Report to:

Minera IRL Patagonia SA

Technical Report
Don Nicolás Gold Project
Santa Cruz, Argentina

Document No. 705-1196070100-REP-R0001-00

TETRA TECH

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Minera IRL Patagonia SA

TECHNICAL REPORT DON NICOLÁS GOLD PROJECT

SANTA CRUZ, ARGENTINA

EFFECTIVE DATE: FEBRUARY 14, 2012

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705-1196070100-REP-R0001-00

REVISION HISTORY

REV. NO	ISSUE DATE	PREPARED BY AND DATE	REVIEWED BY AND DATE	APPROVED BY AND DATE	DESCRIPTION OF REVISION
0	February 14, 2012	All QPs	Hassan Ghaffari	Hassan Ghaffari	Final to Client for publishing on Sedar.

NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Minera IRL Limited (MIRL) and its subsidiary Minera IRL Patagonia S.A. (MIRLP) by Wardrop, a Tetra Tech Company (Tetra Tech), Coffey Mining Limited (Coffey Mining), NCL Ingenierá y Construcción Limitada (NCL), Golder Associates Argentina (Golder), and Ausenco Vector (Argentina).The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Tetra Tech's, Coffey Mining's, NCL's, Golder's, and Ausenco Vector's services, based on: (i) information available at the time of preparation, (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use subject to the terms and conditions of MIRL or MIRLP's contract with Tetra Tech, Coffey Mining, NCL, Golder, and Ausenco Vector. This contract permits MIRL to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party's sole risk.

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GLOSSARY

Units of Measure

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	Cdn$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz

Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min

Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

1.0 SUMMARY

The Don Nicolás Gold Project is an advanced-stage gold-silver project located in the southern Argentine province of Santa Cruz, Argentina, approximately 2,000 km south of Buenos Aires. Santa Cruz is generally regarded as being one of the most mining-friendly provinces in the country.

This Technical Report summarizes the results of a study into the technical and economic feasibility of mining and processing a series of gold-silver vein deposits making up the Project. The study is based on conventional open pit mining and standard Carbon-In-Leach (CIL) process technology with production of gold and silver in doré.

The following consultants have contributed to the Feasibility Study (FS) on which this Technical Report is based:

Consultant	Responsibility
Tetra Tech WEI Inc. (Tetra Tech)	➢ Lead Consultant ➢ Overall Coordination ➢ Process Design & Costing
Coffey Mining Limited	➢ Mineral Resources
NCL Ingeniería y Construcción Limitada (NCL)	➢ Mineral Reserves ➢ Mine Design ➢ Costing
Ingenieros Penta Sur & Tetra Tech	➢ Access Roads ➢ Camp & Buildings ➢ Power Supply ➢ Infrastructure costing
Hidroar S.A. & Tetra Tech	➢ Water exploration ➢ Hydro-geological modelling ➢ Water Supply Design ➢ Costing
Golder Associates (San Juan, Argentina office)	➢ Geotechnical field programs ➢ Open pit slope recommendations ➢ Tailings Storage Facility design ➢ TSF costing
Ausenco Vector (Mendoza, Argentina)	➢ Environmental Baseline Studies ➢ Environmental Impact Assessment, in progress

The Don Nicolás gold and silver mineralization is epithermal in style and occurs within a geological environment common to several other exploration and mining projects found in the Deseado Massif geological setting of Santa Cruz province.

The Project area is typical of Santa Cruz province east of the Andes Mountains: low population density, temperate climate, and often windy conditions, but with good road access via national and provincial highways. There are daily commercial air connections from Buenos Aires to several communities in the region. The Atlantic Ocean is only some 100 km to the east of the Project. Oil exploration and production are the principal economic activities in the region. . With an average elevation of only 100 to 200 m amsl, the terrain around the Project area is flat, treeless, rolling topography.

The mineral resources supporting the Project are focused in two areas separated by some 50 km:

a. La Paloma area, specifically the Sulfuro vein system.

b. Martinetas area, specifically the Cerro Oro, Coyote, and Armadillo vein systems.

Production of gold and silver doré will be based on the following criteria as defined in the FS.

Technical Criteria	
Plant Capacity	350,000 t/y 1,000 t/d nominal
Process Technology	Carbon-In-Leach (CIL)
Metal Products	Gold and Silver in doré
Total Metal Produced	181,000 oz Au 190,200 oz Ag
Annual Metal Production (average of mine life)	52,400 oz Au 56,000 oz Ag
Mine Life	3.6 years
Tailings Facility Capacity	1.2 Mt (in two stages)
Daily Mining Rate (average)	11,400 t/d ore and waste
Waste: Ore Strip Ratio	11.8/1
Manpower Requirements	302 (peak in Year 02 of production)

Mineral Resources	Cutoff g/t Au	Mt	Au g/t	Ag g/t	Contained oz Au	Contained oz Ag
• Measured & Indicated	1.6	1.5	6.0	13	280,000	630,000
	0.3	5.6	2.1	6	381,000	1,140,000
Mineral Reserves						
• Proven & Probable	1.5[1]	1.2	5.1	10	197,000	401,000

Economic Criteria	
Site Operating Costs	US$82.5/t processed
	US$528/oz Au recovered (after Ag credit)
Initial Capital Investment	US$55.5 M
Sustaining Capital, LOM	US$7.55 M
Capital Costs Accuracy	+/- 15%
After-Tax Economic Parameters	
• Net Present Value, 7% (real)	US$21.6 M
• IRR	22.8%
• Payback Period	2.0 years

Infrastructure required to support the short mine life of 3.6 years will include a tailings storage facility, access roads, camp and other buildings, a diesel-fired power station, and a water supply system from wells in the Martinetas and El Cóndor camp areas.

The FS has been based on industry-standard design and estimation methods, mineral resources and reserves that comply with Canadian Institute of Mining and Metallurgy Standards, and pre- and post-tax economic analyses that have been reviewed by a professional accounting firm in Argentina.

The Project has been designed on the basis of an extensive series of geological, geotechnical, hydrogeological and environmental work programs completed principally in 2010 and 2011 by qualified professionals in their fields. Metallurgical testwork completed at the Ammtec laboratories in Perth, Australia forms the basis for process flowsheet design.

Environmental base line investigations have been completed for all the required elements of an Environmental Impact Study which is advancing towards completion. The ensuing permitting process is expected to be completed during the second half of 2012. Project implementation and development is expected to take approximately 12 months with target production of gold-silver doré in Q4 2013.

[1] 1.5 g/t Au cutoff at Martinetas, 1.6 g/t Au at Sulfuro

Exploration for extensions to existing vein systems and for new “blue sky” discoveries are considered promising by MIRLP and will be funded in 2012 by an exploration budget separate from the project investment requirements.

Other upside opportunities for the Project include underground development of the deeper, high-grade (HG) shoots at Sulfuro, and heap leach recovery of lower-grade (LG) material around the Martinetas area where a Measured and Indicated resource of 3.5 Mt grading 0.7 g/t Au and 3 g/t Ag (76,000 oz gold and 348,000 oz silver) has been defined between 0.3 g/t Au and 1.5 g/t Au cutoff grades.

The future possibility of connecting to the provincial power grid would have a positive impact on operating costs; however, both technical and economic aspects of such a project need to be studied in more detail.

The Project is located in a mining-friendly region of Argentina. However, exposure to economic variables in the country could be a risk, particularly related to inflation and the lifting of various national subsidies that began in late 2011.

Based on the work carried out in this FS and the resulting economic evaluation, it is recommended that the Don Nicolás Gold Project proceed to the Detailed Engineering Design stage. Please refer to Section 26.0 Recommendations for more information.

2.0 INTRODUCTION

2.1 BACKGROUND

Minera IRL Limited (MIRL) is an international gold mining company with head office in Lima, Peru, and with shares traded on the AIM London Exchange (AIM), the Toronto Stock Exchange (TSX), and the Lima Exchange (BVL).

MIRL produces approximately 30,000 oz gold per year from its Corihuarmi heap leach mine in Peru. It also has a project at Ollachea in Peru which is currently advancing through the FS stage.

MIRL owns a 100% interest in the Don Nicolás Project in Argentina (the Project) through its wholly owned Argentine subsidiary, Minera IRL Patagonia S.A. (MIRLP).

In May 2011, MIRL commissioned Wardrop, A Tetra Tech Company (Tetra Tech) to assume the lead consultant role and complete certain specific areas of the Don Nicolás Project FS which, at that point, had been underway for some six months under the management of another consultant. The principal objectives of the FS are to demonstrate viability and to set out a basis for obtaining project financing.

This Technical Report presents a summary of the FS completed by Tetra Tech and five other consultants in Argentina and Chile, and represents a summary of the following FS volumes:

Volume	Section	Title
I	1.0	Executive Summary
II	2.0	Introduction
	3.0	Property Description
	4.0	Geology and Mineral Resources
III	5.0	Mine Design and Mineral Reserves
IV	6.0	Metallurgy Testwork
	7.0	Process Plant Design
V	8.0	Geotechnical Site Investigations
	9.0	Tailings and Waste Rock Management
	10.0	Hydrology
VI	11.0	Infrastructure and Services
	12.0	Environmental and Socio Economic
VII	13.0	Capital Cost Estimate
	14.0	Operating Cost Estimate
	15.0	Project Implementation
	16.0	Financial Analysis

2.2 SCOPE OF WORK

This FS summarizes the findings of a FS of the Don Nicolás Project located in the Santa Cruz province in the Patagonia region of Argentina.

