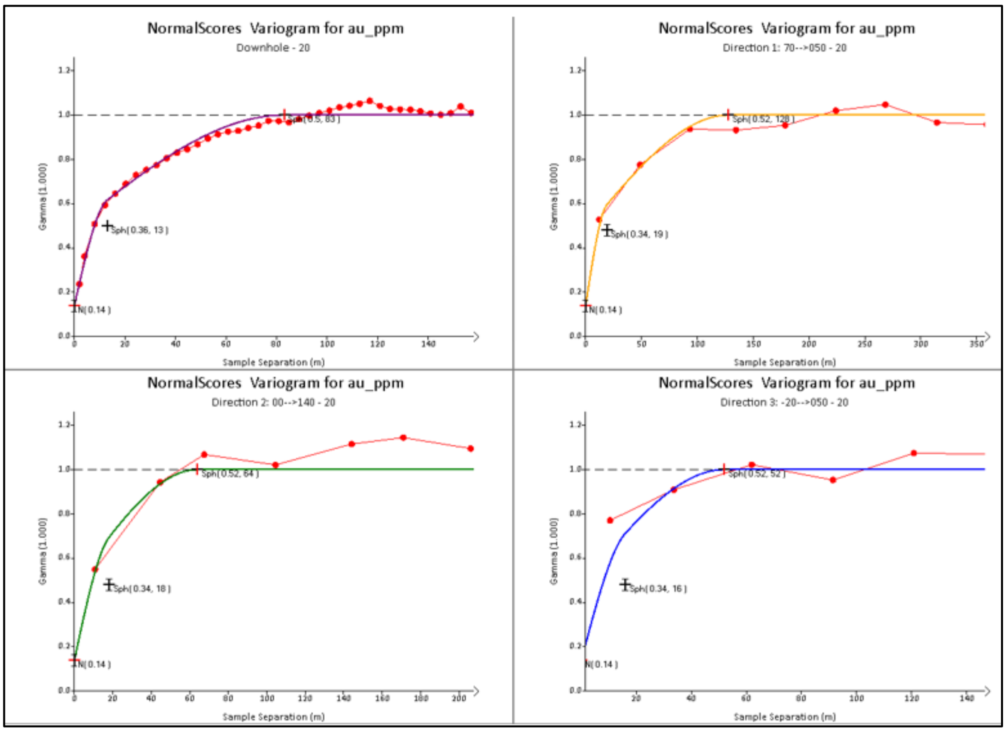


Figure 14-22: Directional Variogram Model, Fenix Central A

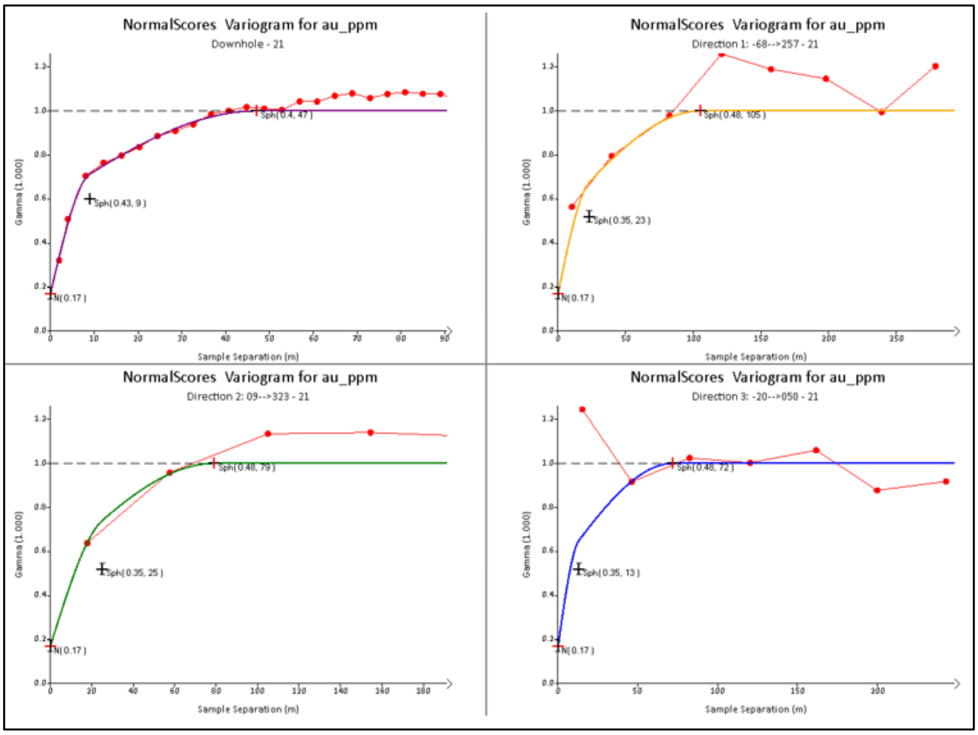


Figure 14-23: Directional Variogram Model, Fenix Central B

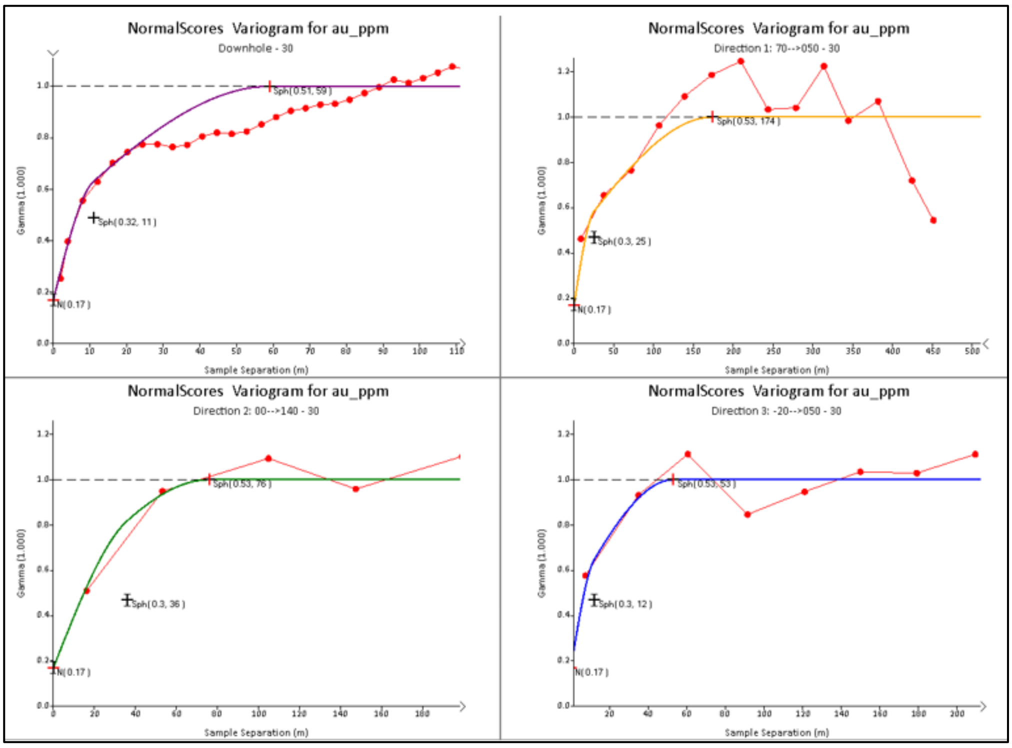


Figure 14-24: Directional Variogram Model, Fenix South

14.6 Block Model and Resource Estimation Plan

A 10m x 10m x 10m block model was created applying the volume model parameters presented in Table 14-4 and Table 14-5.

Table 14-4: Block Model Dimensions

Dimension	X	Y	Z
Origin	480000	7011600	5150
Model Size	2000	3000	1000
Block Size	10	10	10

Table 14-5: Block Model Orientation

Bearing	45°
Plunge	0°
Dip	0°

The most important variables of the model are given in Table 14-6.

Table 14-6: Model Variables

Variable	Description
Density	Block Density
Flag_au	Au Estimation Pass
Domain	0 = Host Rock 10 = Fenix North 20 = Fenix Central A 21 = Fenix Central B 30 = Fenix South
Au	Estimated Au grade using Ordinary Kriging
Cu	Estimated Cu grade using Ordinary Kriging
Au_NN	Estimated Au grade using Nearest Neighbour
Au_nmue	Number of Samples used in Au estimation
Au_dmmc	Cartesian distance to the nearest sample used in estimation
Au_dmmc_anis	Anisotropic distance to the nearest sample used in Ordinary Kriging estimation
Au_davg	Samples Average Distance, used in Ordinary Kriging estimation
Au_davg_anis	Samples anisotropic average distance used in Ordinary Kriging estimation
Kvar_Au	Au Kriging Variance
CBS Au	Au Conditional Bias Slope
Class	Resource classification category

The grade estimation plan for the Fenix Gold Project was carried out in three passes. General settings are detailed below:

- Ordinary Kriging interpolation for both Au and Cu grades.
- Only samples within the mineralized domains were used to estimate blocks, and only samples outside the mineralized domains were used to estimate host rock blocks.
- For the first and second passes the minimum number of samples used for estimation, are higher lower than the maximum allowed per hole. This is used to avoid than blocks are estimated with one drill hole.
- For the first pass, samples are shared between contiguous mineralized domains.
- The search radii for the third kriging pass were set large enough to avoid leaving too many blocks un-estimated within the mineralized envelope.
- Anisotropy rotation angles were used for search ellipsoids.
- A block discretization of 2 x 2 x 5 nodes was adopted for block kriging, this is considered reasonable for the block size and sample length used for resource estimation.

- The maximum composite value in the database is 6.94 g/t Au. High-grade restriction was used in all domains, using different grades and restricted in a 10 m search radius as shown in Table 14-7.
- High grade restrictions were also used for Copper, see the estimation parameters in Table 14-8.

The estimation parameters for Au are shown in Table 14-7, while for Cu are shown in Table 14-8.

Table 14-7: Au Estimation Parameters

Au Domain	Run	Type	Search Angles			Search Distances			Samples x Est.		Restricted Ratios for Threshold High Yield				Max Samples for Drill Hole
			Bearing	Plunge	Dip	Major	Semi	Minor	Min	Max	Au (ppm)	Major	Semi	minor	
Host Rock	1	OK	305	90	90	80	60	40	8	14	0.5	10	10	10	6
	2	OK	305	90	90	120	80	60	6	16	0.5	10	10	10	6
	3	OK	305	90	90	200	140	100	4	18	0.5	10	10	10	-
FN	1	OK	315	92	87	70	50	40	10	14	5	10	10	10	6
	2	OK	315	92	87	130	80	60	8	16	5	10	10	10	6
	3	OK	315	92	87	240	180	100	4	18	5	10	10	10	-
FCA	1	OK	300	90	88	65	45	35	10	14	3.7	10	10	10	6
	2	OK	300	90	88	130	70	50	8	16	3.7	10	10	10	6
	3	OK	300	90	88	240	150	120	4	18	3.7	10	10	10	-
FCB	1	OK	295	89	90	80	60	40	8	14	3	10	10	10	6
	2	OK	295	89	90	105	80	70	6	16	3	10	10	10	6
	3	OK	295	89	90	220	160	140	4	18	3	10	10	10	-
FS	1	OK	300	91	89	80	50	40	10	14	3.7	10	10	10	6
	2	OK	300	91	89	140	90	60	8	16	3.7	10	10	10	6
	3	OK	300	91	89	220	150	80	4	18	3.7	10	10	10	-

Table 14-8: Cu Estimation Parameters

Cu Domain	Run	Type	Search Angles			Search Distances			Samples x Est.		Restricted Ratios for Threshold High Yield				Max Samples for Drill Hole
			Bearing	Plunge	Dip	Major	Semi	Minor	Min	Max	Cu (ppm)	Major	Semi	minor	
Host Rock	1	OK	305	90	90	80	60	40	8	16	600	10	10	10	6
	2	OK	305	90	90	150	90	60	6	18	600	10	10	10	6
	3	OK	305	90	90	300	150	100	4	20	600	10	10	10	-
FN	1	OK	315	92	87	70	60	40	10	16	1110	10	10	10	6
	2	OK	315	92	87	140	90	60	8	18	1110	10	10	10	6
	3	OK	315	92	87	280	180	100	4	20	1110	10	10	10	-
FCA	1	OK	300	90	88	80	60	40	10	16	1250	10	10	10	6
	2	OK	300	90	88	120	90	60	8	18	1250	10	10	10	6
	3	OK	300	90	88	240	150	120	4	20	1250	10	10	10	-
FCB	1	OK	295	89	90	80	60	40	8	16	1200	10	10	10	6
	2	OK	295	89	90	150	120	80	6	18	1200	10	10	10	6
	3	OK	295	89	90	250	180	120	4	20	1200	10	10	10	-
FS	1	OK	300	91	89	80	60	40	10	16	700	10	10	10	6
	2	OK	300	91	89	150	100	80	8	18	700	10	10	10	6
	3	OK	300	91	89	240	150	100	4	20	700	10	10	10	-

14.6.1 In Situ Dry Bulk Density Estimation

The 2019 Resource Model uses the same dry bulk density estimated values from the prior 2014 Model. There are no additional samples with bulk density measurements taken since 2014 (the new drill holes are all RC), so for this model the same bulk density values (block by block) were assigned from the 2014 estimation. In situ density in the Resource Model varies between 2.291 and 2.617 t/m^3.

The estimated bulk density values were assigned using Ordinary Kriging interpolation, and the details are explained in Section 14.10 of the 2014, PFS (PFS, 2014).

14.7 Validations

A series of validations on the estimated grades were carried out, including visual and statistical validations.

14.7.1 Global Bias

The estimate was validated using independent checks including comparison of summary statistics between the Ordinary Kriging estimate and a Nearest Neighbour estimate, visual inspection of estimated grade against samples and drift analysis to detect spatial bias. The

Nearest Neighbour method assigns the Au grade value of the nearest sample to each block. The estimated Au grade values are compared thus to the sample grade values assigned to blocks. It is a proxy for data declustering.

Table 14-9 shows the comparison between Ordinary Kriging and Nearest Neighbour estimates for gold. It shows that the relative error between estimates and database, and between estimates and nearest neighbour model is close to 5%, which Andres Beluzan (QP) considers to be reasonable. It should be noted that this validation procedure was carried out for the Measured and Indicated Resources only.

Table 14-9: Global Bias Validation

Zone	Au OK	Au NN	Composites	OK/NN	OK/Composite
Fenix North	0.42	0.41	0.45	3%	-6%
Fenix Central A	0.42	0.41	0.44	2%	-5%
Fenix Central B	0.28	0.28	0.30	2%	-6%
Fenix South	0.39	0.37	0.44	4%	-13%

14.7.2 Drift Analysis

A drift analysis is used to compare spatial trends between the estimated grades and the NN model (declustered samples) in the east-west, north-south and vertical coordinate directions. Drift analysis was obtained by plotting the average grades from Ordinary Kriging, Nearest Neighbour, and composites within slices of 20m (two blocks) in the north-south and east-west direction and 10 m in vertical direction. The analysis was focused on the Measured and Indicated Resources.

The trend analysis shows an agreement between Ordinary Kriging (red), declustered or NN estimates (green), and composites (blue), since curves follow very similar trends, and therefore, results were considered satisfactory. Drift graphs for each zone in indifferent directions are shown in Figure 14-25 to Figure 14-28.

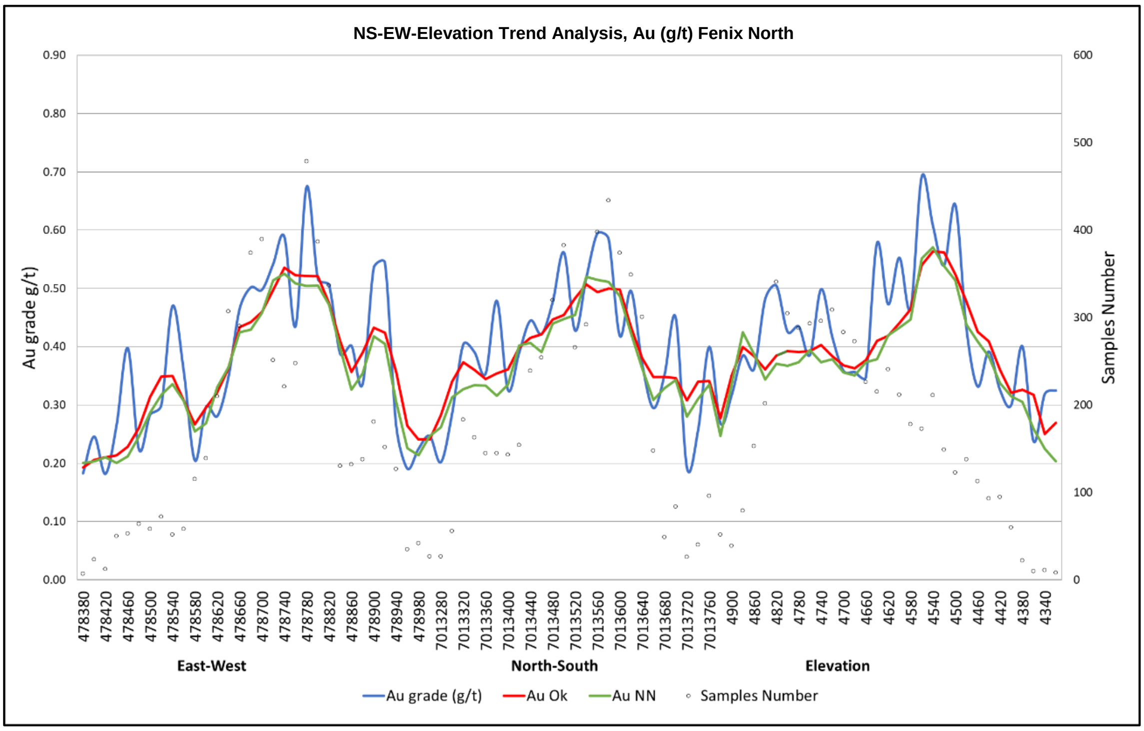


Figure 14-25: Drift Analysis, Fenix North

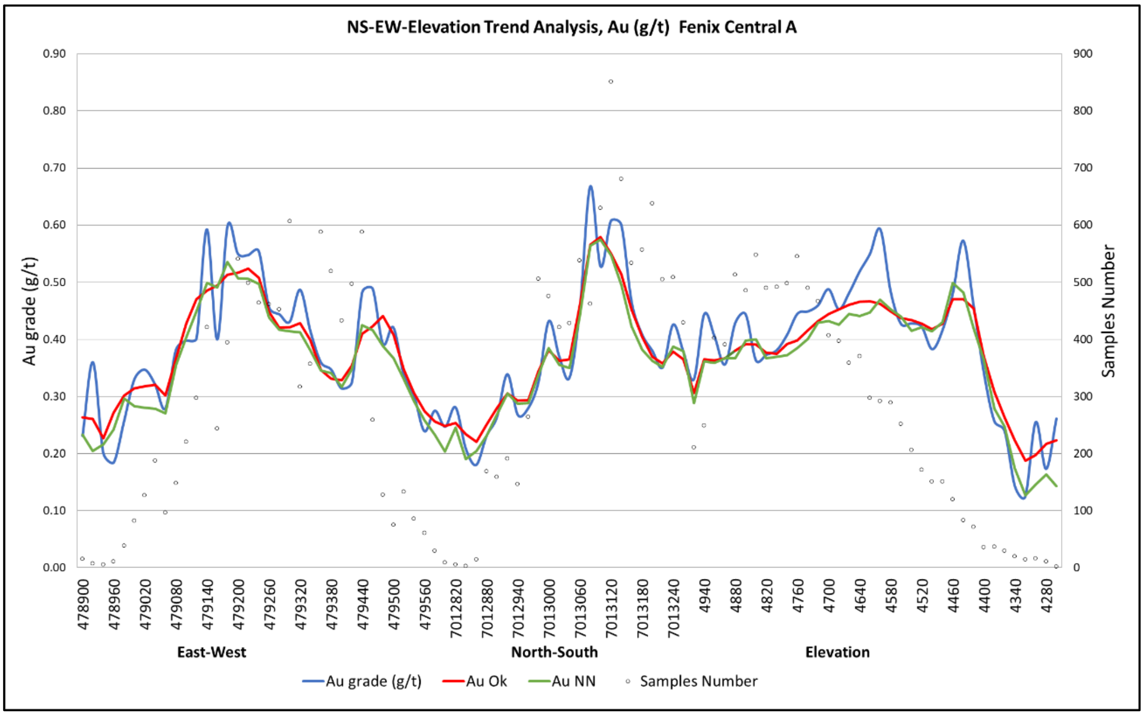


Figure 14-26: Drift Analysis, Fenix Central A

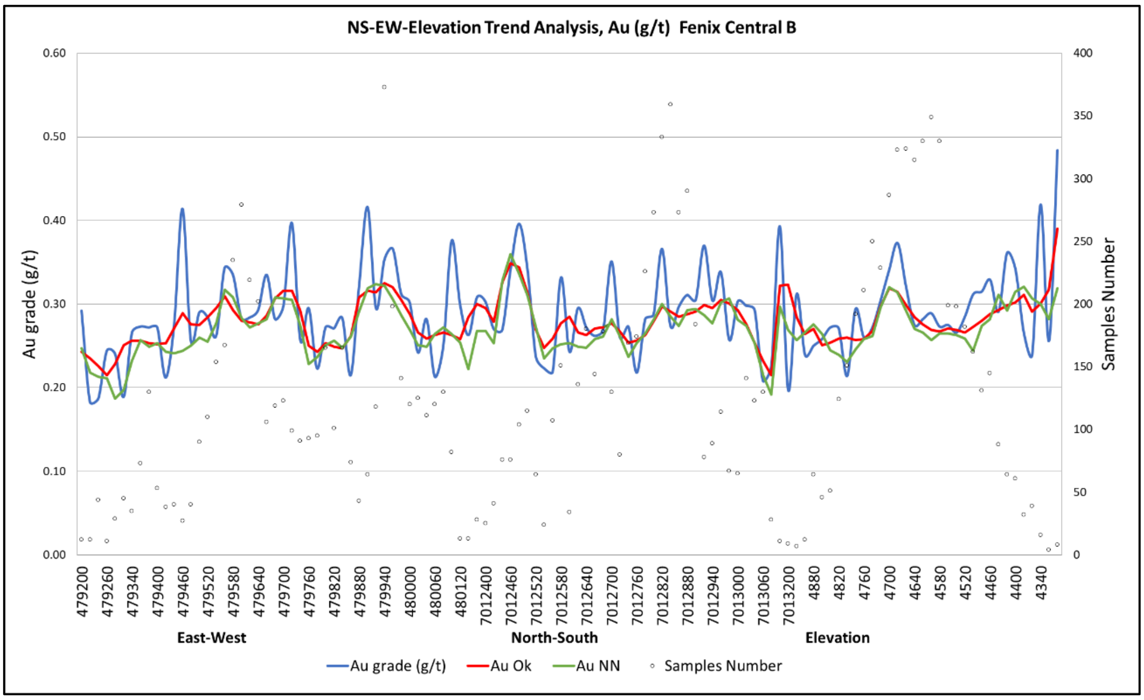


Figure 14-27: Drift Analysis, Fenix Central B

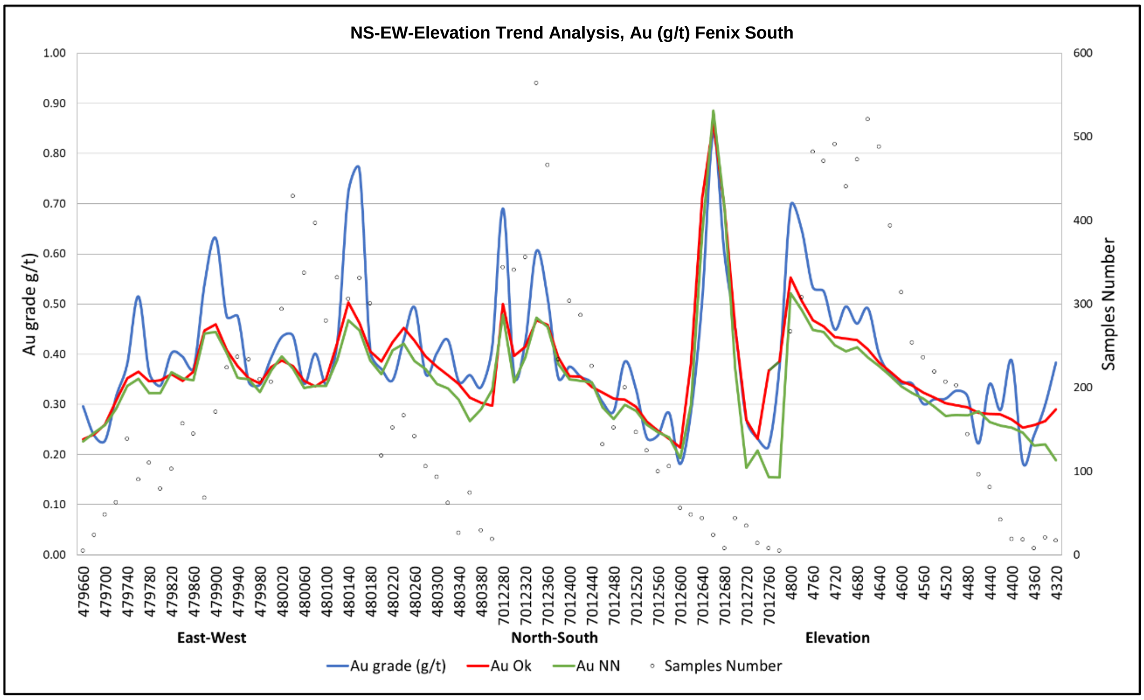


Figure 14-28: Drift Analysis, Fenix South

14.7.3 Visual Validation

Four cross sections of the Fenix Gold Project deposit (Figure 14-29) were reviewed by Andres Beluzan (QP) to compare Au block model estimates against drill hole sample grades using the same colour scheme. The Fenix North and Fenix Central A cross-sections are shown in Figure 14-30.

Similarly, visual validations were completed for the Cu estimates. Andres Beluzan (QP) notes that there is very little Cu in the deposit, generally less than 0.04% Cu, and only a few small areas in the 0.04% Cu to 0.1% Cu range.

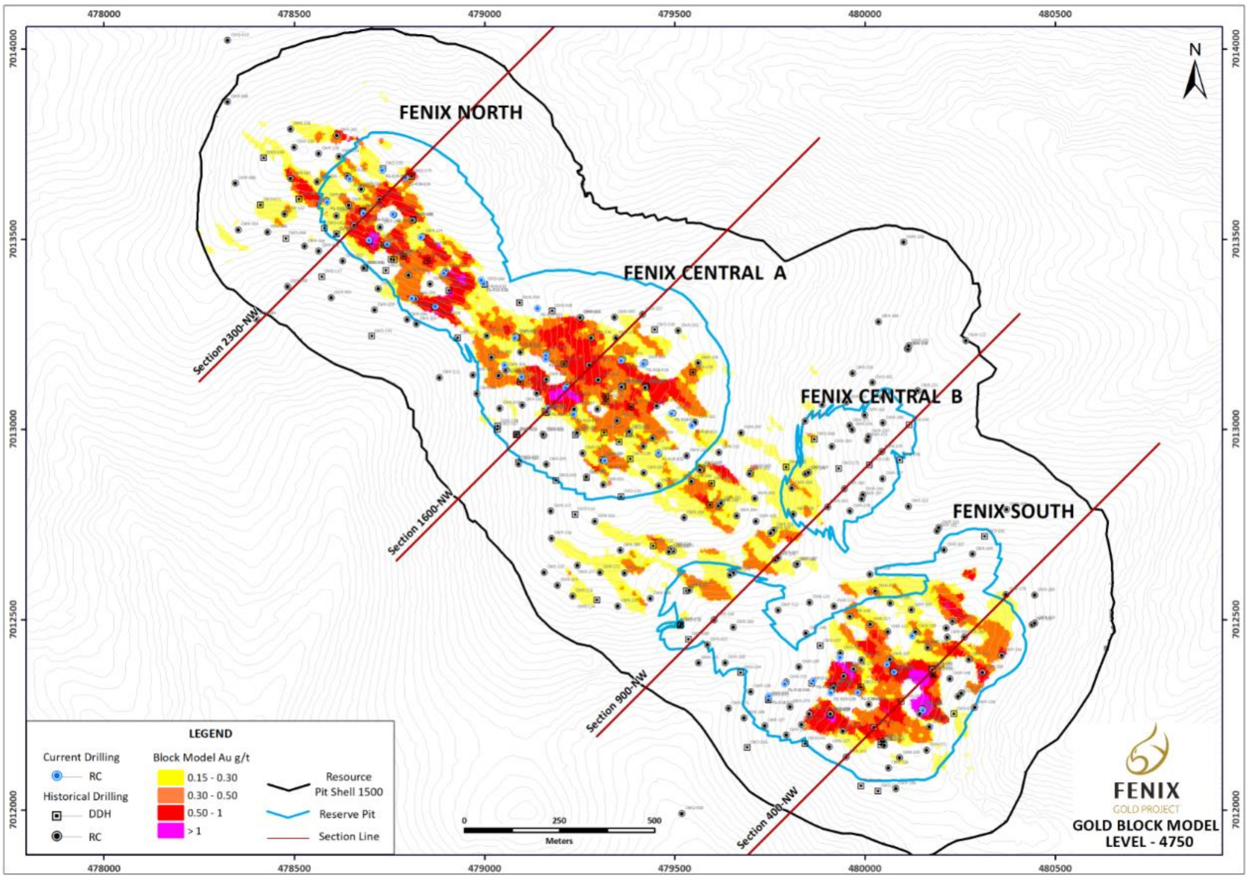


Figure 14-29: Location of the sections reviewed by Andres Beluzan to compare modelled Au grade and drilled grade

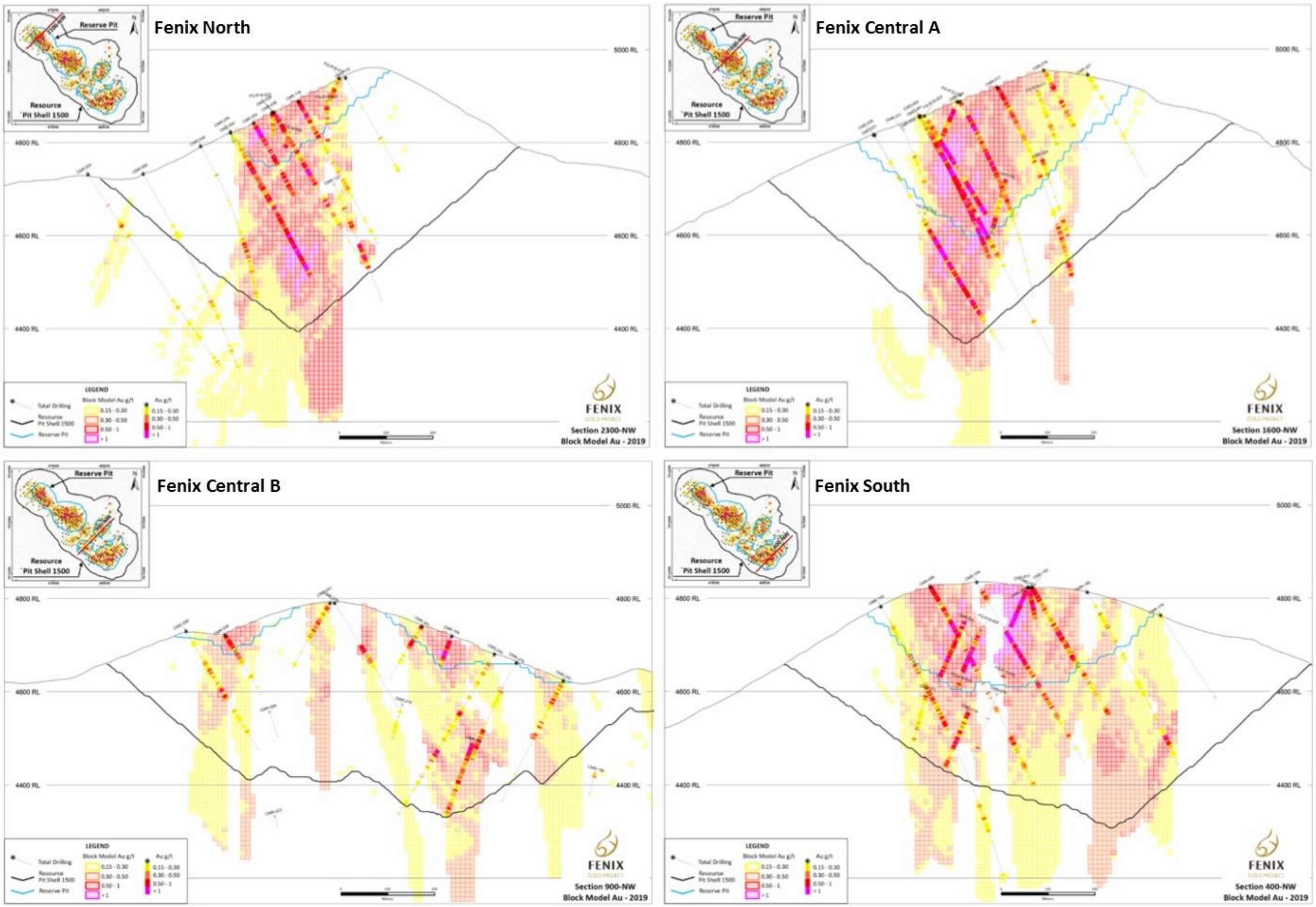


Figure 14-30: Sectional view of the Au estimate (looking NW)

14.8 Resource Classification

Mineral Resources for the Fenix Gold Project were estimated according to the Canadian NI 43-101 (Standards for Disclosure for Mineral Projects, 2011) and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014).

The Resource Classification methodology used in the 2019 model differs from that used in the 2014 Model. The 2014 model used kriging variance on a block basis and based on a mining production rate. This mining production rate is higher than the one currently envisioned, and given this and another characteristics of the method employed in 2014, it can be said that the resource classification in 2014 was overly generous for Measured and Indicated Resources.

Conversely, the 2014 Model was extremely conservative on Inferred Resources, with distances used to project Inferred at roughly 25% of variogram ranges. This has been rectified in the current model where the full variogram range (150 m) has been used to project Inferred Resources at depth.

GeoSystems International has based the current resource classification on a more traditional (geometric) approach, taking advantage of several parameters available from the block estimation process. Among others, the number of drill holes used in the estimation of the block; the minimum number of samples; the anisotropic closest distance to the nearest sample and the anisotropic average distance samples.

After obtaining the classification codes in each block, a manual smoothing process was performed on the Measured and Indicated Resource to improve continuity.

Blocks inside the mineralization envelope that were not classified as Measured, Indicated or Inferred are not considered resources, were classified as potential mineralization, and are not presented in the resource statement.

Mineral resources for the Fenix Gold Project were classified using the following criteria:

Measured Mineral Resources:

1. Portion of block must be contained within interpreted mineralized domain.
2. Anisotropic Closest Distance of samples used to estimate the block must be less than or equal to 40 m.
3. Anisotropic Average Distance of samples used to estimate the block must be less than or equal to 75 m.
4. The block must be estimated with three drill holes or more.

5. The number of samples used to estimate the block must be greater or equal than 10.

Indicated Mineral Resources:

1. Portion of block must be contained within interpreted mineralized domain.
2. Anisotropic Closest Distance of samples used to estimate the block must be less than or equal to 70 m.
3. Anisotropic Average Distance of samples used to estimate the block must be less than or equal to 85 m.
4. The block must be estimated with two drill holes or more.
5. The number of samples used to estimate the block must be greater or equal than 10.

Inferred Mineral Resource:

1. Portion of block must be contained within interpreted mineralized domain.
2. Anisotropic Average Distance of samples used to estimate the block must be less than or equal to 150 m.
3. The block can be estimated with a minimum of two drill holes.

Results of the resource categorization procedure are shown in Figure 14-31 using the same sections presented in Figure 14-30.

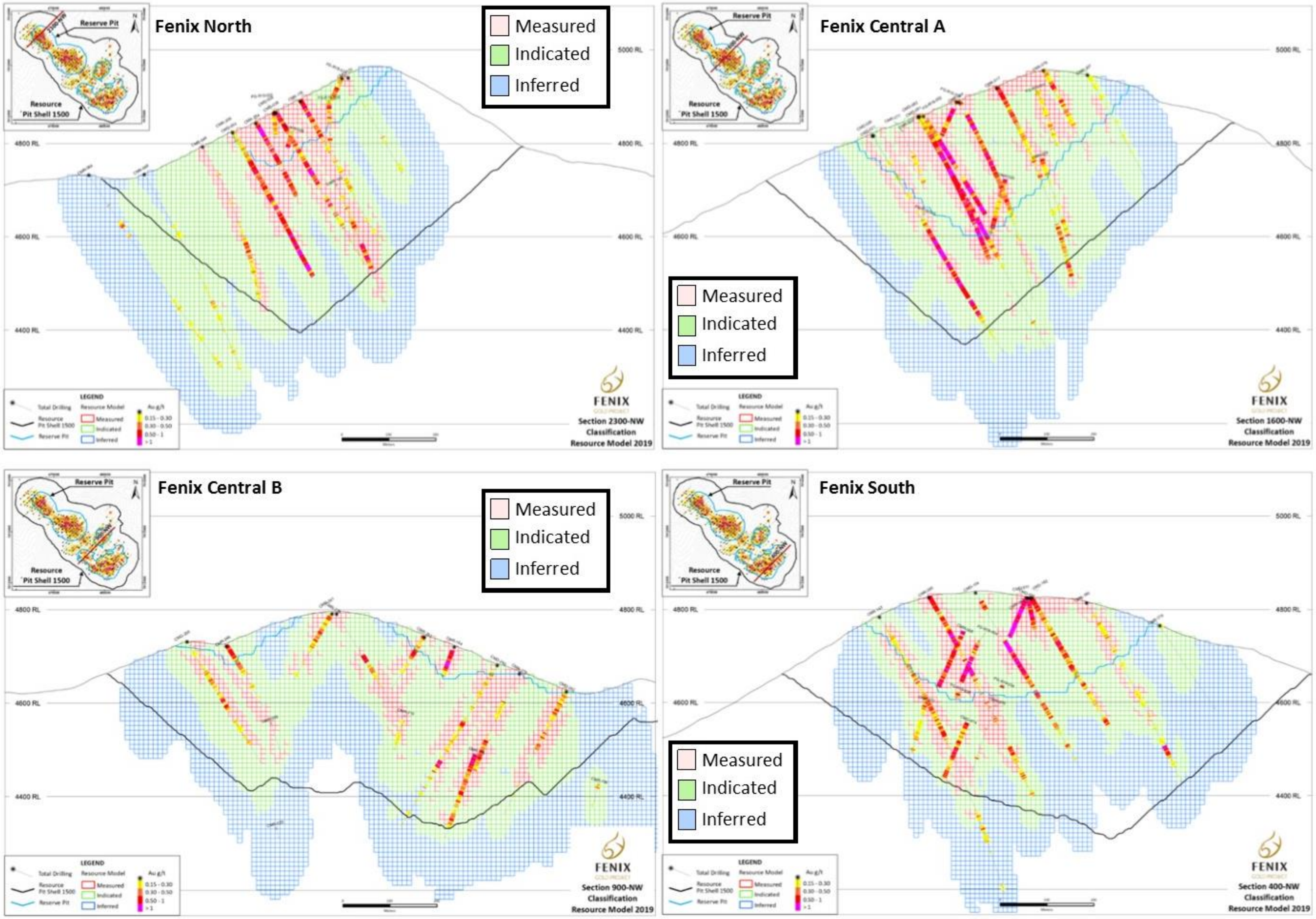


Figure 14-31: Section views of the Resource Classification (looking NW)

14.9 Resource Tabulation

The Mineral Resource has been determined inside a Whittle Open Pit Optimization, using the long term parameters of the 2014 PFS reflecting the longer term potential of the project on the optimization utilizing Measured, Indicated; and Inferred resources. The parameters used to develop the Resource Pit are shown in Table 14-10. Costs have been based on the 2014 PFS, which represents the long term potential of the project. No dilution or mining recovery factor was applied (100% mining recovery, 0% dilution).

Table 14-10: Resource Pit Parameters

Item	Units	Value
Au Price	$/troy ounce	1,500
Au Recovery to Dore	Percent (%)	79.5
Gold Refining Charge	$/troy ounce	10
Average Mining Cost	$/tonne mined	1.45
Processing and G/A Cost	$/tonne processed	3.09
Overall Pit Slope Angle	Degrees	41°
Discount Rate	Percent (%)	5

The Resource Table for the Fenix Gold Project for Measured, Indicated, Measured plus Indicated, and Inferred is shown in Table 14-11, and corresponds to a cut-off grade of 0.15 g/t Au inclusive of Reserves.

Table 14-11: Resource Statement for the Fenix Gold Project, 0.15 g/t Au cut-off grade

Resource Classification	Million Metric Tonnes	Au Grade (g/t)	Au Ounces (x1000)
Measured	122.4	0.41	1,630
Indicated	288.3	0.36	3,355
Total Measured + Indicated	**410.7**	**0.38**	**4,985**
Inferred	136.6	0.32	1,388

1. Mineral Resources reported is inclusive of mineral reserves.

2. Table 14-11 includes all Measured, Indicated, and Inferred Resources contained within the "Resource Pit", which represents the test for eventual extraction applied.

3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.