The FS has been carried out according to the Canadian Institute of Mining & Metallurgy Standards (2010) definition, as follows:

> *A Feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of realistically assumed mining, processing, metallurgical, economic, marketing, legal, environmental, social and governmental considerations together with any other relevant operational factors and detailed financial analysis, that are necessary to demonstrate at the time of reporting that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher than that of a Pre-Feasibility Study.*

The capital cost estimate for the mine, process plant and infrastructure has been prepared in accordance with standard industry practices to give an intended level of accuracy of +/- 15%. Operating costs have also been estimated to +/- 15%, subject to the process and cost assumptions therein. An economic analysis, including taxation during construction and operation of the plant has been carried out for the current projected life of mine of approximately four years by the Client, MIRLP, with Tetra Tech's review.

The physical scope of this FS addresses the proposed operation of the open pit mines and process plant, along with related infrastructure requirements within the property perimeter related to access, power generation, personnel accommodation (construction and permanent), and general administration activities.

Exclusions from the study include:

a. Land Acquisition.
b. MIRL Head Office Costs.

2.3 ROLES AND RESPONSIBILITIES

This FS has been compiled and written by Tetra Tech and subcontractors to MIRLP or MIRLP, or MIRL.

Table 2-1: Summary of Roles and Responsibilities

Item	Report Section	Responsible	QP
1	Summary	Tetra Tech	C. Grant
2	Introduction	Tetra Tech	C. Grant
3	Reliance on Other Experts	Tetra Tech	C. Grant
4	Property Description and Location	Tetra Tech	C. Grant
5	Accessibility, Climate, Local Resources, etc.	Tetra Tech	C. Grant
6	History	Coffey[2]	D. Corley
7	Geological Setting and Mineralization	Coffey	D. Corley
8	Deposit Types	Coffey	D. Corley
9	Exploration	Coffey	D. Corley
10	Drilling	Coffey	D. Corley
11	Sample Preparation, Analyses and Security	Coffey	D. Corley
12	Data Verification	Coffey	D. Corley
13	Mineral Processing and Metallurgical Testing	Tetra Tech	A.de Ruijter
14	Mineral Resource Estimates	Coffey	D. Corley
15	Mineral Reserve Estimates	NCL[3]	C. Guzmán
16	Mining Methods	NCL	C. Guzmán
17	Recovery Methods	Tetra Tech	A. de Ruijter
18	Project Infrastructure	Tetra Tech/Golder[4]	C. Grant/A. Cadden
19	Market Studies and Contracts	Tetra Tech	C. Grant
20	Environmental Studies, Permitting, etc.	AV[5]	A. Sanford
21	Capital and Operating Costs	Tetra Tech/NCL	H.Ghaffari/C.Guzmán
22	Economic Analysis	Tetra Tech	C. Grant
23	Adjacent Properties	Coffey	D. Corley
24	Other Relevant Data and Information	Tetra Tech	C. Grant
25	Interpretation and Conclusions	Tetra Tech	C. Grant
26	Recommendations	Tetra Tech	C. Grant

The site visits have been undertaken by Qualified Persons (QP) as follows:

a. Overall Coordination: Callum Grant, P. Eng. visited the site in May 2011 (two days) and again in September 2011 for an additional two days.

b. Geology and Mineral Resources: Doug Corley, MAIG R.P. Geo., March 2011 for 5 days.

c. Mine Design and Costing: Carlos Guzmán, November 2011 for two days.

[2] Coffey Mining Limited, Australia
[3] NCL Ingeniería y Construcción Limitada, Chile
[4] Golder Associates, San Juan, Argentina
[5] Ausenco Vector, Mendoza, Argentina

d. Process Plant Design and Engineering: Andre de Ruijter and Hassan Ghaffari, September 2011 for two days.
e. Hydrology and Hydrogeology: Steve Osterberg, September 2011 for two days.
f. Infrastructure: Hassan Ghaffari and Callum Grant as noted above.
g. Geotechnical Investigations: Alistair Cadden, C.Eng. in September 2011 for one day.
h. Environmental: Ausenco Vector's QP, Anthony Sanford, will visit the site within 90 days of submission of this Technical Report.

Professional designations of the QPs can be found in their Certificates of Qualifications included at the end of this report.

Tetra Tech has completed this report and the corresponding study work from its offices in Vancouver, Canada and Santiago, Chile.

3.0 RELIANCE ON OTHER EXPERTS

Tetra Tech has relied on the following parties who are not QPs:

a. Backer & Mackenzie, Buenos Aires for verification of the land status and Don Nicolás property title. Memorandum to MIRL dated February 12, 2012.

b. PKF Villagarcía & Asociados, Accountants and Business Advisors, Buenos Aires. Review of applicable taxes in relation to financial analysis included in Section 22.0 of this Technical Report. Memorandum to Tetra Tech dated February 10, 2012.

The Don Nicolás Feasibility Report summarizes the findings of a FS of the Don Nicolás Project. The objective of the study has been to evaluate all aspects of the Project including geology, reserves, mining, metallurgy, processing, infrastructure, environmental requirements, and financial evaluation. The study sets out the costs to construct and operate the mine and includes a financial analysis of the Project.

Substantial parts of the work and reporting are the responsibility of consultants directly contracted by MIRL and/or MIRLP. Tetra Tech has taken reasonable care to review the field work and technical analysis completed by these sub-consultants and has no reason to believe that information collected by them is inaccurate.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Project area consists of over 47,000 ha of mineral rights located in Santa Cruz Province approximately 100 km inland from the South Atlantic Ocean as shown in Figure 4-1. The key data related to the Project's location are listed as follows:

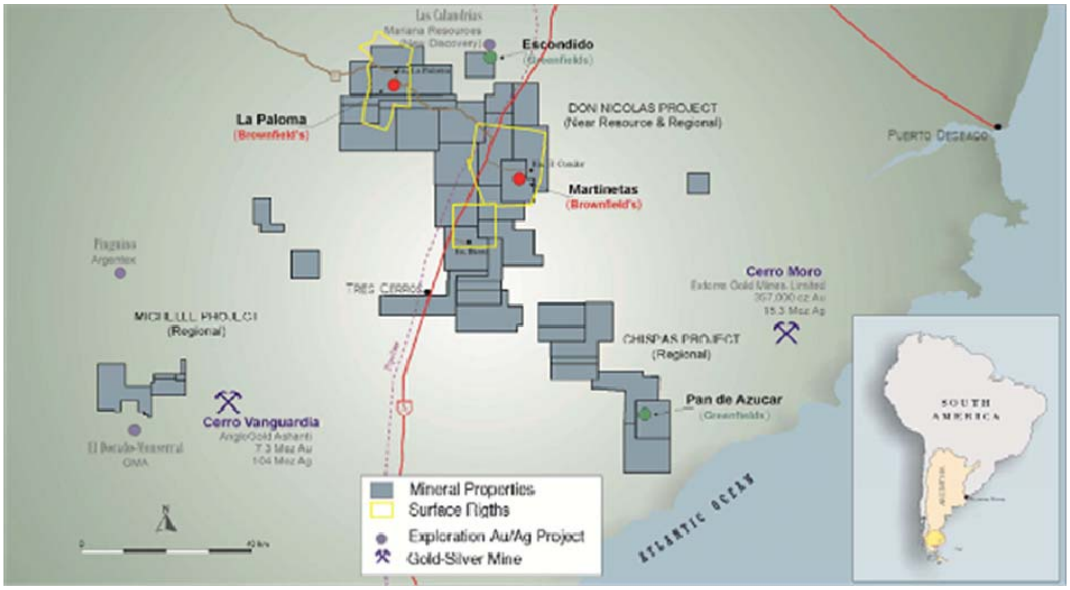
Table 4-1: Location Data for the Don Nicolás Project

	La Paloma	Martinetas
Latitude	47°42'30" S	47°54'23" S
Longitude	67°46'23" W	67°22'44" W
Elevation, m	~120	~160
Distance in km to:		
• Comodoro Rivadavia	275	270
• Rio Gallegos	565	530
• San Julian	212	180
• Puerto Deseado	230	225

The La Paloma area hosts a number of vein systems, the principal economic target being the Sulfuro structure; Martinetas, the mineralized structures at Cerro Oro, Coyote, and Armadillo are the economic targets of principal interest.

The location of the La Paloma and Martinetas areas are shown in Figure 4-1.

Figure 4-1: Project Location Map

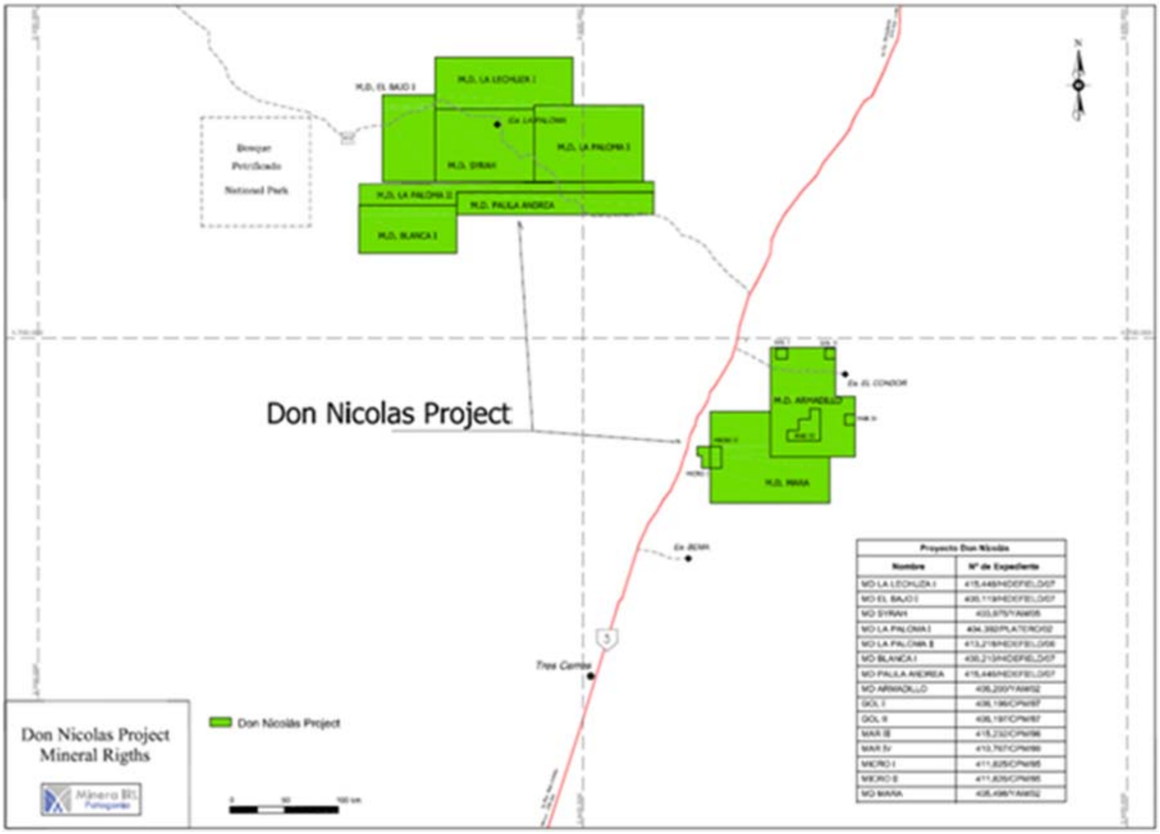


The Project lies in an active and prolific gold mining and exploration region of the world. Established gold operations include Anglo Gold Ashanti's Cerro Vanguardia operation some 90 km to the south west while several advancing exploration projects like Cerro Moro can be found within a 100-km radius of the Project.

4.2 MINERAL TENURE

Legal title to the Don Nicolás Project area is held as a series of "cateos" and manifestaciones de descubrimiento" having a total area of 33,798 ha as shown in Figure 4-2.

Figure 4-2: Property Claim Map, Don Nicolás Project



The company's total Santa Cruz land package covers 270,000 ha (includes the Don Nicolás Project claims) and, in addition, the company holds surface rights of 70,000 ha within the project area.

Mineral rights in Argentina have been broadly regulated by the Federal Government since the enactment of the original *Mining Code* in 1822. Since then the rules and regulations have been modified several times, particularly in 1993 with the introduction of the *Mining Investment Law* that provided an updated legal and taxation framework for mineral exploration and mine production. However, while this framework is broadly regulated at the federal level, in effect it is the individual provinces that hold domain over the mineral resources within their territories, and administer the mineral resources. The provinces therefore are the key to acquiring, owning, producing, and selling mineral products located within specified parcels of land known as cateos, manifestaciones de descubrimiento, and minas:

Cateo: a cateo is a parcel of land measured in units of 500 ha, and can vary in size from a single unit to a maximum of 20 units (10,000 ha). The holding of a cateo is associated with relinquishing ground based on a formula varying from 300 to 700 days and reduction in ground held to 50% of that originally claimed.

Manifestación de Descubrimiento (MD): on discovering a mineral occurrence of interest within a cateo, the owner can apply for an MD around his discovery at any time within the period of the corresponding cateo. The maximum area of an MD is 3,000 ha and remains in force until such time as the property is legally surveyed, an essential prior step to the longer term granting of a “mina”.

Mina: minas are mining concessions or leases which permit mining on a commercial basis. The area of a mina is measured in "pertenencias" and will vary in size according to the distinction between vein and disseminated targets believed to occur on the property. Individual mining authorities (the provinces) may determine the number of pertenencias required to cover the geologic extent of the mineral deposit in question. Once granted, minas have an indefinite term assuming exploration, development or mining is in progress.

All mineral rights described above are considered forms of real property and can be sold, leased or assigned to third parties on a commercial basis.

The mining code contains environmental and safety provisions, administered by the provinces. Prior to conducting operations, operators must submit an environmental impact report to the provincial government, describing the proposed operation and the methods to be used to prevent undue environmental damage.

All permits necessary to carry out exploration work on the Don Nicolás Project area have been obtained by MIRLP. Table 4-2 provides details of the mineral titles:

Table 4-2: MIRLP Mineral Titles

Claim Name	Title Holder	Area, ha
La Lechuza I	MIRLP	5,902
EL Bajo I	MIRLP	3,863
Syrah	MIRLP	5,994
La Paloma I	MIRLP	600
La Paloma II	MIRLP	3,780
Blanca I	MIRLP	3,960
Paula Andrea	MIRLP	3,699
Aramdillo	MIRLP	3,500
GOL I	MIRLP	100
GOL II	MIRLP	100
MAR III	MIRLP	600
MAR IV	MIRLP	100
Micro I	MIRLP	200
Micro II	MIRLP	200
Mara	MIRLP	1,200
Total ha		**33,798**

4.3 ROYALTIES, RIGHTS AND ENCUMBRANCES

Three royalties apply to all or part of the Don Nicolás Project. These are:

1) An ad valorem provincial royalty of up to 3% of mine mouth value will be payable. Such mine head value is defined as the price for the sale of the corresponding mineral deducting certain costs and expenses.
2) A two percent net smelter royalty reserved to Royal Gold Inc., pursuant to agreements dated February 1, 2000 (Polimet Royalty Agreement) and January 1, 2002, with Yamana and associate companies (La Paloma Royalty Agreement). The first agreement includes the following MIRL mineral rights: Gol I, Gol II, Mar III, Mar IV, Micro I and Micro II along with other mining rights that are not included in the Don Nicolás Project or that do not even belong to MIRL. The latter covers the Syrah declaration of discovery.
3) A US$3.00/oz gold royalty to a cap of US$2 M payable to Yamana. This is applicable to all of the current resource areas and, effectively, those key licenses covered by the Royal Gold agreement.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 TOPOGRAPHY, VEGETATION AND LAND USE

The Project area is located on the eastern Patagonian plains and is generally characterized by flat to gently undulating landforms dissected occasionally by incised valleys. Some prospect areas exhibit hilly terrain, but this does not impede easy access to the entire project area. Elevation ranges from 130 to 220 m above sea level. Vegetation is sparse and dominated by grasses and low shrubs. Some cattle and sheep grazing activities persist. However these are limited in extent as the pastoral industry has not recovered from the effects of the ash blanket from the Mount Hudson. Logistics in the region have also been affected by the more recent (2011) Puyuehue volcano. In summary, the area is now largely uninhabited.

5.2 CLIMATE

Don Nicolás is located in a climatically rigorous region of southern Argentina known as the Argentine Patagonia. This region is marked particularly by its strong westerly or south westerly winds that persist throughout much of the year both in the warm summer months but also in the cold southern winter from June to September or October when snow and rain are common.

Average monthly temperatures above 10ºC generally occur in the Project area between November and March; average monthly temperatures below 5ºC generally occur from June through August.

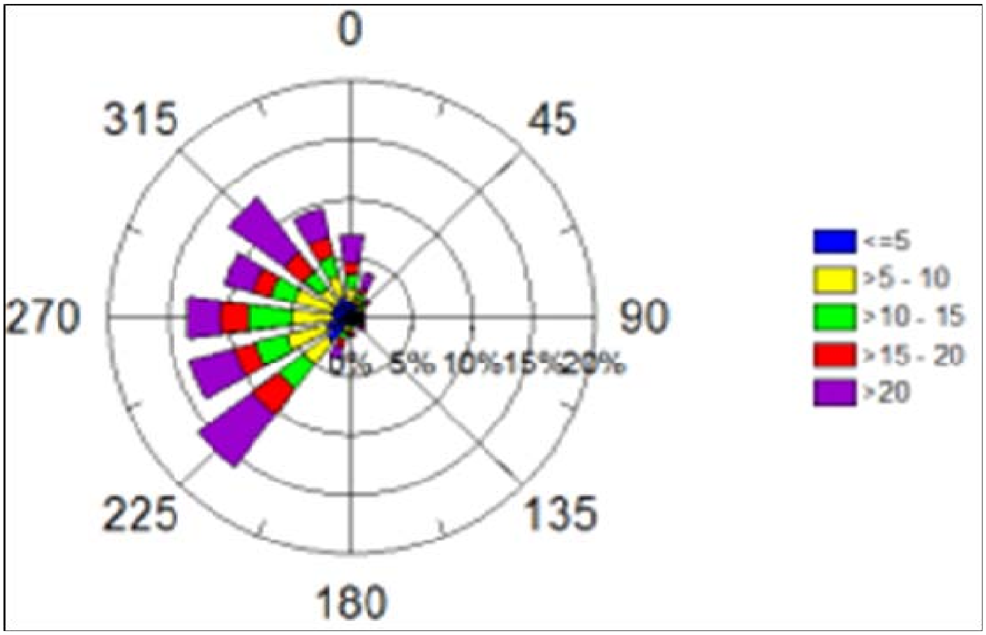
Annual precipitation is from 180 to 300 mm, with occasional heavy snow falls in the winter.

Monthly average wind speed statistics are as follows:

Maximum (March)	88.5 km/h
Maximum (March)	24.6 m/s
Annual average	14.8 km/h
Annual average	4.1 m/s

The prevailing winds are from the west (66.7% for directions between 213° and 326°), but with a marked variability as shown in Figure 5-1.