4. Mineral Resources are reported in accordance with Canadian Securities Administrators (CSA) National Instrument 43-101 (NI 43-101) and have been estimated in conformity with generally accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines.

5. Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

6. The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured mineral resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category.

Figure 14-32 shows the overall tonnage-grade curve for the gold mineral resources considering all mineralized domains within the Fenix Gold Project.

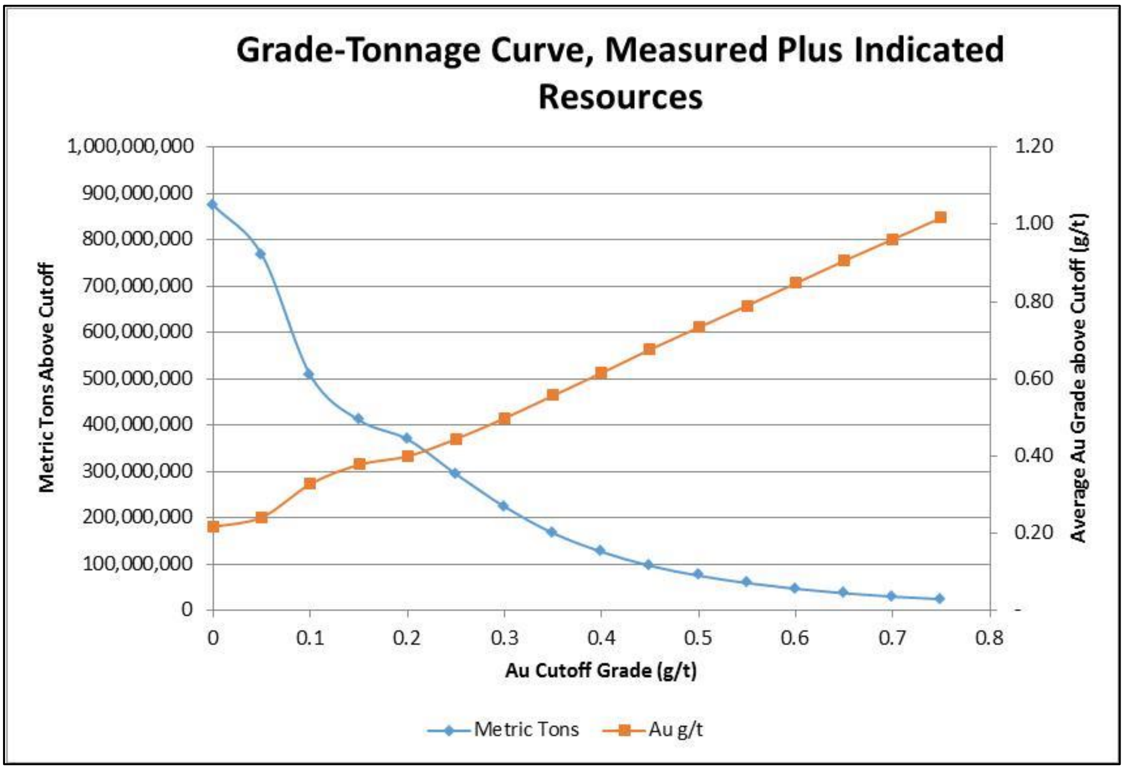


Figure 14-32: Grade-Tonnage Curve, Measured + Indicated Resources, Fenix Gold Project

Table 14-12 shows the grade-tonnage relationships corresponding to for Measured, Indicated, Measured plus Indicated and Inferred resources for the global Fenix Gold Project resources.

Table 14-12: Resources at Different Cut-offs for the Fenix Gold Project

Au Cut-off g/t Au	Measured			Indicated			Measured + Indicated			Inferred		
	Tonnes	Au g/t	Ounces (x1000)	Tonnes	Au g/t	Ounces (x1000)	Tonnes	Au g/t	Ounces (x1000)	Tonnes	Au g/t	Ounces (x1000)
0	217.6	0.27	1,861	657	0.2	4,225	874.6	0.22	6,086	537	0.13	2,245
0.05	196.6	0.29	1,833	570.2	0.23	4,125	766.9	0.24	5,958	395.8	0.17	2,100
0.1	143.7	0.37	1,709	363.4	0.31	3,633	507.1	0.33	5,342	211.1	0.25	1,670
0.15	**122.4**	**0.41**	**1,630**	**288.3**	**0.36**	**3,355**	**410.7**	**0.38**	**4,985**	**136.6**	**0.32**	**1,388**
0.2	112.3	0.44	1,570	257.1	0.38	3,174	369.3	0.4	4,744	112.5	0.35	1,251
0.25	94	0.48	1,439	200.1	0.43	2,760	294.1	0.44	4,199	82.1	0.39	1,032
0.3	74.8	0.53	1,269	148.9	0.48	2,313	223.7	0.5	3,582	57.5	0.44	817
0.35	58	0.59	1,094	108.8	0.54	1,892	166.8	0.56	2,986	40.8	0.49	643
0.4	45.9	0.64	948	80.4	0.6	1,552	126.4	0.62	2,500	27	0.55	477
0.45	36.1	0.7	814	60	0.66	1,275	96.1	0.68	2,089	19.7	0.6	378
0.5	28.8	0.76	702	46.2	0.72	1,064	74.9	0.73	1,766	14	0.65	292
0.55	23.3	0.82	610	35.8	0.77	889	59	0.79	1,499	9.8	0.7	220
0.6	18.9	0.87	529	27.7	0.83	741	46.6	0.85	1,269	7.2	0.75	174
0.65	15.3	0.93	457	21.7	0.89	621	37	0.91	1,077	5.2	0.79	133
0.7	12.5	0.99	396	17.3	0.94	525	29.8	0.96	922	4.1	0.82	108
0.75	10.3	1.05	345	13.9	1	445	24.2	1.02	790	2.6	0.88	73

1. Mineral Resources reported is inclusive of mineral reserves.

2. Table 14-12 includes all Measured, Indicated, and Inferred Resources contained within the “Resource Pit”, which represents the test for eventual extraction applied.

3. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.

4. Mineral Resources are reported in accordance with Canadian Securities Administrators (CSA) National Instrument 43-101 (NI 43-101) and have been estimated in conformity with generally accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines.

5. Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

6. The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured mineral resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category.

14.10 Comparison with the 2014 Resource Model

Andres Beluzan (QP) has compared the 2014 and 2019 Mineral Resource Estimates visually and numerically. The numerical comparisons are based on the grade-tonnage curves, particularly at the 0.15 g/t Au cut-off used as reference.

Table 14-13 to Table 14-15 show that the differences between the two models (positive means that the 2019 value is larger), particularly after considering the combined Measured and Indicated resources, is minimal. Based on the estimation parameters shown in Table 14-8, the current classification criteria are somewhat more conservative in the definition of Measured Resources at a 0.15 g/t Au cut-off. This is compensated by a slightly higher grade and more tonnage in the Indicated category, and thus results in about the same amount of Measured and Indicated Resources.

The differences are more significant for the Inferred Resources. This is due mostly to the introduction of a lithology model and the use of the 0.15 g/t Au, interpreted with more confidence in this since it has been constrained and guided by the lithology model. However, additional drilling is required to confirm and upgrade the Inferred Resources into the Indicated category.

Table 14-13: Comparison of Measured Resources, 2019 and 2014 Block Models, using US$1,500/oz Au Constraining Pit, 0.15 g/t Au

Study	Cut-off Au g/t	Measured		
		Tonnes	Au g/t	Ounces (x1000)
BM2019	0.15	122.4	0.41	1,630
BM2014	0.15	155.9	0.38	1,920
Difference (2019/2014)	**0%**	**-21.49%**	**8.09%**	**-15.10%**

Table 14-14: Comparison of Combined Measured and Indicated Resources, 2019 and 2014 Block Models, using US$1,500/oz Au Constraining Pit, 0.15 g/t Au

Study	Cut-off g/t Au	Measured + Indicated		
		Tonnes	Au g/t	Ounces (x1000)
BM2019	0.15	410.7	0.38	4,985
BM2014	0.15	404.8	0.38	4,929
Difference (2019/2014)	**0%**	**1.46%**	**-0.32%**	**1.14%**

Table 14-15: Comparison of Inferred Resources, 2019 and 2014 Block Models, using US$1,500/oz Au Constraining Pit, 0.15 g/t Au

Study	Cut-off Au g/t	Inferred		
		Tonnes	Au g/t	Ounces (x1000)
BM2O19	0.15	136.6	0.32	1,388
BM2O14	0.15	30.7	0.29	283,205
Difference (2019/2014)	**0%**	**344.95%**	**10.10%**	**-99.51%**

Note that while both models are visually similar with consistent spatial grade distribution, the 2019 Model has better continuity, both in terms of grade as well as resource categories.

The 2019 model and the 2014 model have essentially the same amount of Measured and Indicated resources, although the classification criteria applied to Measured and Indicated resources in 2019 is more conservative than the one applied in 2014.

Therefore the following major conclusions are:

- Additional RC drilling in 2018/19 has confirmed the 2014 resource estimate in the Measured and Indicated resource area.
- The more conservative classification criteria applied in 2019 will give the resource model a more robust nature.

The main difference between the two models is the Inferred category, which shows an increase with respect to the 2014 model. This is a result of using for the first time in the Fenix Gold Project a well-developed and interpreted geologic model, which in turn supports the interpretation of the 0.15 g/t isograde shell. Consequently, isogrades extend further at depth and laterally, supported by both the drilling and the geologic model.

Visual comparisons are shown in Figure 14-33 (Au grade) and Figure 14-34 (Resource Category), which shows side-by-side 4 sections through each of the deposit areas.

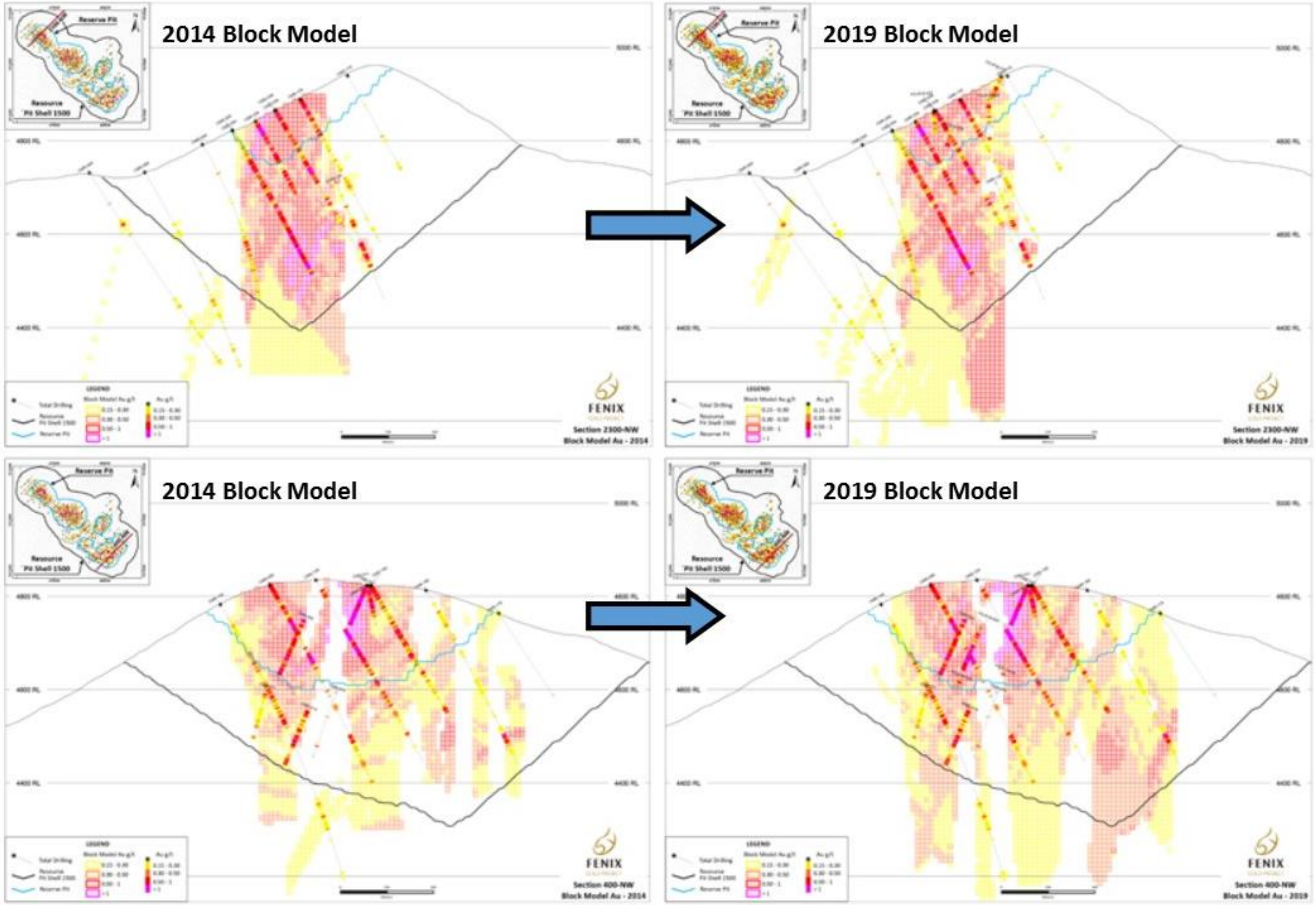


Figure 14-33: Example cross sections through the Fenix Gold Project showing Au grades from the drill holes and Block Model, 2014 Model on the left, 2019 Model on the right

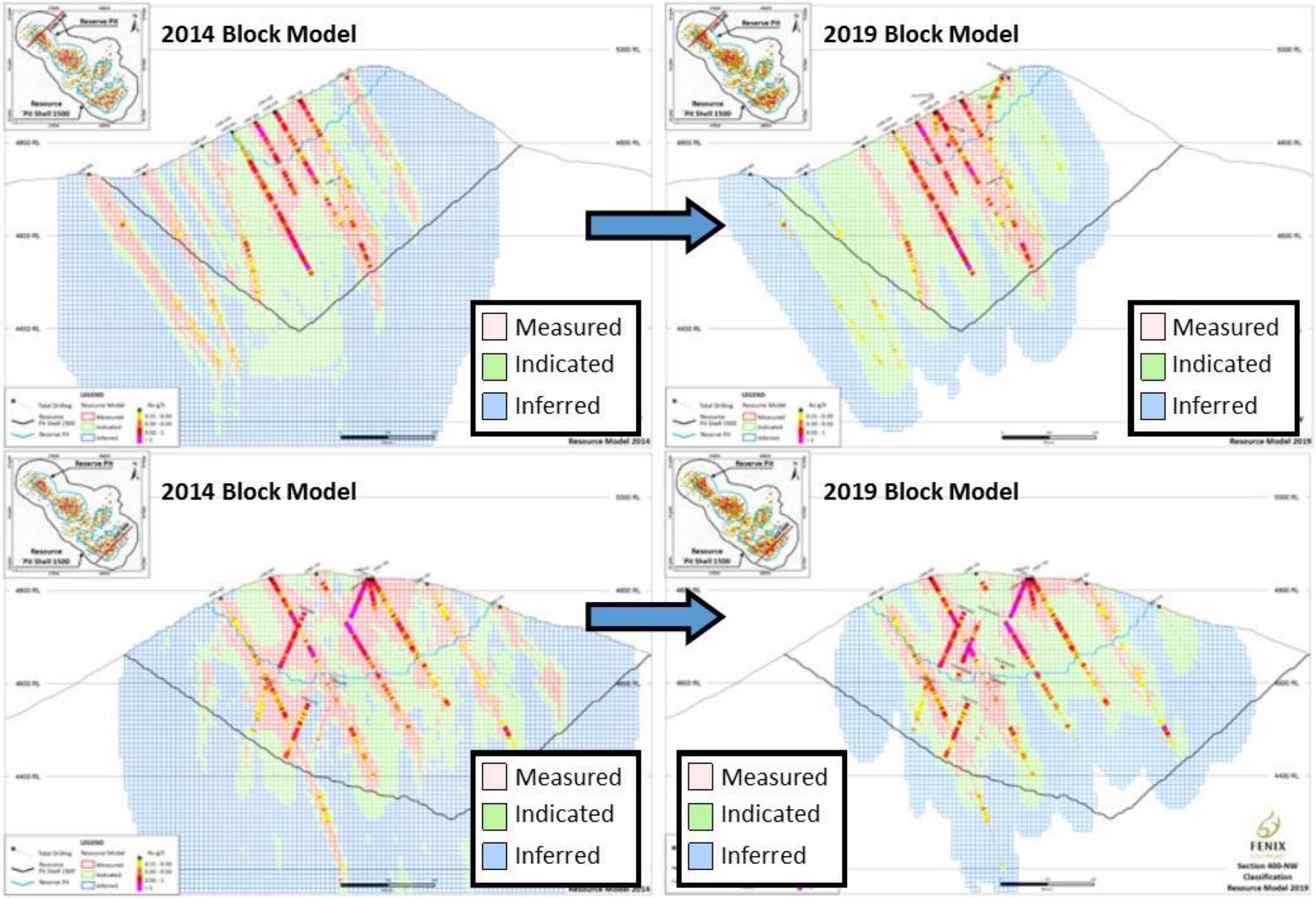


Figure 14-34: Example cross sections through the Fenix Gold Project showing Resource Classification Categories, 2014 Model on the left, 2019 Model on the right

14.11 Copper Content in the Deposit

The estimated Cu values in the block model show that the Fenix Gold Project contains very little Copper (Figure 14-35).

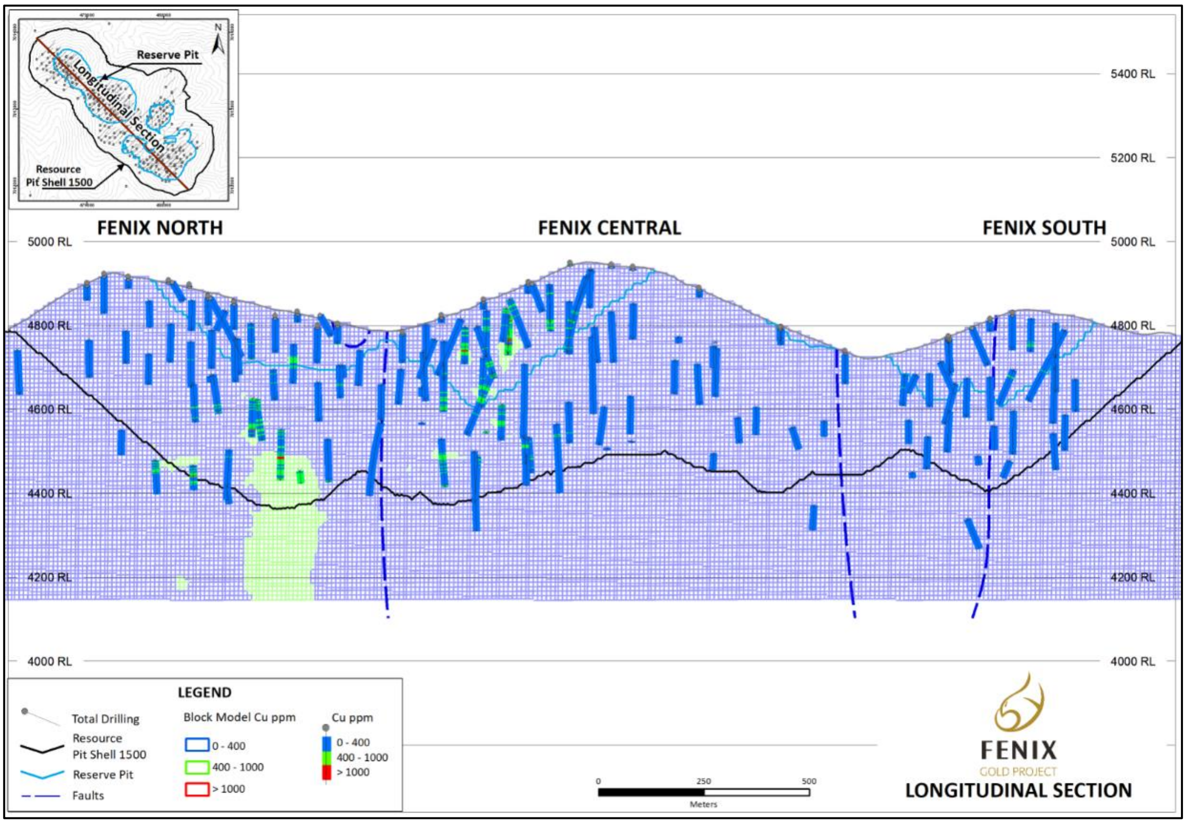


Figure 14-35: Longitudinal Sectional (looking NE), Estimated Cu Grades and Composites

In addition to the visual validations mentioned above, this is also seen when Cu content is tabulated in relation to Au content and based on the Resource Classification of the Block Model. Globally, about 3% of mineralized tonnage contains some minor Cu. Also, it can be seen that Measured Resources contain less copper than the Indicated and Inferred categories, and also that the vast majority of the estimated Cu values are below 0.1% Cu. Table 14-16 shows the details.

Table 14-16: Copper Content by Resource Category

Category	Cu < 400				400 =< Cu < 1000				Cu >= 1000			
	Tonnes	Au g/t	Cu ppm	Au Oz	Tonnes	Au g/t	Cu ppm	Au Oz	Tonnes	Au g/t	Cu ppm	Au Oz
Measured	117,554,170	0.394	171	1,489,670	4,840,030	0.889	508	138,413	-	-	-	0
Indicated	278,154,871	0.349	157	3,117,405	9,839,733	0.737	519	233,149	12,228	1.336	1,118	525
Inferred	132,143,956	0.311	142	1,322,334	4,319,473	0.461	498	63,984	-	-	-	0
Total	**542,164,070**	**0.348**	**155**	**6,067,345**	**18,999,236**	**0.713**	**511**	**435,546**	**12,228**	**1.336**	**1,118**	**525**

Table 14-17: Drill hole used in Mineral Resource Estimate

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
CMA-1	482018.1	7013361	4339	29.15	RC	2012
CMA-2	478034.1	7011445	4432	18.8	RC	2012
CMA-3	478964.1	7014949	4548	31.3	RC	2012
CMA-4	470200.1	7023838	4066	19.3	RC	2012
CMA-5	473655.1	7018864	3602	30	RC	2012
CMA-6	475346.2	7012921	4278.37	150	RC	2013
CMD-001	480178.6	7012357	4823.576	217.35	DDH	2010
CMD-004	478611.6	7013513	4822.83	181.85	DDH	2010
CMD-008	479177.9	7013312	4865.948	321.05	DDH	2010
CMD-009	479572.2	7012899	4911.464	350.2	DDH	2010
CMD-010	479380.6	7012990	4954.829	165.35	DDH	2010
CMD-011	480178.5	7012359	4823.583	224.2	DDH	2010
CMD-012	480073.9	7012364	4818.878	295.6	DDH	2010
CMD-014	480233.5	7012253	4778.408	398.05	DDH	2010
CMD-016	479907.7	7012249	4776.921	310.05	DDH	2010
CMD-019	479530	7012576	4770.79	266.2	DDH	2010
CMD-021	479084	7012986	4815.475	143.15	DDH	2010
CMD-026	479085.9	7012988	4815.7	536.4	DDH	2011
CMD-027	479909.1	7012249	4776.989	278.2	DDH	2011
CMD-031	479746.1	7012289	4697.008	302.35	DDH	2011
CMD-036	478680.1	7013574	4865.491	22.95	DDH	2011
CMD-037	479094.8	7013125	4819.423	424.95	DDH	2011
CMD-038	478680.8	7013574	4865.488	371.2	DDH	2011
CMD-046	479087	7013242	4816.23	310.95	DDH	2011
CMD-049	478736.1	7013485	4826.953	350.3	DDH	2011
CMD-056	479746.6	7012293	4697.283	490.8	DDH	2011
CMD-058	478849.8	7013443	4827.556	275.2	DDH	2011
CMD-065	478907.1	7013366	4798.933	272.35	DDH	2011
CMD-066	478478.5	7013502	4804.077	305.1	DDH	2011
CMD-072	478411.7	7013591	4855.928	416.3	DDH	2011
CMD-073	478930.3	7013241	4743.907	370.9	DDH	2011
CMD-091	479160	7013048	4855	25.95	DDH	2011
CMD-092	479163.9	7013044	4856.945	589.6	DDH	2011
CMD-093	479268.5	7012873	4915.303	531	DDH	2011
CMD-096	479364.4	7013113	4957.437	351.85	DDH	2011
CMD-099	479089.1	7012914	4818.77	700	DDH	2011
CMD-104	480095.3	7012285	4835.374	450	DDH	2011
CMD-108	479385.2	7013060	4965.738	347.65	DDH	2011
CMD-111	478685.8	7013424	4786.66	450	DDH	2011
CMD-119	479990	7012323	4793.986	563	DDH	2012
CMD-121	479443.4	7012694	4840.406	491.45	DDH	2012
CMD-122	478786.7	7013455	4817.729	173.5	DDH	2012
CMD-126	478698	7013494	4826.261	400	DDH	2012
CMD-128	479383.2	7012922	4940.031	600.45	DDH	2012

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
CMD-137	479882.7	7012432	4743.396	450	DDH	2012
CMD-140	479209	7013172	4883.046	400	DDH	2012
CMD-141	479358.7	7012824	4920.369	522.05	DDH	2012
CMD-144	479238.2	7012777	4895.111	630	DDH	2012
CMD-145	479843	7012174	4758.572	338.15	DDH	2012
CMD-147	478572.6	7013402	4756.632	437.1	DDH	2012
CMD-152	479354.6	7012967	4949.84	134.8	DDH	2012
CMD-153	479989.4	7012064	4774.147	600	DDH	2012
CMD-154	479296.3	7012552	4808.172	449.55	DDH	2012
CMD-160	480034.5	7012050	4773.462	351.65	DDH	2012
CMD-165	479593	7012802	4899.123	422.95	DDH	2012
CMD-167	479034.5	7013001	4784.771	281.89	DDH	2012
CMD-169	478420.5	7013714	4897.628	440.85	DDH	2012
CMD-170	478808.3	7013665	4939.656	314.4	DDH	2012
CMD-178	479424.7	7013113	4976.116	107.85	DDH	2012
CMD-182	478741.2	7013418	4789.913	400	DDH	2012
CMD-184	479425.8	7013115	4975.896	153.85	DDH	2012
CMD-187	479314.5	7012992	4944.268	471.2	DDH	2012
CMD-191	478703.8	7013246	4691.677	350.15	DDH	2012
CMD-192	480176.5	7012372	4823.313	320	DDH	2012
CMD-193	479916.6	7012326	4764.747	180	DDH	2012
CMD-196	480149.9	7012264	4823.542	250.02	DDH	2012
CMD-198	478812.9	7013551	4877.852	80.35	DDH	2012
CMD-200	479536.4	7012449	4730.998	300	DDH	2012
CMD-201	479597.3	7012858	4900.272	300.05	DDH	2012
CMD-224	479673.8	7012362	4656.801	230.35	DDH	2012
CMD-228	479860.1	7012333	4739.407	270.25	DDH	2012
CMD-232	479793	7012901	4795.21	317.55	DDH	2012
CMD-236	480092.3	7012920	4640.151	300	DDH	2012
CMD-240	480011.8	7012908	4680.717	400.05	DDH	2012
CMD-242	480115.9	7013012	4623.792	290.3	DDH	2012
CMD-247	479239.7	7012986	4903.804	380.3	DDH	2012
CMD-249	479447.6	7013262	4964.494	650.15	DDH	2012
CMD-256	479092.5	7013333	4833.508	401	DDH	2013
CMD-258	479547.8	7013150	4951.393	530.25	DDH	2013
CMD-262	478579.7	7013529	4827.083	390	DDH	2013
CMD-266	479690.9	7012165	4668.965	300	DDH	2013
CMD-268	479867.2	7012975	4754.412	230.1	DDH	2013
CMD-272	479516.8	7012488	4751.343	250.15	DDH	2013
CMD-275	479931.7	7012898	4721.573	210	DDH	2013
CMD-278	479514.9	7012485	4751.267	200.15	DDH	2013
CMD-281	480314.9	7012719	4711.295	140.1	DDH	2013
CMD-282	478513	7013605	4865.078	270	DDH	2013
CMD-284	479000.7	7013381	4829.057	350	DDH	2013
CMD-287	478868.3	7013322	4763.181	310	DDH	2013
CMD-293	479187.5	7012866	4870.863	450	RC/DDH	2013

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
CMD-295	478733.1	7013686	4934.411	250	DDH	2013
CMD-298	479035.4	7013010	4784.543	464.2	RC/DDH	2013
CME-001	479149.7	7014634	4565.801	300	DDH	2013
CMQ-001	478264.8	7011870	4428.772	150	RC	2013
CMQ-002	479474.5	7011636	4495.626	150	RC	2013
CMQ-003	479519	7011992	4571	100	RC	2017
CMQ-004	479183	7011524	4486	100	RC	2017
CMQ-005	478371	7012092	4450	100	RC	2017
CMQ-006	477966	7011360	4420	90	RC	2017
CMQ-007	477517	7013041	4700	100	RC	2017
CMQ-008	477691	7010945	4358	70	RC	2017
CMQ-009	479292	7014115	4733	100	RC	2017
CMQ-010	478325	7014023	4734	100	RC	2017
CMQ-011	476433	7013127	4387	100	RC	2017
CMQ-012	477291	7013633	4495	100	RC	2017
CMR-002	479164	7013047	4856.871	342	RC	2010
CMR-003	479163.1	7013201	4856.752	228	RC	2010
CMR-005	480143.2	7012253	4823.72	252	RC	2010
CMR-006	480216.5	7012454	4825.818	300	RC	2010
CMR-007	480077.1	7012362	4819.095	300	RC	2010
CMR-013	479417.8	7012957	4948.083	460	RC	2010
CMR-015	479243.5	7012992	4902.571	460	RC	2010
CMR-017	479275	7013169	4918.185	450	RC	2010
CMR-018	479380.4	7012996	4955.024	444	RC	2010
CMR-020	479162.2	7013132	4856.389	456	RC	2010
CMR-022	478627.5	7013443	4788.003	432	RC	2010
CMR-023	479585.4	7012435	4707.38	432	RC	2010
CMR-024	479441.8	7012978	4950.752	194	RC	2011
CMR-025	479253.5	7013295	4905.066	444	RC	2011
CMR-028	479298.7	7013131	4928.372	396	RC	2011
CMR-029	479360.2	7013115	4956.913	416	RC	2011
CMR-030	479360.2	7013112	4957.015	374	RC	2011
CMR-032	479348.3	7013024	4952.513	362	RC	2011
CMR-033	479418.8	7012886	4934.914	348	RC	2011
CMR-034	479311.6	7012922	4939.227	450	RC	2011
CMR-035	479162.5	7012909	4857.285	486	RC	2011
CMR-039	479307.5	7012920	4937.751	360	RC	2011
CMR-040	478682.7	7013425	4786.754	354	RC	2011
CMR-041	479267.8	7012876	4915.431	348	RC	2011
CMR-042	478800.2	7013406	4795.239	354	RC	2011
CMR-043	479250.8	7013295	4904.98	294	RC	2011
CMR-044	478584.4	7013602	4867.67	428	RC	2011
CMR-045	479422.2	7013110	4976.025	312	RC	2011
CMR-047	479458	7012853	4926.248	414	RC	2011
CMR-048	478564.2	7013470	4792.466	350	RC	2011
CMR-050	479566.3	7012894	4911.693	420	RC	2011

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
CMR-051	479312.6	7012856	4924.714	354	RC	2011
CMR-052	479296.4	7013055	4931.798	376	RC	2011
CMR-053	478597.2	7013347	4732.497	400	RC	2011
CMR-054	479215.9	7013116	4886.932	450	RC	2011
CMR-055	479538.1	7012579	4770.399	300	RC	2011
CMR-057	479235.6	7013054	4898.385	350	RC	2011
CMR-059	478710.9	7013315	4730.052	350	RC	2011
CMR-060	479099.5	7013065	4822.954	500	RC	2011
CMR-061	478857.1	7013383	4795.169	500	RC	2011
CMR-062	479257.8	7012939	4914.051	450	RC	2011
CMR-063	479156.2	7012985	4855.826	450	RC	2011
CMR-064	478399.7	7013292	4732.094	500	RC	2011
CMR-067	479089.6	7012910	4819.029	450	RC	2011
CMR-068	478482.4	7013376	4733.033	464	RC	2011
CMR-069	479083.2	7013245	4817.16	450	RC	2011
CMR-070	479006	7013247	4779.912	450	RC	2011
CMR-071	480161.6	7012158	4814.972	500	RC	2011
CMR-074	479362.9	7013185	4958.051	320	RC	2011
CMR-075	480244.3	7012299	4778.208	396	RC	2011
CMR-076	479345.2	7013240	4954.868	192	RC	2011
CMR-077	479153.9	7012988	4855.625	450	RC	2011
CMR-078	479040.3	7013056	4788.609	500	RC	2011
CMR-079	479452.5	7013063	4976.173	450	RC	2011
CMR-080	479018.3	7013190	4777.953	462	RC	2011
CMR-081	479496.7	7012682	4829.901	422	RC	2011
CMR-082	479094	7013204	4819.239	354	RC	2011
CMR-083	479699.9	7012890	4846.798	450	RC	2011
CMR-084	478796.4	7013289	4725.335	452	RC	2011
CMR-085	478490.5	7013660	4891.841	450	RC	2011
CMR-086	478345.3	7013647	4855.891	500	RC	2011
CMR-087	478640.4	7013667	4902.39	350	RC	2011
CMR-088	478325.1	7013861	4803.372	500	RC	2011
CMR-089	478810.5	7013551	4877.712	350	RC	2011
CMR-090	479935.5	7012413	4764.43	450	RC	2011
CMR-094	479908.8	7012253	4776.691	300	RC	2011
CMR-095	480023.6	7012217	4825.289	482	RC	2011
CMR-097	480177.6	7012369	4823.785	200	RC	2011
CMR-098	480150.2	7012266	4823.635	250	RC	2011
CMR-100	480091.1	7012138	4816.316	350	RC	2011
CMR-101	479696.8	7012884	4846.817	300	RC	2011
CMR-102	480050.4	7012180	4825.675	350	RC	2011
CMR-103	479713.8	7012759	4836.821	168	RC	2011
CMR-105	480164.7	7012428	4827.788	320	RC	2011
CMR-106	480213.1	7012479	4823.697	210	RC	2011
CMR-107	480065.2	7012397	4811.833	400	RC	2011
CMR-109	480132.8	7012470	4828.548	350	RC	2011

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
CMR-110	479368.2	7012623	4844.461	400	RC	2011
CMR-112	479918.2	7012537	4764.124	200	RC	2011
CMR-113	480060	7012469	4810.845	230	RC	2011
CMR-114	479941.7	7012208	4794.75	400	RC	2011
CMR-115	479991.1	7012395	4788.657	250	RC	2011
CMR-116	480009.7	7012278	4810.959	496	RC	2011
CMR-117	480013.6	7012489	4792.536	400	RC	2011
CMR-118	479960.4	7012510	4775.002	294	RC	2011
CMR-120	479037.1	7013143	4783.19	400	RC	2012
CMR-123	479854.4	7012546	4741.382	234	RC	2012
CMR-124	478610.6	7013562	4848.306	372	RC	2012
CMR-125	479495.5	7013043	4963.056	200	RC	2012
CMR-127	479531.5	7012932	4930.497	350	RC	2012
CMR-129	479917.5	7012323	4764.647	400	RC	2012
CMR-130	478564.3	7013725	4910.497	250	RC	2012
CMR-131	479853.2	7012254	4754.084	490	RC	2012
CMR-132	478617.3	7013718	4915.355	210	RC	2012
CMR-133	478675.7	7013632	4892.144	480	RC	2012
CMR-134	480230.4	7012496	4817.073	350	RC	2012
CMR-135	478724.9	7013605	4889.601	350	RC	2012
CMR-136	479279.9	7013241	4921.935	360	RC	2012
CMR-138	478489.7	7013790	4895.271	350	RC	2012
CMR-139	478499.8	7013742	4917.479	200	RC	2012
CMR-142	478474.2	7013567	4842.087	474	RC	2012
CMR-143	479950.4	7012141	4782.939	650	RC	2012
CMR-146	479844.7	7012465	4732.025	300	RC	2012
CMR-148	480223.6	7012346	4805.798	214	RC	2012
CMR-149	480282.9	7012673	4736.134	300	RC	2012
CMR-150	480111.7	7013213	4666.107	450	RC	2012
CMR-151	480169.6	7012219	4811.771	350	RC	2012
CMR-155	479966.7	7013000	4698.13	428	RC	2012
CMR-156	480113.1	7013212	4666.145	332	RC	2012
CMR-157	479736.6	7012223	4695.783	300	RC	2012
CMR-158	480013.3	7012620	4751.83	250	RC	2012
CMR-159	478720.5	7013370	4761.021	396	RC	2012
CMR-161	479457.5	7012936	4943.363	300	RC	2012
CMR-162	479553.4	7013019	4939.721	394	RC	2012
CMR-163	479341.9	7013295	4939.226	266	RC	2012
CMR-164	479945.5	7012845	4719.989	250	RC	2012
CMR-166	478527.6	7013482	4793.352	430	RC	2012
CMR-168	479682.4	7012243	4667	306	RC	2012
CMR-171	479959.3	7013011	4697.941	200	RC	2012
CMR-172	479303.4	7012625	4847.953	400	RC	2012
CMR-173	480005.9	7012973	4681.094	228	RC	2012
CMR-174	478837.1	7013508	4857.879	300	RC	2012
CMR-175	479641.1	7012268	4640.065	270	RC	2012

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
CMR-176	479561.5	7013176	4943.373	200	RC	2012
CMR-177	479244.7	7012643	4853.836	400	RC	2012
CMR-179	480370	7012567	4764.409	250	RC	2012
CMR-180	480261.2	7012454	4813.549	420	RC	2012
CMR-181	480189.1	7012734	4691.54	270	RC	2012
CMR-183	478898.4	7013416	4824.549	250	RC	2012
CMR-185	479436.2	7012557	4806.464	500	RC	2012
CMR-186	480080.6	7012057	4784.47	300	RC	2012
CMR-188	480149.2	7012267	4823.75	250	RC	2012
CMR-189	479826.6	7012377	4721.239	434	RC	2012
CMR-190	480035.8	7013284	4704.484	400	RC	2012
CMR-194	479994.4	7012829	4695.196	300	RC	2012
CMR-195	480253.3	7012308	4778.053	264	RC	2012
CMR-197	479991.3	7012819	4695.207	300	RC	2012
CMR-199	479421.6	7013176	4969.09	400	RC	2012
CMR-202	479509.3	7013260	4950.467	200	RC	2012
CMR-203	480372.3	7012790	4671.724	400	RC	2012
CMR-204	479950.9	7013072	4698.077	400	RC	2012
CMR-205	480194.2	7012742	4690.833	250	RC	2012
CMR-206	479843.9	7012884	4769.103	300	RC	2012
CMR-207	479415.5	7013303	4945.533	250	RC	2012
CMR-208	478979.2	7013095	4747.286	482	RC	2012
CMR-209	478610.8	7013514	4823.073	450	RC	2012
CMR-210	479156.2	7012625	4826.614	550	RC	2012
CMR-211	478881.4	7013137	4686.311	408	RC	2012
CMR-212	478761.9	7013565	4875.001	400	RC	2012
CMR-213	479192	7012591	4821.453	600	RC	2012
CMR-214	479289.9	7012759	4901.632	400	RC	2012
CMR-215	479350.6	7012537	4805.76	400	RC	2012
CMR-216	479176.2	7012715	4862.349	500	RC	2012
CMR-217	479174.5	7012786	4862.841	400	RC	2012
CMR-218	479231.2	7012563	4820.989	510	RC	2012
CMR-219	479851.1	7012888	4768.488	300	RC	2012
CMR-220	480009.1	7012982	4680.527	350	RC	2012
CMR-221	480101.6	7013493	4692.244	300	RC	2012
CMR-222	480264.5	7013235	4608.817	260	RC	2012
CMR-223	480114	7012799	4652.273	216	RC	2012
CMR-225	480051.1	7012171	4824.748	258	RC	2012
CMR-226	480061.2	7012111	4806.149	250	RC	2012
CMR-227	479906	7012167	4777.859	310	RC	2012
CMR-229	479944.2	7012353	4769.445	310	RC	2012
CMR-230	480288.7	7012271	4751.402	276	RC	2012
CMR-231	480138.7	7013103	4637.993	300	RC	2012
CMR-233	479793.3	7012337	4710.226	220	RC	2012
CMR-234	480045.8	7013085	4659.592	400	RC	2012
CMR-235	480114.7	7013220	4665.188	180	RC	2012