Figure 5-1: Wind Rose Diagram, 2008 to 2010



5.3 ACCESS

The Project is easily accessible from major centres in the region by paved public highways and secondary gravel roads. The larger population centres in the area include the coastal port city of Comodoro Rivadavia (actually in Chubut province) which is serviced daily by commercial jet flights from Buenos Aires approximately 1,750 km to the north. Comodoro Rivadavia is a regional centre of approximately 140,000 inhabitants servicing the Argentine oil and gas industry, a major employer in this region of Argentina. From Comodoro Rivadavia, the Project is accessed by driving south along paved National Route No.3 for approximately 280 km. This section of Route No. 3 is part of the principal north-south road link traversing the length of the country, and is therefore well maintained by the provincial and national road authorities

From Route No. 3, the La Paloma estancia is accessed by turning west onto the unpaved Route No. 49. Access to the Martinetas mineral deposits on the El Cóndor estancia is found approximately 7 km further south turning east off Route No. 3. Average driving time between Comodoro Rivadavia and the property is about four hours.

In the Project area, off-highway driving around the individual exploration sites is facilitated by the gentle topography, mainly dry climate, and sparse vegetation that are typical characteristics of this region of the Argentine Patagonia. In the winter months, heavy rain can occur, making for more difficult conditions, but these are typically of short duration.

5.4 INFRASTRUCTURE AND SERVICES

Don Nicolás is located in a very sparsely populated region of Argentina where the original settlers were dedicated to sheep ranching from scattered ranches each covering large areas, typically measured in square kilometres. These "estancias" and related rural activities have now largely been abandoned as the result of the 1991 volcanic eruption of Cerro Hudson which led to the near extinction of the sheep industry.

Oil is a significant industry in Santa Cruz province (and neighbouring Chubut to the north) principally from onshore production and exploration. Oil service centres with port facilities are located at Caleta Olivos (Chubut) and Rio Gallegos (the provincial capital of Santa Cruz). Nearby settlements to Don Nicolás include Puerto Deseado towards the east and Puerto San Julian towards the south where basic services and supplies are readily available.

A gas pipeline running parallel to National Route No. 3 transects the Project area and there are proposals to eventual extending power grid access from the main line that currently runs from Puerto Deseado to Caleta Olivia to the north-west. However, in the short term, power requirements for the mining project at Don Nicolás will be provided by diesel generators operated by a third-party supplier, such as Sullair or Atlas Copco.

Communication at site is currently provided by satellite link. Accommodation and office facilities with domestic power and water services are available at the El Cóndor estancia which has served as the main Don Nicolás exploration camp for several years.

Water exploration in the area has successfully identified sufficient water to support a mining operation and related infrastructure at Don Nicolás.

6.0 HISTORY

6.1 HISTORICAL BACKGROUND

MIRL acquired the Don Nicolás Project and surrounding properties in December 2009 as part of the acquisition of Hidefield Gold Plc (Hidefield) in an all-stock transaction on the AIM London Exchange. These holdings are now held by MIRL wholly-owned subsidiary MIRLP with head office in Buenos Aires.

The targets in and around Don Nicolás were originally identified in the early 1990s by regional exploration activities following the discovery and subsequent development of the Cerro Vanguardia gold mine by AngloGold Ashanti Limited and Fomicruz (a Santa Cruz provincial mining holding entity) that currently produces approximately 150,000 oz gold per year. Companies conducting initial exploration in the region included Newcrest Limited, Yamana Resources, and various joint ventures.

The claim areas (cateos) within and around the Don Nicolás Project were originally explored by a number of companies including Newcrest, Minas Buenaventura, Yamana Gold, Rio Algom, Hochschild and Hidefield. This exploration work included surface sampling, trenching and limited drilling, both core and percussion.

Early historical drilling on the Martinetas vein showings consisted of:

a. 1996: 20 RC drillholes completed by Yamana.
b. 1997: 86 RC and 46 core holes completed by Yamana.
c. 1999: 20 RC and 20 DDH holes completed by Yamana.
d. 2003: 18 DDH holes completed by Recursos Yamana S.A. (RYSA), a joint venture between Yamana Resources and Compañia Minera Buenaventura.

Early exploration in the La Paloma area included 12 RC holes completed by Newcrest in 1996.

Between 2006 and 2009, Hidefield completed a total of 152 core holes (HQ) at a number of vein showings in La Paloma (85 holes for 11,100 m) and Martinetas 67 holes for 7,743 m). In addition, a substantial number of trenches were completed in both project areas.

In 2010, MIRL announced the funding for a FS to establish the viability of the Don Nicolás Project. This included various field programs of drilling to upgrade mineral resources, geotechnical investigations to establish mining and construction parameters, environmental base line studies, water resource investigations, and technical studies into mining and processing alternatives.

6.2 HISTORICAL RESOURCES

In June 2007, an independent technical report was prepared for Hidefield by Independent Engineers (Australia) Pty Limited with mineral resources published (at a 1 g/t Au cutoff grade) as shown in Table 6-1.

Table 6-1: Mineral Resources, Independent Engineers, 2007

	Indicated			Inferred		
	t	**Au g/t**	**Au Oz**	**t**	**Au g/t**	**Au Oz**
Sulfuro Vein	613,000	8.0	156,600	117,000	10.3	39,000
Arco Iris vein				250,000	4.1	32,700
Coyote Norte Veins	19,000	22.8	13,600	24,000	12.7	9,900
Coyote Sur Veins	59,000	10.6	20,100	6,000	12.4	2,400
Armadillo Veins				66,000	5.9	12,500
Cerro Oro Veins				60,000	7.7	14,800
Total	**690,000**	**8.6**	**190,300**	**524,000**	**6.6**	**111,200**

Note: Figures have been rounded

In December, 2008, an updated mineral resource estimate was completed by Runge Ltd. of Perth, Australia, also at a 1 g/t Au cutoff grade. A summary of the estimate is shown in Table 6-2.

Table 6-2: Mineral Resources, Runge, 2008

Prospect	Indicated			Inferred		
	Tonnes	**Au g/t**	**Au Oz**	**Tonnes**	**Au g/t**	**Au Oz**
Sulfuro Vein[1]	930,000	5.5	165,700	134,000	2.0	8,500
Rocio vein				93,000	4.1	12,400
Arco Iris vein				310,000	5.5	55,100
Coyote Norte Veins	44,000	7.6	10,600	66,000	6.3	13,300
Coyote Sur Veins	63,000	8.7	17,600	71,000	8.6	19,600
Armadillo Veins				157,000	3.4	17,300
Cerro Oro Veins	41,000	5.1	6,800	245,000	4.1	32,300
Total	**1,078,000**	**5.8**	**200,700**	**1,075,000**	**4.6**	**158,400**

1. Ramal Vein included.

Note: Figures have been rounded.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL SETTING

The Don Nicolás Project is located in the Patagonia region of Argentina within an uplifted fault block area known as the Deseado Massif of Santa Cruz province. Covering a surface area of approximately 60,000 km^2, the Deseado Massif is predominantly underlain by volcanic rocks of Jurassic age and is host to several epithermal Au-Ag deposits such as Don Nicolás.

The Deseado Massif is dominated by rhyolitic and andesitic volcanic and tuffaceous volcaniclastic lithologies of Middle to Upper Jurassic age (130 to 170 ma). It is criss-crossed by numerous extensive fault and fracture zones which served as conduits for hydrothermal activity during periods of Jurassic volcanism. The result of this activity is a widespread network of shallow level mineralized "epithermal" fissure veins, breccias, and stock-work systems, many of which carry potentially economic Au and Ag mineralization.

7.2 PROJECT GEOLOGY

Broad similarities exist between the two main Don Nicolás project areas of La Paloma and Martinetas. Each are hosted within rhyolitic to andesitic volcaniclastic lithologies which are interpreted to be flat to shallow dipping. Gold and silver deposits occur as low sulphidation, epithermal mineralization within sub-vertically oriented quartz-breccia veins.

At La Paloma, the Sulfuro-Rocio vein system comprises narrow, arcuate, steeply dipping quartz-breccia veins. Drilling has defined four areas of interest:

a. The Sulfuro vein is the principal deposit of economic interest and is represented by a single, well developed quartz vein typically 2 to 4 m in thickness and has a primarily northwest-south southeast orientation with a steep southwestwardly dip. Associated sulphide minerals include pyrite and minor galena and sphalerite.

b. The Ramal Sulfuro vein occurs at the northern end of the main Sulfuro vein and is strongly curved from a north-south orientation to east-west, and is typically 2 to 4 m in thickness.

c. A third vein at Rocio occurs to the west of the main Sulfuro vein. The Rocio vein is typically 2 to 5 m in thickness and dips steeply to the east. It is also arcuate in shape and sub-parallels the Sulfuro vein.

d. A near-surface oxidized resource has been estimated for the Arco Iris vein, located towards the north of the Sulfuro vein. It is represented by a series of narrow, structurally displaced, sub-parallel aligned, sheared quartz veins hosting erratic precious metal mineralization.

The La Paloma veins remain open-ended at depth. Geological interpretation of the results of recent geophysical studies strongly suggests that the main Sulfuro vein is additionally open-ended towards the south.

At Martinetas, multiple mineralized structures occurring as “vein swarms” with minor intervening stockwork development occur. Five resource areas have been delineated. The main resource is at the Coyote and Cerro Oro deposits comprising a series of narrow, sub-parallel, anastomosing quartz veins varying in width from tens of centimetres to several metres, and typically averaging one metre or less in thickness. Au/Ag mineralization is variable within the veins with some minor stockwork mineralization extending into the host volcanic lithology. Conceptually, near-surface oxidized stockwork precious metal mineralization might provide a low-grade, conventional, heap-leachable resource.

Other resource areas at Martinetas include the Lucia, Calafate and Armadillo deposits. Precious metal mineralization associated with these deposits is also hosted by narrow to moderately thick, steep dipping quartz veins of variable tenor.