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
CMR-237	479833.4	7012225	4750.74	120	RC	2012
CMR-238	479699.3	7012312	4667.503	130	RC	2012
CMR-239	479764.1	7012658	4790.856	160	RC	2012
CMR-241	479772	7012664	4790.741	260	RC	2012
CMR-243	479710.1	7012819	4843.593	262	RC	2012
CMR-244	478612.3	7013774	4918.279	150	RC	2012
CMR-245	478560.2	7013651	4891.891	110	RC	2012
CMR-246	480272.5	7012396	4797.652	200	RC	2012
CMR-248	479603	7012501	4722.354	150	RC	2012
CMR-250	478643.1	7013590	4866.056	280	RC	2012
CMR-251	478429.8	7013520	4819.018	200	RC	2012
CMR-252	479137.2	7013096	4842.687	420	RC	2012
CMR-253	478818	7013344	4762.885	330	RC	2012
CMR-254	478657.5	7013537	4842.3	180	RC	2013
CMR-255	479616.2	7012940	4892.035	380	RC	2013
CMR-257	479674.6	7012992	4865.645	390	RC	2013
CMR-259	479967.4	7013149	4715.087	300	RC	2013
CMR-260	479842.8	7013023	4764.013	160	RC	2013
CMR-261	479999.8	7013038	4675.021	250	RC	2013
CMR-263	479912.4	7012955	4729.665	330	RC	2013
CMR-264	480359.2	7012407	4757.568	370	RC	2013
CMR-265	479818	7012645	4771.967	210	RC	2013
CMR-267	479821.8	7012647	4771.907	160	RC	2013
CMR-269	480446	7012566	4725.924	270	RC	2013
CMR-270	479802.5	7012272	4725.067	320	RC	2013
CMR-271	479653.7	7012624	4791.621	230	RC	2013
CMR-273	479645.3	7012618	4791.816	190	RC	2013
CMR-274	479970	7012371	4778.599	260	RC	2013
CMR-276	479960.8	7012786	4707.442	130	RC	2013
CMR-277	480046.2	7012872	4665.067	180	RC	2013
CMR-279	480044.5	7012942	4662.603	230	RC	2013
CMR-280	480047.3	7013018	4652.429	350	RC	2013
CMR-283	479654	7012481	4712.489	200	RC	2013
CMR-285	480066.4	7012544	4798.54	180	RC	2013
CMR-286	479543.8	7012864	4921.509	180	RC	2013
CMR-288	480309.1	7012363	4769.301	100	RC	2013
CMR-289	479356.9	7012683	4871.225	160	RC	2013
CMR-290	479524.4	7012769	4885.059	340	RC	2013
CMR-291	479902	7012798	4740.509	130	RC	2013
CMR-292	479887.1	7013066	4734.702	210	RC	2013
CMR-294	479662.9	7012774	4863.429	180	RC	2013
CMR-296	478726	7013532	4851.372	300	RC	2013
CMR-297	479793.2	7012198	4729.786	220	RC	2013
CMR-299	479614.8	7012800	4888.814	150	RC	2013
CMR-300	479807.8	7012847	4791.384	200	RC	2013
CMR-301	479621.7	7012807	4888.397	180	RC	2013

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
CMR-302	478970	7013144	4742.282	200	RC	2013
CMR-303	480122.1	7012527	4809.172	150	RC	2013
CMR-304	478821.4	7013277	4724.981	200	RC	2013
CMR-305	480019.7	7013124	4683.361	290	RC	2013
CMR-306	479055.1	7013157	4794.996	300	RC	2013
CMR-307	480207.7	7012684	4719.888	100	RC	2013
CMR-308	479811.7	7012778	4787.433	200	RC	2013
CMR-309	479632.2	7012388	4669.994	300	RC	2013
CMR-310	480026.9	7012576	4779.874	150	RC	2013
CMR-311	479562.4	7012387	4691.203	300	RC	2013
CMR-312	479771.3	7012526	4734.783	150	RC	2013
CMR-313	478352.9	7013525	4828.042	300	RC	2013
CMR-314	479489.3	7012685	4830.287	150	RC	2013
CMR-315	480438.9	7012488	4729.201	90	RC	2013
CMR-316	480446.1	7012492	4729.059	230	RC	2013
CMR-322	479911.4	7012313	4764.449	400	RC	2013
CMR-323	479759.8	7012736	4815.323	250	RC	2013
CMR-325	479751.7	7012728	4815.588	260	RC	2013
CMR-326	479484.3	7012679	4830.25	180	RC	2013
MET-01	478755	7013447	4832	150	DDH	2017
MET-02	479319	7013087	4938.5	300	DDH	2017
MET-03	480042	7012180	4831.5	250	DDH	2017
MGT-01	478763	7013446	4831	300	DDH	2017
MGT-02	479320	7013074	4938.5	300	DDH	2017
MGT-03	480042	7012172	4831.5	300	DDH	2017
T-1650	479244.8	7013201	4902.262	44	TRENCH	2013
T-400	480207.1	7012400	4822.657	54	TRENCH	2013
T-500	480179.3	7012513	4808.047	92	TRENCH	2013
T-700A	479962.1	7012571	4766.227	40	TRENCH	2013
T-7008	480064.1	7012681	4714.221	36	TRENCH	2013
FG-R18-001	478696.5	7013498	4826.3	300	RC	2018
FG-R18-002	479167.1	7013057	4857.16	348	RC	2018
FG-R18-003	480152.4	7012263	4823.49	250	RC	2018
FG-R18-004	480076	7012363	4819.09	210	RC	2018
FG-R18-005	479980.8	7012310	4793.72	170	RC	2018
FG-R18-006	479933.8	7012402	4765.64	148	RC	2018
FG-R18-007	479747.2	7012300	4697.71	100	RC	2018
FG-R18-008	479788.8	7012332	4710.74	100	RC	2018
FG-R18-009	479863.8	7012341	4740.63	120	RC	2018
FG-R18-010	479417.5	7013174	4969.35	250	RC	2018
FG-R18-011	479359.1	7013182	4957.81	240	RC	2018
FG-R18-012	479491.8	7013043	4963.65	250	RC	2018
FG-R18-013	479545.3	7013011	4940.34	120	RC	2018
FG-R18-014	479213.3	7013111	4887.06	230	RC	2018
FG-R18-015	479160.7	7013185	4856.79	250	RC	2018
FG-R18-016	479161.4	7013195	4856.83	200	RC	2018

Hole_ID	Easting	Northing	Elevation	Final_Depth	Hole_Type	Year
FG-R18-017	479097.5	7013138	4819.7	200	RC	2018
FG-R18-018	479079.7	7013241	4817.09	160	RC	2018
FG-R18-019	478792.3	7013662	4938.72	180	RC	2018
FG-R19-020	478731.3	7013682	4933.97	200	RC	2019
FG-R19-021	478645.7	7013659	4902.78	130	RC	2019
FG-R19-022	478681.7	7013568	4865.54	160	RC	2019
FG-R19-023	478586.2	7013598	4867.6	100	RC	2019
FG-R19-024	478893	7013411	4824.49	180	RC	2019
FG-R19-025	478809.8	7013345	4762.64	100	RC	2019
FG-R19-026	478870.9	7013323	4763.57	100	RC	2019
FG-R19-027	479052.5	7013170	4795.19	140	RC	2019
FG-R19-028	478996.8	7013384	4829.06	120	RC	2019
FG-R19-029	478759.6	7013565	4875.36	100	RC	2019
FG-R19-030	478832.6	7013506	4857.62	100	RC	2019
FG-R19-031	479139.2	7013319	4851.79	250	RC	2019
FG-R19-032	478991.2	7013392	4829.4	180	RC	2019
FG-R19-033	478744	7013487	4827.34	110	RC	2019
FG-R19-034	480124	7012458	4828.56	230	RC	2019
FG-R19-035	479457.8	7012939	4942.79	250	RC	2019
FG-R19-036	479316.4	7012919	4938.21	250	RC	2019
FG-R19-037	479234.7	7013041	4899.015	200	RC	2019
FG-R19-038	480057.6	7012383	4811.689	230	RC	2019
FG-R19-039	479909.8	7012309	4764.427	110	RC	2019

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

The Proven Mineral Reserve is based on Measured Mineral Resources and the Probable Mineral Reserve is based on Indicated Mineral Resources after consideration of all economic, mining, metallurgical, social, environmental, statutory and financial aspects of the Project.

The Mineral Resources have been converted to Mineral Reserves based upon the following modifying factors:

- Only Measured and Indicated Resources may be included.
- The Mineral Resources within an optimized pit limits are considered.
- Mining Dilution and Mining Recovery are applied.
- The mineralized rock is economically and technically feasible to extract.

Each of these requirements was addressed in establishing the Mineral Reserves. The Mineral Reserves statement has been prepared according to the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) standards.

15.2 Base Case Considerations

Annual production will commence at 11.7 Mt (waste and ore), ramping up to 20.5 Mt (waste and ore) in year seven. The Base Case considers a 20,000 tpd production (mineral only) rate based on available water.

The mine is scheduled to work seven days per week, 365 days per year. Each day will consist of two 12-hour shifts with four mining crews required to cover the operation.

15.3 Block Model

The resource Block Model presented in section 14 (file name "bmfx1904r10_incl_tr_edi2fin_rv") was used for all mine planning work. A three dimensional block model was generated to estimate grades into 10 x 10 x 10 m blocks. Sufficient variables were included in the block model construction to enable grade estimation and reporting.

The resource block model that was the basis of the pit optimization and scheduling work reported in this Study was prepared by third parties, and was provided to Mining Plus. The grid coordinate system used for this block model is the PSAD56 coordinate system.

The bounds of the block model are shown in Table 15-1.

Table 15-1: Block model framework (Model Origin is in the PSAD56 coordinate system)

	East	North	Elevation
Model origin	480000	7011600	2,500
Cell size meters	10	10	10
Rotation Angle		45°	

For areas of fill above the current topography surface (principally, waste dumps and leach pads), a nominal placed fill density of 1.7t/m^3 was assumed.

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:

- Bulk density (t/m^3).
- Gold grade in grams per tonne (g/t Au).
- Copper grade (ppm)
- A class code to distinguish Measured, Indicated, and Inferred resource blocks.

Raul Espinoza (QP) 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 2019 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.

15.4 Material Types (Mineralization)

All mineralization is oxide material, there is no transition or sulphide material.

Ore is typically supergene enriched and gold often occurs with iron oxides. Minor localized areas of relatively enriched copper and magnesium, are recognized in the deposit; copper and magnesium grades in these areas are not sufficiently high to affect processing.

15.5 Cut-off Grade

The cut-off grade was established to maximize revenue. The minimum cut-off grade was derived from an existing reference equation listed below:

$$\text{COG (Au g/t)} = \frac{(\text{Treatment Cost} + \text{G\&}A)}{(\text{Recovery})\ \text{x}\ (\text{Price} - \text{Sell Cost})}$$

**Treatment costs = processing and rehandling.*

The cut-off grade, calculated using the above formula, was used to determine the Mineral Reserves shown in Table 15-6. The by-products recovered in the concentrate or in the doré bars were considered to have too little value contribution to be included in the cut-off grade calculation. The economic assumptions presented in Table 15-3 from the previous section (pit optimization) were used to calculate a cut-off grade of 0.24 g/t Au.

As stated above, a cut-off grade of 0.24 g/t Au was used to define the Mineral Reserve; however, Rio2 raised the cut-off grade value used in the first production period to 0.4 g/t Au in order to obtain higher head grades in the schedule and improve cash flow as a result. This is a common industry practice in mine operations depending upon the grade distribution and stockpile capacity on site. The material between the elevated cut-off grade of 0.4 g/t Au and the marginal cut-off grade (0.24 g/t Au) is stockpiled for processing later when the capital costs have been retired. In the case that an alternative water source can be incorporated in to the project later, the stockpiled material will be able to be processed at a higher rate, which will improve the economics. In this mine production schedule all the material sent to the stockpile is reclaimed for processing at the end of mining. The raised cut-off grade for the Project is detailed in Table 15-2 and presented graphically in Figure 15-1.

Table 15-2: Cut-off Grade (COG) for the Mine Schedule

Year	High-Grade Cut-off Au g/t	Mid-Grade Cut-off Au g/t	Low-Grade Cut-off Au g/t
1	0.46	0.34	0.24
2	0.31	0.30	0.24
3	0.32	0.30	0.24
4	0.33	0.30	0.24
5	0.38	0.30	0.24
6	0.37	0.30	0.24
7	0.30	0.30	0.24
8	0.32	0.32	0.24
9	0.36	0.27	0.24
10	0.35	0.35	0.24
11	0.31	0.31	0.24
12	0.31	0.31	0.24
13	0.24	0.24	0.24
14	0.24	0.24	0.24
15	0.24	0.24	0.24
16	0.24	0.24	0.24
17	0.24	0.24	0.24

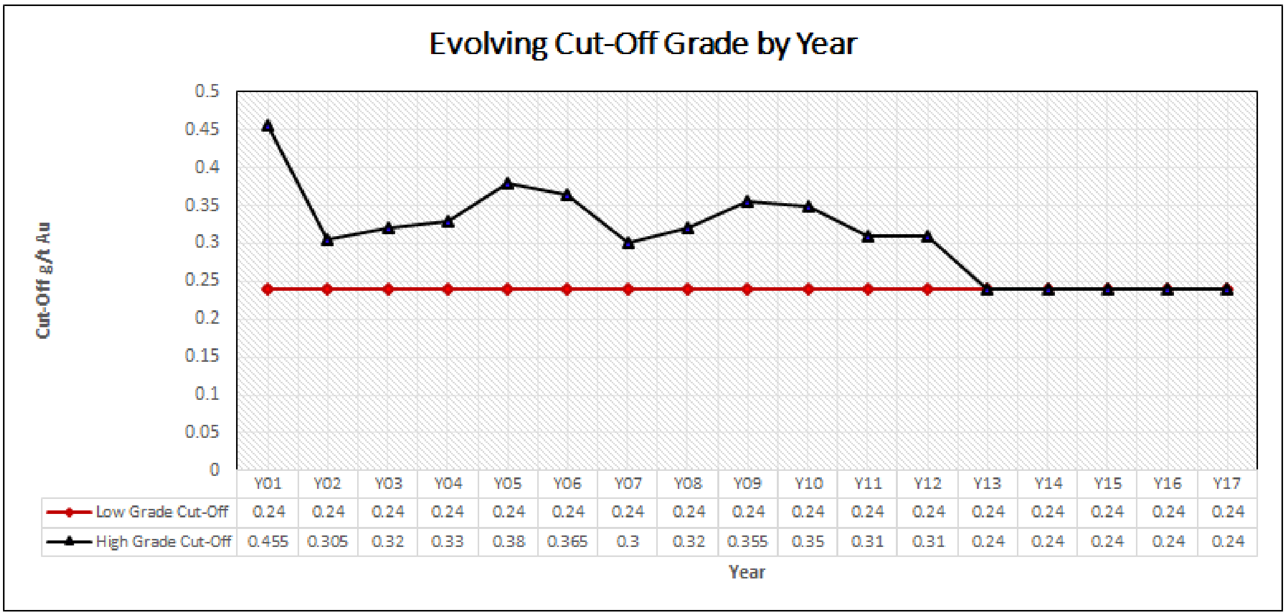


Figure 15-1: Cut-Off Grade by Year g/t Au

15.6 Pit Optimization

The pit optimization was conducted using the Whittle® software package. Whittle is a well-known commercial product that uses various geologic, mining, and economic inputs to determine the pit shell with the maximum profit and cash flow. The optimized economic pit shells were selected as the basis of open pit designs which were created using this software. Whittle is a well-known commercial product that uses various geologic, mining, and economic inputs to determine the pit shell with the maximum profit and cash flow.

The Mineral Reserves are constrained by a pit geometry that has been determined by technical and both cost and recovery economic inputs. The lists of assumptions used for the oxide pits are presented in Table 15-3.

Table 15-3: Pit Optimization Parameters for Mineral Reserves

Whittle Parameters	Value	Unit	Source
Characteristics of Block Model			
Dimensions	10x10x10	m	Coordinate System PSAD 56
Density	Model	t/m^3	According to Rock Type
Au grade	Model	(g/t)	Gold Grade
Pit Optimization			
Operating Costs			
Mine Operating Costs (ore)	1.82	US$/t	STRACON
Mine Operating Costs (waste)	1.82	US$/t	STRACON
Plant Costs to Process 20 ktpd			
Processing Costs	4.1	US$/t	HLC / STRACON
Rehandling Cost	0.91	US$/t	STRACON
Operational Support (G&A)	1.99	US$/t	Internal Rio2
Economic Parameters			
Discount factor	5	%	NI 43-101 20141006 Cerro Maricunga - Page 303
Refining Charge Au	10	US$/Oz	NI 43-101 20141006 Cerro Maricunga - Page 32
Mining Recovery	97	%	NI 43-101 20141006 Cerro Maricunga - Page 32
Dilution	3	%	NI 43-101 20141006 Cerro Maricunga - Page 32
Incremental cost per bench (10m)	0.021	$/t	Benchmark
Metallurgical Recovery			
Metallurgical Recovery of Au	75	%	HLC (Metallurgical QP)
Base Prices			
Base Price per Ounce of Au	1250	$/Oz	Au Price Benchmarking

The price of gold used for all optimization studies is $1,250 /oz. The optimal pit shell ultimately selected equates to a $1,225 /oz gold price and is discussed in section 15.6.1. Discussion on gold sales price forecasts is presented in Section 19.

15.6.1 Optimal Pit Shell Selection

Raul Espinoza (QP) highlights that economics alone did not drive the selection of the optimal pit at Fenix. Principal considerations when determining the optimal pit at Fenix included; water availability limited to 20,000 tpd, minimizing capital investment and minimizing stripping ratio´s.

Raul Espinoza used Geovia´s Whittle software and the Lerchs-Grossmann algorithm to determine the optimal pit. The Whittle software determines three scenarios, “Best Case”, “Worst Case”, and “Specific Case”.

The “Best Case” considers the sequential extraction of successive nested pits in totality prior to commencing of the extraction of the next. The “Best Case” scenario generates the best NPV but is rarely practically achievable as the approach implies very narrow mining widths.

The "Worst Case" considers the extraction of all mineral from single pit, this approach considers mining benches from the top of the pit to the bottom. The "Worst Case" scenario is nearly always achievable, but compared to the "Best Case" produces as greatly reduced NPV as it implies the mining of large volumes of waste earlier than might be needed.

The "Specific Case" NPV is the mining sequence somewhere between the "Best" and "Worst" cases. In general, the optimal pit, is defined by selecting initial phases with low Revenue Factors (RF) followed by selecting subsequent phases with higher RF´s. This approach balances viability and maximizing value.

Having run the Whittle optimization, Pit 35, with a RF of 0.98 was chosen. This equates to an optimized shell at a gold price of $1,225 /oz.

The final pit selection was focused on minimizing the movement of large amounts of waste and keeping close to the maximum NPV. The mine plan will be prepared annually and will be aimed at maximizing NPV and delivering and achievable rate of vertical advance (benches by phase by pit by year).

Figure 15-2 shows a "Pit by Pit Graph" analysis and Table 15-4 summarizes values for each Whittle pit shells.

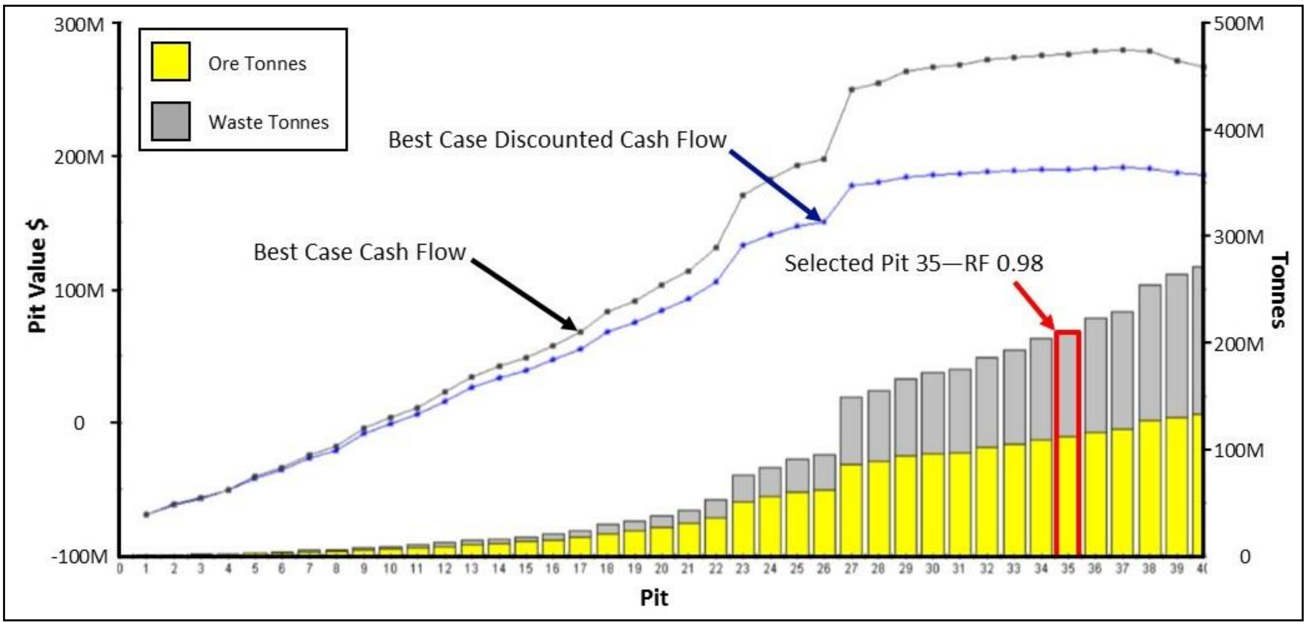


Figure 15-2: Pit by Pit Graph

Table 15-4: Summary of Whittle Shells (Pit Shell 35 was selected)

Pit Shell	Revenue Factor	Cash Flow "Best Case"	Cash Flow Specified Case	Ore (Mt)	Waste (Mt)
1	0.3	-69.1	-69.1	0.61	0.14
2	0.32	-61.2	-61.2	1.08	0.28
3	0.34	-56.8	-56.8	1.38	0.35
4	0.36	-50.7	-50.7	1.79	0.62
5	0.38	-41.3	-41.3	2.65	0.82
6	0.4	-35.3	-35.3	3.22	1.11
7	0.42	-26.7	-26.7	4.18	1.48
8	0.44	-21.2	-21.2	4.77	1.68
9	0.46	-7.9	-8.1	6.25	2.03
10	0.48	-0.9	-1.2	6.99	2.41
11	0.5	6.1	5.6	7.85	2.69
12	0.52	16.3	15.5	9.38	3.16
13	0.54	26.4	25.3	10.96	3.74
14	0.56	33.2	31.9	12.21	4.08
15	0.58	39.2	37.7	13.55	4.51
16	0.6	46.9	44.8	15.31	5.29
17	0.62	55.3	52.3	17.57	6.24
18	0.64	68.2	63.9	21.41	8.36
19	0.66	75.1	70.1	23.6	9.44
20	0.68	84.1	77.7	27.05	10.96
21	0.7	92.4	84.7	30.55	12.45
22	0.72	105.6	95.7	36.5	16.05
23	0.74	132.9	113.9	50.99	24.77
24	0.76	141	119.6	55.69	27.75
25	0.78	147.6	123.9	59.69	31.17
26	0.8	150.7	126	61.92	33.24
27	0.82	178.3	133.5	86.4	62.92
28	0.84	180.7	134.1	89.08	65.78
29	0.86	184.7	134.3	93.66	72.45
30	0.88	186.4	134	95.92	75.82
31	0.9	187.1	133.5	97.35	77.8
32	0.92	189	130.8	101.99	85.13
33	0.94	189.6	129.2	104.83	88.25
34	0.96	190.3	125.4	108.95	94.82
35	**0.98**	**190.4**	**124**	**111.64**	**97.96**
36	1	191.1	119.1	116.23	106.95
37	1.02	191.3	117.3	118.78	110.67
38	1.04	191	105.2	126.85	127.84
39	1.06	189.1	99.1	130.33	133.63
40	1.08	187	93.8	132.74	138.29
41	1.1	161.5	22.8	159.05	228.87
42	1.12	158.9	15.9	162.21	237.16
43	1.14	157.4	12.5	163.96	241.67

15.7 Mine Design

The final pit was designed based on pit 35 with ramp widths of 14 m in accordance with width requirements of the Chilean mining regulations. Pit ramps have a gradient of 10% for two-way traffic haul road. Mine design parameters are given in Table 15-5.

Table 15-5: Mine Design Parameters (Mining-plus)

Parameter	Unit	Value
Haul Road Width	m	14
Haul Road Gradient	%	10
Bench Height	m	10
Stacked Bench Height with 2 Benches Stacked	m	20
Nominal Minimum Mining Phase Width	m	50
Batter Angle	°	75
Berm Width	m	9.5
Inter-ramp Angle	°	53
Safety Berm Width Every 160 vertical m	m	30

The final pit design is presented in Figure 15-3 and Figure 15-4. There are planned two exits on the west side of the pit that gives access to the primary crusher and to the waste dumps.

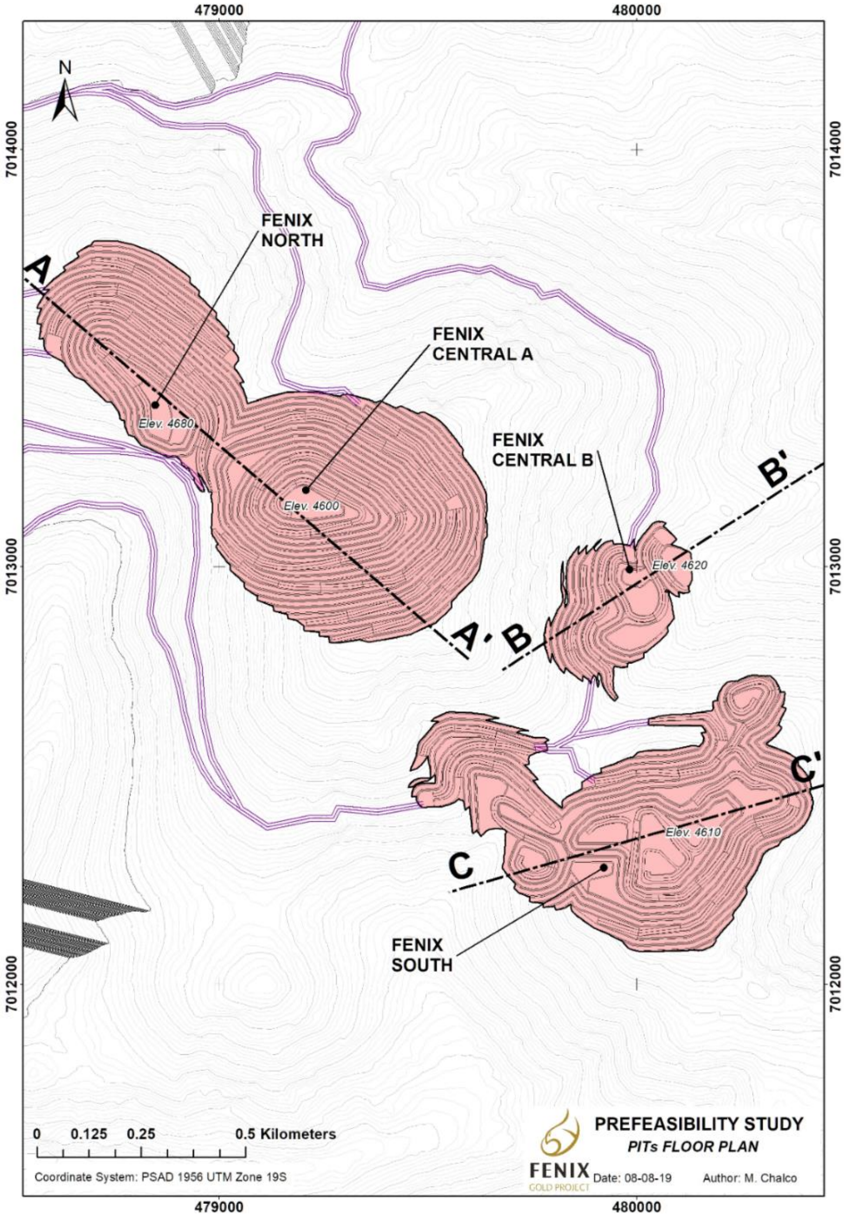


Figure 15-3: Plan view of final pit design

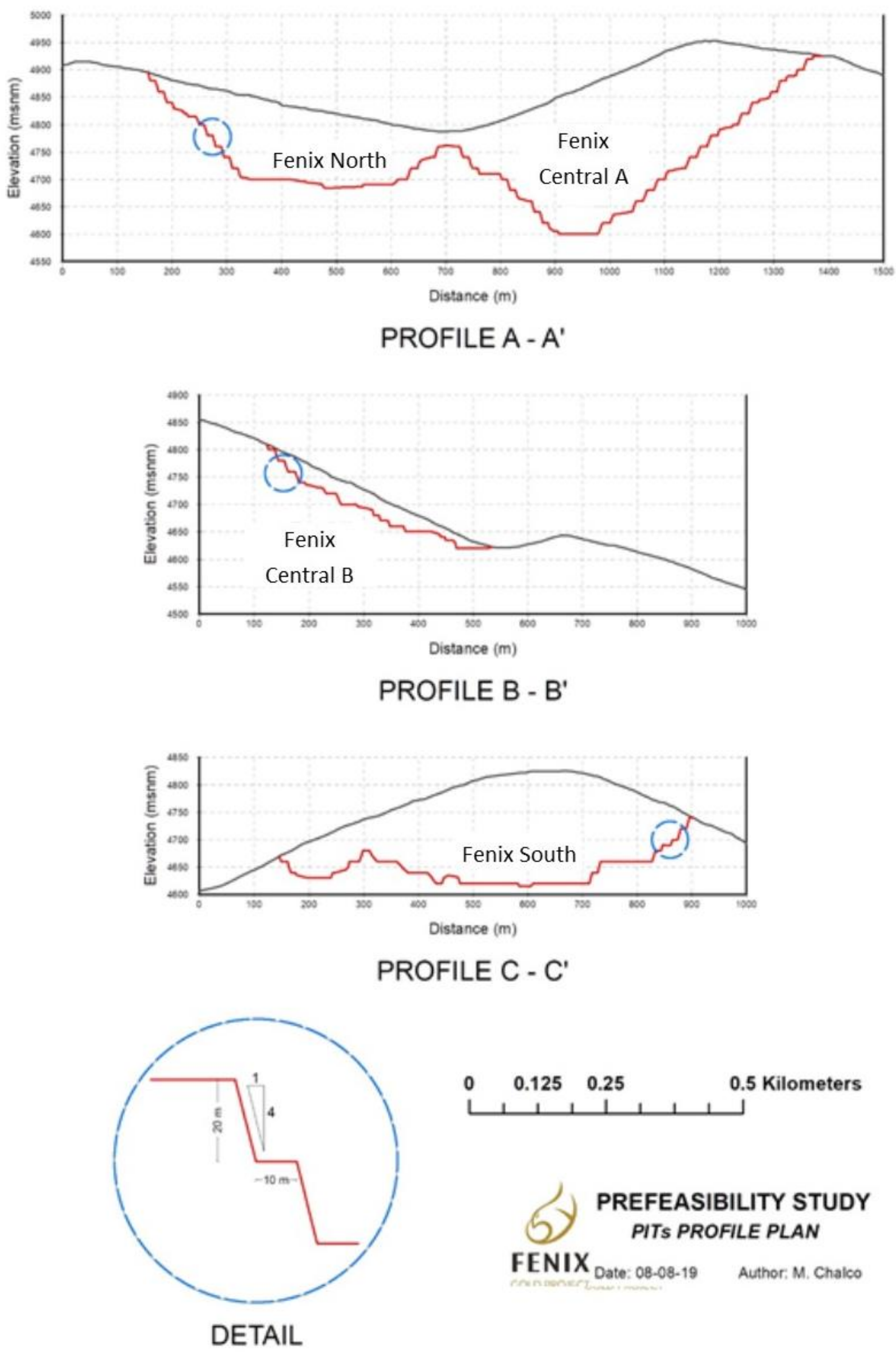


Figure 15-4: Example cross section (looking NW) through the final pit design

15.8 Mineral Reserve Statement

The Resource Estimate discussed in Section 14 was prepared using industry standard methods. Raul Espinoza (QP) has reviewed the reported resources, production schedules, and cash flow analysis to determine if the resources meet the CIM Definition Standards for Mineral Resources and Mineral Reserves, to be classified as reserves. Based on this review, the assessment of the Measured and Indicated Mineral Resource occurring within the final pit design contains mineralization (gold oxide) that can be classified as Proven and Probable Mineral Reserves.

The open pit Proven and Probable Reserves include existing and future stockpiles scheduled for processing and inventory.

The oxide ore will be sent to the leach pad for gold recovery by hydrometallurgical methods.

Mineral Reserves

The Mineral Reserve estimate is shown in Table 15-6 and is effective as of August 15, 2019. The Mineral Reserves are reported as in-situ dry million tonnes and include 3% mining dilution and 97% mining recovery using a cut-off grade of 0.24 g/t Au.

Table 15-6: Mineral Reserves (Mining Plus, 2019)

Reserve Category	Million Tonnes	Grade Au g/t	Contained Ounces Au x1000	Recoverable Ounces Au x1000
Proven	53	0.52	866	650
Probable	63	0.47	962	722
Proven and Probable	**116**	**0.49**	**1,828**	**1,372**

The Mineral Reserve Statement contains the total minable reserve for the deposits described in Section 15.1. The Mineral Reserve passed an economic test conducted on the production schedule. The results of the economic analysis are shown in Section 22.

16 MINING METHODS

Rio2 acquired the Cerro Maricunga Oxide Gold Project and renamed it Fenix in 2018.

In 2014, a Pre-Feasibility Study detailed a conceptual mine design for the Project that considered an 80,000 tpd operation that would produce approximately 3 Moz of Au over 13 years. The 2014 mine plan relied on the installation of a 149 km, capital intensive, water pipeline and associated environmental baseline studies. Permission for the pipeline is dependent on the negotiation of easements with numerous parties. Absence of a defined route and associated permitting timeline for the pipeline led to Rio2 exploring alternative project development options.

As such, Rio2 has developed a mine plan for a 20,000 tpd operation that negates the need to install and permit the water pipeline. The benefits include:

- Commencement of production in the shortest possible timeframe, estimated in Q4 2021, as the permitting footprint and associated requirements are greatly reduced.
- Minimizing the upfront capital requirements.
- Maximizing free cash flow by year five of the mine plan.
- Maintaining the potential to reconfigure and expand operations within the optimized pit shell.

The new mine plan makes use of an approved supply of trucked industrial water. This approach ensures that the environmental baseline study is focused within the area of the proposed operations and removes possible uncertainties related to the routing of a water pipeline and the associated environmental permitting. Environmental baseline studies for the area of operations are advanced and scheduled for completion by October 2019.

Rio2 recognizes that the revised mine plan does not necessarily optimize the NPV of the Project, but highlights that the intention is to reduce the upfront capital requirements, commencing production in the shortest possible timeframe, and maximizing free cash flow by year five. There is scope to reconfigure the mine design to expand operations within the optimized pit shell. The anticipated evolution of the mine is presented graphically from Figure 16-1 to Figure 16-3, and a computer generated representation of the mine layout is show in Figure 16-4.

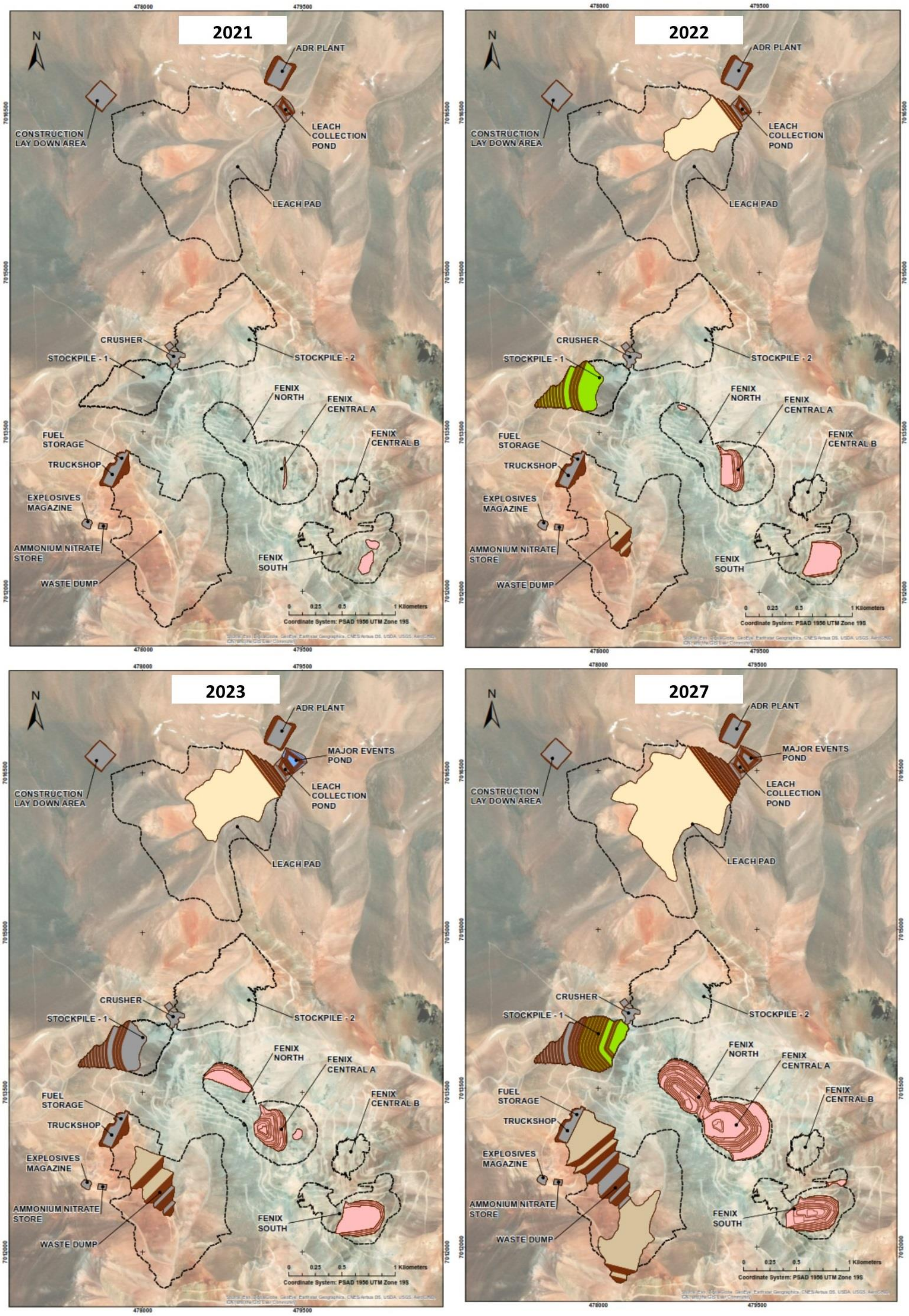


Figure 16-1: Evolution of the Mine Plan (2021 to 2027)

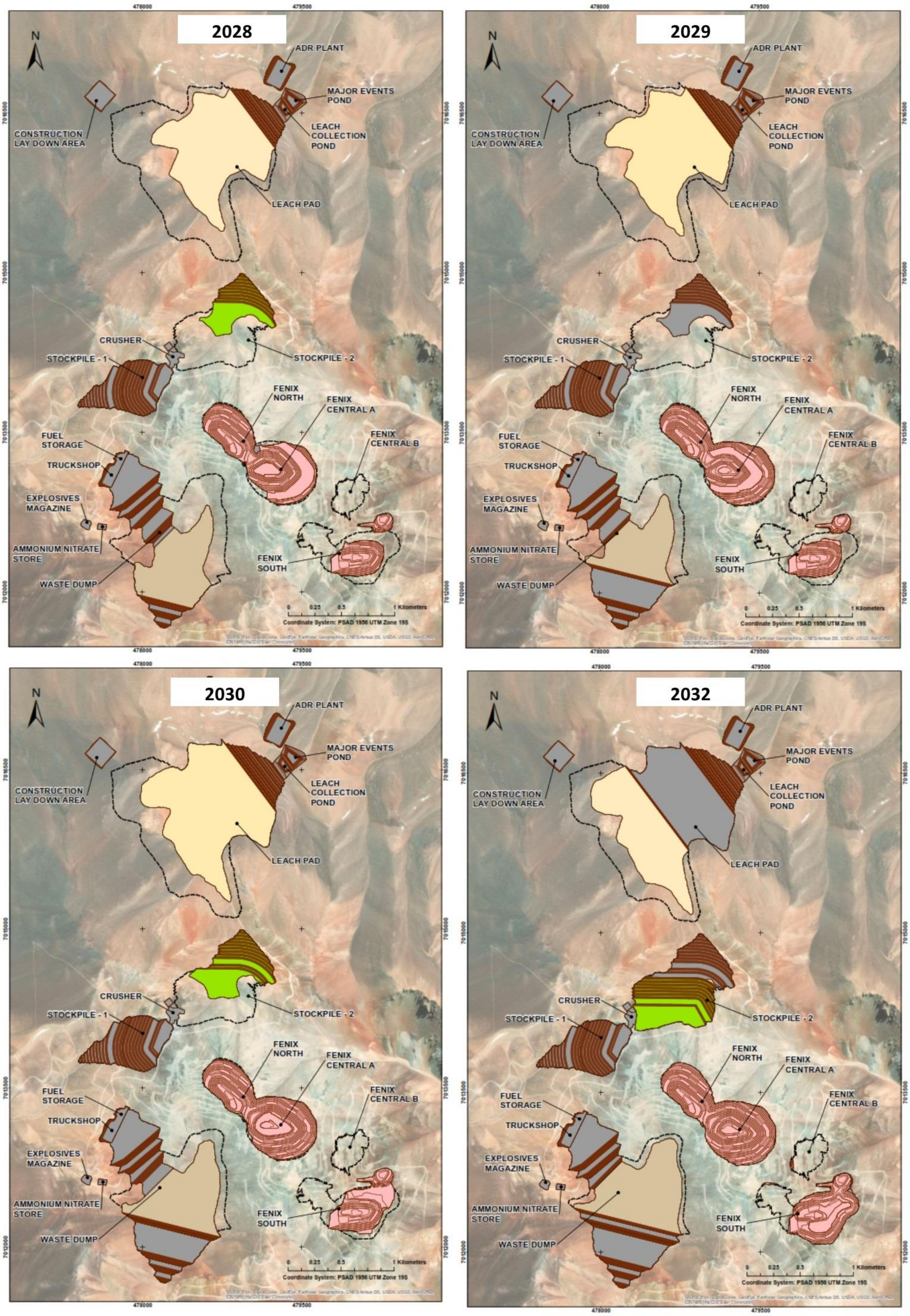


Figure 16-2: Evolution of the Mine Plan (2028 to 2032)

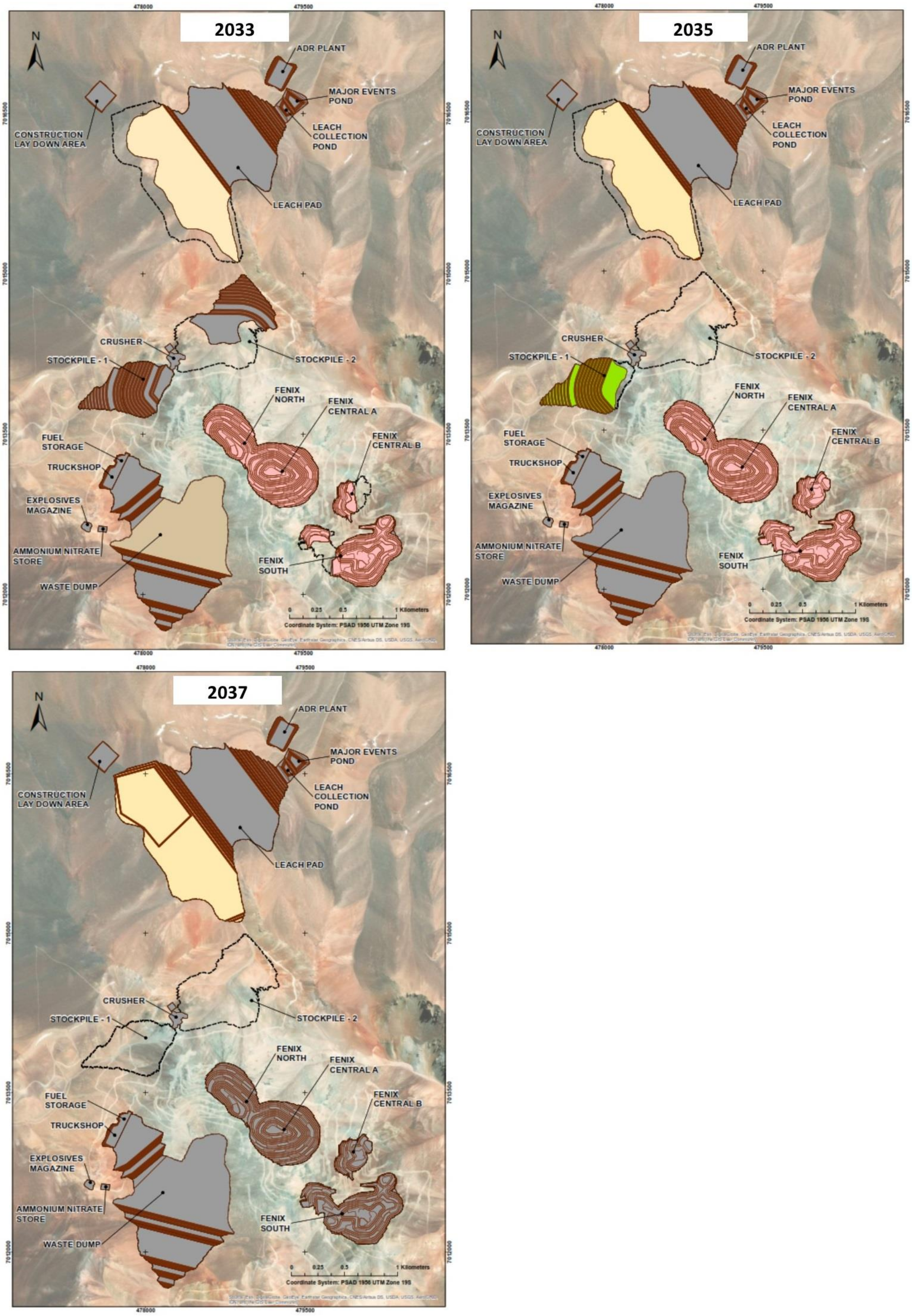


Figure 16-3: Evolution of the Mine Plan (2033 to 2037)

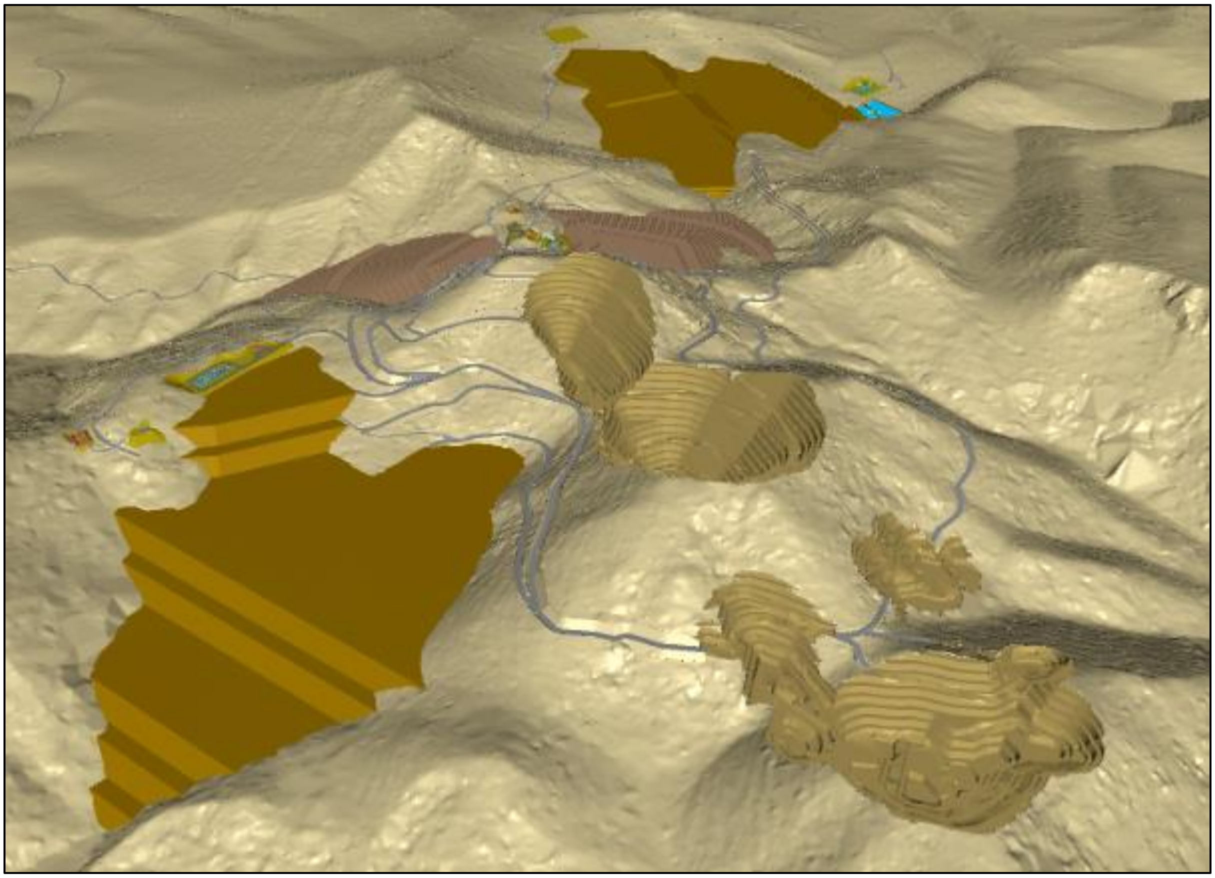


Figure 16-4: Computer generated representation of proposed mine layout

Parameters for the 2014 and the updated PFS are presented in Table 16-1.

Table 16-1: Comparison of assumed parameters (Mining Plus 2019)

	2014 PFS	2019 PFS
Cut-off Grade Au (g/t)	0.15	0.24
Contained Au/Oz (x1000)	3743	1829
Proven & Probable Reserve(Mt)	294.4	116
Gold Price	$1350	$1250
Stripping Ratio	1.76:1	0.81:1
Estimated Start of Production CAPEX and OPEX		
Initial CAPEX ($M)	398.9	111
Sustaining CAPEX ($M)	187.6	95
Operating Costs($/t)	6.88	11.1
IRR %		
Pre Tax	28.6	31.9
After Tax	25	27.40
NPV @ 5% $M		
Pre Tax	521	168
After Tax	409	121
Payback Period (years)		
Pre Tax	2.75	3
After Tax	3	4.3

16.1 Geotechnical Parameters

Historical geotechnical analysis at PFS level for the Cerro Maricunga Project (now the Fenix Gold Project) considered a larger and deeper pit than has been considered by Rio2. Raul Espinoza (QP) has reviewed historical analysis by DERK (DERK, 2014), which was used to support the 2014 PFS, and considers that this analysis is technically sound and suitable for PFS level studies.

With respect to the pit designs presented in this PFS, Raul Espinoza (QP) notes the following:

- Pits are within the footprint of the previous pit design.
- Pits are smaller and shallower than the previous pit design.
- Pits conform to slope angles and dimensioning deemed stable by DERK (Table 16-2).

Accordingly, the QP is satisfied that pit designs presented in this PFS will not reduce ground stability compared to previous designs. The reader is referred to the DERK 2014 report for further detail.

Table 16-2: Parameters established by DERK for slope stability

Bench - Shoulder Slope				Inter-Ramp Slope		Overall Slope	
Bench Height (m)	Bench Face Angle (°)	Rupture (m)	Shoulder (m)	Bench Face Angle (°)	Maximum Inter Ramp Height (m)	Ramp Width (m)	Maximum Overall Height (m)
20	75	5.4	9.5	53.4	160	14	330

Based on DERK´s geotechnical investigations, a 30 m safety berm is required when pit walls exceed 160 m; and in all cases, the optimized design complies with this geotechnical requirement.

16.2 Hydrogeology and Hydrology

Hydrogeology and hydrology considerations for this PFS have been taken from Atacama´s 2014 PFS study for the Cerro Maricunga Project (2014, PFS).

No groundwater has been encountered in any drilling. The behaviour of the aquifers in the basin or sub-basin in the project area should be studied to ensure any water resources of the area are not impacted by mining activities.

Hydrogeological information reported in the 2014 PFS indicated that the project area is underlain by minor aquifers of little hydrogeological importance and this was confirmed by Rio2 during its 2019 drilling campaign.

The results of underground water quality testing in the project area reflect local lithological characteristics and the rock mineralogy. Ground water has high conductivity that is indicative of it carrying significant dissolved solids. It is worth outlining that the 2014 PFS reports high concentrations of aluminium, cobalt, iron and manganese in groundwater.

16.3 Mine Production Schedule (Phasing)

Mine production schedules for the Fenix Gold Project are summarized in Table 16-3. Tonnages are reported as in-situ dry tonnes after the application of 3% ore loss and a 3% dilution factor.

The Fenix Gold Project consists of an open pit mine which will be developed using conventional drill and blast techniques, with an excavator and truck configuration. The mining rate is 20,000 tpd of ore to the crusher, with low-grade to one of two stockpiles, and waste to the waste dump. The mining rate has been determined based on the processing rate, which is primarily a function of the available water. The water supply will be delivered to site by a fleet of trucks from Copiapo. The water supply rate was determined primarily by assessing the practical and sustainable limit to the number of trucks that can make the continual cycle from Copiapo to the mine.

For this mining rate and cost structure, the cut-off grade was determined using Whittle software to be 0.24 g/t Au. To further improve the economics and increase cash flow, a medium grade and low-grade stockpile will be used. The medium grade material is between 0.30 – 0.4 g/t Au and low-grade material is between 0.24 – 0.30 g/t Au. Both stockpiles will be located adjacent to the crusher, however the low-grade stockpile will be located close to the pad and maybe treated directly as ROM mineral if metallurgical work indicates an acceptable recovery.

The first year of production has an average production rate of 12,000 tpd of material placed on the leach pad, which reflects the ramp up from 0 to 20,000 tpd. This is due to the starter pit initial phases being located on topographic high points where there is limited work space. The mined grade is 0.76 g/t producing 80 koz of recovered gold.

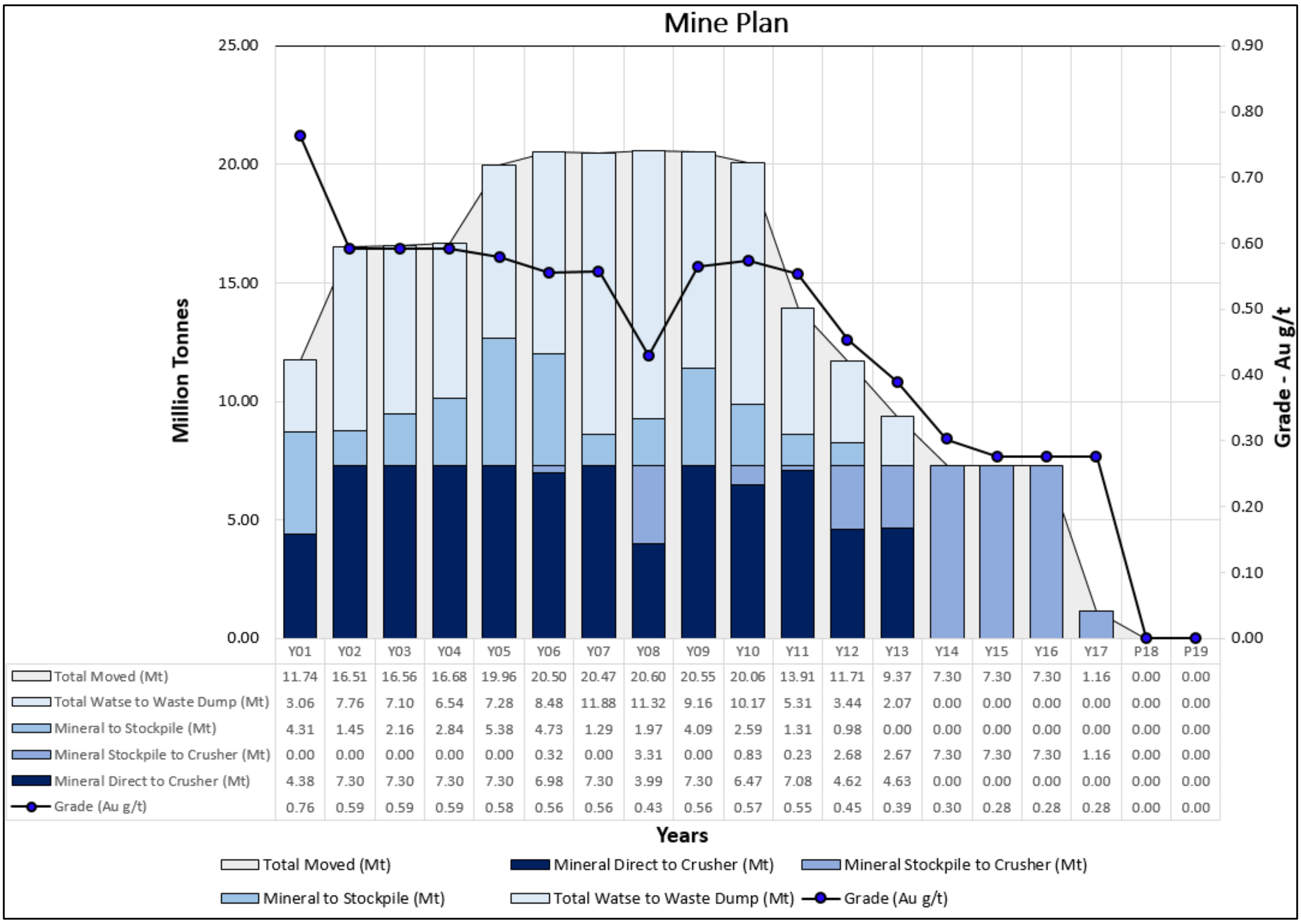
Mining operates at the full 20,000 tpd capacity between years 2 and 13, and if required, minor top up from stockpiles will ensure that the mine operates at the total mining rate (ore and waste), set at 20.5 Mtpa. The grade mined during this period is 0.536 g/t Au recovering an average of 94.4 koz each year.

Between years 12 and 13 approximately 30% of production is rehandled from the low and medium grade stockpiles to achieve a 20,000 tpd processing rate, and between years 14 to 17 feed to pad is from 100% stockpile rehandle, with an average grade of 0.285 g/t Au recovering an average 39.6 koz from the leach pad each year.

The projected production schedule for the Fenix Gold Project is presented in Table 16-3 and Figure 16-5.

Table 16-3: Projected Production Schedule for the Fenix Gold Project

Physicals	Units	Totals	Phase 1 12 ktpd	Phase 2 20ktpd												Phase 3 Stockpile			
Mine Plan			Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Mineral to crusher	Mt	81.9	4.38	7.3	7.3	7.3	7.3	6.98	7.3	3.99	7.3	6.47	7.08	4.62	4.63				
Grade	g/t	0.57	0.76	0.59	0.59	0.59	0.58	0.55	0.56	0.5	0.56	0.6	0.56	0.52	0.42				
Mineral to LG	Mt	33.3	4.31	1.45	2.16	2.84	5.38	4.73	1.29	1.97	4.09	2.59	1.31	0.98	0.23				
Grade	g/t	0.3	0.35	0.27	0.28	0.32	0.31	0.3	0.27	0.28	0.3	0.29	0.28	0.27	0.22				
Total Mineral	Mt	115.2	8.69	8.75	9.46	10.14	12.68	11.71	8.59	5.96	11.39	9.06	8.39	5.60	4.86				
	g/t	0.49	0.56	0.54	0.52	0.51	0.47	0.45	0.52	0.43	0.47	0.51	0.52	0.48	0.41				
Waste	Mt	93.55	3.06	7.76	7.1	6.54	7.28	8.48	11.88	11.32	9.16	10.17	5.31	3.44	2.07				
Strip Ratio	Ratio	0.81	0.35	0.89	0.75	0.64	0.57	0.72	1.38	1.90	0.80	1.12	0.63	0.61	0.43				
Reclaim from Stocks	Mt	33.1	0	0	0	0	0	0.32	0	3.31	0	0.83	0.23	2.68	2.67	7.3	7.3	7.3	1.16
Grade	g/t	0.3	0	0	0	0	0	0.59	0	0.35	0	0.33	0.33	0.33	0.33	0.3	0.28	0.28	0.28
Mineral to Pad	Mt	115.04	4.38	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	1.16
Grade	g/t	0.49	0.76	0.59	0.59	0.59	0.58	0.56	0.56	0.43	0.56	0.57	0.55	0.45	0.39	0.3	0.28	0.28	0.28
Gold Placed	Moz	1.83	0.11	0.14	0.14	0.14	0.14	0.13	0.13	0.1	0.13	0.13	0.13	0.11	0.09	0.07	0.07	0.07	0.01
Recovery	4" Crush																		
Recovered koz	75%	1,370	0.081	104	104	104	102	98	98	76	99	101	97	80	69	53	49	49	8



	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	P18	P19
Total Moved (Mt)	11.74	16.51	16.56	16.68	19.96	20.50	20.47	20.60	20.55	20.06	13.91	11.71	9.37	7.30	7.30	7.30	1.16	0.00	0.00
Total Watse to Waste Dump (Mt)	3.06	7.76	7.10	6.54	7.28	8.48	11.88	11.32	9.16	10.17	5.31	3.44	2.07	0.00	0.00	0.00	0.00	0.00	0.00
Mineral to Stockpile (Mt)	4.31	1.45	2.16	2.84	5.38	4.73	1.29	1.97	4.09	2.59	1.31	0.98	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mineral Stockpile to Crusher (Mt)	0.00	0.00	0.00	0.00	0.00	0.32	0.00	3.31	0.00	0.83	0.23	2.68	2.67	7.30	7.30	7.30	1.16	0.00	0.00
Mineral Direct to Crusher (Mt)	4.38	7.30	7.30	7.30	7.30	6.98	7.30	3.99	7.30	6.47	7.08	4.62	4.63	0.00	0.00	0.00	0.00	0.00	0.00
Grade (Au g/t)	0.76	0.59	0.59	0.59	0.58	0.56	0.56	0.43	0.56	0.57	0.55	0.45	0.39	0.30	0.28	0.28	0.28	0.00	0.00

Figure 16-5: Fenix Gold Project, mine plan

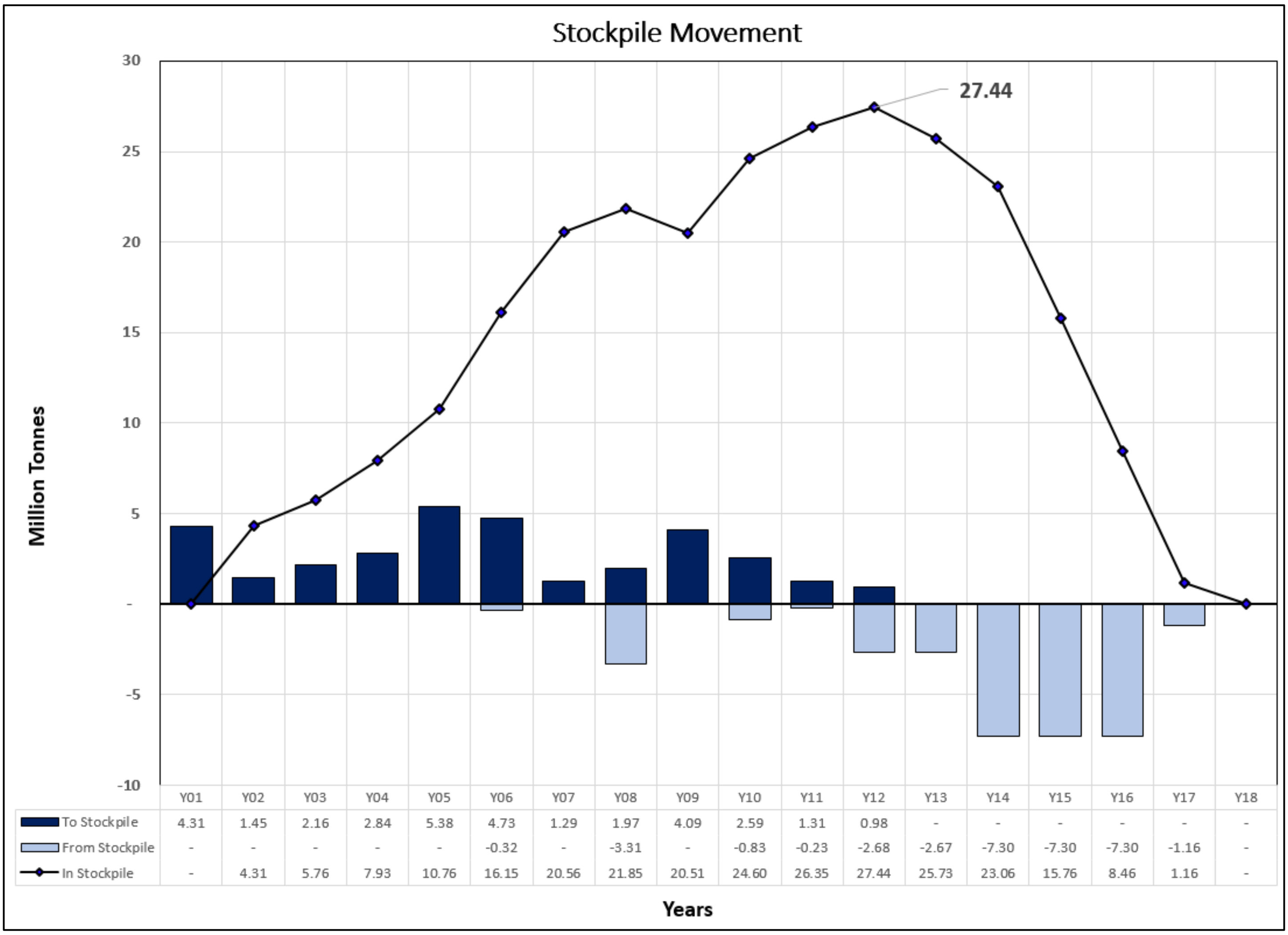


Figure 16-6: Fenix Gold Project, stockpile movement

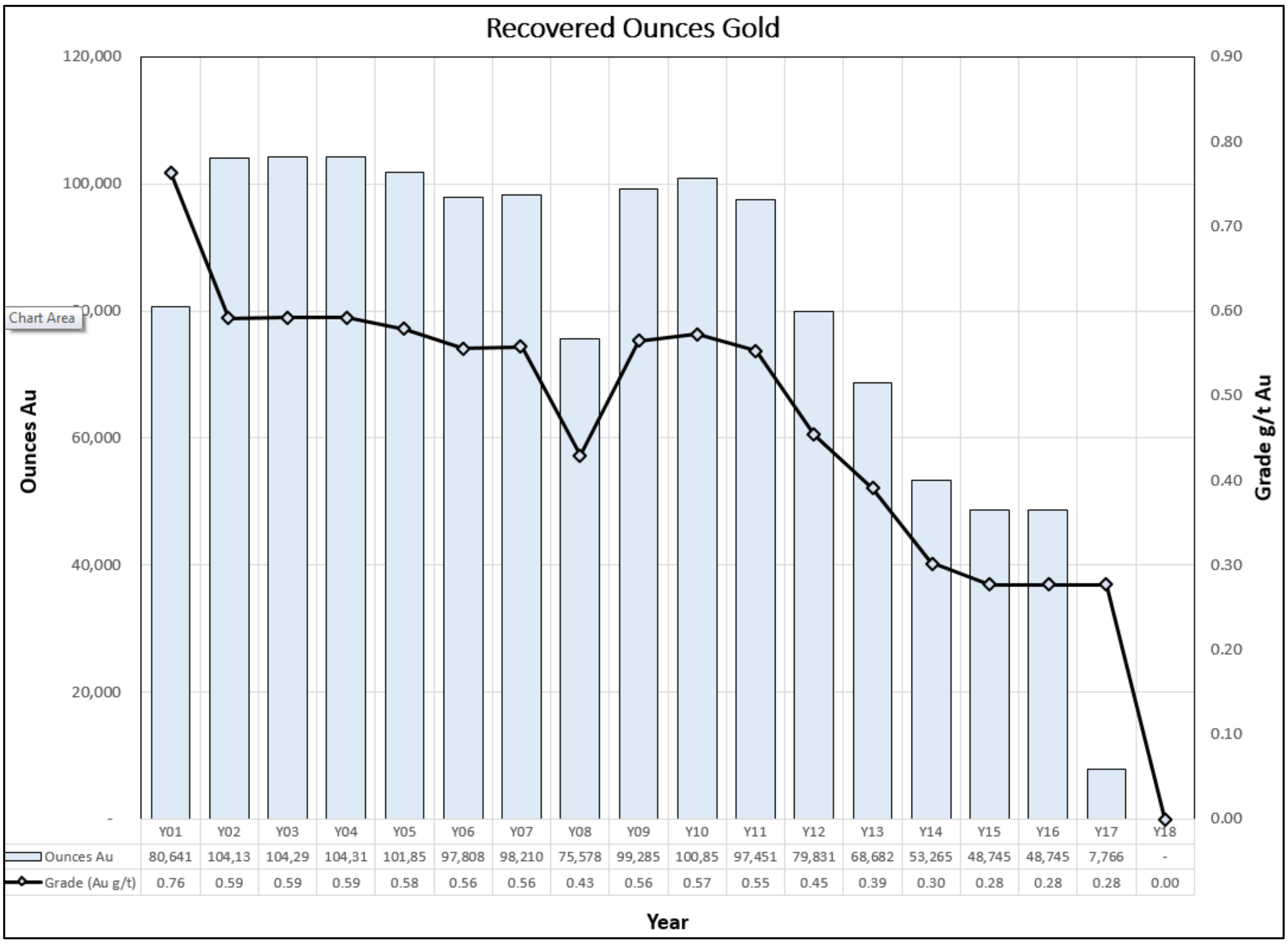


Figure 16-7: Fenix Gold Project, recovered ounces

16.3.1 Alliance Mining Contract

Rio2 is considering an alliance style mining contract to mine and manage all the material handling to the pad, stockpiles and waste dump. This style of contract is similar to a partnership to achieve a specified scope of work with shared risk on equipment, personnel and materials. This is an 'open book' partnership with an underlying premise of 'no conflict or claims'. The contractor charges a fee as a percentage of the overall in-scope costs. The fee structure is designed to give the contractor an incentive to manage a high level of safety and environmental compliance and to keep costs down.

16.4 Mining Equipment

16.4.1 Mining Fleet

The proposed Mining fleet has two DM45 drills or equivalent, a total of 5 x Cat 90 t Excavators, and 3 x CAT 966 Loaders for re-handling, the fleet also includes 53 x 43 t payload haul trucks and associated ancillary fleet.

This mining contractor will purchase and mobilize the fleet to site.

The main equipment list and distribution by year required to meet the mine plan are shown in Table 16-4.

The total required machine hours are calculated based on machine availabilities, productivities, and haul profiles generated from the mine design and layout.

Table 16-4: Primary Mining Equipment (Mining Plus, 2019)

Primary Mining Fleet Summary	Max Count	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Drill: DM45	2	2	2	2	2	2	2	2	2	2	2	2	1	1	0	0	0	0
Excavator: Cat 390	5	3	4	4	4	4	5	5	5	5	5	4	3	3	2	2	2	1
Tip Truck: 8x4 43t	53	31	42	40	38	46	49	53	49	46	53	41	32	27	15	17	18	4
Front-end Loader C966	2	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	1

16.4.2 Drill and Blast

The Pit will be drilled and blasted on 10 m high benches using 155 or 171 mm diameter blast holes and a powder factor of 0.42 - 0.61 Kg/bcm (kilograms per bank cubic meter). Loading of ore and waste would be with 70 t – 90 t excavators into 43 t payload rigid frame 8m x 4m heavy tipper dump trucks. Ore will be hauled to the primary crusher and waste is hauled to the waste dump.

Technical drilling parameters considered for the Fenix Gold Project are given in Table 16-5.

Table 16-5: Technical Drilling Parameters (STRACON, 2019)

	Units	Ore Rock	Waste Rock
Hole Diameter	Inches	6 3/4	6 3/4
Drilling Pattern	Burden/Spacing (m)	4.6/5.3	5.2 / 6.0
Bench Height	Metres	10	10
Sub Drilling	Metres	1.2	1.2
Re-Drilling	m	2.70%	2.70%
Penetration Rate	m/hr	43.0	43.0

An Unsensitised Gassable Bulk Emulsion Matrix is the primary explosive proposed for blasting. The emulsion matrix is shipped as an oxidizer and must be sensitized with a chemical gassing technology to become detonable prior to use. The mine is being developed using a conventional load and haul truck open pit mining method. Ore grade control has been considered in the mining method, so proposed mining will be conducted in 20 m benches with double blasting (2 x 10 m) to minimize dilution and ore losses. The operative bench height is in the order of 11-12 m due to the swell factor applied after the blasting. Table 16-6 shows the operative loading heights.

Table 16-6: Loading Technical Parameters (STRACON, 2019)

Items	Units	Value
Nominal Bench Height	m	10.0
Operative Bench Height	m	11.0 - 12.0

16.4.3 Loading, Hauling, and Ore Rehandling

The Fenix Gold Project will operate under an over-trucking model, which means that the production will be limited by the loading fleet, not the truck availability.

The running surface on the haul roads of 14 m wide has been designed to be at least three times the width of haulage trucks (3.5 m) as this is aligned to international operational practices and Chilean safety regulations. Haul road designs include a 0.5 m drain either side of the running surface and a safety berm, and so considering the running surface, drains and safety berms, haulage roads are 24 m wide.

A fleet of two drills, three excavators (390F with a 14.25t bucket), 43 t payload haul trucks and associated load and the haul support equipment will be purchased by the contractor. The equipment list and distribution by year is shown in Table 16-7 and Figure 16-8.

Table 16-7: Primary Mining Equipment (Mining Plus, 2019)

Year	Truck 8x4 ht43t	Exc-390
Yr 1	31	3
Yr 2	42	4
Yr 3	40	4
Yr 4	38	4
Yr 5	46	4
Yr 6	49	5
Yr 7	53	4
Yr 8	49	5
Yr 9	46	4
Yr 10	53	5
Yr 11	41	4
Yr 12	32	3
Yr 13	27	3
Yr 14	15	2
Yr 15	17	2
Yr 16	18	2
Yr 17	4	1

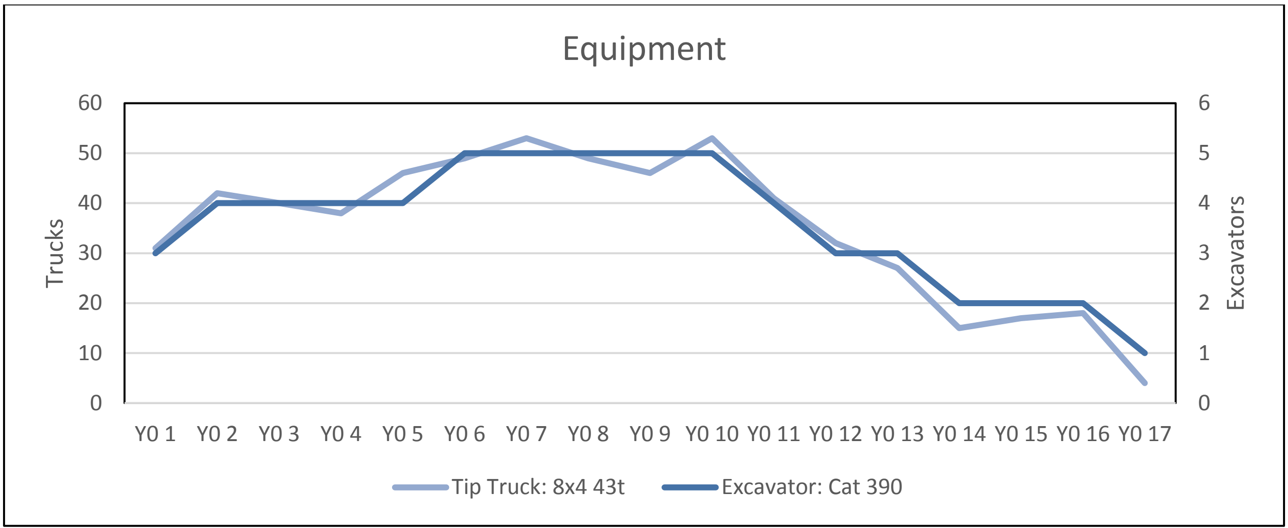


Figure 16-8: Primary Mining Equipment Units (Mining Plus, 2019)

16.4.4 Ancillary Work and Equipment

A general overview of the ancillary equipment needed for the production stages of mining is outlined in Table 16-8.

Table 16-8: Ancillary Equipment Fleet Size (equipment count is maximum for any year) (STRACON, 2019)

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Excavator C336	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Front end Loader C966	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2
Bulldozer CD8T	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0
Bulldozer CD6T	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheeldozer 834	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader C14M/140K	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Backhoe C420	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck 6000G	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Fuel Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Lube Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Roller 10t	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Screen	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0	0
Pickup	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	1	1
Lighting Plant	9	7	8	8	8	8	9	8	9	8	9	8	6	6	3	3	3
Operators	303	211	259	251	243	275	284	303	290	272	300	245	203	184	101	105	110
Mechanics	70	62	66	62	62	70	70	70	66	66	66	70	70	66	48	46	46

16.5 Leach Pad, PLS Pond and Major Event Pond

Anddes (2019) designed the leach pad, PLS pond and major event pond.

The leach pad has been designed in four phases (Figure 16-9) and has a combined capacity of 129 Mt (Table 16-9). The base of the leach pad is inclined at 2% towards the PLS and major event pond.

The PLS pond will have a double geomembrane liner system and will have an installed capacity of 40k m^3 from the first year of the mine plan.

The major event pond will have a single liner and its capacity will increase in phases: 20k m3 in year 2, 50k m^3 in year 5, 90k m^3 in year 10 and 120k m^3 in year 17.

When Phase 4 has been constructed, the combined leach pad, PLS pond and major event pond will extend over a combined area of 159 hectares. The leach pad, PLS Pond and Major Event Pond relative to the pits, is presented in Figure 16-10.

Table 16-9: Parameters of the leach pad, PLS pond and major event pond

Parameter	Unit	Value
Designed Leach Pad Capacity	Mt	129
Typical Bench Height	m	10
Lift Slope		1.4H:V
Berm Width	m	10
Overall Slope		2.5H:V
PLS Pond Capacity	m^3	40,000
Major Event Pond	m^3	120,000

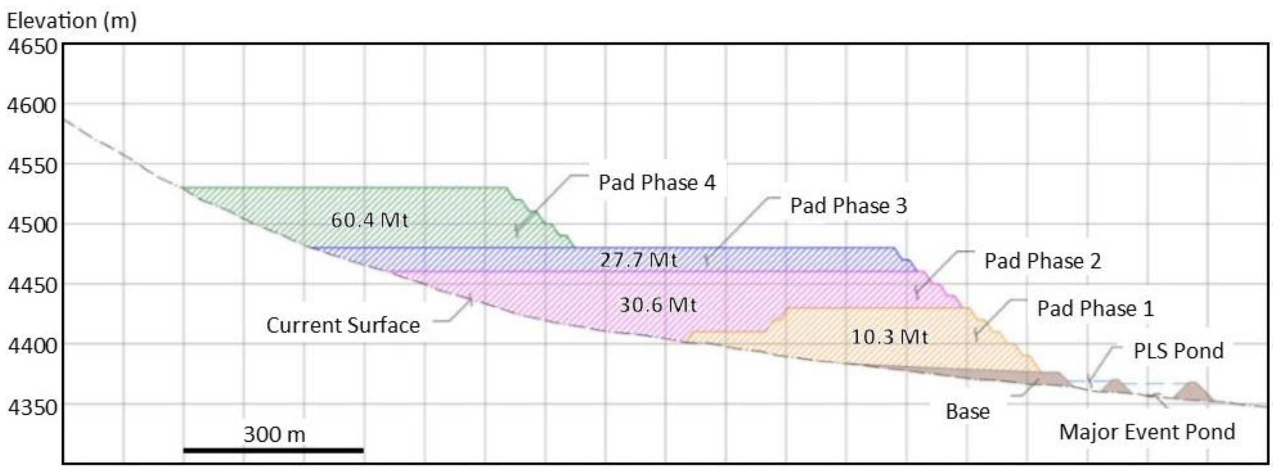


Figure 16-9: Cross-section of proposed leach pad, PLS pond and major event pond

16.6 Waste Storage Area

Anddes designed the waste storage area and evaluated the stability of proposed waste storage areas (Anddes, 2019). Geotechnical recommendations for the waste storage area design are summarized in Table 16-10.

Table 16-10: Waste Dump Design Parameters (Anddes, 2019)

Parameter	Unit	Value
Capacity	Mt	100
Layer Height	m	40
Batter Angle	ratio	1.4H:1V
Berm Width	m	44
Overall Angle	ratio	2.5H:1V

The proposed waste storage area, relative to the pits, is presented in Figure 16-10.

16.7 Low-grade Ore Stockpiles

Two stockpiles have been planned for storing low-grade ore between Year 1 and 12 with a total required capacity of 25.7 Mt. From Year 13 ore will be recovered from the stockpiles and will be taken to the crusher until Year 17.

Stability analysis of the global slope has been performed for the two stockpiles and proper stability conditions have been found for both of them. The dimensions of stockpiles 1 and 2 are summarized in Table 16-11. A step out berm of 50 m is required for every 100 vertical meters.

The location of the Stockpile -1 and Stockpile - 2 is shown in Figure 16-10.