7.3 MINERALIZATION

The La Paloma and Martinetas epithermal systems are classic “low sulphidation” type deposits consisting mainly of quartz with adularia and free Au, minor amounts of sulphides (except at Sulfuro where a relatively high sulphide content occurs), and weak alteration haloes. Classic low sulphidation textures like lattice textures, crustiform-colloform banding, comb quartz, etc., are common. They may also be associated with anomalous amounts of arsenic (As), mercury (Hg), or antimony (Sb).

8.0 DEPOSIT TYPES

The important Au-Ag occurrences in Santa Cruz Province are confined to the Deseado Massif.

The potential for important gold-silver mineralization was recognized in the Deseado Massif some 25 years ago. The Deseado Massif is a plateau consisting of Middle- to Upper Jurassic acid volcanic and volcaniclastic rocks that host fracture-controlled epithermal Au-Ag mineralization. Precious metal mineralization is related to Late-Jurassic magmatism. A number of world-class deposits have been identified in this region of Patagonia. These include Cerro Negro, Cerro Vanguardia and Cerro Moro. Many smaller deposits have been discovered (Mina Marta, Don Nicolás, Pinguino, etc.), and there are also many recently discovered exploration projects being actively explored.

The gold-silver deposits are hosted in silicic volcanic and volcano-sedimentary Jurassic rocks related to arc or back-arc settings in Andean or extra-Andean settings. The ore geology, textures, mineralogy, restricted alteration and geochemistry of these mineralized occurrences indicate that they belong to the epithermal class of precious metal deposits. The deposits are mainly associated with quartz +/- calcite +/- adularia +/- illite alteration assemblages interpreted to represent low and intermediate sulphidation epithermal type deposits.

Known deposits represent diverse levels of erosion ranging from sinter formed at the paleo-surface, to intermediate Au-Ag rich quartz veins, to base-metal bearing Au-Ag veins that represent deeper levels of the epithermal systems. Based on metallic associations, the different deposits can be divided into;

a. Au-Ag or Ag>Au;
b. polymetallic with Ag-Au or only Ag or;
c. complex polymetallic with Ag-Au.

Precious metal deposits generally occur in tectonically controlled structures with quartz fill, frequently brecciated in nature. Mineralization is generally associated with discrete banded quartz vein structures ranging in width from 0.5 to over 5 m, although a disseminated form of weak stockwork quartz veining of less than 0.5 m width intermediate to the principal quartz structures does also occur in some instances. Strong lithological control may occur. Dating in a few instances suggests that hydrothermal activity is several million years younger than peak magmatism. The presence of surface features like silica-sinters, veins interpreted to be feeder structures, carbonate-lacustrine deposits and silicic lithocaps related to steam-heated waters in the water-table, indicates that the tops of some systems have been preserved. Erosion of younger sedimentary and volcanic cover is now exposing these systems and is suggesting that large regions or tectonic blocks are highly prospective for epithermal mineralized systems at depth.

Extension fracturing developed in the Jurassic volcanics and the influx of meteoric waters into geothermal systems is considered the main control on ore genesis. Mineralizing fluids are dilute to low-intermediate salinity waters with temperatures ranging between 160 to 330˚C. The geological characteristics of the region and the presence of major Au-Ag deposits provide much optimism for continuing exploration with new discoveries to be made in the Deseado Massif.

9.0 EXPLORATION

9.1 EARLY EXPLORATION

The Don Nicolás Project areas were acquired from Hidefield who in turn had acquired them from Yamana Resources Inc. (Yamana). Yamana identified the areas as having potential for Au-Ag mineralization and commenced exploration work in the region in the early 1990s.

On the Martinetas prospect, the discovery of Au mineralization at surface led to the commencement of drilling in 1996. Since that time, extensive surface trenching, as well as various campaigns of reverse circulation (RC) and/or diamond drilling have been completed. The majority of drilling was completed by Yamana between 1996 and 1999. An additional program was carried out in 2003. Between 2006 and 2009, Hidefield completed drilling and trenching at the project. From 2010, after the late-2009 acquisition of Hidefield, MIRL invested significant funding into an extended campaign of infill drilling with some minor extension drilling of the known resources.

Several zones of satellite Au-Ag mineralization have been identified on the Martinetas project area. Systematic exploration of these areas commenced under the management of MIRL and is described in more detail below.

At the La Paloma Project, initial drilling was carried out in 1996 by Newcrest Minera Argentina SA (Newcrest). No further drilling was carried out until Yamana resumed exploration in 2003 and drilled a series of holes in that year. Hidefield then purchased the project and completed drilling between 2006 and 2009. MIRL continued a program of infill DDH at Sulfuro and Arco Iris during 2010 and into 2011.

9.2 MIRL EXPLORATION

Additional to the resource infill drilling and surface trenching described above, MIRL's brownfield exploration work in the Martinetas area also focused on targeting shallow, oxidized satellite pit resources (potentially amenable to heap leaching), located peripheral to the central Martinetas deposits. Approximately 11,000 m of RC drilling has been planned in the central Martinetas area. Also, over 1,000 m of RC drilling has been planned for the currently open-ended Armadillo deposit and both areas are scheduled to be drilled in early 2012. Approximately US$ 2.4 M has been budgeted for this drilling.

There is potential to add 1 to 1.5 MT at a grade range of 1.2 to 1.6 g/t Au, above a 0.3 g/t Au cutoff, to the combined Measured and Indicated mineral resource. *The potential quality and grade is conceptual in nature and it is uncertain if future exploration will result in the target being delineated as a mineral resource. The potential target, quantity and grade was based on approximately 60% of the current* *delineated Inferred mineral resource being upgraded to a Measured or Indicated category and 40% from down-dip extensions of current mineralization and areas between known mineralized zones.*

Current results have also shown that the Choique prospect, located approximately 1 km southwest of the principal Coyote-Cerro Oro vein swarms, could host additional ounces to add to the Martinetas resource. There is good potential to add more brownfield ounces to the Martinetas resource base in the future.

Brownfield exploration in the La Paloma vein area will focus on testing the down dip and south-eastern strike extension of mineralization along the Sulfuro vein structure. Several other parallel vein structures were also identified as prospective targets; Ramal Sulfuro, Veta Esperanza, Rocio and Rocio Oeste. Based on preliminary exploration geophysical results, it is conceptualized that the principal Sulfuro vein structure potentially extends significantly south-eastwards from the known Sulfuro vein outcrop.

10.0 DRILLING

10.1 INTRODUCTION

Drilling at the Don Nicolás Project has been carried out by several companies over the period October 1996 to June 2011 and summarized as follows:

1996: Yamana Resources (Yamana) begins drilling in the Martinetas Region.

1996: Newcrest begins an initial program at the Arco Iris deposit (La Paloma area).

1996 to1999: Yamana completes drilling (DDH and RC) and trenching in the Martinetas area.

2003: RYSA (a joint venture between Yamana and Compañia Minera Buenaventura) completes drilling (DDH) at Martinetas.

2006 to 2009: Hidefield Gold Plc (Hidefield) continues drilling (DDH) and trench sampling in the La Paloma and Martinetas regions.

2010 to Present: Since the acquisition of the project in 2009, MIRLP has continued infill drilling (DDH and RC) and trench sampling at all the deposits within the Don Nicolás Project area.

10.2 DRILLING HISTORY

A summary of the different phases of drilling and trenching completed at the Don Nicolás Project is presented in Table 10-1.

Table 10-1: Historical Drilling and Trenching Summary

Region	Company	Deposit	Period	Type	No. of Holes/ Trenches	Metres
La Paloma	Newcrest[2]	Arco Iris	1996	RC	5	605
	RYSA	Sulfuro	2003	DDH (HQ)	4	621
			2003	Trench	24	320
		Arco Iris	2003	DDH (HQ)	3	377
			2003	Trench	19	204
	Hidefield	Arco Iris	2006	DDH (HQ)	15	1,833
		Sulfuro	2006 - 2008	DDH (HQ)	62	8,589
			2007	Trench	43	775
	MIRL	Sulfuro	2010	DDH (HQ)	81	10,829
			2010	RC (pre-collar)/ DDH (HQ)	3	725
		Arco Iris	2010	Trench	4	93
			2010	DDH (HQ)	19	1,149
Martinetas	Yamana	Central Zone[1]	1996 - 1999	RC	33	2,767
			1997 - 1999	DDH (HQ,NQ)	18	1,538
			1997	RC	60	1,267
			1997	Trench	6	839
		Armadillo	1996 - 1997	RC	4	571
			1997 - 1999	DDH (HQ,NQ)	12	829
	RYSA	Central Zone[1]	2003	DDH (HQ)	9	1,025
			2003	Trench	56	2,476
		Armadillo	2003	DDH (HQ)	3	346
			2003	Trench	10	575
	Hidefield	Central Zone[1]	2006	Trench	6	642
			2006 - 2008	DDH (HQ)	62	7,199
		Armadillo	2006	DDH (HQ)	5	547
	MIRL	Central Zone[1]	2010	DDH (HQ)	28	2,336
			2010	Trench	11	2,249
			2011	RC	68	6,211
		Armadillo	2011	RC	17	743
Total Metres (Drilling and Trenching)						**58,281**

Notes:

1. Central Zone – contains Cerro Oro, Coyote, Lucia and Calafate Deposit.

2. Newcrest has reportedly drilled 12 holes into the La Paloma area in 1996; only 5 holes are included in the drilling database provided.

Note: Figures have been rounded.

Notes:

a. All surveying, plotting and mineral resource modelling utilises the Gauss-Kruger – Zone 2 Datum Grid.

b. Drillholes were surveyed on approximately 50 m downhole intervals using a Sperry-Sun magnetic single shot instrument. More recent surveying (from 2008 drill campaigns) used a Reflex EZ-Shot®, an electronic single shot instrument manufactured by Reflex of Sweden.