Table 16-11: Stockpile Design Parameters (Anddes, 2019)

Parameter	Unit	Stockpile 1	Stockpile 2
Capacity	Mt	15.7	31.9
Layer Height	m	100	100
Batter Angle	ratio	2H:1V	2H:1V
Berm Width	m	10	10
Overall Angle	ratio	3H:1V	3H:1V

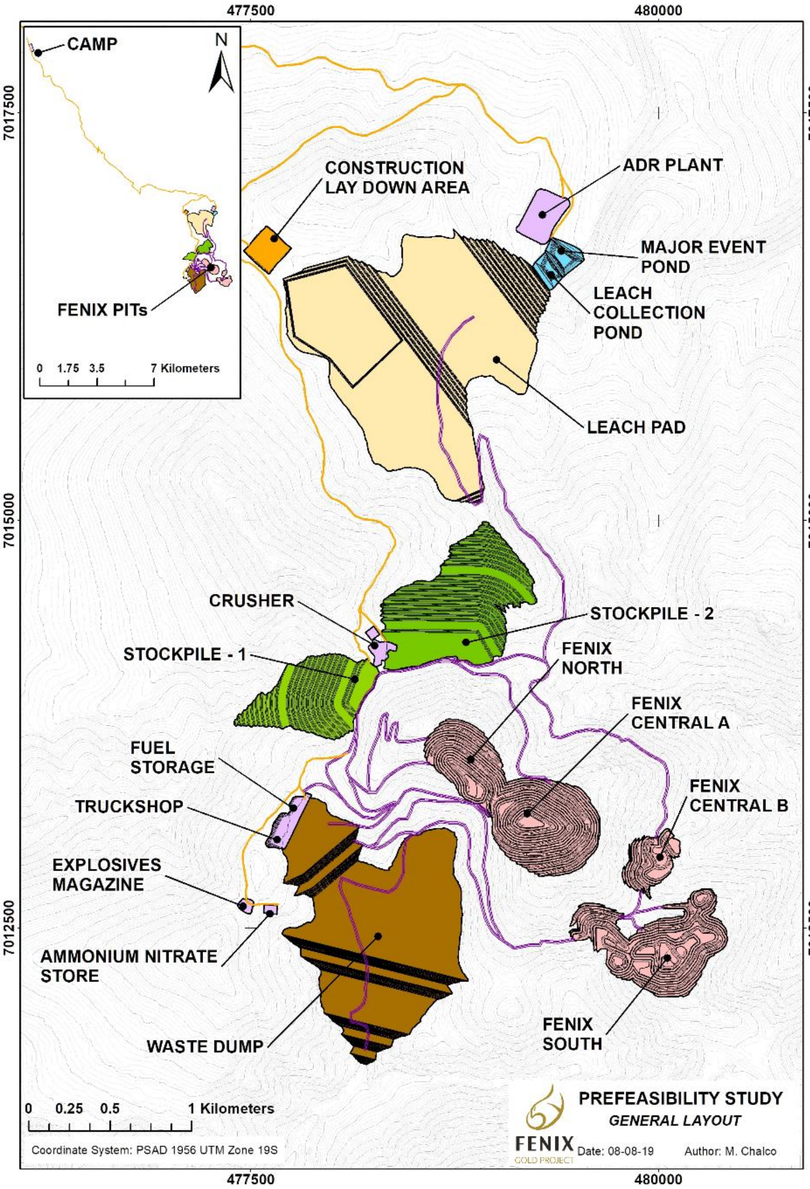


Figure 16-10: Leach Pad, PLS Pond, Major Event Pond and Waste Storage Areas related to other mine infrastructure

16.8 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.

16.9 Salaried Staff

Mine salaried staff requirements over the project life are shown in Table 16-12. 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 shown in Table 16-12. 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 16-12: Salaried Staff Labour Requirements (Mining Plus, 2019)

Area	$/Yr	PP	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Administration and Accounting		**10.1**	**8**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**8**	**8**	**8**	**7**
Superintendent of administration and accounting	150,742	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Head of administration and accounting mine	75,843	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1	1	1	1
Mine administration & accounting coordinator	53,262	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Financial accountant	70,340	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Accounting and tax analyst	57,299	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Accounting assistant	27,716	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Administrative assistant and reception - Copiapo office	27,716	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Camp assistant	22,994	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Pickup driver	22,994	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Civil Construction		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**0**	**0**	**0**	**0**
Head of civil construction	75,843	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Civil construction engineer	65,188	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Contracts and Logistics		**11**	**8**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**8**	**8**	**8**	**5**
Head of logistics	75,843	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Contracts administrator	65,188	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Head of Warehouse	70,340	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Purchaser	57,299	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Warehouse supervisor	51,254	1	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Warehouse assistant	27,716	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Dispatcher	25,355	3	2	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	1
Costs & Budgets		**3**	**3**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**2**	**2**	**2**	**2**
Head of costs and budgets	75,843	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine cost accountant	57,299	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Cost and budget analyst	57,299	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Equipment and productivity control assistant	25,355	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Mine Geology		**10**	**7**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**4**	**4**	**4**	**3**
Mine geology superintendent	150,742	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Mine geologist	65,188	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Modelling geologist	65,188	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Ore control technician	34,883	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	0
Blast hole sampler	19,452	3	2	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2
Logging and mapping assistant	19,452	1	1	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0
Management		**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Mine manager	247,217	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Environment		**7**	**5**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**4**	**4**	**4**	**4**
Superintendent environment	150,742	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Head of environment	70,340	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1	1	1	1
Environmental coordinator	57,299	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Environmental technician	37,354	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Water treatment assistant	19,452	3	2	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2
Community Relations		**3**	**2**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**2**	**2**	**2**	**2**
Head of community relations	70,340	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Communication and information assistant	25,355	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Mine Maintenance		**10**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**8**	**8**	**8**	**7**
Mine maintenance superintendent	150,742	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Mine maintenance chief	75,843	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Head of maintenance workshop	70,340	3	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	1
Maintenance analyst	57,299	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Monitoring supervisor	51,254	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance programmer	51,254	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
AMT assistant	25,355	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Operations		**11**	**11**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**7**	**7**	**7**	**7**
Mine superintendent	150,742	1	0	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Head of geotechnical	70,340	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Head of mine	75,843	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shift boss mine	70,340	3	3	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2
Heavy equipment monitor	41,765	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0
Drilling and blasting technician	49,356	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Auxiliary services technician mine	41,765	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Geotechnical assistant	19,452	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Organization and methods		**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**0**	**0**	**0**	**0**
Head of organization & methods	70,340	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Mine Planning		**7**	**5**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**4**	**4**	**4**	**3**

Area	$/Yr	PP	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17
Mine planning superintendent	150,742	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Head of mine planning	75,843	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1	1	1	1
Head of topography	70,340	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Mine planning engineer	65,188	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	0
Surveyor	34,883	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Topography assistant	19,452	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Patrimonial Security		**5**	**3**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**3**
Head of security	70,340	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Security supervisor - gold room	51,254	4	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Occupational Health and Safety		**6**	**5**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**6**	**6**	**6**	**3**
Occupational health & safety superintendent	150,742	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Security engineer	65,188	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Emergency response supervisor	51,254	2	0	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	0
Systems		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**1**	**1**	**1**	**1**
Head of IT	70,340	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Information systems and communications coordinator	53,262	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Human Talent		**8**	**6**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**5**	**5**	**5**	**3**
Head of labor relations	75,843	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Communications & culture analyst	57,299	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Recruitment and selection coordinator	57,299	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Training Coordinator	57,299	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	0
Payroll coordinator	57,299	2	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Social worker	57,299	2	1	3	3	3	3	3	3	3	3	3	3	3	3	1	1	1	1
Total Salaries - Staff (INDIRECT)		**97**	**80**	**110**	**110**	**110**	**110**	**110**	**110**	**110**	**110**	**110**	**110**	**110**	**110**	**65**	**65**	**65**	**51**

17 RECOVERY METHODS

The Fenix Gold Project has approximately 116 million tonnes of ore with an average grade of 0.49 g/t Au, which will be processed at a maximum rate of 20,000 tpd (7.3 million tonnes per year), giving a life of mine of approximately 16 years.

The results of the metallurgical test work carried out between 2010 and 2017 show that the mineral in the Fenix Gold Project is amenable for the recovery of gold by heap leaching.

The mineral will be mined by open pit methods; it will be crushed to 80% passing 4" in a primary crushing circuit and transported in trucks to heap leach pads with a lift height of 10 m. The feed will be leached with a dilute sodium cyanide solution and the gold will dissolve, the gold will be recovered from the pregnant leach solution (PLS) in an adsorption circuit with activated carbon and then recovered in pressure desorption and electro-deposition circuits. The electrolytic precipitate will be filtered and sent to a retort furnace and finally smelted in a furnace to obtain doré bars. The project includes the following unit operations:

- Crushing
    - Primary crushing.
    - Mineral transport and stacking.
- Heap leach
    - Management of solutions.
- ADR plant
    - Adsorption.
    - Desorption and electro-deposition.
    - Acid wash.
    - Thermal regeneration.
    - Smelting.

The criteria used for the design of the plant are summarized in Table 17-1. Figure 17-1 shows a general flow diagram of the plant. The design considers a recovery of 75% gold in the leaching process.

Table 17-1: Main Design Criteria for the Plant (Mining Plus, 2019)

Description	Unit	Value	Source
General			
Reserves	t	116,000,000	Defined by the client
Daily throughput	t/d	20,000	Defined by the client
Au design head grade	g/t	0.49	Defined by the client
Crushing Plant			
Availability	%	75	Defined by the client and HLC S.A.C.
Operating	h/d	18	Defined by the client and HLC S.A.C.
Density of the ROM feed	t/m^3	1.7	Defined by the client
Feed size to leach (P80)	Inches	4	Defined by the client and HLC S.A.C.
Mineral moisture	%	3	Defined by the client
Leach pad			
Lift height	m	10	Defined by the client
Nominal flow (Operation)	$L/h/m^2$	10	Defined by the client and HLC S.A.C.
Cyanide strength	ppm	100	Defined by the client and HLC S.A.C.
Leaching time	days	90	Defined by the client and HLC S.A.C.
Solution application method	-	Drip	Defined by the client and HLC S.A.C.
Recovery			
Gold	%	75	Result of metallurgical test work.
Reagents for leaching			
Sodium cyanide	kg/t	0.27	Result of metallurgical test work.
Lime	kg/t	4	Result of metallurgical test work.
Adsorption			
Gold concentration in the solution	ppm	0.27 to 0.30	Defined by the client and HLC S.A.C.
Number of circuits	No.	1	Defined by the client and HLC S.A.C.
Number of columns per circuit	No.	5	Defined by the client and HLC S.A.C.
Concentration of gold in activated carbon	g/t	2,000	Defined by the client and HLC S.A.C.

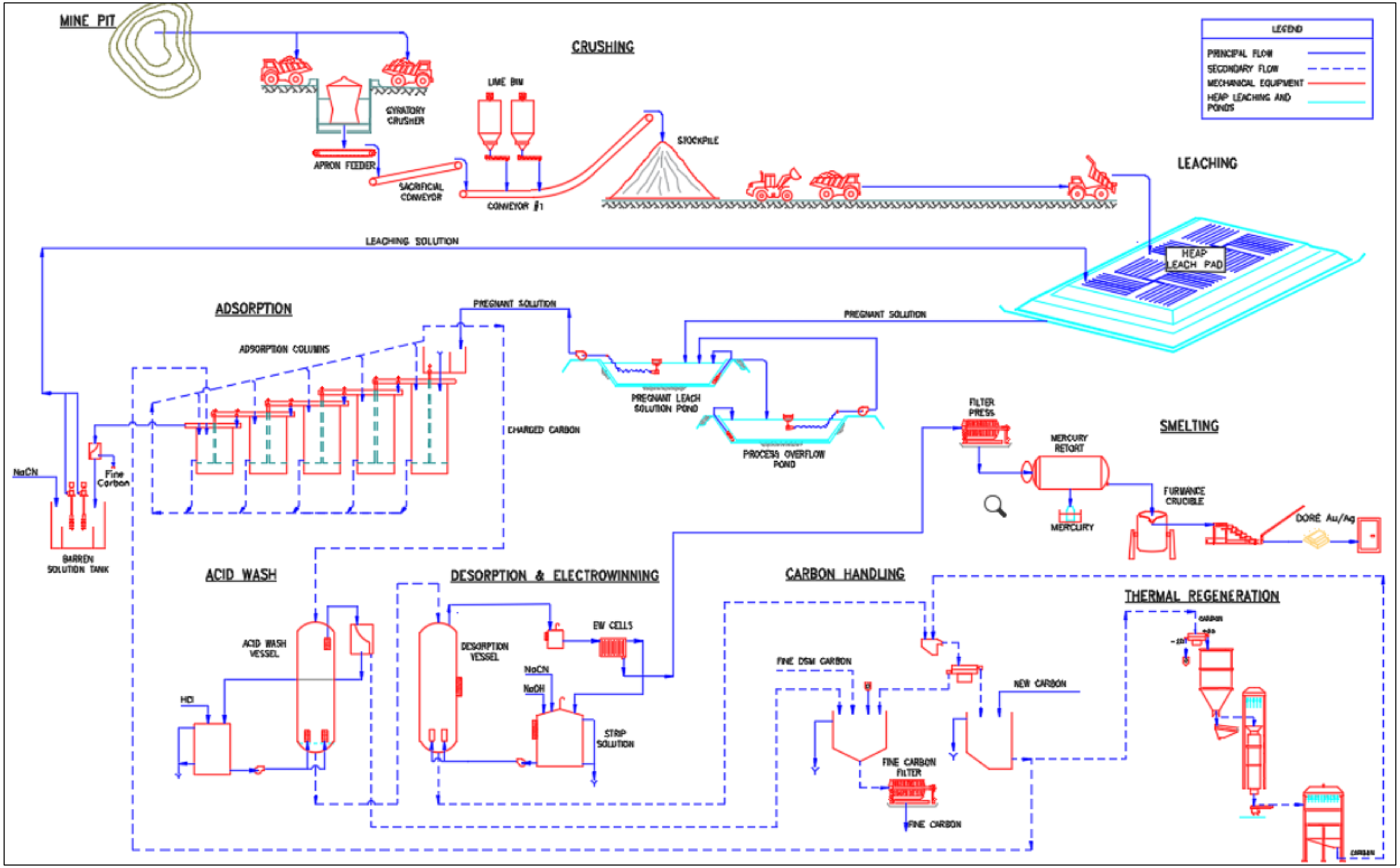


Figure 17-1: Conceptual Flow Diagram of the Process

17.1 Crushing Plant

The location and general layout of the crushing plant are shown in Figure 17-2 and a computer generated representation of the crushing plant is shown in Figure 17-3.

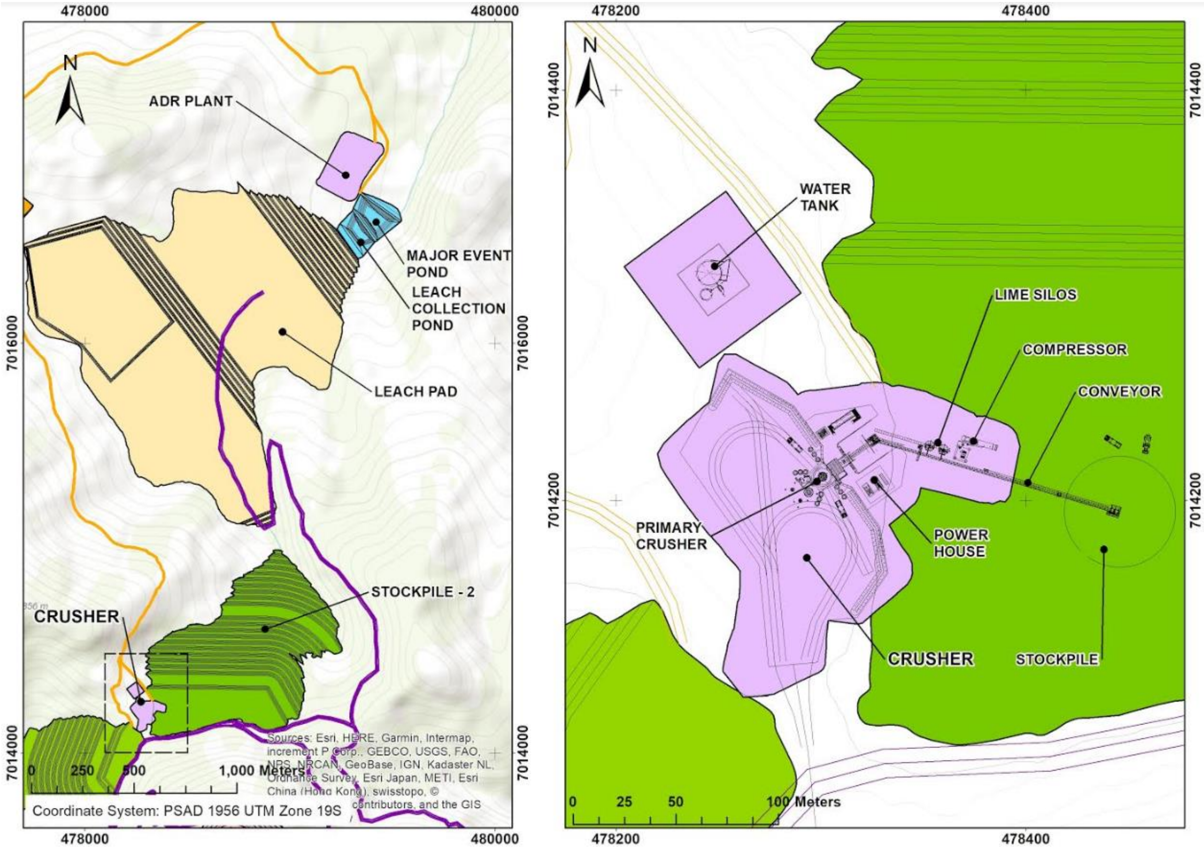


Figure 17-2: Location and general layout of the crushing plant

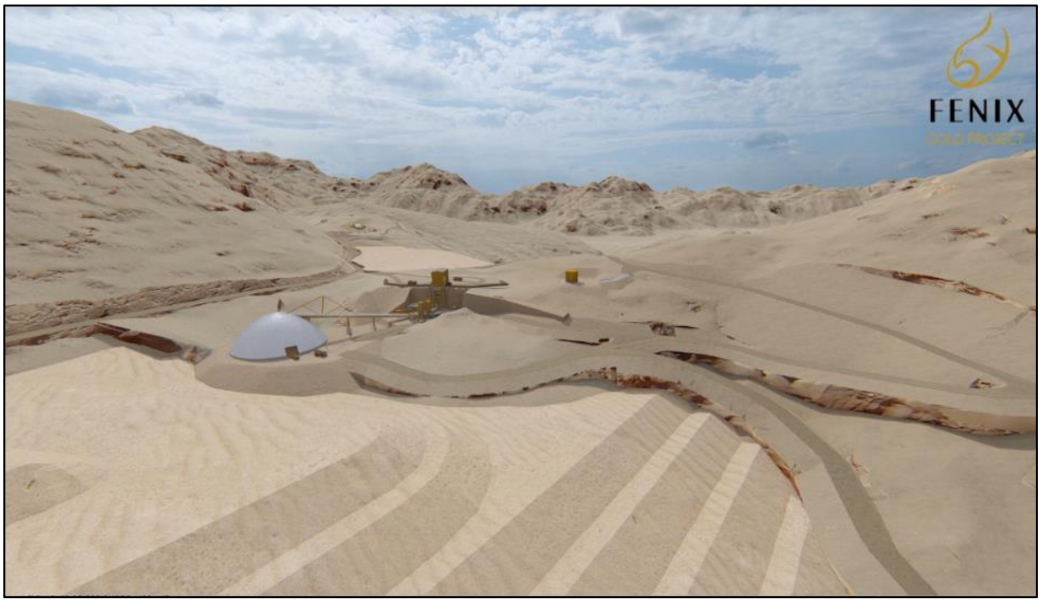


Figure 17-3: Computer generated representation of crusher plant

17.1.1 Primary Crushing

The mineral from the mine (ROM) will be transported by 43 t capacity trucks (Actros or Volvo type) to the feed hopper of the primary gyratory crusher (Metso 42' x 63', or equivalent) at a rate of 1,111 t/h with an availability of 75% (18 hours per day of operation). The feed is crushed to a P80 size of 4”. The feed will then pass to the crusher discharge hopper from where it will be discharged by an apron feeder to a short sacrificial conveyor. This conveyor will transfer the feed to conveyor No. 1 that will transport the crushed mineral to the stockpile. An electromagnet will be located on the sacrificial belt to collect tramp metal from the mine. A rock breaker will be installed at the primary crusher feed hopper to break oversize rocks and prevent blockages in the crusher. A dust suppression spray system will be installed at the crusher feed hopper to reduce the amount of dust in the atmosphere.

17.1.2 Stockpile and Lime Addition

The crushed feed is transported by conveyor No. 1 to a stockpile, which will have a total storage capacity of approximately 30,000 t. Lime will be added to conveyor No. 1 at a rate of 4.0 kg/t of feed using a variable speed screw feeder. The stockpiled material is loaded by front-end loaders into trucks for transport to the leach pads.

The lime is delivered to the mine in dry bulk carrier trucks. The lime is transferred pneumatically from the trucks to two 200 t capacity lime storage silos. Each silo will be equipped with an activator that will supply a continuous flow of lime to the screw feeder that will dose the lime onto conveyor belt No. 1. The primary crushing, stockpile and lime addition stages are shown in Figure 17-4.

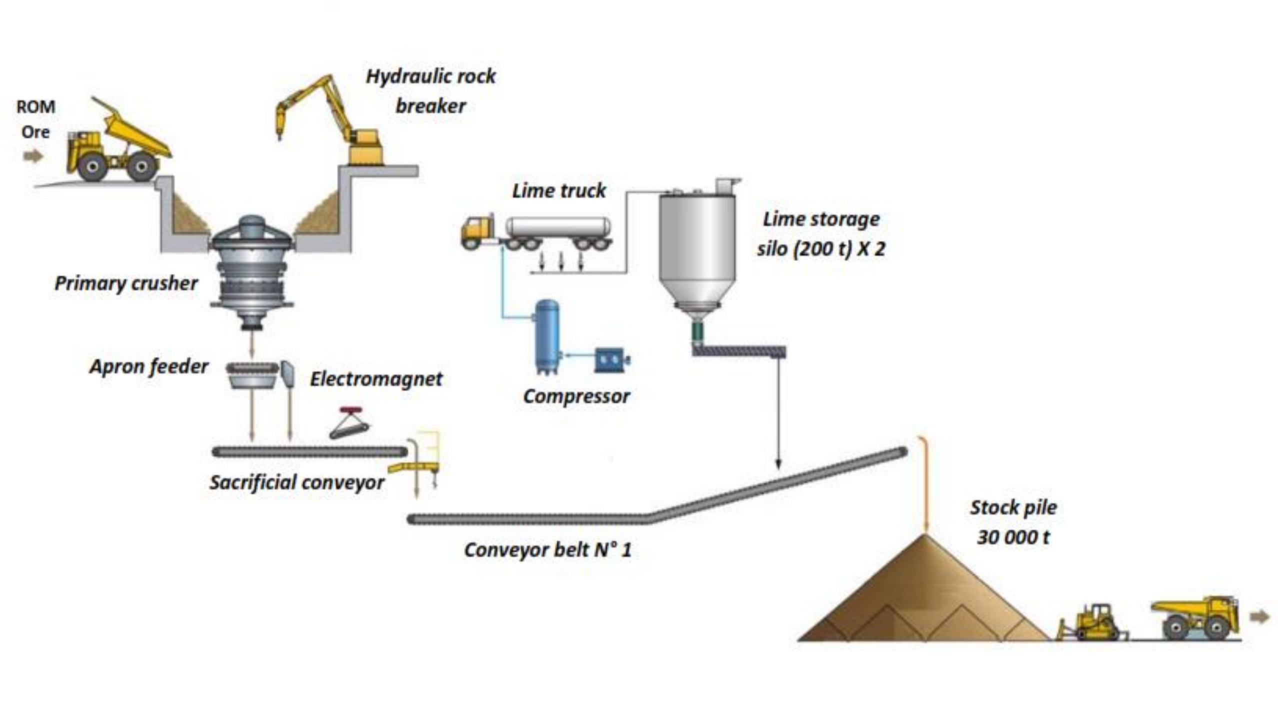


Figure 17-4: Crushing Stage and Stockpile

17.2 Heap Leaching

17.2.1 Hauling and Stacking Ore

The crushed feed is then transported from the crushing plant to the heap leach pad by mine trucks. The feed will be stacked in lifts of 10 m. Access ramps are built between lifts.

After the trucks unload the feed, they will leave high mounds of feed that is pushed to the edge of the pile by a front loader or a crawler tractor, leaving the feed at the topographically controlled level. Once a leaching cell is completed, the surface will be scarified using the ripper of a crawler tractor to eliminate compaction caused by the mine trucks and mobile equipment. The cell will then be ready for leaching.

17.2.2 Heap Leach Pad

The leach pad area will be prepared and covered with an impermeable liner. Corrugated, perforated drainage piping to be laid on the liner for collection of the pregnant leach solution. A protective layer of finely crushed, permeable mineral is then placed on top of the liner to prevent damage from the mobile equipment and during loading with feed.

The heap leach pad is located 4 km from the pit, at an elevation of 4,376 masl as illustrated in Figure 17-5. Development of the pad is in four stages with a stacking volume for Stage 1 of

10.3 Mt; 30.6 Mt for Stage 2; 27.7 Mt for Stage 3 and 60.4 Mt for the final stage. The total pad capacity will be 129 Mt.

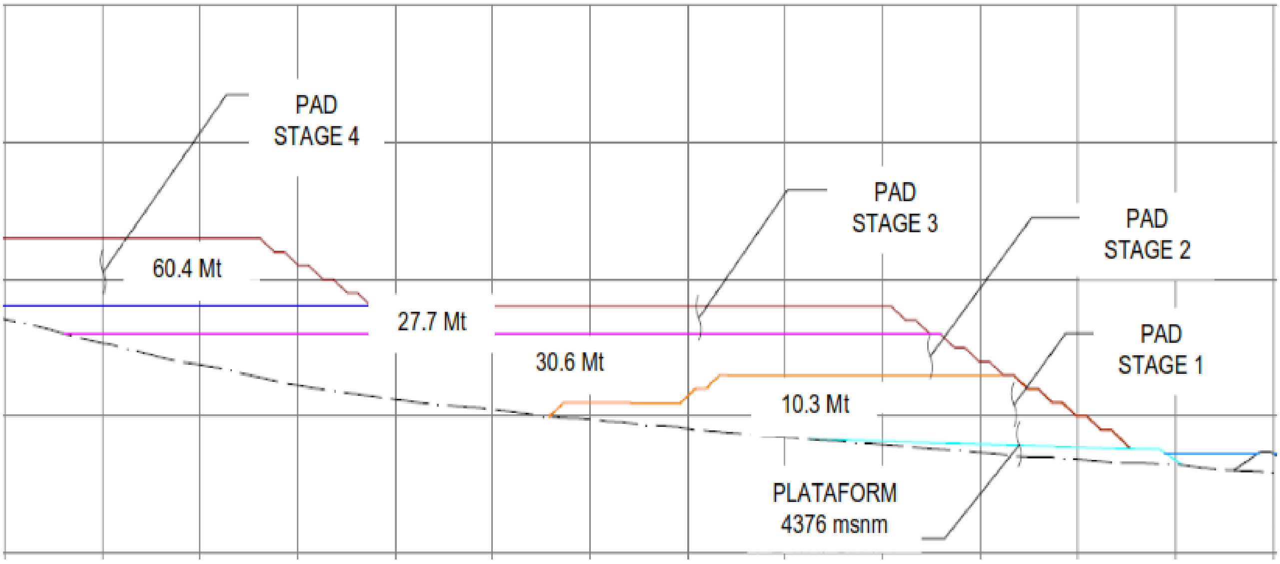


Figure 17-5: Heap Leach Pad Section (Anddes, 2019)

17.2.3 Heap Leach Irrigation System

The irrigation system will uniformly apply cyanide solution directly onto the levelled surface of the leach pile through a drip irrigation system, at an irrigation rate of 10 l/hm^2 (litres per hour per meter squared) with an irrigation cycle of 90 days. Irrigation of the stack feed is by a sodium cyanide solution pumped from the barren solution tank.

The percolation rate through the heap will depend on the viscosity and specific gravity of the solution, the mineral void space, the percentage of fines, mineral affinity for the solution and air entrapment.

Once the heap is irrigated and the ore reaches absorption moisture, the gold rich solution will drain to the lowest part of the pad.

17.2.4 Drainage and Pumping System for the PLS

<u>Collection of leach Solution</u>

The slope of the pad will allow the PLS to flow by gravity through the drainage piping to the PLS pond. The PLS pond will have an overflow system to direct solution downstream to the process overflow pond in the event of a large process upset or a major storm. The pond will be designed with an impermeable double geomembrane to prevent filtration and loss of solution and with a leak detection and recovery system installed between the two geomembranes. Pumps will be installed to return any leaked solution to the ponds.

The capacity of the PLS pond will be 40,000 m^3; the major event pond will have a capacity on 120,000 m^3.

PLS Pumping

The PLS solution will be pumped from the PLS pond to the ADR plant at a flow rate of 1,058 m^3/h as illustrated in Figure 17-6. There will be submersible pumps for the detection of leaks and underground drainage pumps.

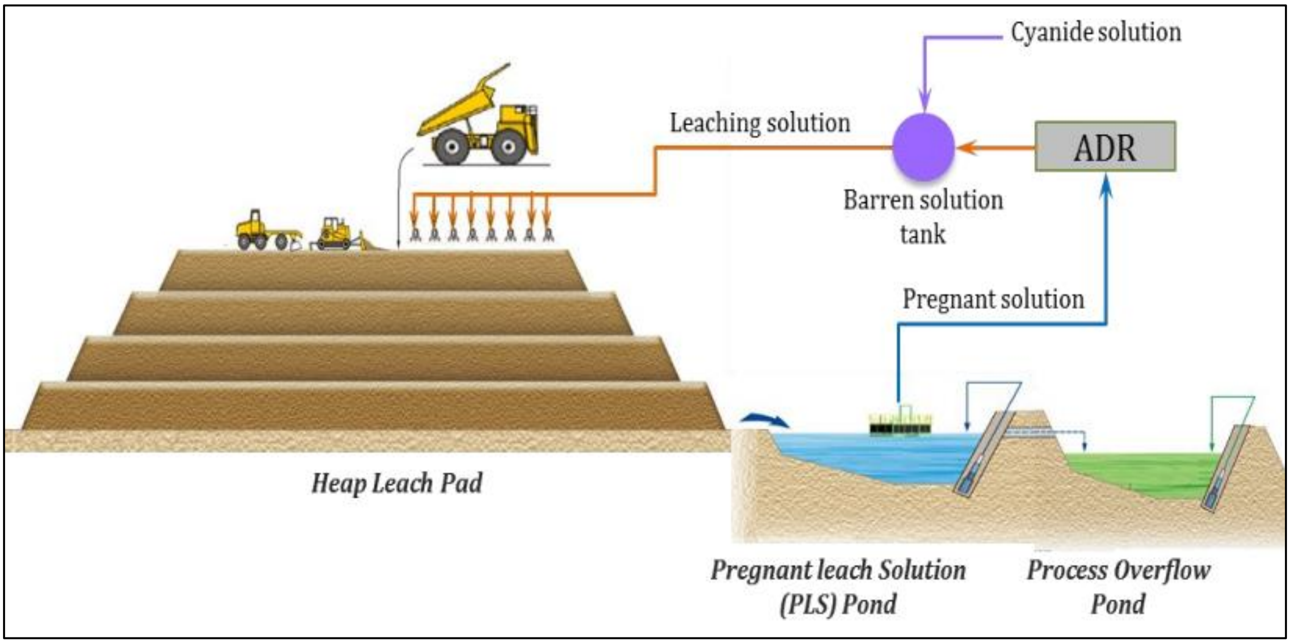


Figure 17-6: Heap Leaching System (HLC, 2019)

17.3 ADR, EW and Smelting

The location and general layout of the ADR plant are shown in Figure 17-7 and a computer generated representation of the crushing plant is shown in Figure 17-8.

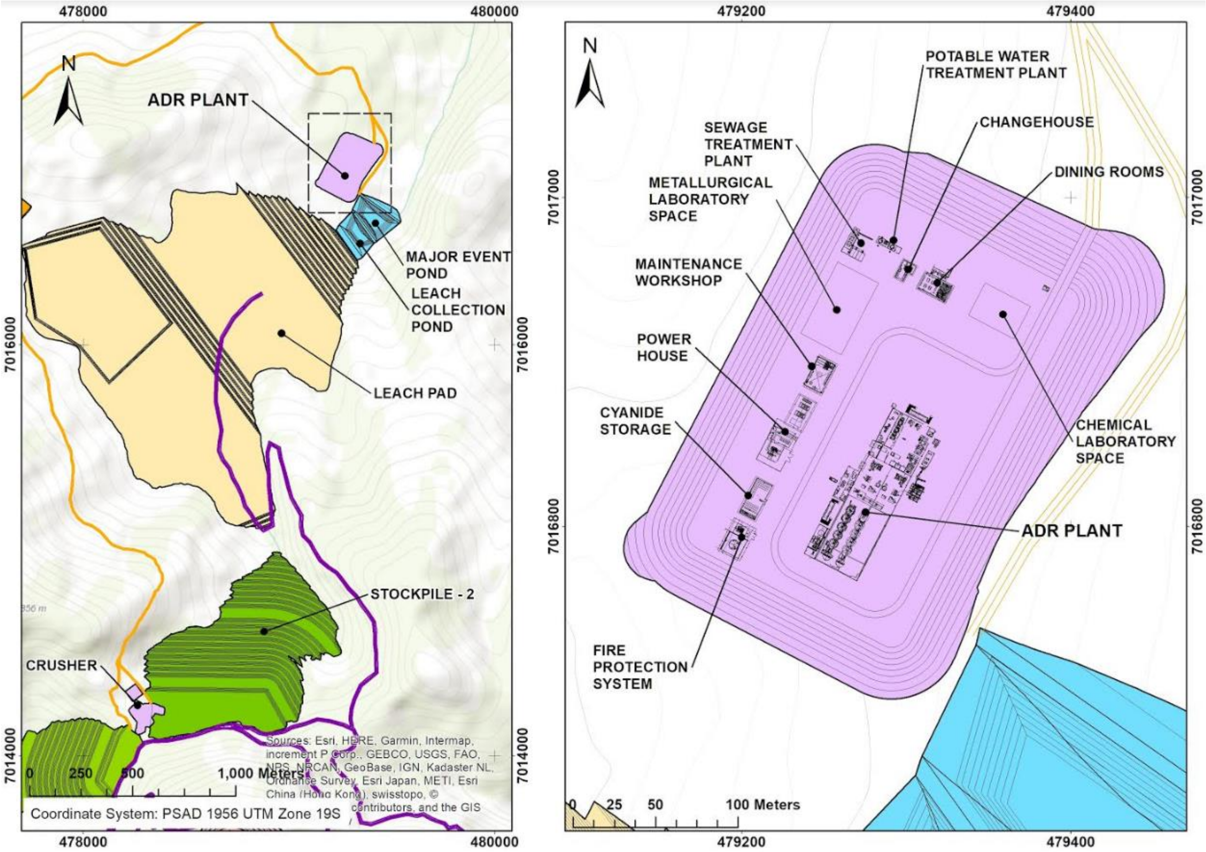


Figure 17-7: Location and general layout of the ADR plant

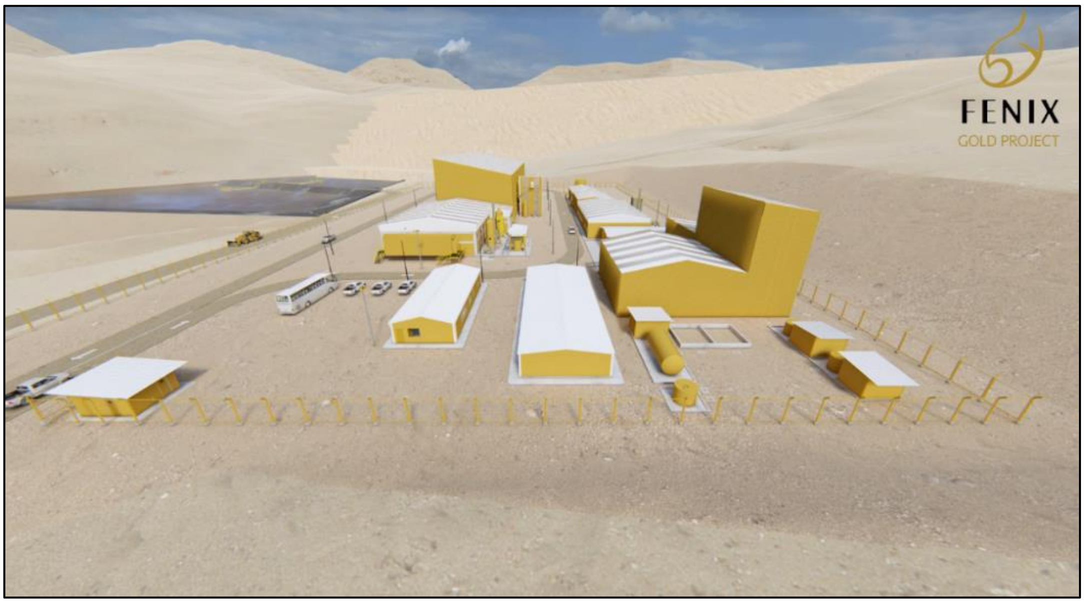


Figure 17-8: Computer generated representation of the ADR plant

The process to recover gold and silver from the PLS solution will involve the following stages: adsorption, desorption, electro-deposition, acid washing, thermal regeneration and smelting.

17.3.1 Adsorption

The PLS solution will be pumped to the adsorption stage, where the gold and silver dissolved in the PLS solution are adsorbed onto the activated carbon.

The adsorption circuit will consist of a train of five adsorption columns, designed for a capacity of 6 tonnes of carbon. In these columns the PLS will pass counter-current through the activated carbon.

The PLS solution will enter a distributor box and then drain by gravity to the first column through a central downpipe and will flow upwards through a distribution plate that will prevent the return of carbon and will fluidize the carbon bed before overflowing from the top of the adsorption tank. The solution will overflow into a channel that will lead the solution to the next adsorption column of the series.

The discharge solution from the last adsorption column will pass through a vibrating screen that will recover any carbon that escapes from the adsorption circuit.

The carbon in the adsorption circuit moves counter-current to the PLS solution flow until it reaches the first adsorption column, where the carbon will be harvested by an eductor that will transfer the carbon to the acid wash stage.

17.3.2 Acid Wash

The acid wash reactor will have a capacity of 6 tonnes of carbon. The acid wash will remove carbonates and sulphates adsorbed by the activated carbon in the adsorption stage. This process is carried out in a closed circuit, with a 4% solution of hydrochloric acid (HCL). This solution will pass through the carbon until the pH stabilizes below two.

Once the acid wash is complete, the carbon is rinsed with water and then with diluted caustic solution to remove any residual acid. The total time required for the acid wash of a batch of 6 tonnes of carbon is 4 to 6 hours. Once the acid wash is complete, the carbon is then transferred by an eductor to the pressure desorption reactor with a capacity of 6 tonnes.

The system will have stationary screens to recover any carbon that enters the solution during the acid wash process.

17.3.3 Desorption

Carbon containing the valuable metals (free of carbonates) will be loaded by an eductor into the 6 tonnes capacity pressure desorption reactor from the acid wash reactor. The solution for desorption will be prepared in the strip solution tank and consists of an alkaline solution of caustic soda (NaOH) and sodium cyanide (NaCN). The desorption process will be carried out by recirculating strip solution at a temperature of 130°C at a pressure of 50 psi. At the end of this process, the strip solution containing the valuable metals will pass to a tank where it will be depressurized and distributed to the electrodeposition circuit. The desorbed carbon is sent by an eductor to the thermal regeneration area.

17.3.4 Electro-deposition

The electro-deposition process is carried out in a circuit consisting of four electro-deposition cells where a direct current is applied to deposit gold and silver onto the stainless steel mesh cathodes. The barren solution from the electro-deposition cells will flow by gravity to the strip solution storage tank, creating a closed circuit with the desorption stage. The stainless steel mesh cathodes are then washed in the cells, to recover the electrolytic precipitate containing the gold and silver. The precipitate is pumped to a pressure filter to remove the excess water and the cake generated in the press filter will be loaded into retort oven trays.

17.3.5 Thermal Regeneration

The process will have a thermal regeneration system with a barren activated carbon processing capacity of 125 kg/h. The barren activated carbon from the storage hoppers will be discharged onto a circular vibrating screen to separate the fine carbon, discharging the fines into the fine carbon tank and the coarse carbon will be discharged into the carbon feed hopper.

The carbon from the feed hoppers will be discharged to the thermal regeneration feed hopper by a pre-drying vibrating screen. The fine carbon particles from this screen is then directed to the fine carbon stockpile.

Thermal regeneration will take place at ~ 500°C and the operation time of the process will be from 28 to 34 hours approximately.

Finally, the carbon will be discharged into a cooling tank by the electromagnetic feeders located under the thermal regeneration furnace.

The regenerated carbon from the cooling tanks will be transported by portable eductors to the carbon storage hoppers.

17.3.6 Refining and Smelting

The filtered electrolytic precipitate from the electro-deposition area will be treated in two retort ovens to recover any mercury that may be present in the precipitate. The sludge will be placed in trays and heated in the retorts for approximately 10 hours at a temperature of approximately 480°C to volatilize the mercury.

The steam generated from the retort oven will pass through a water cooled condenser. Chilled steam depleted in mercury will then pass through a scrubber containing carbon impregnated with sulphur to remove residual mercury and ensure that final emissions meet environmental standards.

Furnace

After extraction of mercury, the electrolytic precipitate is mixed with fluxes including borax, potassium nitrate, silica and sodium carbonate. This mixture will be charged to a crucible to be melted. The main product of the smelting furnace will be molten metal (doré), which will be poured into moulds for cooling, and then stored in a vault until transport (Figure 17-9). The doré bars are the final product of the valuable metals recovery process.

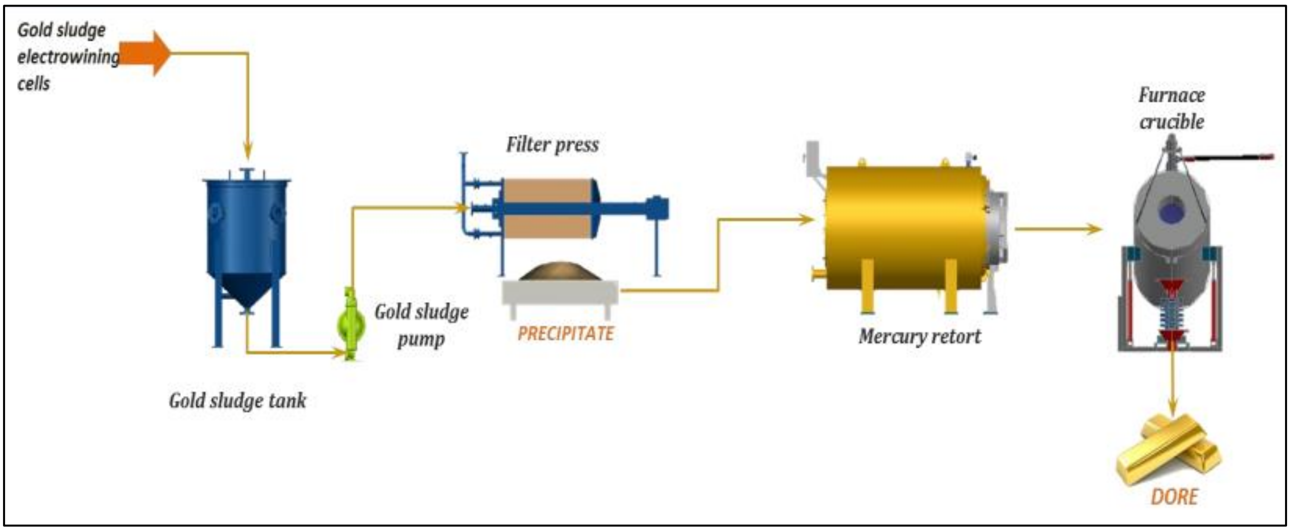


Figure 17-9: Gold Room (HLC, 2019)

17.4 Main Processing Equipment

The main equipment selected for the process is outlined in Table 17-2 with equipment dimensions and capacities.

Table 17-2: Main Process Equipment (HLC, 2019

Equipment	Unit	Characteristics
Primary crushing		
Gyratory crusher	1	Metso 42 X 63, Cap. = 1,111 t/h
Conveyor belt to stockpile	1	Length = 132 m
Heap Leaching		
PLS pond	1	40,000 m^3
Overflow pond	1	120,000 m^3
ADR Plant		
Adsorption columns	5	1 train, capacity = 1,058 m^3/h
Desorption reactor	1	Capacity = 6 t of carbon
Acid washing reactor	1	Capacity = 6 t of carbon
Electrodeposition		
Cells	4	Capacity = 28.8 m^3/h
Thermal Regeneration		
Thermal regeneration reactor	1	Capacity = 125 kg/h
Refining and Foundry		
Electric retort oven	2	300 kg/batch
Tilting furnace	1	600 kg/batch

17.5 Process Reagents and Consumables

Table 17-3: Consumption of Reagents and Consumables.

Table 17-3: Consumption of Reagents and Consumables

Description	Unit	Quantity
Crusher liners		7
Sodium cyanide	g/t	270
Lime	kg/t	4
Carbon	t/month	0.5
Borax	kg/t ppt*	300
Sodium nitrate	kg/t ppt*	100
Silica	kg/t ppt*	70
Sodium hydroxide	g/t	36
Hydrochloric acid (32%)	g/t	20
Water (Process only)**	m^3/t	0.09
Energy (Overall)	Wh/t***	2.32
Primary Crushing	*Wh/t*	*0.66*
Leaching	*Wh/t*	*0.73*
ADR Plant	*Wh/t*	*0.72*
Other	*Wh/t*	*0.20*

(*) t ppt: tonnes of precipitates

(**) Rock moisture assumed at 2% (same as 2014 PFS).

(***) May not sum due to rounding

18 PROJECT INFRASTRUCTURE

The Fenix Gold Project requires significant infrastructure for the mining and processing. The infrastructure includes roads, power supply, water supply, workshops, warehouses, offices, laboratories, site establishment, camp and other facilities as shown in Figure 18-1.

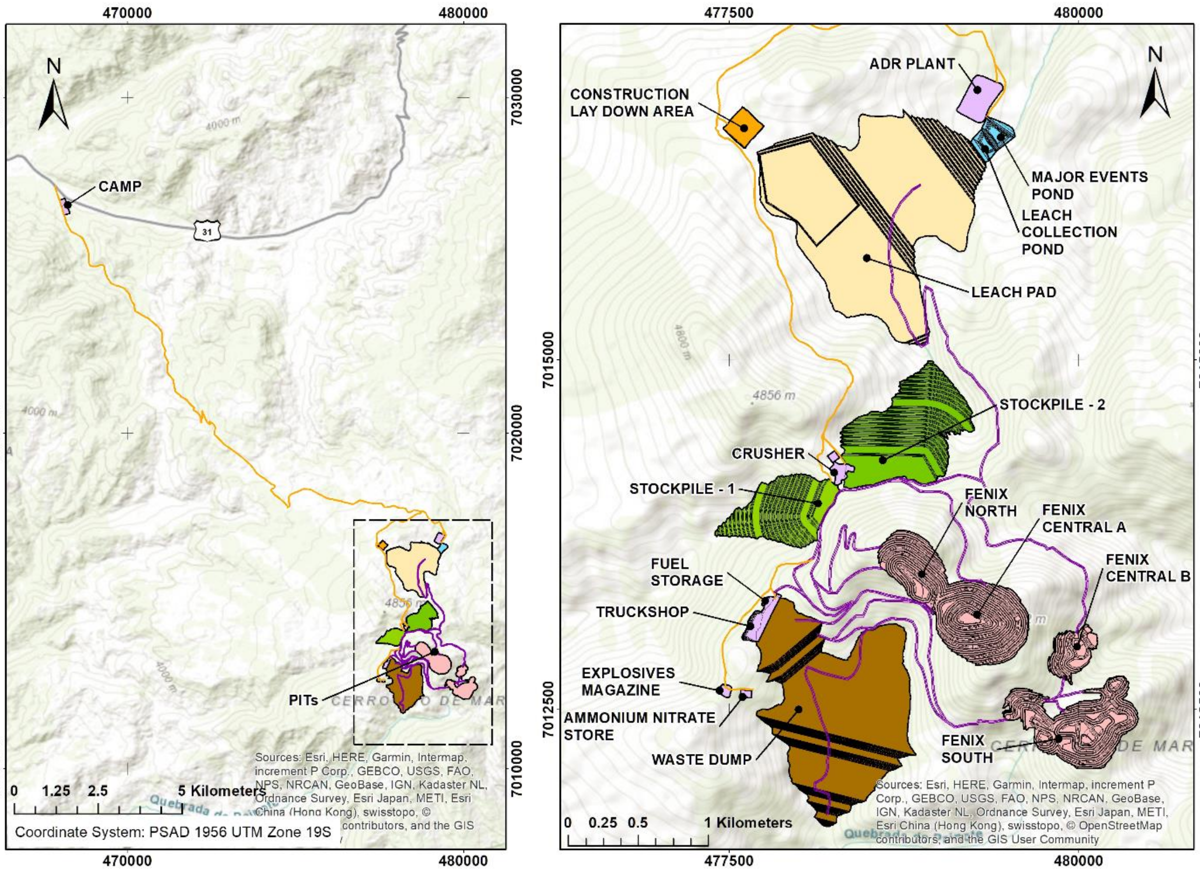


Figure 18-1: Key Infrastructure Facilities

18.1 Water Supply

The Project requires a fresh water supply of up to 24 l/s. The Fenix Gold Project has access to water via a contract signed with Aguas Chañar, the major water supplier in Copiapo, to supply up to 80 l/s of treated industrial water from its Piedra Colgada treatment facility located to the west of Copiapo.

The water will be transported in 30 m^3 water tankers from the Aguas Chañar facility and discharging to the Process Plant located in the Project, a distance of approximately 158 km. The water transport route via international highways 31 is illustrated below in Figure 18-2.

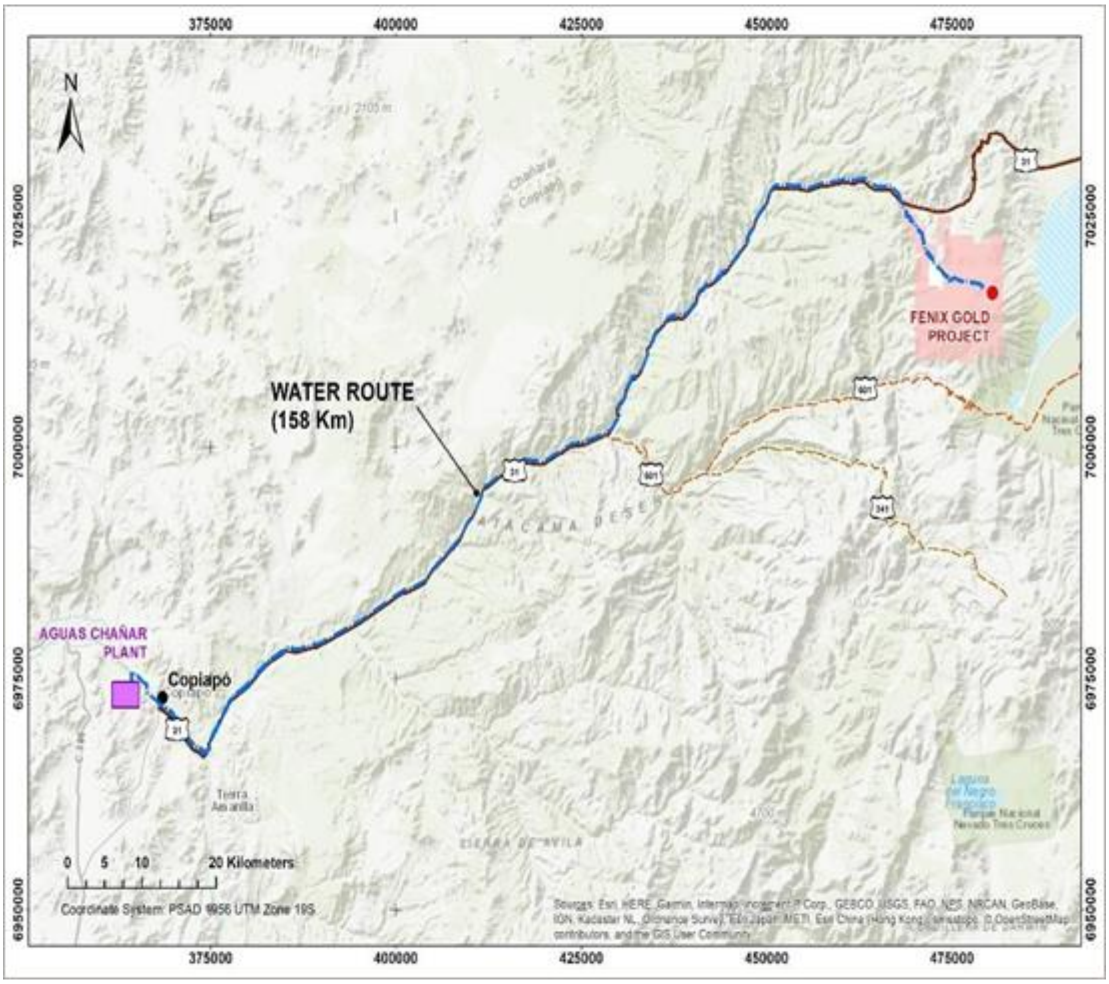


Figure 18-2: Trucked Water Route

The primary water use on-site is for mineral processing. Section 17 considers that the mineral on the pad will need 11% by weight saturation per tonne. The ore placed on the pad from the mine is considered to have a humidity of 2% equal to that considered in the 2014 PFS, leaving 9%, or 90 litres required per tonne for leaching. This equates to approximately 90% of the site water requirement (approximately 21.8 l/s for 20,000 tpd).

The remaining 2.2 l/s (188,000 litres per day) has been allocated for camps, dust control and evaporation.

All wastewater from the camp will be treated and reused in either process or dust control. Other than the operational work areas which will be controlled with water trucks, all permanent haul roads will be treated with Magnesium Chloride surfacing so as not to require dust control.

It is recommended in the next stage of study to complete a detailed water balance to optimise the use of water on site.

18.2 Power Supply

The power supply for the Project will require three generators, two in continuous operation and one installed and on standby in the power plant located in the ADR plant. The generators will produce 1,410 kW each (at the working altitude); the generators feed a synchronization panel located in the electrical room that will feed the plant ADR panel and the distribution panel for plant infrastructure.

The power supply for low voltage areas: ADR plant, dining room, change room, potable water treatment plant, wastewater treatment plant, metallurgical laboratory and chemical laboratory and will be by an overhead, aluminium line supported on poles.

Two generators will provide the power supply to the primary crushing plant, one in continuous operation and one on standby installed in the crushing plant power plant. These generators will feed at synchronization panel located in the electrical room that feeds the panels for primary crushing equipment, stockpile, conveyors and lime storage silos.

18.3 Roads

18.3.1 Access Roads

The location of the Fenix Gold Project is convenient for construction as national road CH 31 passes close to the project. There is good access from CH31 to the project currently, however there are some sections that will need regrading and reworking for a working mine operation. This includes an improved access road bypassing the mine site directly to the process plant of 5.7 km that will require upgrading to an acceptable level.

Figure 18-3 illustrates the planned site access road along with the plant access diversion road. Both access roads will have a width of 8 m and maximum gradients of 10%, constructed with compacted road base.

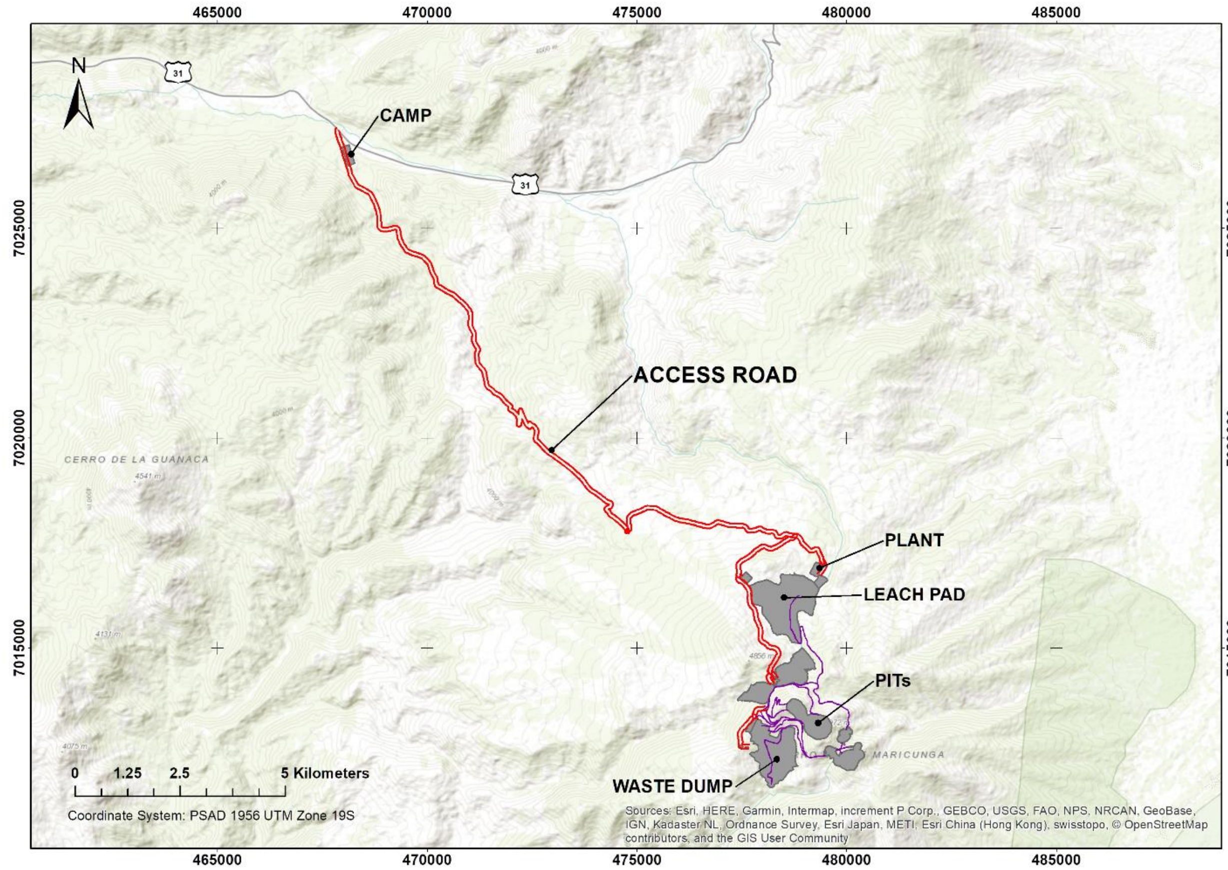


Figure 18-3: Access Road

18.3.2 On-Site Roads

The design of the Project includes the on-site roads that will connect the major facilities as shown in Figure 18-4. The roads connect to the site access road. All haul roads on the site will be constructed to a mining criteria of 14 m design width to deliver 12 m operational width, after safety berms and water channels are installed.

These access roads include:

- Mine – Crusher, Mine - Waste Dump, Mine to Low-grade Stockpiles.
- Primary crusher – Leach pad.
- Leach pad - ADR Plant and ponds.
- ADR Plant to Construction lay down area and crusher.
- Access to Mine maintenance workshop and installations, Wash-down & Fuel Farm and magazine.

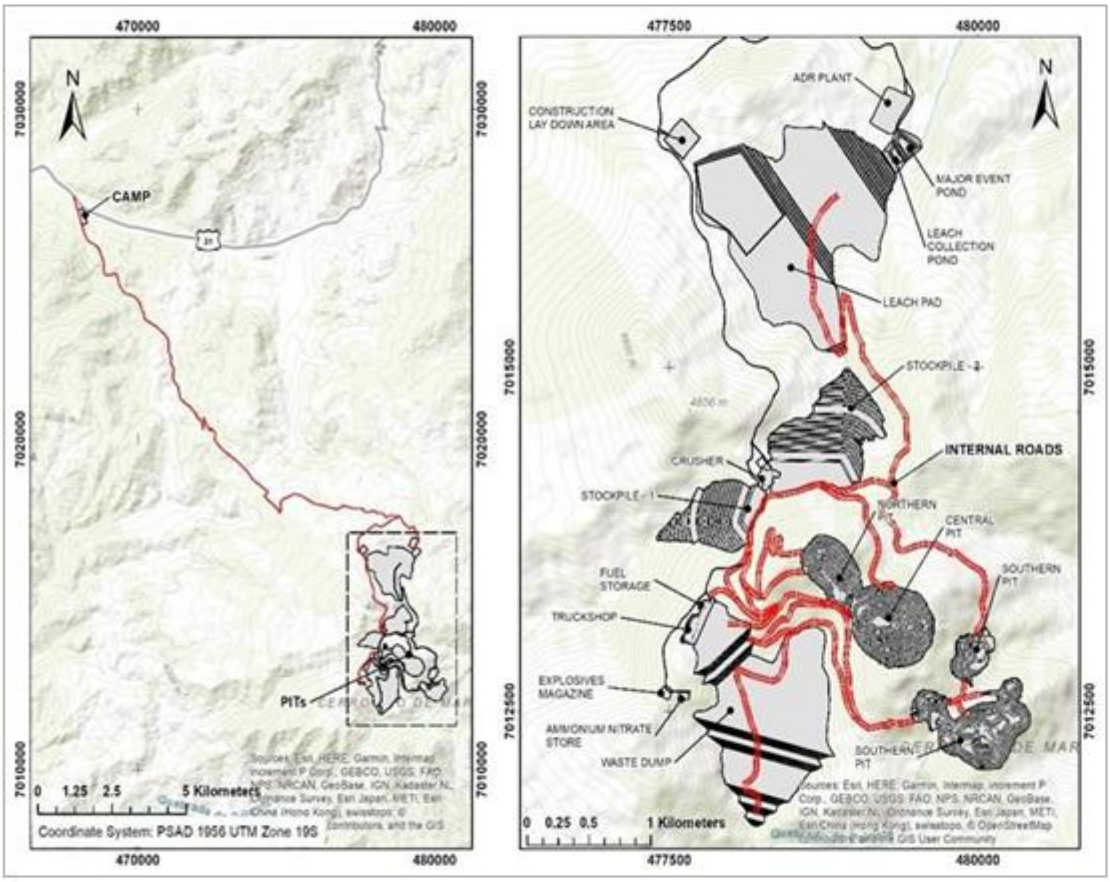


Figure 18-4: Internal Roads

18.4 Camp

The camp facilities include the following buildings: office, medical centre, dining room, bedrooms for senior staff and workers, a recreation room, sports field and parking. These accommodation facilities will initially be used for construction contractors and later for the operations and administration staff.

The camp is located about 12 km northwest of the access road to the project at 3,415 masl, as shown in Figure 18-5. The camp is designed to accommodate up to 310 people over a period of 17 years, and covers an area of approximately 76,300 m^2 and includes the security checkpoint area and parking. The camp will be built using modular prefabricated units.

A security checkpoint located at the camp entrance will control entry to the Project.

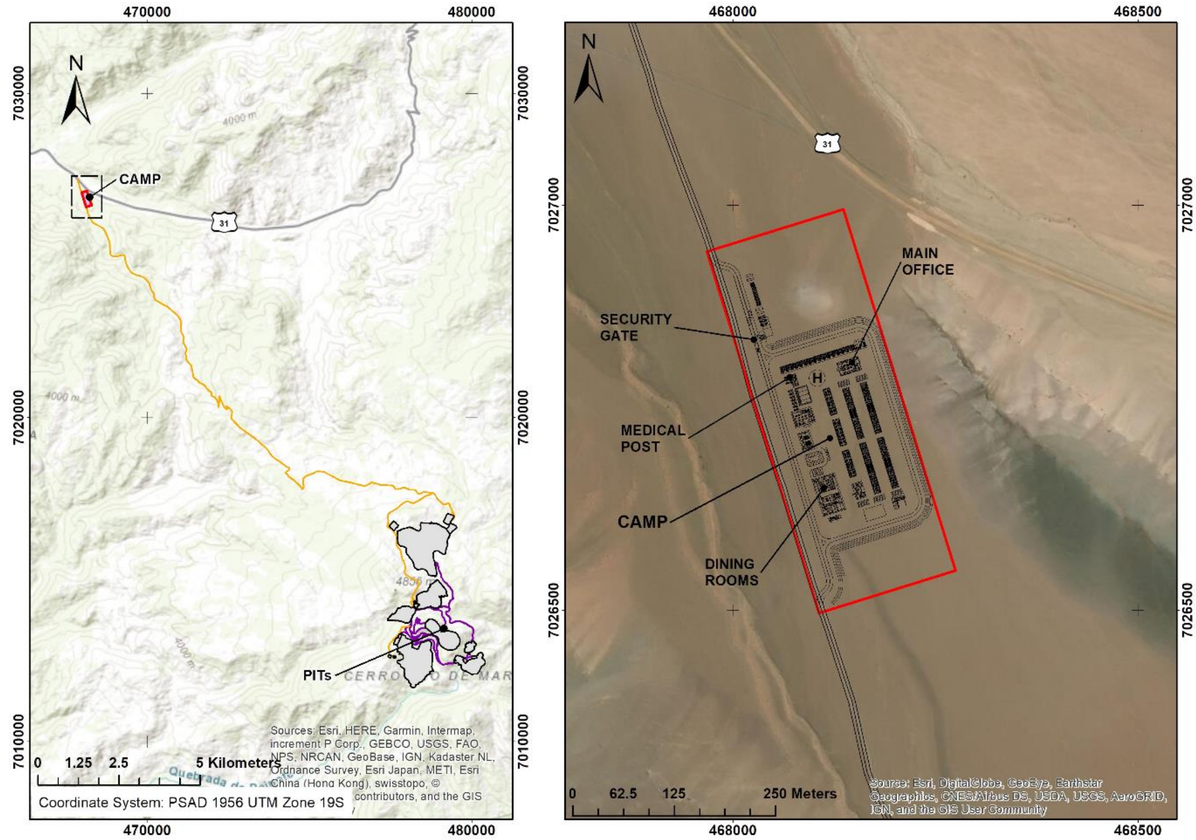


Figure 18-5: Camp Location

18.5 Potable Water Supply

The project includes a potable water treatment plant to process the water trucked to site for consumption.

18.6 Sewage Treatment

Sewage from the camp will be directed to a wastewater treatment plant sized for 310 employees.

Sewage generated in the ADR plant and plant infrastructure will be directed to the wastewater treatment plant located at the plant.

Septic tanks will collect sewage from infrastructure areas that are not connected to the treatment plant. Septic sludge will be collected and transported by truck to treatment facilities off site.

18.7 Waste Management

All solid waste, industrial waste and toxic waste generated by the mine will be temporarily stored and classified prior to transportation to a final disposal destination.

18.8 Main Offices

The main administration office will be located at camp. This building will have offices for mine managers, technical staff and administration.

The entrance is via a reception area and an area for administration staff. There will be areas for the main offices, meeting room, document and plan filing, IT and communications, and a kitchenette. There will also be toilets for male and female staff.

The building will have a central air conditioning system. Utilities include electricity, water and a sewage system. The main switchboard, servers and network operation will be located in this office.

The majority of the staff will be located in the main office as support for the operation visiting site as required.

An operational office on site will be located in the maintenance workshop for mining, geology and maintenance personal to coordinate day-to-day activities.

18.9 Plant infrastructure

Plant infrastructure is summarized in Figure 18-6 and a computer generated representation of the plant infrastructure is provided in Figure 18-7.

18.9.1 Plant Offices

The offices will be located within the ADR plant; they will consist of one steel frame bay 20.50 m long by 7.70 m wide, with a peaked roof.

The offices include airlock entrance, reception and waiting room, meeting room for 10 people, communications room, management offices, supervisors' office, document file area, workstations for operators, control room, kitchenette and toilets for male and female staff members.

18.9.2 Plant Dining Room

The dining facility will have shelves for hard hats, tables and chairs and fixed furniture for food service. The food service area will include a cold bar, microwave, blender, electric water heater, hot water servers, sauce server and table for support for service. It will also include toilets for male and females.

The dining room will have an area of 175 m^2 with a capacity for 32 people.

18.9.3 Change Rooms

Entry to the showers and change rooms will be through a swing door leading to an access airlock for the men's area and another for the women's area. At the sides of the airlock, there are two spaces: one for the general electric panel and the other for the electric water heating (a horizontal floor mounted tank of 304 L capacity. The women's area will have sinks, toilets, lockers and showers. The men's area will have sinks, urinals, toilets, lockers and showers.

18.9.4 Plant Maintenance Workshop

The plant maintenance workshop will be designed to perform maintenance and with space for planning, to improve availability and reduce maintenance costs for the crushing plant and ADR plants.

The maintenance shop will have access for forklifts and a pedestrian entrance equipped with an anti- panic bar. The workshop will include metal shelving, welding machines, drill stand, plane, lathe, metal filing cabinets and worktables.

18.9.5 Chemical Laboratory

A service provider for drill sample analysis and process plant sampling will implement the Laboratory on site. This sampling cost has been included into the mining cost and process cost estimate. The process plant platform has been designed leaving space for the installation of the laboratory facilities.

18.9.6 Metallurgical Laboratory

The metallurgical laboratory was removed from the initial CAPEX. A design has been completed and space left in the Process plant platform. Sample testing would be completed by a service provider in Copiapo.

18.9.7 Plant Power House

The powerhouse will house three generators, two in operation and one on standby.

The low voltage synchronization panel receives power from the generators and performs the synchronization using controllers that operate in master-slave mode. These controllers will perform the following functions:

- Synchronization of the sets.
- Load sharing.
- Switching off.

The synchronization panel will evaluate the number of generators needed in operation, depending on the power demand and the amount of power to be delivered by each generator.

The powerhouse will consist of an area for the generators and an area for the fuel storage tank.

18.9.8 Reagent Storage

Reagent storage will consist of two areas: an area to store cyanide and an area for carbon storage; the latter will have an area contained by concrete curbs in which hydrochloric acid cylinders will be stored.

Access to the cyanide storage area is through a sliding metal gate; there will be a central passage for the forklift. Access to the carbon storage will also be through a sliding metal gate.

Lime to be added to the feed, will be stored in lime silos, located in the crushing plant. Lime will be added by a metering screw feeder onto the feed before transporting and stacking on the heap leach.

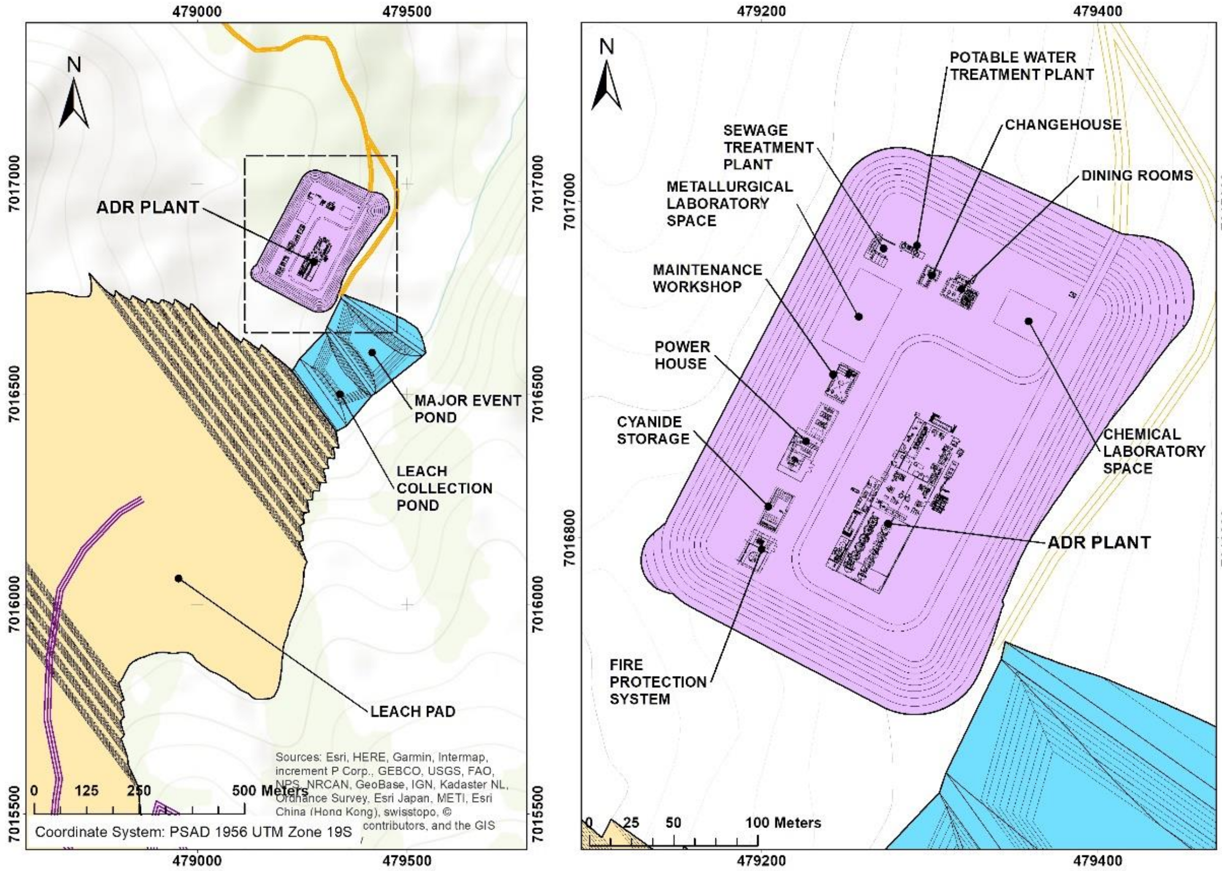


Figure 18-6: Plant Infrastructure

Figure 18-7: Computer generated representation of plant infrastructure

18.10 Communications

18.10.1 Off-Site Communications

Basic telephone service will initially be provided by a satellite phone communication. Cell phone coverage will be established as soon as possible to cover the construction offices and camp construction area and eventually mine site construction areas.

Internet will be connected and distributed via a satellite system located at the camp installations. As the infrastructure is constructed internet will be extended to include the plant, crushing and workshop areas.

18.10.2 On-Site Communications

Cell phone repeaters will be installed to give coverage to the principal infrastructure on site, Radio towers will be implemented to cover operational areas including the Pits, Waste Dump, Crusher, Leach Pad, Plant and Workshops, and camp.

18.11 Mine Facilities

Mine workshop infrastructure is described in sections 18.11.1 to 18.11.3 and is displayed in Figure 18-8.

18.11.1 Explosives

Warehouse storage of explosives will be located near the waste dump. Detonators, detonating fuses and cable will be stored in protected 20 ft. containers. Each container will be isolated by containment walls of compacted material and there will be no electrical installations to avoid the generation of sparks. The containing walls will be surrounded by a metal fence with barbed wire and a locked gate.

Ammonium nitrate will be stored in a warehouse in a nearby isolated and protected area. The warehouse floor will be cement, and the walls and peaked roof will be steel. Emulsion storage silos supported on metal structures will be located in the same area, set up for direct loading into the MMU truck.

18.11.2 Truckshop

The workshop for truck maintenance and auxiliary equipment maintenance will be located adjacent the waste dump with the upper level of the waste dump forming the platform. The location is central to the mining and crushing operations. The truck shop will have the following areas: truck maintenance area, tire shop, welding shop, lubricant storage, compressor room, truck wash and offices and materials and components storage. The foundations will be reinforced concrete and the structure will be steel with metal sidings and roofing.

18.11.3 Fuel Storage and Delivery

Fuel storage comprises two tanks with a capacity of 40,000 gallons each.

The tanks will be located on a slab and enclosed by high perimeter wall to contain any fuel spillage.

The fuelling station will receive fuel from pumps connected directly to the storage tanks via a buried piping system. Fuel will be able to be supplied to both heavy and light vehicles at the Fuel Farm.

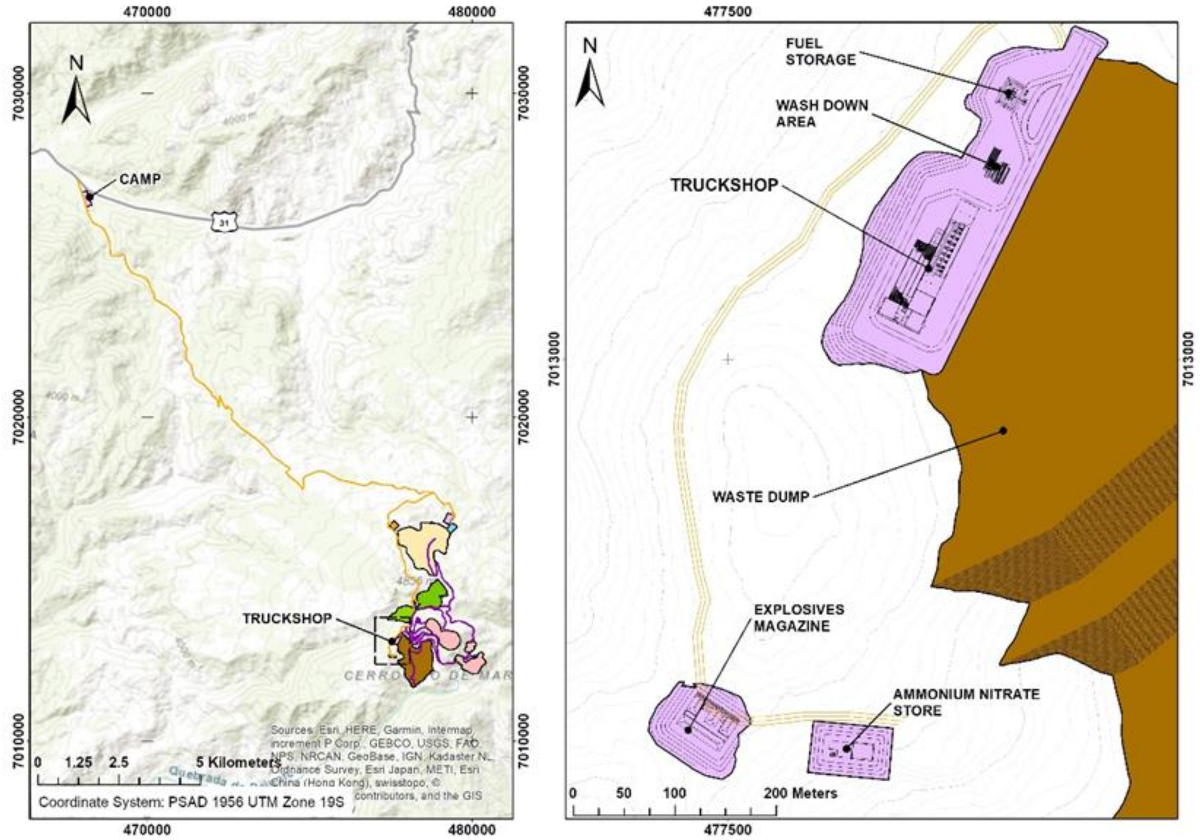


Figure 18-8: Mine Workshop Infrastructure

19 MARKET STUDIES AND CONTRACTS

19.1 Market studies

Rio2 has not conducted a market study in relation to the gold and silver doré that will be produced by the Fenix Gold Project. Gold and silver are freely traded commodities on the world market for which there is a steady demand from numerous buyers.

A sale price of US $1300 Oz of Au has been used. Raul Espinoza (QP) considers that this price is reasonable and notes that gold has been trading above this price since before the beginning of 2019.

19.2 Contracts

There are no refining agreements or sales contracts currently in place that are relevant to the Technical Report.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

In 2011, Atacama filed the Environmental Impact Declaration (DIA) for drilling. This DIA is titled "Mining Examination Cerro Maricunga" and was filed with the Environmental Impact Assessment System (SEIA), which was approved through the Exempt Resolution No. 232/11 by the Atacama Region Commission of Environmental Assessment.

In October 2018, the National Geology and Mining Service, (SERNAGEOMIN) was informed about the beginning of grade control (GC) drilling operations in 39 pre-existing drill platforms, using the existing DIA, which was completed in February 2019.

The preparation of a new Environmental Impact Declaration (DIA) to perform geotechnical, geometallurgical and sterilization drilling operations began in October 2018. In April 2019, the Environmental Impact Declaration was filed with the Environmental Assessment Service (EAS) and is expected to be approved later in 2019.

The environmental baseline (EIA) for the Fenix Gold Project has been developed since November 2018. The baseline is currently in the information gathering stage and the EIA will be submitted SEIA in February 2020. The RCA (Resolution Calificación Ambiental) or approval is expected to be April 2021.

20.2 Environmental Work Completed

Atacama Pacific completed the Environmental work between 2010 and 2014. The weather station and other data collection devices at the project were dismantled in 2015, and no other environmental monitoring was completed until Rio2 purchased the project in June 2018. Since then Rio2 has re-installed data collection devices and has re-initiated data collection.

20.2.1 Air Quality

Data collection by Atacama between 2011 and 2014 shows low levels of particulate matter, within accepted levels, inside the project area (Baseline EIA-MYMA, Rio2 Limited, 2019). This is to be expected, as there is no significant mining activity in the project area. Rio2 is continuing with a program of air quality measurements and these confirm the previous low levels of particulate matter.

20.2.2 Noise Levels

Based on the fieldwork carried out, no potentially sensitive noise receptors were detected at the project site; there are no human groups or activities close to the project.

20.2.3 Hydrology and Hydrogeology

No permanent superficial water flows were detected in the project 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 behaviour of the aquifers in the basin or sub-basin in the project site in order to prevent damaging the water resources of the area.

The 2014 PFS reported that the project area is locally underlain by aquifers of little hydrogeological importance (in agreement with the drilling campaigns). Aquifers formed of rock or unconsolidated deposits, essentially devoid of underground water resources.

20.2.4 Flora and Vegetation

The Project area is lacking vegetation, due to adverse climatic conditions.

Vegetation in the area of influence of the Project consists of nine species distributed in six families.

After reviewing the current Chilean regulations and the scientific-technical proposals with legal importance, none of the registered species are protected by law under any official conservation category. All nine species registered in the influence area are of 100% native origin.

Regarding its growth form, 89% of the registered flora correspond to herbaceous plants and 11% correspond to shrubs. Regarding its growth form, 88.9% of the registered flora corresponds to herbaceous and 11.1% corresponds to shrubs (Table 20-1).

Table 20-1: Vascular flora of the Study area according to biological type and origin

Biological type	Autochthonous		Allochthonous	Total	%
	Native	Endemic			
High woody	0	0	0	0	0.00%
Short woody	1	0	0	1	11.10%
Herbaceous	8	0	0	8	88.90%
Succulent	0	0	0	0	0.00%
Total	**9**	**0**	**0**	**9**	**100%**

Source: Baseline - MYMA, 2019.

20.3 Fauna

Fauna has been characterized in the field from forty-one sampling locations. This included:

- Twenty-nine locations for amphibians, reptiles, birds and mammals.
- Seven micromammal trapping lines.
- Three locations with trap cameras, and
- Two night monitoring locations (chiropteran detector).