The Figures 10-1 to 10-3 show the extent of drilling and trenching at the Paloma (Sulfuro), Martinetas Central Area, and Armadillo deposits respectively.

Figure 10-1: Paloma (Sulfuro) Drill and Trench Plan

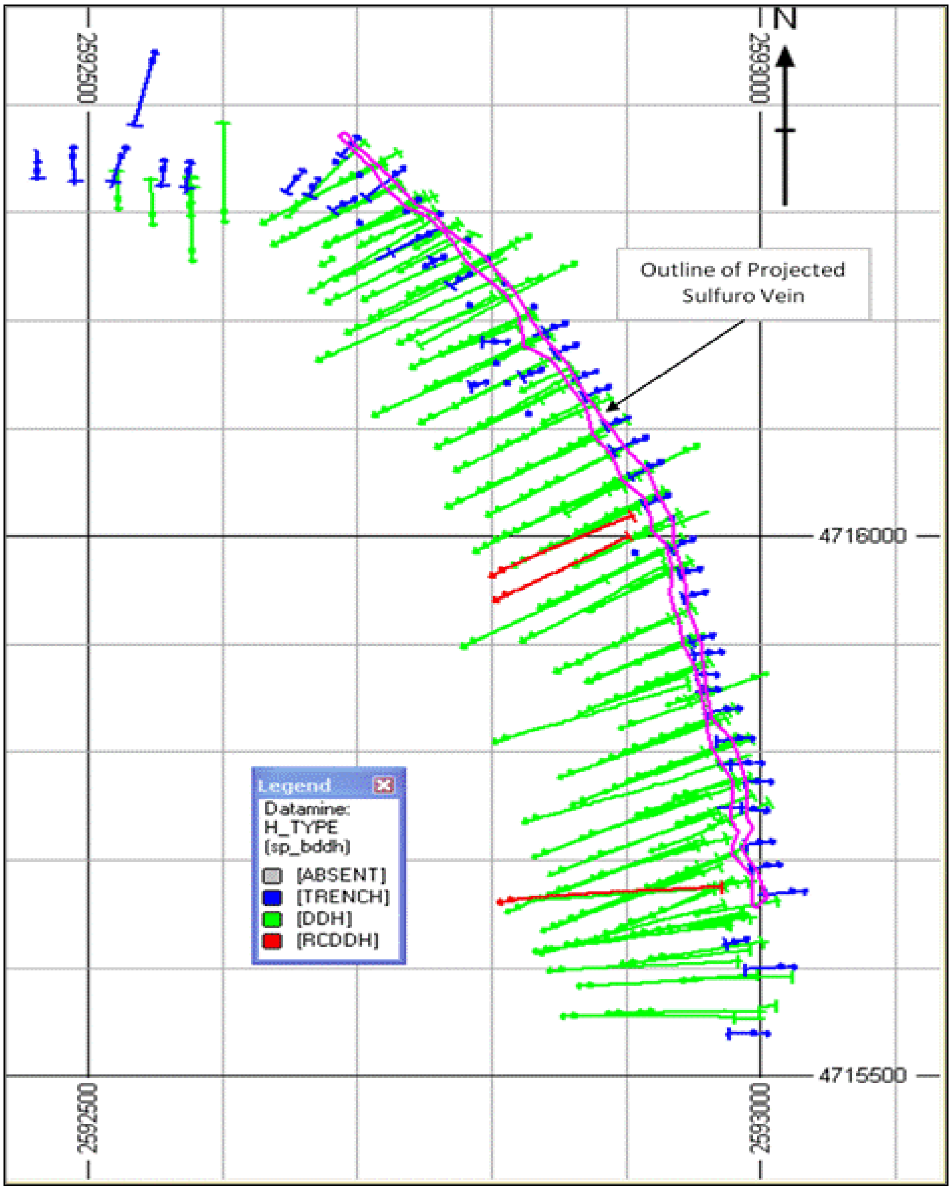


Figure 10-2: Martinetas Central Area Drill and Trench Plan

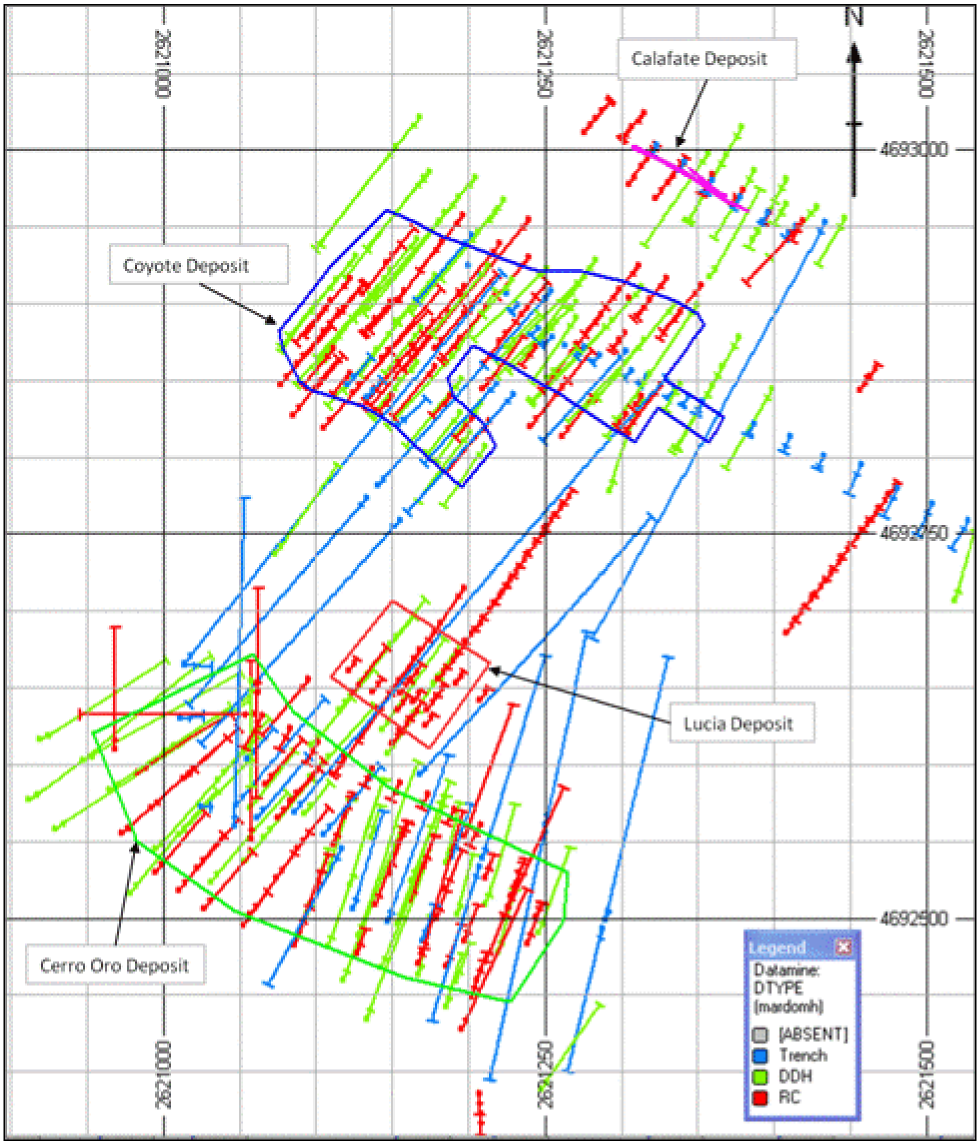
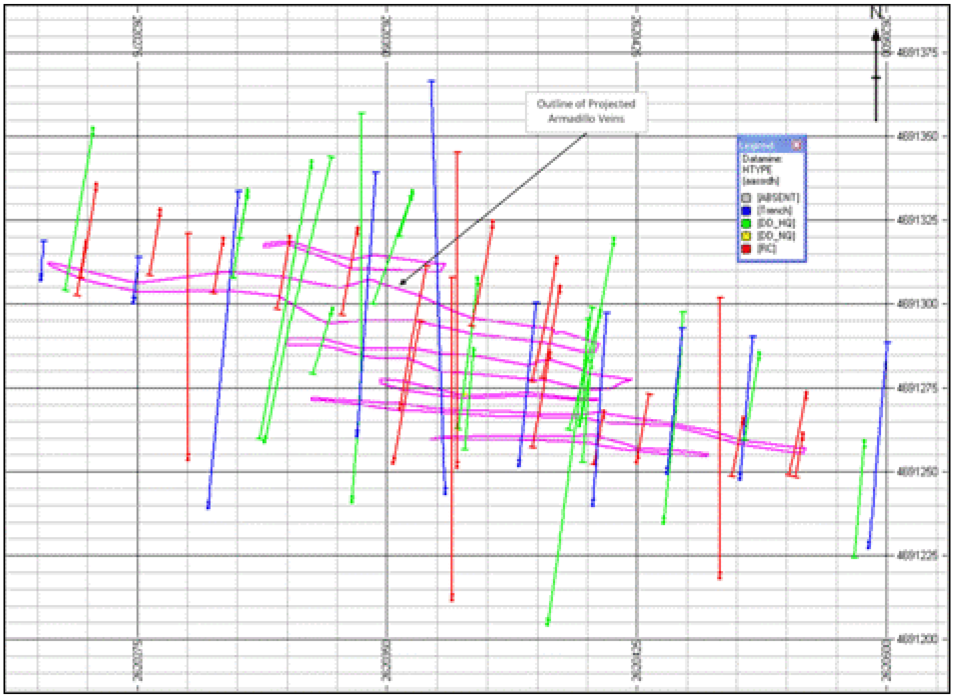


Figure 10-3: Armadillo Drill and Trench Plan



Whist on site in 2010, Coffey Mining chose several drill collars and verified their location using a hand-held GPS unit. All drillholes checked were within +/- 5 m of the reported location (within the accuracy limits of the device).

Survey reports show that the topographic survey was generated by surveyors using DGPS total station. This topography was reported with sub-centimetre accuracy and compares well with the drillhole collar survey data. Coffey Mining considers the topography to be of high confidence.

Within the Don Nicolás Project, drilling has been orientated in order to be perpendicular to the strike of the known vertical to sub-vertical mineralization.

Within the Don Nicolás Project area, generally samples have been taken at 1 to 2 m lengths within the known mineralized zones (some core samples have been taken at a minimum length of 0.4 m based on geological interpretation); average sample length is approximately 1 m.

The relationship between drilling and mineralization is defined in further detail is Section 14. Drillholes typically intersect mineralization orthogonally, and the true mineralized intercepts are typically 60 to 80% of the intersected mineralization.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 TRENCHING

Trenching took place across all deposits within the Don Nicolás Project from 1997 (Yamana), to as recent as 2010 (MIRL) at Martinetas. Trenches are typically 1.2 m wide, 0.5 to 1.5 m deep and of various lengths. The trenches were channel sampled along one wall in 1 m intervals except where dictated by lithology, where samples could be less than 1 m.

Coffey Mining observed the sampling of the trenches whilst on site in June 2010, and considers the samples appropriate for use in the mineral resource estimate.

11.2 DRILLING AND SAMPLE QUALITY

11.2.1 RC DRILLING – NEWCREST

No information is available on the sampling procedure used by Newcrest. Runge (2009), reported that 12 RC holes were drilled by Newcrest in 1996, however only five RC drillholes were included in the database provided to Coffey Mining.

11.2.2 RC DRILLING – YAMANA

Doe (1997), conducted a review of RC sampling by Yamana. Within the document it is stated that the routine sample interval for the project was set to 2 m, however it also states that at the beginning of the program sample intervals were changed to 2.033 m, to account for the fact the drilling contractor only had 6.10 m (20 ft) rods. This information has not been captured in the database provided to Coffey Mining, however Coffey Mining considers it would make no material difference to the respective data. Several RC holes were sampled at 0.5 m intervals and a very small number of samples were taken at 1 and 1.5 m intervals. RC field duplicate samples are routinely collected to allow assessment of the field sampling error (or bias).

Dry hole sampling procedure:

a. Cyclone discharge is collected in 20-L pails.
b. Sample is passed through a modified Jones 3-tier splitter[6], where the material is split twice and replicate samples each consisting of 25% of the total sample volume.

[6] Modified by replacement of the middle tier with a pan which would collect 50% of the total sample for introduction into the lower tier; resulting in replicate 25% samples.

Wet hole sample procedure:

a. Cyclone discharges directly into the rotary wet splitter which has no covers in place and thus delivers 50% to the sample stream.
b. The sample port discharges into the bottom-tier of the modified Jones 3-tier splitter, which cuts the stream into replicate samples each containing 25% of the total cyclone discharge;
c. The respective sample bag was placed in a 20-L pail, and then positioned under the sample discharge chute, it the first pail becomes full, it was set aside and a second pail was set in place to collect the balance of the sample interval.
d. The pails are set aside and a flocculent (DD 2000 polymer) was added to accelerate settling, after settling excess water was decanted off, and contents of second pail, were used, was added to the samples bag and closed.

Coffey Mining considers the stated sample procedures are adequate and of industry standard, however the correct sampling of wet samples is problematic and recommends that if water is a problem, that drilling be switched to DDH, as was recommended by Doe (1997).

There was no record of wet and dry sampling within the database for this program in the database provided to Coffey Mining, so there is no way of quantifying the effect of wet sampling during this time.

11.2.3 DDH Drilling - Yamana

A review conducted by Small (1997), states that except where hole conditions force the reduction in core size, all drilling should be done to a minimum of HQ core size. Three holes were drilled at NQ core size (02-021C, 02-022C and 02-023C).

Within the report Small (1997) states core was sawed into two equal halves (using an electric diamond saw), one half of the core was assayed and the other half was archived and stored on site. Selecting the sample intervals was determined by the geologist based on lithology/geology. No minimum sample length was specified, but the procedure states the geologist should refrain from selecting too many short intervals, and in general, intervals of more or less 1 m should be selected. Field duplicate samples we taken on quarter core (half of the remaining reference core sample).

Selective sampling was carried out in many drillholes; this resulted in some portions of the core remaining un-sampled.

Core photos where kept from all drilling and available for viewing by Coffey Mining. No core orientation was performed. Coffey Mining considers the sampling methods for DDH is of acceptable industry standard.

11.2.4 DDH Drilling – RYSA, Hidefield and MIRL

Coffey Mining was unable to find any specific documentation for DDH drilling during the RYSA period (2003). But as RYSA was a joint venture with Yamana, it is expected that a similar procedure would have existed

Both Hidefield and MIRL, maintained similar DDH sampling procedure to Yamana. As reported in Runge (2009), Hidefield transported all DDH core to the El Cóndor ranch, core was laid out in a well-lit shed. The core was photographed then geological logging carried out. Geologists define sample intervals on geological boundaries with a minimum length of 0.4 m, with samples generally around 1 m in length. Core is then cut in half using a diamond saw. Half core samples are then bagged and dispatched for preparation and analysis. The left side of the core is uniformly taken for analysis. Field duplicate samples we taken on quarter core (half of the remaining reference core sample).

No drill core orientation was performed on the exploration drill core.

Whilst on site in June 2010 and March 2011, Coffey Mining observed a similar DDH sampling procedure being adopted by MIRL. Coffey Mining considers the procedures meet an acceptable industry standard.

11.2.5 RC Drilling – MIRL

In 2011, MIRL preformed an infill RC drill campaign in the Martinetas area, to improve the mineral resource estimate. All sampling was taken at 1-m downhole intervals.

Dry hole sampling procedure:

a. Cyclone discharge is collected in 20-L pails.
b. The entire sample interval is placed onto the hopper of a Gilson single-tier splitter, and the split is taken all at once across the face of the splitter by opening the hopper and allowing the sample to fall evenly across the face of the splitter.
c. The sample was re-split until a final sample weight of 7 to 5 kg was returned.

Wet hole sample procedure:

a. A similar procedure to what was used by Yamana; which was also was recommended by Smee (2010).

A log of the RC sample condition was made by MIRL personnel and provided in the database to Coffey Mining (the log indicated which of the samples was dry, damp or wet).

Sampling procedures have been reviewed on several occasions by Barry Smee, P.Geo., whose reports are listed in the references to this technical report.

Coffey Mining considers that the sampling procedure used by MIRL meets acceptable industry standards.

11.3 Sample Recovery

As mentioned, there is no record of some historical RC sampling; however recent RC drilling by MIRL (2011) at Martinetas has shown that less than 0.5% of all RC holes were logged as wet.

Core recovery was detailed by Runge (2009), it was noted that there was a reasonably high amount of intervals returning less than 90% core recovery in mineralized zones at La Paloma. Recent core drilling at La Paloma by MIRL is collected using a triple-tube system to ensure recovery is as good as possible. Core recovery at Martinetas is considered to be reasonable and satisfactory (with a majority of the core having recoveries of greater than 90%).

A weathering profile exists to a depth of 20 to 50 m resulting in the formation of clay rich lithologies; however examination of core recovery in these areas was reasonable.

11.4 Sample Security

The close scrutiny of sample submission procedures by MIRL technical staff, and the rapid submission of samples from drilling for analysis, provides little opportunity for sample tampering. Smee (2010) noted the following; sample bags are placed into rice bags for shipment to the preparation laboratory. A transport truck is used to deliver the samples to the preparation laboratory in Mendoza, Argentina. Each of the rice bags is sealed with a sequential number sample tag.

A review of the previous company's procedures used in the La Paloma and Martinetas regions show that similar sample security measures were used.

Reference material is retained and stored at the MIRL exploration camp at El Cóndor Ranch, as well as chips derived from RC drilling, half-core and photographs generated by Diamond drilling, and duplicate pulps and residues of all submitted samples. Assessment of the data indicates that the assay results are generally consistent with the logged alteration and mineralization, and are entirely consistent with the anticipated tenor of mineralization.

11.5 Analytical Laboratories

Preparation and assaying of the primary samples from the Don Nicolás Project has been carried out at three principal independent laboratories since exploration commenced in 1996:

a. SGS Laboratories (Santiago, Chile) operated the on-site Bema Polimet Laboratory (from 1996 to 1997 and 1999), used by Newcrest and Yamana.
b. ACME Analytical Laboratories (Santiago, Chile) S.A. (1997 and 1999), used by Yamana
c. ALS-Chemex Argentina S.A. (from 2003 to current), used by RYSA, Hidefield and MIRL.

Umpire Laboratory testing was carried out by Yamana from 1996 to 1999, at various South American laboratories, including American Assay Laboratory (AAL) Mendoza and ALS-Geolabs in Chile.

11.6 Sample Preparation and Analytical Procedure

11.6.1 SGS Laboratories (Bema Polimet)

Sample Preparation:

a. Samples are dumped into individual sample drying pans for drying (the original drill sample bag carrying the sample number is placed in the pan with the sample).
b. After drying, the samples are roll crushed to -10-mesh (90% passing).
c. A Jones splitter is used to produce a representative split of approximately 300 to 500 g. The remaining material is bagged and labelled and stored on-site as a “coarse reject”.
d. The 300- to 500-g split is pulverised to -150-mesh (90% passing).