Work at these sample stations determined:

- Five environments for wild animals within the Project influence area: steppe scrubland, areas without vegetation, scarce vegetation areas and industrial areas.
- Nine species were encountered, of which one corresponds to reptiles, four to birds and four to mammals.
- Regarding the origin of the identified species, one of them is endemic, one is introduced and seven are native.
- Of the total registered species, three are in the preservation category: one reptile species (*Liolaemus rosenmanni*) and two mammal species (*Lama guanicoe and Lycalopex culpaeus*), with both being in the vulnerable category.
- Regarding mobility, two low mobility species were registered, of which one is under the vulnerable preservation category (*Liolaemus rosenmanni*).

20.3.1 Fauna Monitoring Program

Within the Environmental Impact Study framework, Fenix Gold Limited will commit to the monitoring of the biological component for fauna, flora and vegetation of the identified species within the Project area, in order to monitor any variation.

20.4 Archaeological Heritage

In the area surrounding the projected location of the pit and waste dump, the presence of one site of low heritage value was verified, corresponding to stone wall fenced areas with evidence of historical occupation.

During the archaeological assessment carried out in 2019 for the EIS (Environmental Impact Statement) Baseline, eleven archaeological sites have been detected alongside the Project's access road. 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 stone structures and one rocky shelter.

Fenix Gold will define the protection measures for these archaeological findings within a management plan.

In case there is a detected presence of anthropological/archaeological or subsurface historic cultural remains not registered in the present survey, procedures should comply with provisions in articles 26° and 27° of Law 17.288 of National Monuments and articles 20° and 23° of its Regulations, with the purpose of designing and carrying out the appropriate archaeological salvage activities. Likewise, the Council of National Monuments must be notified in writing immediately, so that it may authorize the specific procedures to be followed.

20.5 Landscape and Tourism

The scenic and tourist areas near the Project area are the Nevado Tres Cruces National Park, the Portezuelo de Maricunga, the viewpoint of the Laguna Santa Rosa, the Virgen de La Candelaria and the viewpoint of the Salar de Maricunga.

The Project development is planned so that the visual impact on landscapes is minimized.

20.6 Human Environment

It is important to note that mining activities in the Fenix Gold Project are located between 4,400 and 4,900 masl, where the altitudinal and climatic conditions of the area impose natural restrictions for the establishment of human settlements, plants and wild animals, with predominantly arid soils.

The Colla Communities closest to the Project are located in Quebrada San Andres and Quebrada Paipote, where they carry out their main productive activities: breeding, grazing and agriculture for self-consumption, creation of handicrafts and collecting medicinal herbs with the latter being the basis of their family income, their cultural manifestations and ancestral customs.

Regarding the territorial distribution of the sites of cultural importance for the communities, it is worth mentioning that most of them are grouped around the means of communication including routes C-341, C-601 and C-607, and with less use of international road CH-31 (Figure 20-1). However, territory occupation by the Colla Indigenous Communities is discontinuous and dispersed. Occupations vary, according to each community's perspective, as a reflection of their ancestral usage and current variants, mainly linked to ceremonial practices and migratory habits. The conceptual socialization plan is shown in Figure 20-2.

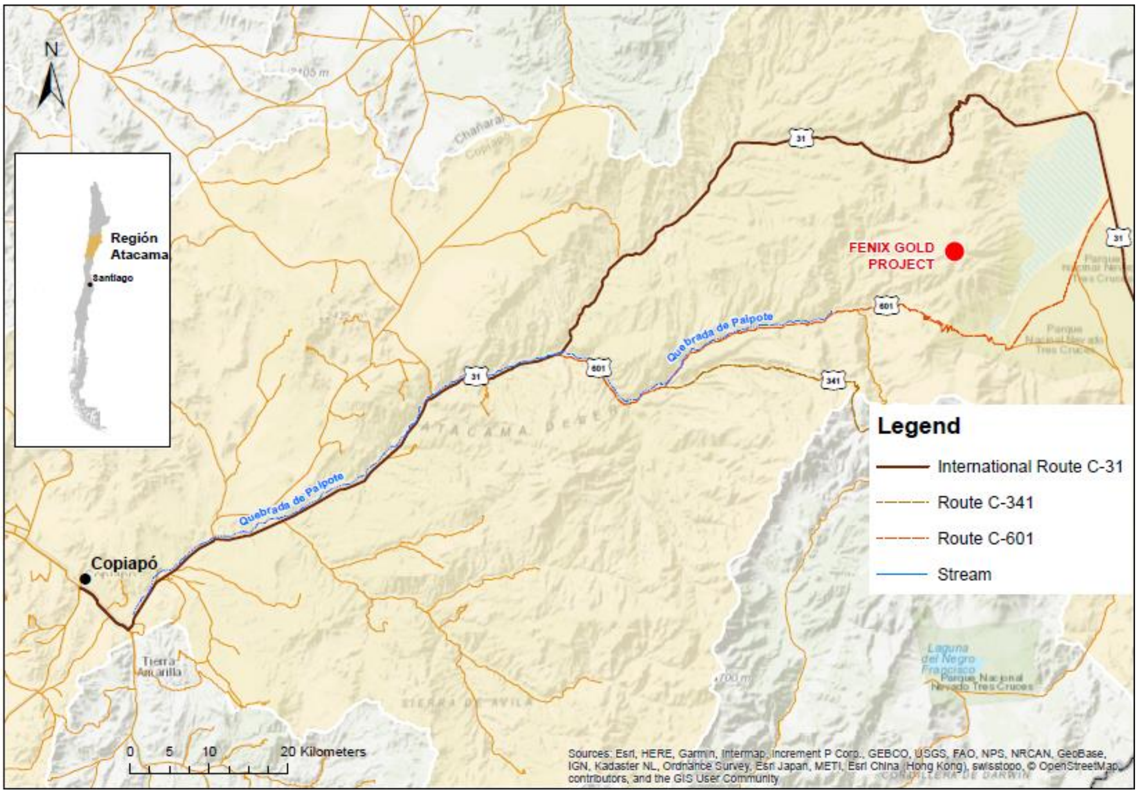


Figure 20-1: Travel Ways of the Colla Community

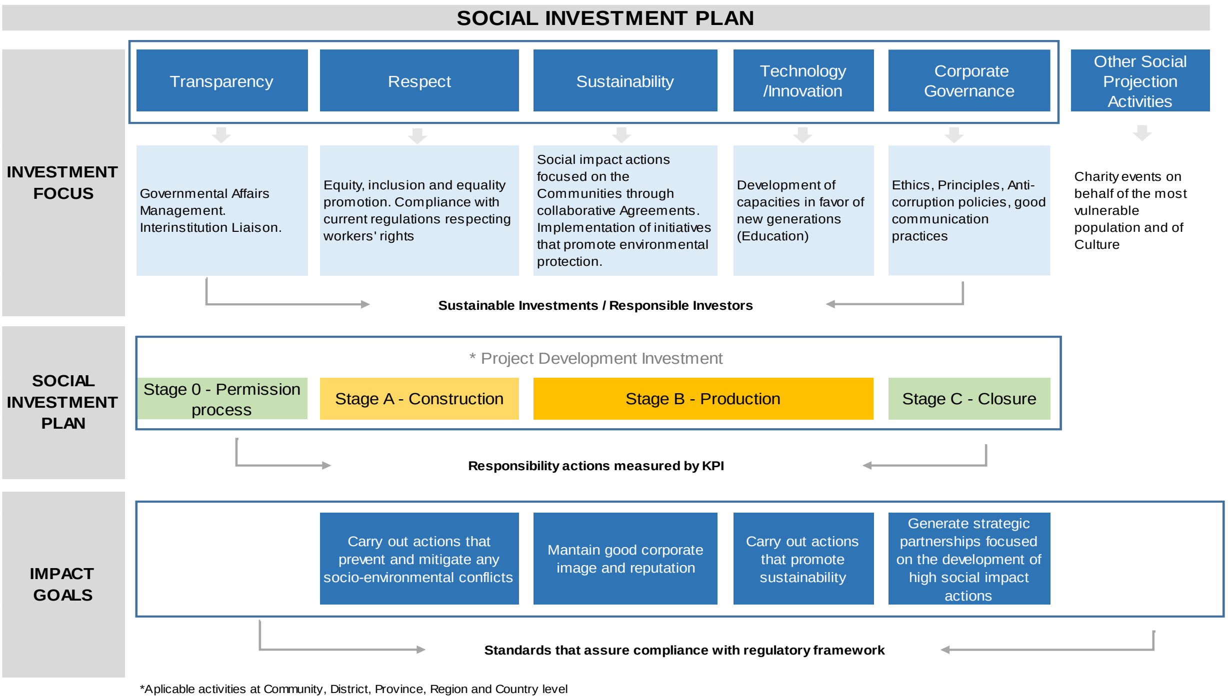


Figure 20-2: Conceptual Social Investment Plan

20.7 Protected Areas and Priority Sites

There is no territorial or spatial overlap between the project area and the protected areas, priority sites, protected wetlands, glaciers or sites with environmental value.

In the Atacama Region there is a total of ten Protected Areas, corresponding to the following categories: three (3) National Parks, three (3) National Protected Goods, one (1) National Reserve, one (1) Nature Sanctuary, one (1) Marine Reserve, and one (1) Marine Coastal Protected Area. The closest site is the Nevado Tres Cruces National Park, approximately 3.5 km away in a straight line from the Project (Figure 20-3).

According to the Instructions on Priority Sites for Conservation in the Environmental Impact Assessment System (Ordinary Official Letter N° 100143 de 2010, of the Executive Directorate of the Environmental Assessment Service), the Priority Site for the Conservation of Biodiversity closest to the Project corresponds to the site known as “Pedernales Salt Flat and its surroundings”, located more than 36 km away in a straight line from the Project’s central point. In this regard, the analysis made from the defined influence areas for noise, vibrations and dust, show that the emissions of the Project will not generate any effects on this site.

According to the List of Wetlands of International Importance (Ramsar Convention), in the Atacama Region there is a Ramsar site corresponding to Laguna del Negro Francisco and Laguna Santa Rosa, which are part of Nevado de Tres Cruces National Park.

According to the “Inventory of Chilean Glaciers of the General Directorate of Water Management” (GDWM, 2014), 884 glaciers are identified in the Atacama Region, of which 191 are in the Copiapo district. In this regard, the Project is not positioned or close to any glacier. The closest one is located approximately 29 km south of the Project and corresponds to a Rocky Glacier located in the Tierra Amarilla district.

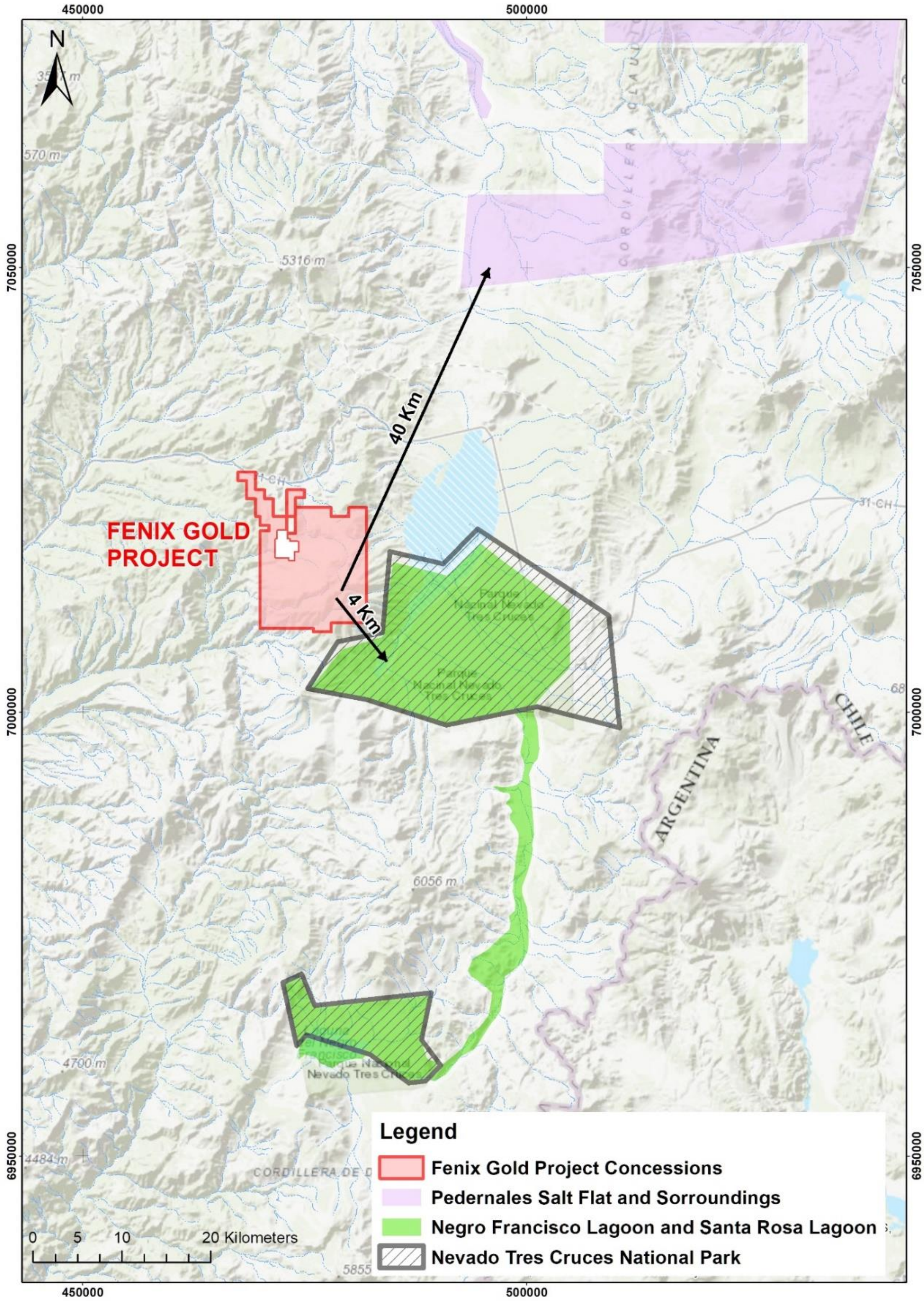


Figure 20-3: Protected Areas and Priority Sites

20.8 Potential Emissions, Waste and Effluents Generated by the Project

The development of Project works will generate emissions, effluents and waste in all its stages, for which environmental control measures will be issued.

20.8.1 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 and potentially ore crushing.

Fenix Gold will establish measurements to reduce the negative effects of emissions through the setup of an environmental management plan.

20.8.2 Noise and Vibrations

During the construction stage, noise would be generated by heavy-duty machinery and trucks performing earthworks. 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 and potentially by crushers. 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.

Fenix Gold will establish measurements to reduce the negative effects of noise and vibrations through the setup of an environmental management plan.

20.8.3 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 static heap leaching pad process, waste generated will be managed on site.

The waste dump will comply with the regulatory and technical requirements to ensure physical and chemical stability.

The presence of little precipitation in the area makes this component of practical management.

Control measures for the waste dump will be expanded within the development of the environmental management plan of the Environmental Impact Study.

Conditions for the generation of acid rock drainage are very low due to the low rainfall in the area and high evaporation rate. However, Rio2 will develop hydrogeological and hydrochemical studies as a matter of normal course.

20.8.4 Industrial Waste

Normal minor hazardous waste will be produced during construction, such as old pipes, oil and grease and during the operation of the project. These wastes will be temporarily stored in the transfer zones and later taken to their final disposal in authorized places that have the required environmental security measures.

Industrial waste will be handled by an officially authorized company, and recycling and re-use segregation will be promoted before its final disposal to minimize available volume.

20.8.5 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.

Domestic waste will be segregated and recycled as much as possible from the beginning and the remaining will be disposed in authorized places for final disposal.

Domestic waste will also have a management plan to control any possible impact that it may generate.

20.9 Closure and Abandonment Stage

20.9.1 Closure Plan

During preparation of the Environmental Impact Study, the Sectorial Environmental Permit 137: Permission for the approval of the mine site closure plan will also be requested.

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: Law 20.551 Mine Site Closure), 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 because 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 and ensuring that the spent leach pad is 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.

20.9.2 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 to adopt the necessary corrective measures in case any contingent event was to happen.

The objective of the post-closure is to verify that the physical and chemical stability of the closure procedures applied to the mining components was achieved.

20.10 Summary of Main Environmental topics for the Project

The location of the Project has characteristics that make it favourable to develop the Project, diminishing possible impacts when considering the characteristics of the area. The use of water from a source that does not generate any significant environmental impact, the scarcity of fauna and flora due to the altitude of the project, the absence of human activities and the low rainfall of the area make the project attractive from an environmental perspective.

This situation is highly favourable 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 Flat.

No significant environmental issues have been identified which could hamper or halt the development of a mining and associated heap leach processing facility at the Fenix Gold Project.

It is considered that during operations, Fenix will obtain the International Code Certification on Cyanide Management, which will give greater control for the environmentally responsible use of the product.

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 modelling. Consequently, Fenix Gold Limited should conduct studies to assess the impacts.

During the next stage of the project, it is recommended that the Environmental Authorities and the neighbouring 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 neighbouring communities to enhance communications and facilitate the environmental permitting. It should be noted that Fenix Gold Limitada 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.

During the Project closure, it will be verified that all actions contribute to restore the environment after the operation, so that it is restored in the same or better environmental conditions than those found prior to the operation.

During closure and post closure, continuous monitoring will be carried out to identify deviations and propose the necessary measures to achieve an optimum closure in accordance with the environmental and social responsibility of the Company.

21 CAPITAL AND OPERATING COSTS

Capital and operating costs for the Fenix Gold Project were developed based the mine plan, production schedule, process plant design, and required infrastructure. The capital costs were estimated based on designs for the infrastructure, including, equipment materials, labour, and services required for the fabrication and assembly of the various components. Operating costs were estimated for equipment, labour, materials, power, supplies, fuel, and explosives with supporting costs from consultants and potential suppliers to operate the mine and plant as designed.

The capital and operating cost estimates have been prepared by HLC Ingeniería y Construcción (HLC), Anddes Asociados (Anddes), STRACON and Rio2.

Capital and operating costs are quoted as United Stated Dollars ($).

21.1 Capital Costs

The capital cost estimate (CAPEX) presented in this Report is for a gold mine capable of producing and processing an average of 20,000 tpd of ore (dry basis). The total life of mine capital investment (Initial and Sustaining Capital) for the Fenix Gold Project is estimated to be $206.2M (Table 21-1). Initial Capital expenditure has been estimated over a two year period, approximately 86% ($95.8M) of the capital will be used in the first year for construction and 14% ($15.4M) will be used the following year.

Table 21-1: Capital Cost Summary

Area	Capex $M	Sustaining $M	Total $M
Mining	8.58	0.85	9.43
Process Plant	35.37	16.27	51.64
Civil Construction	41.18	44.09	85.27
Contingency	14.23	13.81	28.04
Owner costs	11.84	4.57	16.41
Closure Costs		15.4	15.4
Total	**111.2**	**95**	**206.2**

21.1.1 Currency

The estimate is expressed in US Dollars, as at August 2019. No additional funds have been allocated in the estimate for further escalation or to offset potential currency fluctuations.

21.1.2 Estimates Exclusions

Items not included in the capital estimate, include:

- Foreign currency exchange rate fluctuations.
- Interest and financing cost.
- General sales and withholding taxes (included in the financial analysis).
- Working capital has been excluded from the estimate.

Risks due to political upheaval, government policy changes, labour disputes, permitting delays, or any other force majeure occurrences are also excluded. Operational costs consider ten days per annum of no operation for bad weather.

21.1.3 Mine Capital Costs

Mining will be undertaken using contract mining, via an alliance agreement similar to Rio2 management experience in Peruvian projects.

Initial mining capital costs are reported in Table 21-2. Mining capital costs are associated with mining equipment mobilization, demobilization, workshop, minor auxiliary equipment, fuel farm and magazine construction and communication's equipment. Mobilization and demobilization costs are estimated to be (at approximately $500k (Table 21-3).

Table 21-2: Mine Capital Cost

Area	Capex $M	Sustaining $M	Total $M
STRACON Mob & Demob	0.25	0.25	0.5
Workshop Equipment (crane truck, IT, Compressor)	1.404	0	1.404
Workshops and Infrastructure	3.82	0	3.82
Fuel Farm	0.65	0	0.65
Magazine	0.86	0	0.86
Communications	1.6	0.6	2.2
Total	**8.58**	**0.85**	**9.43**

Table 21-3: Mobilization and demobilization

Description	Unit	$/unit	Total $
Mobilization - Equipment - Year 1	1	72,594	72,594
Mobilization - Equipment - Year 2 -10	9	13,883	124,947
Mobilization - Other - Year 1	1	50,000	50,000
Mobilization Total			**247,541**
Demobilization - Equipment - Year 3-18	16	12,347	197,552
Demobilization - Other - Year 1	1	50,000	50,000
Demobilization Total			**247,541**

21.1.4 Process Plant

The capital cost estimate of the processing plant comprises:

- Direct cost of construction, fabrication and assembly. Acquisitions are considered part of supply of equipment, labour and ancillary equipment for the construction and building materials.

Table 21-4 details estimated capital costs for the processing plant.

Table 21-4: Summary of Process Plant Capital Costs

Area	Capex $M
ROM Primary Crusher	12.42
Stockpile Dome	4.94
Leaching	3.94
ADR Plant	13.35
Spare Parts	0.72
Total	**35.37**

21.1.5 Civil Construction Capital Costs

Table 21-5 lists the cost areas that are considered as capital civil construction costs.

Table 21-5: Civil Construction Capital Costs

Civil Construction	Capex $M	Sustaining $M	Total $M
Support Facilities	13.41	0	13.41
Indirect Cost support facilities	16.22	0	16.22
Leach Pad, Waste dump, PLS and Major Event Ponds	11.55	44.09	55.64
Total	**41.18**	**44.09**	**85.27**

21.1.5.1 Supporting Facilities Capital Costs

The capital cost estimate of supporting facilities includes:

- Spare part costs: required to maintain the installation and one year of operation.
- Contingencies are calculated as a percentage of direct cost (15%).

Table 21-6 show estimated capital costs for the Support Facilities.

Table 21-6: Summary of Support Facilities Capital Costs

Area	Capex $M
Plant maintenance	0.54
Cyanide storage	0.79
Mess and changing rooms	0.41
Drinking and grey water treatment plants	0.23
Camp infrastructure	5.95
Power House Gensets	3.65
Power generation Crushing planta and stockpile	1.84
Total	**13.41**

21.1.5.2 Indirect Cost Support Facilities

Indirect costs include plant construction, contractor costs for items such as mobilization and demobilization, temporary construction facilities, construction quality assurance / quality control, topography support, operation of camps during construction, rental of generator sets and fuel during construction, construction warehouse and fenced yards, support equipment, security and equipment commissioning. These costs have been estimated based on percentages of direct cost on similar projects carried out by HLC.

Indirect costs are summarized in Table 21-7.

Table 21-7: Summary of Indirect Cost

Area	Capex $M
Internal Freight of Equipment and Materials	1.35
Customs	0.23
General Expenses	7.58
Commissioning supervision	0.57
EPCM	6.49
Total	**16.22**

21.1.5.3 Internal Freight of Equipment and Materials

The cost of freight was factored as 2.5% of the direct cost for all equipment and materials purchased in Chile and for imported equipment placed in a Chilean port.

21.1.5.4 Customs Clearance

Import duties were factored as 2.5% of the imported equipment supply cost.

21.1.5.5 General Expenses

General expenses include contractor costs for items such as mobilization and demobilization, temporary construction facilities, construction quality assurance / quality control, survey support, operation of camps during construction, rental of generator sets and fuel during construction, construction warehouse and fenced yards, support equipment and security. These costs have been estimated based on percentages of direct cost on similar projects carried out by HLC. The estimated general expenses represent 14% of the direct cost.

21.1.5.6 Vendor representative

Vendor representatives will be required at the Project site during construction to verify that the installation of the main equipment has been performed in compliance with technical specifications. Representatives will also be required during the pre-commissioning and commissioning stages.

The cost of vendor representatives is estimated at 4% of the supply of electromechanical equipment.

21.1.5.7 EPCM

The estimated cost for engineering, procurement and construction management (EPCM) for the development, construction, and commissioning was based on estimates from HLC. The total estimated EPCM cost is $6.49M, or 12% of the total project direct costs not including mining costs.

The EPCM costs cover services and expenses for the following areas:

- Project management.
- Process engineering, international procurement assistance, technical oversight of detailed engineering / construction management, and commissioning.
- Detailed engineering, heap leach pad and ponds, Crushing plant, site utilities and infrastructure, and Recovery plant.
- Procurement.
- Construction management.

21.1.5.8 Leach Pad, Waste dump and PLS and Major Event Ponds Construction

Costs for Leach Pad, Waste dump and PLS and Major Event Ponds construction have been developed based on available information and historical data pertaining to similar projects in Chile (Table 21-8). Contractor costs and profits are considered unit pricing.

Multiplication of quantities by unit pricing gives a partial price for the construction of Leach Pad, Waste dump and PLS Ponds. Summing partial prices for each item gives the total direct costs for the Leach Pad, Waste dump and PLS Ponds.

The pad construction costs consider levelling and cleaning the surface and placement of the impermeable geomembrane. The processing, transport and placing of sorted material over the liner system is considered as an operational cost.

Indirect project costs have been calculated by applying commonly used percentages for similar projects. Estimated EPCM costs assume a 15% direct cost whilst the cost of the owner assumes 5% of the direct costs.

Table 21-8: Capex Summary -leach pad, waste dumps and PLS ponds

Area	Capex $M	Sustaining $M
Early Works	0.62	1.49
Waste Dump	0.48	0.67
leach Pad	7.93	34.9
PLS Pond	1.01	-
Major Events Pond	-	1.28
EPCM 15%	1.51	5.75
Total	**11.55**	**44.09**

Sustaining capital relates to the construction of phase 2, 3 and 4 of the elements shown in Table 21-9.

Table 21-9: Summary of Construction del pad, waste dumps and PLS ponds

Item	Description	Phase 1 Years 0 - 2 $M	Phase 2 Years 3 - 6 $M	Phase 3 Years 7 - 10 $M	Phase 4 Years 11-16 $M
1	Preliminary and Provisional Work	0.62	0.65	0.35	0.49
2	Leach Pad	7.93	14.43	8.42	12.06
3	Waste Dump	0.48	0.67	0	0
4	PLS Ponds	1.01	0	0	0
5	Major Event Pond	0	0.96	0.32	0
	Works Budget (PO)	**10.04**	**16.72**	**9.08**	**12.54**
	EPCM (S), 15%PO	1.51	2.51	1.36	1.88
	Owners Costs (CP), 5%PO	0.5	0.84	0.45	0.63
	Contingency (CO), 30%*(PO+S+CP)	3.61	6.02	3.27	4.52
	Total	**15.66**	**26.09**	**14.17**	**19.57**
	Area (hectares)	27.5	60.5	36.3	50.6
	Capex by Area ($/m²)	56.9	43.1	39	38.7

21.1.6 Contingency

An estimate of contingency (from -20% to 30%) has been made equivalent to class IV estimation according to the American Association of Cost Engineers (AACE) based on the accuracy and level of detail of the cost estimate. The purpose of the contingency provision is to make allowance for uncertain cost elements which are predicted to occur, but which are not included in the cost estimate. These cost elements include uncertainties concerning completeness and accuracy of material take-offs, accuracy of labour and material rates, accuracy of labour productivity expectations, and accuracy of equipment pricing (Table 21-10).

Table 21-10: Summary of Contingency Costs

Contingency	Capex $M	Sustaining $M	Total
Process Plant and Infrastructure	10.63		10.63
Leach Pad, Waste dump and PLS Ponds Construction	3.6	13.81	17.41
Total	**14.23**	**13.81**	**28.04**

21.1.7 Capital Cost

The current CAPEX includes an estimate for Owner's Costs as shown in Table 21-11. These costs include estimates for Owner's staffing during construction, site communications, Owner's camp, Owner's commissioning, operator training, environmental compliance, community development, land acquisitions, consultants, legal expenses and further metallurgical testing.

Table 21-11: Owners Capital Costs Summary

Area	Capex $M	Sustaining $M	Total $M
Rio2 Overhead @ 100%	4.14	0	4.14
Permitting	0.24	0	0.24
Social & Legal	0.66	0	0.66
Copiapo Office	0.5	0	0.5
Survey, Lv´S, Environ, camp, etc	1.26	2.65	3.91
Earthworks	3.0	0.0	3
Road construction	1.0	0.0	1
Owner Infrastructure (HLC)	0.5	0.0	0.54
Owner Leach Pad (Anddes)	0.5	1.92	2.42
Total	**11.84**	**4.57**	**16.41**

21.1.8 Mine closure

Mine closure costs have been calculated in three phases:

- Closure between 2034 and 2038.
- Final closure 2039 to 2041.
- Post Closure 2042 to 2043.

Mine closure considers profiling of pits, waste dumps, dismantling of built structures; estimated closure costs are presented in Table 21-12.

Table 21-12: Mine Closure

Activity	$Millions
Progressive Closure	**3.2**
Profiling of Slopes and Waste Dump	1.5
Profiling of Pit	1.7
Final Closure	**6.41**
Profiling of Pit	0.3
Dismantling Process Plant	0.8
Dismantling Workshops	0.25
Dismantling of Offices and Camp	0.15
Washing Leach Pad	0.24
Profiling Leach Pad Slope	0.9
Profiling Pit	1.3
Profiling Waste Dump	2.47
Post Closure	**0.39**
Road Closure	0.2
Power House Closure	0.01
Dismantling Pumping System	0.04
Maintenance	0.1
Monitoring	0.04
Sub - Total	**10**
Direct Costs 25%	2.5
Contingency 10%	1
VAT 19%	1.9
Total Closure	**15.4**

21.2 Operating Costs

Operating costs averaged over the life of mine are presented in Table 21-13.

Table 21-13: Summary of Operating Costs

Area	LOM Cost $M	US $/t ore
Mining	505.8	4.4
Processing	467.2	4.1
G&A	228.6	1.99
Off-site Overhead	41.5	0.36
Gold Sales, Insurance, Legal and Social	27.4	0.24
Royalty	1.3	0.01
Total	**1,271.8**	**11.1**

21.2.1 Mine Operating Cost

Mine Operating costs summary incurred over the life of mine are presented Table 21-14. Mining in pit, refers to the cost of extracting in-situ material, that is material that requires drilling and blasting:

- Rehandle stockpile to crusher - Refers to the cost of hauling low-grade material previously deposited on the low-grade stockpile to the crusher.
- Rehandle crusher to pad – Refers to the costs implied by hauling material crushed to the required size to the leach pad.

Table 21-14: Mine Operating Cost Summary

Description	Units	Value
Material mined	$/t mined	1.8
Reclaimed Ore from Stockpile	$/t ore stock	0.65
Crusher to Pad	$/t ore	0.91
Blast Holes Assay	$/t mined	0.02
Total	**$/t ore**	**4.4**

Mining costs were estimated by STRACON, a specialist earth-moving contractor with significant mining experience. STRACON determined equipment counts and productivities based on operational usage mechanical availability and its experience of similar equipment (see Table 21-15 and Table 21-16).

Table 21-15: Mining Equipment Productivity

Productivity	Unit	Value
Drill (Atlas Copco DM45)	m/h	37
Excavator 90t (Mine)	t/h (wet)	847
Excavator 90t (Rehandle Stockpile)	t/h (wet)	847
Loader Cat966 (Rehandle Crusher)	t/h (wet)	596

Table 21-16: Availability and Utilization

Equipment	Availability %	Utilization %
Drill (Atlas Copco DM45)	85	65
Excavator 90t (Mine)	90	79
Excavator 90t (Rehandle Stockpile)	90	79
Loader Cat966 (Rehandle Crusher)	90	78
Tip Truck 43t	90	78

Table 21-17 summarizes calculated costs for different Mining activities like drilling, blasting, loading, hauling and ancillary.

Table 21-17: Mining Cost

Activity	Mining in Pit $/t (dry)	Rehandle Stockpile to Crusher $/t (dry)	Rehandle Crusher to Pad $/t (dry)
Drilling	0.15		
Blasting	0.29		
Loading	0.23	0.24	0.12
Hauling	0.78	0.32	0.75
Ancillary	0.35	0.10	0.04
Total	**1.81**	**0.65**	**0.91**

21.2.2 Process Plant

The process operating costs were developed by HLC based on: interpretation of metallurgical test work, supplier quotes for reagents and consumables, HLC's cost database; and calculations from first principles.

The operating cost estimate is expressed in second quarter 2019 United States dollars.

The monthly process plant operating costs are summarized in Table 21-18, and Figure 21-1 shows the distribution of the costs. Rental costs are not included.

Table 21-18: Operating Costs per item of Expenditure

Area	Unit Cost		
	$/Month	$/t	$/Oz
Manpower	178,779	0.3	27
Materials and Supplies	967,847	1.613	149
Services	317,121	0.529	49
Maintenance	60,000	0.1	0
Total	**1,523,747**	**2.54**	**234**

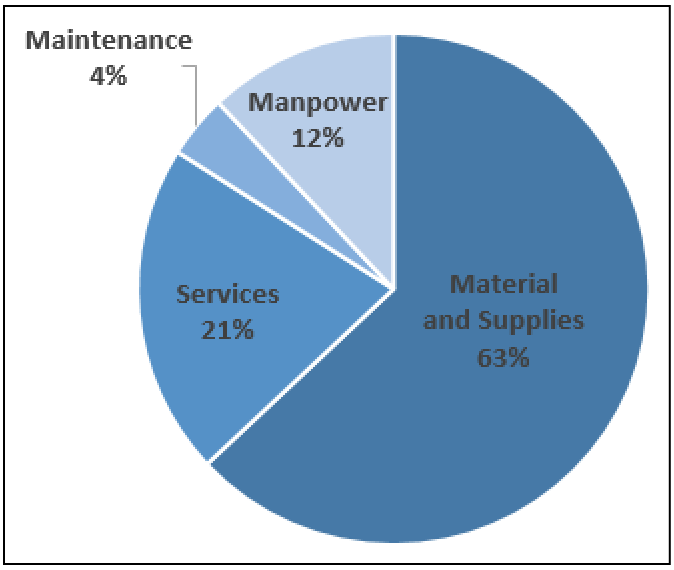


Figure 21-1: Distribution of Operating Costs for the Processing Plant by Category

Water for mineral processing needs to be trucked to site from Copiapo; this will be managed using 30,000 litre trucks, with a gross weight not exceeding 45t.

Pricing includes equipment and its maintenance, a maintenance workshop and yard based in Copiapo, supervision, operators, maintenance labour and support equipment. Pricing is based on $0.70/l diesel price. Pricing includes the cost of water to be supplied by Rio2 in Copiapo, at a cost of $0.75/m^3.

The price for the first four years is $14.42/m^3.

The price thereafter is $14.02/m^3.

The cost of water per tonne of processed mineral is estimated at $1.56/t for the first four years of the mine plans, reducing to $1.51/t in subsequent years. Total processing costs are summarized in Table 21-19.

Table 21-19: Total Processing Cost

Area	Unit Cost		
	$M/year	$/t	$/Oz
Crush and ADR	18.54	2.54	213.0
Water cost	11.12	1.52	127.8
Total	**29.66**	**4.06**	**340.8**

21.2.3 General and Administrative Expenses

Rio2 has developed estimated administrative cost; estimates based on the number of operational personnel at site according to the mine plan. The number of personnel at site affects supply requirements such as; camp requirements, personal protective equipment, fuel, power, medical exams, auditing, vehicle administration and maintenance, personnel and transport. Workers' salaries account for approximately 40% of administrative costs.

Table 21-20 details estimated general and administrative costs by category for different periods of the mine plan.

Figure 21-2 breaks down the costs of the ten most costly General and Administrative costs. General and Administrative costs totalling less than 1% are included as other items.

Table 21-20: Administrative Costs

$Millions	Year 1	Year 2 to 5	Year 6 to 11	Year 12 to 13	Year 14 to 16	Year 17	Percentage
Salaries	5.11	6.43	6.43	6.43	3.64	1.22	41.6%
Food	2.79	3.45	3.65	2.89	1.79	0.23	21.0%
Accommodation and Laundry	0.88	1.09	1.15	0.91	0.55	0.07	6.6%
Materials	0.87	0.81	0.80	0.80	0.52	0.26	5.8%
Buses	0.65	0.65	0.65	0.65	0.32	0.05	4.2%
Collaboration Agreements with Communities	0.20	0.50	0.50	0.50	0.20	0.10	2.8%
First Aid Services	0.62	0.62	0.62	0.62	0.57	0.10	4.5%
Workplace Welfare Programs	0.20	0.30	0.30	0.25	0.20	0.10	1.9%
Fuel	0.25	0.25	0.25	0.25	0.13	0.03	1.7%
IT (Software)	0.25	0.20	0.20	0.20	0.20	0.00	1.5%
Regional Social Projection Activities	0.15	0.15	0.15	0.15	0.15	0.10	1.2%
Energy	0.12	0.12	0.12	0.12	0.12	0.05	0.9%
EPP´S	0.10	0.13	0.14	0.12	0.07	0.01	0.8%
I.C. STRACON	0.11	0.11	0.11	0.11	0.11	0.04	0.8%
Vehicle Maintenance	0.12	0.12	0.12	0.12	0.06	0.01	0.8%
Occupational Medical Examinations	0.08	0.10	0.12	0.12	0.06	0.03	0.7%
IT (Internet and Mobile)	0.09	0.10	0.10	0.10	0.09	0.02	0.7%
Other Institutional Activities	0.10	0.08	0.08	0.08	0.08	0.05	0.6%
Service Consulting, Audits and Outsourcing Collaboration	0.10	0.10	0.10	0.10	0.00	0.00	0.6%
Air Transport	0.07	0.07	0.07	0.07	0.03	0.02	0.5%
Payroll Management Service	0.05	0.07	0.07	0.07	0.05	0.02	0.5%
Fire Extinguishers	0.02	0.02	0.02	0.02	0.02	0.00	0.1%
Land Transport	0.01	0.01	0.01	0.01	0.01	0.00	0.1%
IT (Printers)	0.01	0.01	0.01	0.01	0.01	0.00	0.0%
Total	**12.93**	**15.46**	**15.75**	**14.68**	**8.97**	**2.52**	**1.00**

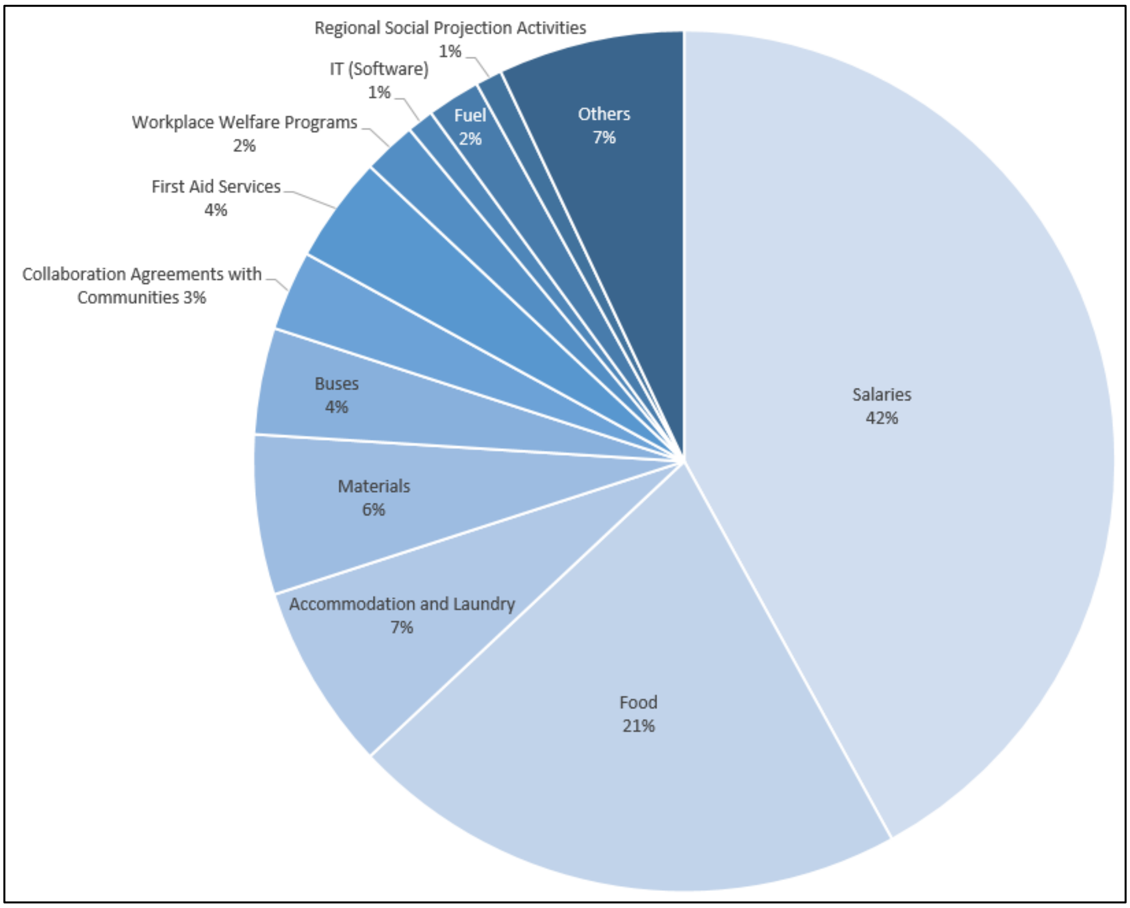


Figure 21-2: Distribution of G&A

22 ECONOMIC ANALYSIS

MP has reviewed and verified the economic model generated by Rio2 with capital and operating cost inputs from Anddes, HLC, STRACON, and Rio2 and that the model was prepared with sound engineering and financial principles and is correct. The financial indicators stated have improved slightly over the 2014 PFS.

22.1 Introduction

The financial evaluation presents the determination of the NPV, payback period (time in years to recapture the initial capital investment), and the IRR for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production cost, and sales revenue. Revenues are based on the gold production. The estimates of capital expenditures and site production costs were developed specifically for this project and have been presented in earlier sections of this report.

22.2 Mine Production Statistic

Mine production is reported as ore high grade, ore low-grade and waste from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report. A total of 115 million tonnes of ore are mined at an average grade of 0.49 g/t Au. A total of 93.55 million tonnes of waste are mined for a stripping ratio of 0.81:1.

22.3 Plant Production Statistic

The design basis for the process plant is:

- 20,000 tpd production.
- 75% gold recovery.

Estimated life of mine gold production is 1.37 million ounces.

22.4 Capital Expenditure

22.4.1 Initial Capital

The base case financial indicators have been determined using the assumption of 100% equity financing of the initial capital. The total initial capital estimate for the project, which includes construction of infrastructure, owners' costs and contingencies, is $111.2 million. A breakout of the capital cost is shown in Section 21. Approximately 87% of capital will be spent during the first year, the remaining will be spent during the second year.

22.4.2 Sustaining Capital

Sustaining capital, estimated at $95M, considers infrastructure maintenance, demobilization of equipment, construction of phases 2, 3 and 4 of the leach pad and respective contingencies.

22.4.3 Revenue

Annual revenue is determined by applying estimated gold prices to the annual payable metal estimated for each operating year (Table 22-1). Sales prices have been applied to all life of mine production without escalation or hedging. Revenue is the gross value of payable gold. The gold price assumptions used in the economic model is $1300/oz.

Table 22-1: Revenue Summary

Description	$M
Revenue	1782.9
Royalty	1.3
Refining & Transportation	27.4
Net Revenue	**1,754.2**

22.5 Total Operating Cost

The average Total Operating Cost over the life of the mine has been estimated on a "per tonne of ore" processed basis at $11.06 (Table 22-2). Total Operating Costs include; mine operations (reclamation of low-grade mineral from the stockpile, and from the crusher to the leach pad), process plant operations, general and administrative costs, off-site overheads, selling costs and royalties.

Table 22-2: Operating Cost Summary

Description	$/t ore
Mining	4.4
Processing	4.06
G&A	1.99
Off-site Overhead	0.36
Gold Sales, Insurance, Legal and Social	0.24
Royalty	0.01
Total	**11.06**

22.5.1 Total Cash Cost

The average Total Cash Cost over the life of the mine is estimated to be $927 per ounce of payable gold. Total Cash Cost for the project is summarized in Table 22-3.

Table 22-3: Cash Cost Summary

Description	$M	$/Oz Au*
Mining	505.8	368.8
Processing	467.2	340.6
G&A	228.6	166.7
Off-site Overhead	41.5	30.3
Gold Sales, Insurance, Legal and Social	27.4	20
Royalty	1.3	1
Total	**1,271.8**	**927.4**

*$/Oz Gold recovered

22.6 All In Sustaining Cost

Average all in Sustaining Costs over the life of the mine are estimated at $997 per ounce of payable gold.

22.6.1 Reclamation & Closure

Cash flow projections include an allowance of $15.4M for closure costs. Table 22-4 details estimated mine closure costs.

Table 22-4: Estimated Mine Closure Costs

Year	Unit	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Yr 21	Total $M
Progressive Closure	M$	**0.55**	**0.55**	**0.55**	**0.60**	**0.95**	**-**	**-**	**-**	**-**	**-**	**3.20**
Profiling of Slopes and Waste Dump	M$	0.25	0.25	0.25	0.30	0.45	-	-	-	-	-	1.50
Profiling of Pit	M$	0.30	0.30	0.30	0.30	0.50	-	-	-	-	-	1.70
Final Closure	M$	**-**	**-**	**-**	**-**	**-**	**1.50**	**2.12**	**2.79**	**-**	**-**	**6.41**
Profiling of Pit	M$	-	-	-	-	-	0.30	-	-	-	-	0.30
Plant Dismantling	M$	-	-	-	-	-	-	-	0.80	-	-	0.80
Workshop Dismantling	M$	-	-	-	-	-	-	-	0.25	-	-	0.25
Office and Camp Dismantling	M$	-	-	-	-	-	-	-	0.15	-	-	0.15
Pad cleaning	M$	-	-	-	-	-	-	0.12	0.12	-	-	0.24
Profiling of Pad Slopes	M$	-	-	-	-	-	0.30	0.30	0.30	-	-	0.90
Profiling of Pit	M$	-	-	-	-	-	0.30	0.50	0.50	-	-	1.30
Profiling of Waste Dump	M$	-	-	-	-	-	0.60	1.20	0.67	-	-	2.47
Post Closure	M$	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**0.32**	**0.07**	**0.39**
Road Closure	M$	-	-	-	-	-	-	-	-	0.20	-	0.20
Closure of Power Plant	M$	-	-	-	-	-	-	-	-	0.01	-	0.01
Pumping System Dismantling	M$	-	-	-	-	-	-	-	-	0.04	-	0.04
Maintenance	M$	-	-	-	-	-	-	-	-	0.05	0.05	0.10
Monitoring	M$	-	-	-	-	-	-	-	-	0.02	0.02	0.04
Sub Total	M$	0.55	0.55	0.55	0.60	0.95	1.50	2.12	2.79	0.32	0.07	10.00
Direct Costs 25%	M$	0.14	0.14	0.14	0.15	0.24	0.38	0.53	0.70	0.08	0.02	2.50
Contingency 10%	M$	0.06	0.06	0.06	0.06	0.10	0.15	0.21	0.28	0.03	0.01	1.00
VAT 19%	M$	0.10	0.10	0.10	0.11	0.18	0.29	0.40	0.53	0.06	0.01	1.90
Total per Year	M$	**0.85**	**0.85**	**0.85**	**0.92**	**1.46**	**2.31**	**3.26**	**4.30**	**0.49**	**0.11**	**15.40**
Total Post Closure	M$	4.93	-	-	-	-	9.87	-	-	0.60	-	15.40

22.7 Depreciation

Tax depreciation was calculated at an annual rate of 25%.

22.8 Taxes

22.8.1 Royalty Tax

In accordance with Chilean law, royalty tax is applied to operating profit at progressive rates from 0% to 1.93% based on operating margin (operating profit divided by sales). Total royalties during the operation of the mine are estimated at $1.3M.

22.8.2 Tax Rate

An income tax of 27% has been applied.

22.9 Project Financing

The financial model has been prepared on the assumption that the project will be financed 100% with equity.

22.10 Net Present Value, Internal of Return, Payback

The economic analyses for the project are summarized in Table 22-5.

Table 22-5: Financial Analysis Results

$M	After Tax	Pre Tax
NPV @ 0%	222	305
NPV @ 5%	121	168
IRR	27.40%	31.90%
Payback Years	4.3	3

22.11 Sensitivity Analysis

The results of the sensitivity analysis for the project after taxes are shown in Table 22-6, Table 22-7, Table 22-8, Figure 22-1 and Figure 22-2.

Table 22-6: NPV and IRR Sensitivity Analysis @5% - After taxes – Gold Price Variation

Sensitivity to Gold Price			
Gold Price ($/oz)	*$ 1,200*	*$ 1,300*	*$ 1,400*
NPV (5% after tax)	$60M	$121M	$181M
IRR (after tax)	17.50%	27.40%	36.10%

Table 22-7: NPV and IRR Sensitivity Analysis @5% - After taxes – Capital Costs Variation

Sensitivity to Capital Costs			
Capital Costs	*-10%*	*$111M*	*+10%*
NPV (5% after tax)	$128M	$121M	$113M
IRR (after tax)	31.20%	27.40%	24.30%

Table 22-8: NPV and IRR Sensitivity Analysis @5% - After taxes – Operating Costs Variation

Sensitivity to Operating Costs			
Operating Costs	*-10%*	*$1,272M*	*+10%*
NPV (5% after tax)	$176M	$121M	$65M
IRR (after tax)	34.90%	27.40%	18.80%

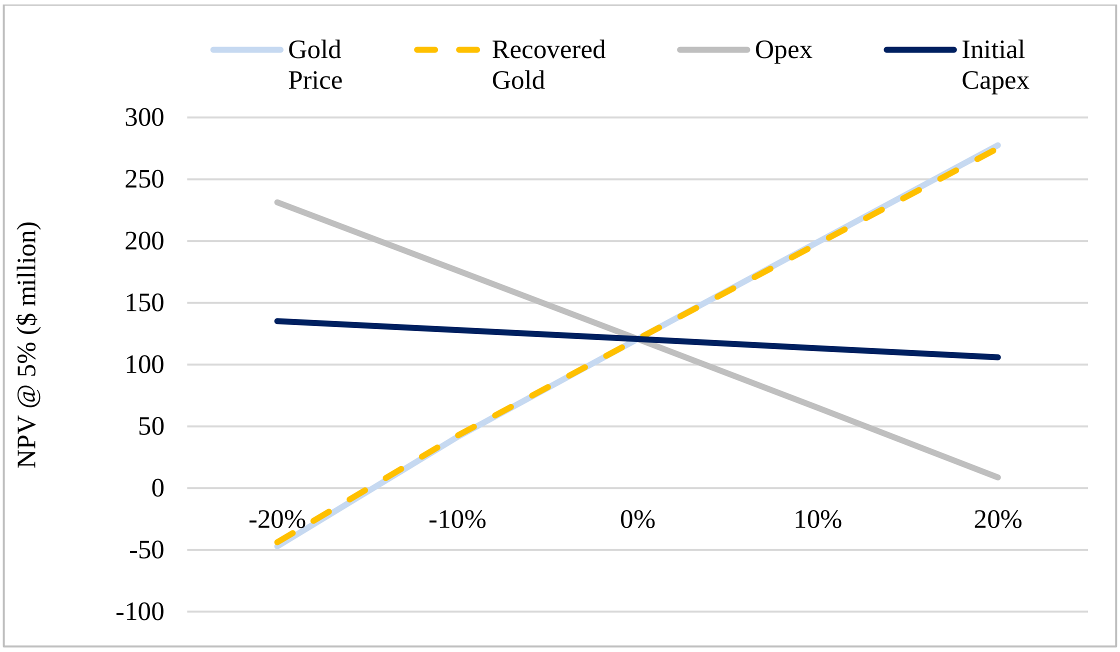


Figure 22-1: NPV Sensitivity Analysis @5% - After taxes ($Millions)

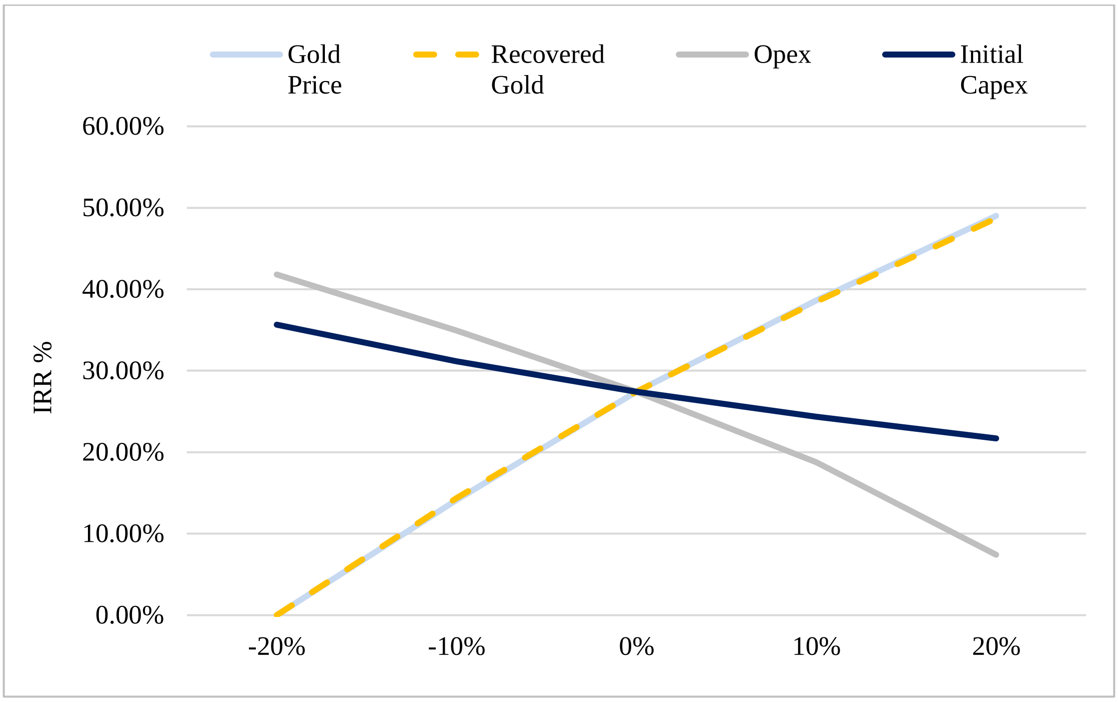


Figure 22-2: IRR Sensitivity Analysis @5% - After taxes

22.12 Detailed Financial Model

		Total /	Year Ended December 31																									
		Average	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
Production Schedule																												
MINING SCHEDULE																												
Ore mined	(000's Tonnes)	81,936	--	--	--	--	--	4,377	7,300	7,300	7,300	7,300	6,980	7,300	3,988	7,300	6,466	7,075	4,621	4,629	--	--	--	--	--	--	--	--
Head grade Au - mined	(g/t)	0.57	--	--	--	--	--	0.76	0.59	0.59	0.59	0.58	0.55	0.56	0.50	0.56	0.60	0.56	0.52	0.42	--	--	--	--	--	--	--	--
Low grade Ore to Stockpile	(000's Tonnes)	33,331	--	--	--	--	--	4,311	1,452	2,164	2,838	5,382	4,729	1,291	1,972	4,093	2,590	1,306	977	226	--	--	--	--	--	--	--	--
Head grade Au - Ore to stockpile	(g/t)	0.30	--	--	--	--	--	0.35	0.27	0.28	0.32	0.31	0.30	0.27	0.28	0.30	0.29	0.28	0.27	0.22	--	--	--	--	--	--	--	--
Waste Mined	(000's Tonnes)	93,549	--	--	--	--	--	3,055	7,756	7,097	6,544	7,277	8,475	11,876	11,324	9,156	10,174	5,306	3,435	2,074	--	--	--	--	--	--	--	--
Strip Ratio	(Waste:Ore)	0.81	--	--	--	--	--	0.35	0.89	0.75	0.65	0.57	0.72	1.38	1.90	0.80	1.12	0.63	0.61	0.43	--	--	--	--	--	--	--	--
Reclaimed Ore from Stockpile	(000's Tonnes)	33,104	--	--	--	--	--	--	--	--	--	--	320	--	3,312	--	834	225	2,679	2,671	7,300	7,300	7,300	1,163	--	--	--	--
Head grade Au - from stockpile	(g/t)	0.30	--	--	--	--	--	--	--	--	--	--	0.59	--	0.35	--	0.33	0.33	0.33	0.33	0.30	0.28	0.28	0.28	--	--	--	--
Total Processed Ore	(000's Tonnes)	115,040	--	--	--	--	--	4,377	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	1,163	--	--	--	--
Head grade Au - processed ore	(g/t)	0.49	--	--	--	--	--	0.76	0.59	0.59	0.59	0.58	0.56	0.56	0.43	0.56	0.57	0.55	0.45	0.39	0.30	0.28	0.28	0.28	--	--	--	--
PROCESS SCHEDULE																												
Tonnes ore to the leach pad	(000's Tonnes)	115,040	--	--	--	--	--	4,377	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	7,300	1,163	--	--	--	--
Head grade Au	(g/t)	0.49	--	--	--	--	--	0.76	0.59	0.59	0.59	0.58	0.56	0.56	0.43	0.56	0.57	0.55	0.45	0.39	0.30	0.28	0.28	0.28	--	--	--	--
Contained Metals from Processing Feed																												
Gold in Ore Processed	(000's oz)	1,829	--	--	--	--	--	108	139	139	139	136	130	131	101	132	134	130	106	92	71	65	65	10	--	--	--	--
Recovery Rates																												
Gold	%	75.0%	--	--	--	--	--	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	--	--	--	--
Recovered Metals																												
Gold	(000's oz)	1,371	--	--	--	--	--	81	104	104	104	102	98	98	76	99	101	97	80	69	53	49	49	8	--	--	--	--
Payability																												
Gold	%	100.0%	--	--	--	--	--	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	--	--	--	--
Total Payable Production																												
Gold	(000's oz)	1,371	--	--	--	--	--	81	104	104	104	102	98	98	76	99	101	97	80	69	53	49	49	8	--	--	--	--
Revenue																												
Total Revenue	**(US$MM)**	**$1,782.9**	--	--	--	--	--	**$104.8**	**$135.4**	**$135.6**	**$135.6**	**$132.4**	**$127.1**	**$127.7**	**$98.3**	**$129.1**	**$131.1**	**$126.7**	**$103.8**	**$89.3**	**$69.2**	**$63.4**	**$63.4**	**$10.1**	--	--	--	--
Royalty																												
NSR Royalty	%		--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Government Royalty	%		--	--	--	--	--	$0.1	$0.2	$0.2	$0.2	$0.1	$0.1	$0.1	$0.0	$0.1	$0.1	$0.1	$0.0	$0.0	--	--	--	--	--	--	--	--
Total Royalty	**(US$MM)**	**$1.3**	--	--	--	--	--	**$0.1**	**$0.2**	**$0.2**	**$0.2**	**$0.1**	**$0.1**	**$0.1**	**$0.0**	**$0.1**	**$0.1**	**$0.1**	**$0.0**	**$0.0**	--	--	--	--	--	--	--	--
Gold Sales, Insurance, Legal and Social																												
Gold Sales	(US$/oz)	$10.0	--	--	--	--	--	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	--	--	--	--
Insurance, Legal and Social Costs	(US$/oz)	$10.0	--	--	--	--	--	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	$10.00	--	--	--	--
Gold Sales, Insurance, Legal and Social	**(US$MM)**	**$27.4**	--	--	--	--	--	**$1.6**	**$2.1**	**$2.1**	**$2.1**	**$2.0**	**$2.0**	**$2.0**	**$1.5**	**$2.0**	**$2.0**	**$1.9**	**$1.6**	**$1.4**	**$1.1**	**$1.0**	**$1.0**	**$0.2**	--	--	--	--
NSR Revenue																												
Gross Revenue	(US$MM)	$1,782.9	--	--	--	--	--	$104.8	$135.4	$135.6	$135.6	$132.4	$127.1	$127.7	$98.3	$129.1	$131.1	$126.7	$103.8	$89.3	$69.2	$63.4	$63.4	$10.1	--	--	--	--
Royalty	(US$MM)	($1.3)	--	--	--	--	--	($0.1)	($0.2)	($0.2)	($0.2)	($0.1)	($0.1)	($0.1)	($0.0)	($0.1)	($0.1)	($0.1)	($0.0)	($0.0)	--	--	--	--	--	--	--	--
Refining & Transportation	(US$MM)	($27.4)	--	--	--	--	--	($1.6)	($2.1)	($2.1)	($2.1)	($2.0)	($2.0)	($2.0)	($1.5)	($2.0)	($2.0)	($1.9)	($1.6)	($1.4)	($1.1)	($1.0)	($1.0)	($0.2)	--	--	--	--
NSR Revenue	**(US$MM)**	**$1,754.1**	--	--	--	--	--	**$103.2**	**$133.1**	**$133.3**	**$133.3**	**$130.3**	**$125.1**	**$125.6**	**$96.7**	**$127.0**	**$129.0**	**$124.6**	**$102.1**	**$87.9**	**$68.2**	**$62.4**	**$62.4**	**$9.9**	--	--	--	--
Operating Costs																												
Total Operating Costs																												
Mining	(US$MM)	$505.8	--	--	--	--	--	$25.3	$36.7	$36.7	$37.0	$42.9	$43.5	$43.8	$40.2	$44.0	$42.1	$31.7	$24.8	$21.0	$11.4	$11.4	$11.4	$1.8	--	--	--	--
Processing	(US$MM)	$467.2	--	--	--	--	--	$17.9	$29.9	$29.9	$29.9	$29.6	$29.6	$29.6	$29.6	$29.6	$29.6	$29.6	$29.6	$29.6	$29.6	$29.6	$29.6	$4.7	--	--	--	--
G&A	(US$MM)	$228.6	--	--	--	--	--	$12.9	$15.5	$15.5	$15.5	$15.5	$15.8	$15.8	$15.8	$15.8	$15.8	$15.8	$14.7	$14.7	$9.0	$9.0	$9.0	$2.5	--	--	--	--
Off-site Overhead	(US$MM)	$41.5	--	--	--	--	--	$2.9	$2.9	$2.9	$2.9	$2.9	$2.9	$2.9	$2.9	$2.9	$2.9	$2.9	$2.9	$2.9	$1.0	$1.0	$1.0	$0.3	--	--	--	--
Gold Sales, Insurance, Legal and Social	(US$MM)	$27.4	--	--	--	--	--	$1.6	$2.1	$2.1	$2.1	$2.0	$2.0	$2.0	$1.5	$2.0	$2.0	$1.9	$1.6	$1.4	$1.1	$1.0	$1.0	$0.2	--	--	--	--
Royalty	(US$MM)	$1.3	--	--	--	--	--	$0.1	$0.2	$0.2	$0.2	$0.1	$0.1	$0.1	$0.0	$0.1	$0.1	$0.1	$0.0	$0.0	--	--	--	--	--	--	--	--
Total Operating Costs	**(US$MM)**	**$1,271.8**	--	--	--	--	--	**$60.8**	**$87.3**	**$87.4**	**$87.6**	**$93.1**	**$93.9**	**$94.2**	**$90.0**	**$94.4**	**$92.6**	**$82.1**	**$73.6**	**$69.6**	**$52.0**	**$51.9**	**$51.9**	**$9.5**	--	--	--	--
Cash Costs Per Ounce	**(US$/oz)**	**$927**	--	--	--	--	--	**$754**	**$838**	**$838**	**$840**	**$914**	**$960**	**$959**	**$1,191**	**$951**	**$918**	**$842**	**$923**	**$1,013**	**$977**	**$1,065**	**$1,065**	**$1,221**	--	--	--	--

		Total / Average	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
			Year Ended December 31																									
Fixed Asset Schedule			2018E	2019E	2020E	2021E	2022E	2023E	2024E	2025E	2026E	2027E	2028E	2029E	2030E	2031E	2032E	2033E	2034E	2035E	2036E	2037E	2038E	2039E	2040E	2041E	2042E	2043E
Capital Expenditures																												
Owners Costs	(US$MM)	$5.5	--	--	--	--	$5.5	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Mine Capex	(US$MM)	$6.0	--	--	--	--	$4.3	$1.8	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Infrastructure and Equipment	(US$MM)	$99.7	--	--	--	--	$86.0	$13.6	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capex	(US$MM)	$95.0	--	--	--	--	--	--	$27.3	$1.2	$1.4	$1.4	$15.5	$1.4	$1.2	$1.2	$21.2	$1.3	$2.0	$2.2	$2.0	$2.3	$2.6	$2.7	$3.3	$4.3	$0.5	$0.1
Total Capital Expenditures	**(US$MM)**	**$206.2**	**--**	**--**	**--**	**--**	**$95.8**	**$15.4**	**$27.3**	**$1.2**	**$1.4**	**$1.4**	**$15.5**	**$1.4**	**$1.2**	**$1.2**	**$21.2**	**$1.3**	**$2.0**	**$2.2**	**$2.0**	**$2.3**	**$2.6**	**$2.7**	**$3.3**	**$4.3**	**$0.5**	**$0.1**
AISC Cash Costs Per Ounce	**(US$/oz)**	**$997**	**--**	**--**	**--**	**--**	**--**	**$754**	**$1,100**	**$849**	**$853**	**$928**	**$1,118**	**$974**	**$1,207**	**$963**	**$1,128**	**$855**	**$948**	**$1,045**	**$1,015**	**$1,113**	**$1,119**	**$1,572**	**--**	**--**	**--**	**--**
Depreciation Schedule																												
Opening balance	(US$MM)		--	--	--	--	--	$95.8	$107.0	$125.4	$117.6	$110.0	$102.2	$107.1	$97.8	$88.2	$78.4	$85.7	$72.9	$60.4	$47.5	$33.9	$19.4	$3.0	--	--	--	--
Additions	(US$MM)	$205.6	--	--	--	--	$95.8	$15.4	$27.3	$1.2	$1.4	$1.4	$15.5	$1.4	$1.2	$1.2	$21.2	$1.3	$2.0	$2.2	$2.0	$2.3	$2.6	$2.7	$3.3	$4.3	$0.5	$0.1
Depreciation	(US$MM)	($205.6)	--	--	--	--	--	($4.2)	($8.9)	($8.9)	($9.0)	($9.2)	($10.5)	($10.7)	($10.8)	($11.0)	($13.9)	($14.1)	($14.5)	($15.0)	($15.7)	($16.8)	($19.0)	($5.8)	($3.3)	($4.3)	($0.5)	($0.1)
Closing balance	(US$MM)	--	--	--	--	--	$95.8	$107.0	$125.4	$117.6	$110.0	$102.2	$107.1	$97.8	$88.2	$78.4	$85.7	$72.9	$60.4	$47.5	$33.9	$19.4	$3.0	--	--	--	--	--
Tax Depreciation Schedule																												
Straight Line Depreciation (4-Years)																												
Total Depreciation	(US$MM)	($200.1)	--	--	--	--	($24.0)	($27.8)	($34.6)	($34.9)	($11.3)	($7.8)	($4.9)	($4.9)	($4.9)	($4.8)	($6.2)	($6.2)	($6.4)	($6.7)	($1.9)	($2.1)	($2.3)	($2.4)	($2.7)	($3.2)	($2.7)	($2.0)
Taxes																												
Corporate Taxes																												
Revenue	(US$MM)	$1,782.9	--	--	--	--	--	$104.8	$135.4	$135.6	$135.6	$132.4	$127.1	$127.7	$98.3	$129.1	$131.1	$126.7	$103.8	$89.3	$69.2	$63.4	$63.4	$10.1	--	--	--	--
Operating Costs	(US$MM)	($1,271.8)	--	--	--	--	--	($60.8)	($87.3)	($87.4)	($87.6)	($93.1)	($93.9)	($94.2)	($90.0)	($94.4)	($92.6)	($82.1)	($73.6)	($69.6)	($52.0)	($51.9)	($51.9)	($9.5)	--	--	--	--
Depreciation	(US$MM)	($205.6)	--	--	--	--	--	($4.2)	($8.9)	($8.9)	($9.0)	($9.2)	($10.5)	($10.7)	($10.8)	($11.0)	($13.9)	($14.1)	($14.5)	($15.0)	($15.7)	($16.8)	($19.0)	($5.8)	($3.3)	($4.3)	($0.5)	($0.1)
EBIT	(US$MM)	$305.5	--	--	--	--	--	$39.8	$39.3	$39.3	$39.0	$30.2	$22.7	$22.8	($2.6)	$23.7	$24.6	$30.5	$15.6	$4.7	$1.5	($5.3)	($7.6)	($5.1)	($3.3)	($4.3)	($0.5)	($0.1)
EBIT	(US$MM)	$305.5	--	--	--	--	--	$39.8	$39.3	$39.3	$39.0	$30.2	$22.7	$22.8	($2.6)	$23.7	$24.6	$30.5	$15.6	$4.7	$1.5	($5.3)	($7.6)	($5.1)	($3.3)	($4.3)	($0.5)	($0.1)
Add: Book Depreciation	(US$MM)	$205.6	--	--	--	--	--	$4.2	$8.9	$8.9	$9.0	$9.2	$10.5	$10.7	$10.8	$11.0	$13.9	$14.1	$14.5	$15.0	$15.7	$16.8	$19.0	$5.8	$3.3	$4.3	$0.5	$0.1
Less: Tax Depreciation	(US$MM)	($200.1)	--	--	--	--	($24.0)	($27.8)	($34.6)	($34.9)	($11.3)	($7.8)	($4.9)	($4.9)	($4.9)	($4.8)	($6.2)	($6.2)	($6.4)	($6.7)	($1.9)	($2.1)	($2.3)	($2.4)	($2.7)	($3.2)	($2.7)	($2.0)
Less: NOL Used	(US$MM)	($36.4)	--	--	--	--	--	($16.3)	($13.5)	($6.6)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Taxable Income	(US$MM)	$274.6	--	--	--	--	($24.0)	--	--	$6.7	$36.7	$31.5	$28.4	$28.5	$3.4	$29.9	$32.3	$38.4	$23.7	$13.0	$15.3	$9.3	$9.2	($1.8)	($2.7)	($3.2)	($2.7)	($2.0)
Tax Rate	(%)		--	--	--	--	--	--	--	27%	27%	27%	27%	27%	27%	27%	27%	27%	27%	27%	27%	27%	27%	--	--	--	--	--
Estimated Taxes Payable	(US$MM)	$82.7	--	--	--	--	--	--	--	$1.8	$9.9	$8.5	$7.7	$7.7	$0.9	$8.1	$8.7	$10.4	$6.4	$3.5	$4.1	$2.5	$2.5	--	--	--	--	--
NOL Schedule																												
Beginning Balance	(US$MM)		$12.5	$12.5	$12.5	$12.5	$12.5	$36.4	$20.2	$6.6	--	--	--	--	--	--	--	--	--	--	--	--	--	--	$1.8	$4.5	$7.8	$10.5
Add: NOL Created	(US$MM)		--	--	--	--	$24.0	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	$1.8	$2.7	$3.2	$2.7	$2.0
Less: NOL Used	(US$MM)		--	--	--	--	--	($16.3)	($13.5)	($6.6)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Ending Balance	(US$MM)		$12.5	$12.5	$12.5	$12.5	$36.4	$20.2	$6.6	--	--	--	--	--	--	--	--	--	--	--	--	--	--	$1.8	$4.5	$7.8	$10.5	$12.5
Unlevered Free Cash Flow																												
EBIT	(US$MM)	$305.5	--	--	--	--	--	$39.8	$39.3	$39.3	$39.0	$30.2	$22.7	$22.8	($2.6)	$23.7	$24.6	$30.5	$15.6	$4.7	$1.5	($5.3)	($7.6)	($5.1)	($3.3)	($4.3)	($0.5)	($0.1)
Book depreciation	(US$MM)	$205.6	--	--	--	--	--	$4.2	$8.9	$8.9	$9.0	$9.2	$10.5	$10.7	$10.8	$11.0	$13.9	$14.1	$14.5	$15.0	$15.7	$16.8	$19.0	$5.8	$3.3	$4.3	$0.5	$0.1
Taxes	(US$MM)	($82.7)	--	--	--	--	--	--	--	($1.8)	($9.9)	($8.5)	($7.7)	($7.7)	($0.9)	($8.1)	($8.7)	($10.4)	($6.4)	($3.5)	($4.1)	($2.5)	($2.5)	--	--	--	--	--
Capital expenditures	(US$MM)	($205.6)	--	--	--	--	($95.8)	($15.4)	($27.3)	($1.2)	($1.4)	($1.4)	($15.5)	($1.4)	($1.2)	($1.2)	($21.2)	($1.3)	($2.0)	($2.2)	($2.0)	($2.3)	($2.6)	($2.7)	($3.3)	($4.3)	($0.5)	($0.1)
Unlevered free cash flow	**(US$MM)**	**$222.8**	**--**	**--**	**--**	**--**	**($95.8)**	**$28.7**	**$20.9**	**$45.2**	**$36.7**	**$29.4**	**$10.1**	**$24.3**	**$6.1**	**$25.4**	**$8.6**	**$33.0**	**$21.7**	**$14.0**	**$11.1**	**$6.6**	**$6.4**	**($2.1)**	**($3.3)**	**($4.3)**	**($0.5)**	**($0.1)**

Economics		
After-tax NPV @ 0%	(US$MM)	$222.2
After-tax NPV @ 5%	(US$MM)	$120.6
After-tax IRR	(%)	27.4%
Payback	(years)	4.3

Pre-tax NPV Calculation		
Pre-tax NPV @ 0%	(US$MM)	$304.9
Pre-tax NPV @ 5%	(US$MM)	$167.8
Pre-tax IRR	(%)	31.9%
Payback	(years)	3.0

23 ADJACENT PROPERTIES

Figure 23-1 depicts the location of projects/mines, which are adjacent to the Fenix Gold Project.

The Fenix Gold Project is centred at latitude 27°0'7.00"S and longitude 69°12'58.00"W; approximately 20 km south of Kinross Gold’s La Coipa Au-Ag mine (currently on standby), 60 km north of Kinross’s Maricunga Gold Mine (currently on residual leaching) and 40 km north of Hochschild’s Volcan Gold Project.

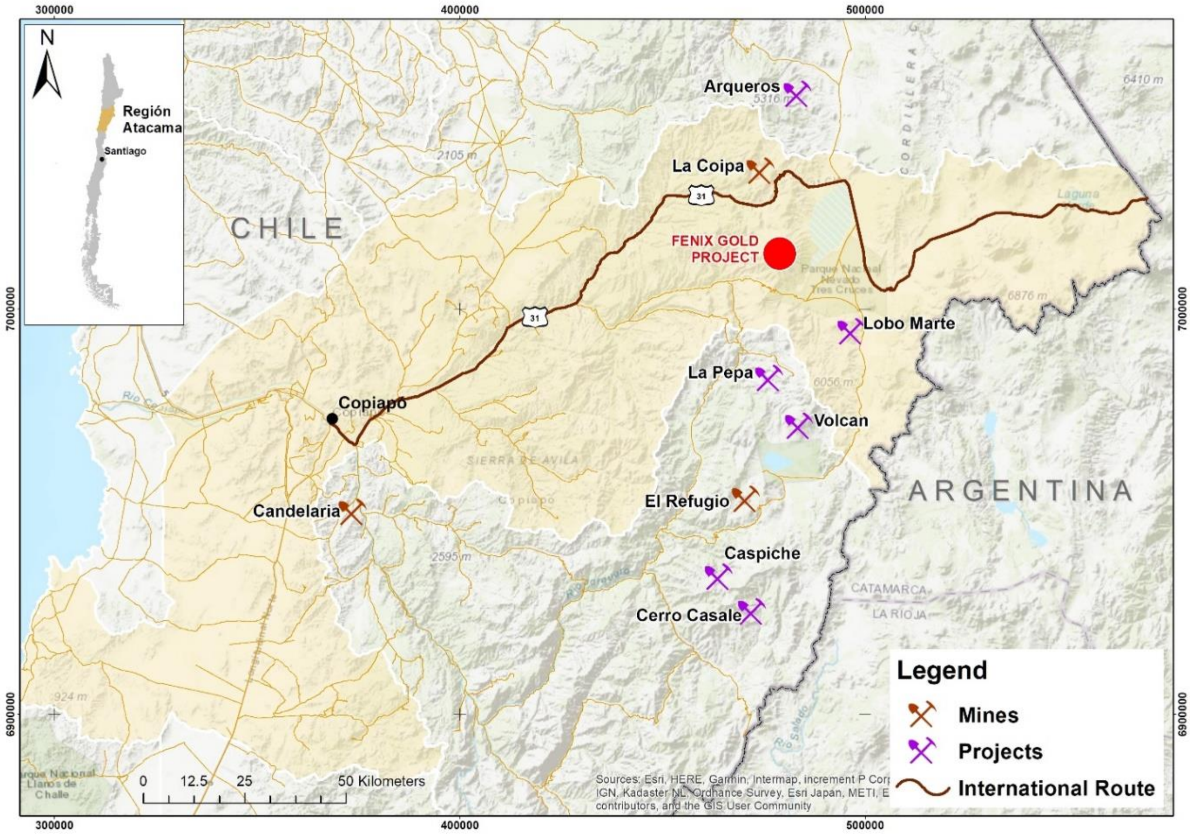


Figure 23-1: Adjacent Properties

24 OTHER RELEVANT DATA AND INFORMATION

As of the effective date of this report, all relevant data and information has been reported.

25 INTERPRETATION AND CONCLUSIONS

The following conclusions have been made:

- The Fenix Gold Project has a 16 year LOM and will produce 1.37M ounces of gold with strong economic returns:
  - LOM AISC $997/Oz.
  - After-tax NPV5 of $121M.
  - After-tax IRR of 27.4% using a base case gold price of $1300/Oz.
  - The project is expected to generate annual after-tax profits of $15.1M.
  - Cumulative LOM after-tax net cash flow of $222M.
- The use of trucked water in place of a piped water supply offers the following advantages when compared to the 2014 FS:
  - Reduced permitting time.
  - Reduced timeline to production.
  - Reduced CAPEX requirements.
- The mine design allows for a reconfiguration and upscaling of mine operations if a piped water supply becomes available.
- The identification of alternative water supplies closer to mine operations offers the potential to reduce operating costs and improve project economics.
- Connection to the Chilean power network (SIC) would improve project economics.
- The plant is designed for easy upscaling from 20,000 tpd to 40,000 tpd and 80,000 tpd.
- Gold recovery of 75% is achievable with simple processing; ore crushed to a P80 size of 4 inches via a single stage Gyratory crusher with lime dosing prior to placement on the stockpile.

26 RECOMMENDATIONS

These are the recommendations for further work to advance the Fenix Gold Project and to prepare for a full construction decision for the 20,000 tpd starter project Rio2.

Recommendations are estimated to cost $3.54M to complete Table 26-1:

Table 26-1: Estimated cost to complete Recommendations

Item	Estimated Cost $ Millions
Complete EIA including studies	1.20
Complete Mechanical and Electrical Engineering	1.00
Investigate Connection to SIC	0.02
Geotechnical Drilling and design	0.60
Condemnation Drilling	0.35
Optimise Mine Schedule	0.02
Model Mg Distribution	0.01
Column Leach Testing of P80 4"	0.15
Mineralogical Analysis of Head Samples	0.02
Trade-off Study Truck v Conveyor to move ore to stockpile	0.02
Production scale pilot tests of ROM	0.15
Total	**3.54**

26.1 General

- Complete environmental baseline studies and begin the environmental impact study presentation process.
- Complete the hydrological and geochemical study required for the EIA.
- Complete mechanical, electrical and geotechnical engineering for all the components of the project to the level adequate to apply for relevant permits.
- Maintain and enhance relationships with relevant social and community groups throughout the EIA process.
- Plan for a phase of trial mining.

26.2 Engineering

- Continue to develop civil engineering for the waste dump, process and heap leach areas to level of construction ready.

26.3 Water

- Continue to review water supply options; new water sources offer the potential to provide time and cost savings and improve project economics, and the potentially expand the project.
- Continue discussions with Trends Industrial SA on their ENAPAC Project to build a desalination plant and a pipeline from the coast to partner mining projects.

26.4 Power Supply

- Investigate the potential to connect the Project to the Chilean power grid (SIC).

26.5 Mining

- Complete a geotechnical drilling program and study to confirm pit design parameters.
- Complete condemnation drilling in pad and waste dump footprints.
- Optimize mine planning and scheduling in order to improve costs.
- Optimization of waste dump, pad and stockpiling distances.
- Define the terms of the proposed mining alliance agreement.
- Source quotes for supply for diesel and explosives.

26.6 Mineral Processing

- Undertake geotechnical laboratory testing in leached ore samples taken after column tests are completed to better understand geotechnical properties such as shear strength and permeability.
- Model Mg distribution to better understand lime consumption.
- A column leach test campaign is recommended on mineral crushed to a P80 size of 4” from Fenix North Fenix Central and Fenix South to optimize gold recovery and reagent consumption. Two tests per zone are recommended for Fenix North and Fenix Central. For Fenix South one test is recommended with Fenix South material and another with a blend of the material from the three zones to confirm the results indicated above in 13.7 item 3. The estimated cost of the campaign is US$ 150,000.
- Mineralogical analyses should be carried out on the head samples at the start of the tests and the residues at the end of the tests.
- It is recommended to carry out the mineralogical analysis on the remaining head samples from the KCA 2017 Fenix South leach tests to determine if chalcocite or other cyanide soluble copper minerals are present or if there are other causes for the higher refractory behaviour.
- For tests with the Fenix South material, copper extraction should be measured at the same frequency as the gold extraction to determine if there is any correlation between the two.

- The Mine Plan currently shows trucking ore from crusher to pad. A trade off study for Trucking from Crusher stockpile vs Conveyor system to Pad needs to be completed. This trade off study should consider the re-handle of the future stockpile material to the Leach Pad also.
- Production scale pilot tests of ROM material for recoveries in the first year of mine production to determine the cost benefit of crushing vs ROM.
- Obtain formal Process plant reagent quotes from Suppliers.
- Continue to develop engineering solutions to manage the impact of the climatic conditions, specifically cold weather and high winds, on the operation of the pad and the ADR Plant at the Fenix Gold Project.
- During the production scale pilot tests and future column tests quantify the as mined moisture content as a percentage of ROM and 4" crushed material. During these tests measure and capture the saturation percentage required for solution to percolate through the mineral, which will help confirm the water requirement for "wetting" mineral, also conduct tests on leached material to capture the residual moisture percentage retained in the mineral.
- Undertake evaporation measurements in the Pad location to confirm the evaporation rate that should be applied to the Leach Pad Water Balance.

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