Analytical Procedure:

a. Pulps are split to 2.5 g; for the atomic adsorption (AA) reading (digested with aqua regia and diluted to 50 ml; for reading of Ag, As and Sb).
b. The 30-g split is analyzed via fire assay for Au. The operator makes a visual inspection of the bead produced, if less than 20 ppm Au, then it is digested with aqua regia and Au value is read via AA, if greater than 20 ppm Au, then a gravimetric assay for Au is applied.

Bema Polimet on-site laboratory, is an independent laboratory and was operated by global group SGS Laboratory Group with ISO 9001:2000 accreditation.

11.6.2 ACME Analytical Laboratories

Sample Preparation:

Prepared by the Bema Polimet on-site laboratory; using the sample preparation procedure as described above.

Analytical Procedure:

a. Pulps are split to 0.5 g; for the ICP (4-acid digest and diluted to 10 ml with diluted aqua regia) with a final AES reading of 34 elements including Ag).

b. The 30-g split is analyzed via fire assay for Au. There is no documentation provided to Coffey Mining, which states whether or not a gravimetric determination was performed on high Au values.

ACME Analytical Laboratory (AAL) is an independent laboratory. In 1996, AAL became the first commercial geochemical analysis and assaying lab in North America to be accredited under ISO 9001. AAL in Santiago, Chile received ISO 9001:2000 accreditation in 2005.

11.6.3 ALS Chemex Argentina

On arrival, the samples are weighed and assigned a barcode number for tracking through the process.

Sample Preparation Procedure (Mendoza, Argentina):

a. The samples are dried in a gas oven, at a temperature of 105°C.

b. Samples are crushed to a <2-mm (70% passing or better).

c. Samples are riffle split to approximately 1000 g.

d. Sample then pulverised to <75 µm (85% passing or better).

Prepared pulp samples are flown to the ALS-Chemex laboratory at La Serena in Chile for analysis where the analytical procedure is as follows:

a. Au/Ag; 50-g charge fire assay with an AA Finish (Au/Ag-AA24), where Au values greater than 10 ppm Au detected, they were re-analyzed using a gravimetric method (Au-GRA22).

b. Where Ag returned values greater than 100 ppm Ag, they were re-analyzed using a gravimetric method (Ag-GRA22).

c. Trace Hg was analyzed by aqua-regia digestion, cold vapour with AAS finish (Hg-CV41).

d. A further 27 elements were analyzed by 4-acid digest with ICP-AES finish (ME-ICP61) or with AAS finish if greater than upper detection limit for Mo, Pb and Zn.

ALS-Chemex is an independent laboratory and is part of the global group ALS Laboratory Group with ISO 9001:2000 accreditation.

Whilst on site in June 2010, Coffey Mining visited the ALS-Chemex sample preparation facilities in Mendoza. No announcement of the visit was made prior, and during the inspection, the facility was seen to be in a clean and tidy state and all equipment was in good repair. Coffey Mining considers the sample preparation laboratory to be of industry standard. Coffey Mining has not visited the ALS-Chemex analytical laboratory in Chile.

11.7 Analytical Quality Control Procedures

Quality control sampling and assaying at Don Nicolás follows a clearly defined procedure which has been documented in Bloom (2008) (and was repeated by Runge (2009)). The program used by Hidefield (since 2006), and continued by MIRL is summarized below:

a. Insertion of a coarse blank sample at a ratio of 1 in 50 samples.
b. Insertion of reference material at a ratio of 1 in 25 samples.
c. Routine duplicate assay of pulps as part of the laboratory QAQC program.
d. Insertion of prepared duplicate samples at a ratio of 1 in 25 samples.
e. Insertion of a duplicate drill core sample at a ratio of 1 in 25 samples.

Coffey Mining was unable establish the actual quality control protocol used by previous operators (Newcrest and RYSA), however memos from Yamana (Doe, 1997 and Small, 1997), mentions that blanks and field duplicates are inserted at the ratio of 1 in 60; standards and prepared duplicate samples are inserted at the ratio of 1 in 20. It also states that external laboratory checks are done at the ratio of 1 in 40, and coarse rejects are submitted to external laboratories in the ratio of 1 in 60.

It is assumed that these procedures (for Yamana) were carried on for RYSA (as the company was a joint venture with Yamana). There has been no documentation provided on the procedures used by Newcrest, however only five drillholes from Newcrest are included in the database (Arco Iris), and is considered to have a limited effect on the mineral resource database, plus this data has been spatially verified by subsequent drilling at the Arco Iris deposit.

11.8 Standards and Blanks

Certified reference material “standards” were used by Yamana, RYSA and Hidefield were purchased from; Rocklabs Ltd., New Zealand (OxC58 and OxN49), Geostats Pty Ltd., Western Australia (G397-2 and G398-2), Ore Research and Exploration Pty Ltd., Victoria, Australia (OREAS 10b, 51P, 52Pb, 53P and 62Pa).

MIRL have used their own in-house standards, which have been certified by ACME Analytical Laboratories (Chile) Ltd., as part of a 5-laboratory certification process involving; ACME Analytical Laboratories (Chile) Ltd., Actlabs Skyline Peru S.A.C, ALS Peru S.A., CIMM Peru S.A and SGS Peru S.A.C. Twelve samples from each standard (8006 to 8012) were sent to each laboratory, which used a 30-g Fire Assay analysis, with an Atomic Absorption finish (except for standard 8009, which had a gravimetric finish). The 60 resulting assays for each standard were compiled by ACME Analytical Laboratories (Chile) Ltd., to determine the appropriate confidence intervals.

11.9 Laboratory Duplicates and Standards

11.9.1 SGS and ACME

Yamana implemented a procedure at the SGS Bema laboratory where 1 out of every 10 pulps is re-assayed by the internal laboratory, 1 out of every 20 pulps is a standard or blank. Approximately 1 out of every 20 pulps (selected randomly), is re-numbered and submitted to the internal lab and external labs.

A QAQC report for the 1996/1997 drill campaigns (Doe, 1997), showed that 157 laboratory pulp duplicates, displayed very good correlation, better than 0.9 for Au, Ag and As, and a low standard error of correlation (less than 0.007).

11.9.2 ALS Chemex

The laboratory performs internal Quality Assurance/Quality Control (QAQC) checks to the following description from ALS Chemex.

> *"The Laboratory Information Management System (LIMS) inserts quality control samples (reference materials, blanks and duplicates) on each analytical run, based on the rack sizes associated with the method. The rack size is the number of sample including QC samples included in a batch. The blank is inserted at the beginning, standards are inserted at random intervals, and duplicates are analyzed at the end of the batch."*

The laboratory staff analyses quality control samples at least at the frequency specified above. If necessary, laboratory staff may include additional quality control samples above the minimum specifications.

Between 2006 and 2008, a total of 303 preparation duplicates were prepared routinely by ALS Chemex and 91.1% of them agree within a precision of +/- 10%... Also, a total of 306 laboratory pulp duplicates were re-analyzed, the reproducibility of 89.5% these assays was within +/- 10%, which is considered acceptable.

Data for the internal laboratory checks, for preparation duplicates and laboratory pulp duplicates was not available for the 2010 and 2011 drilling campaigns.

11.10 Umpire Check Assays

11.10.1 SGS and ACME

A QAQC report for the 1996/1997 drill campaigns (Small, 1997), showed the comparison of pulps assayed by an independent labs (American Assay Laboratories, 887 pulps; ACME, 23 pulps; Geolabs, 61 pulps), are shown to have a reasonable correlation. Greater than 90% of all data lie within two standard deviations of the original SGS Bema assays, there appears to be low negative bias reported by the check laboratories (a -8% bias, was recorded with the American Assay Laboratory). The author goes on to explain that variability is quite high, especially in samples near the detection level, but results are within industry standards.

11.10.2 ALS Chemex

No umpire laboratory checking was conducted for the 2006 and 2008 drill campaigns.

Bloom (2008) reports that in 2007; 205 previously sampled pulps were submitted to Alex Stewart Laboratories, Mendoza, Argentina, for check analysis. 75% of this data falls within a precision of +/- 10% (89% with a precision of +/- 20%). From all the samples comparisons it appears that there is a good correlation with the umpire laboratory.

No umpire laboratory checking was conducted for the 2010 or 2011 drill campaigns.

Coffey Mining recommends that umpire laboratory checking and blind pulp sample submissions are re-instated for all future drill campaigns at the Don Nicolás Project.

11.11 Field Duplicates

11.11.1 SGS and ACME

A QAQC report for the 1996/1997 drill campaigns (Small, 1997), showed that 18 field duplicates from RC drilling (drill rig “B” splits), all results were within +/- 2 standard deviations of the original gold assay value, and that there was good correlation in the results (mean percentage difference of + 5.2%).

11.11.2 ALS Chemex

A total of 305 gold field duplicates were submitted to ALS Chemex between 2006 and 2008 (DDH ½ core versus ¼ core), and were evenly distributed across the different deposits (88 at Sulfuro, 139 at Martinetas and 78 at Arco Iris). The field duplicates show good reproducibility, representitivity and no significant bias, with 84.3% are within a precision of +/- 30%. In values of greater than 10 g/t Au, there is an apparent bias (which is only based on four data pairs), though statistically not valid, this bias could be a result of the volume-variance effect from comparing ½ core against a ¼ core field duplicates, in possibly high grade, nugget samples.

The field duplicates for the data for the 2010 to 2011 drilling (278 gold assay pairs) as a total is relatively poor (only 72.3% are within a precision of +/- 30%). The reason for this, is the assays selected for field repeats at Arco Iris (43 gold assay pairs) appear to have all been, with one exception, below 1.0 g/t (86% <0.5 g/t), and therefore relatively small differences may be exaggerated in correlation statistics and distort the results.

At Martinetas (147 gold assay pairs), 80.1% of the data was within a precision of +/- 30%, there was a slight bias in values greater than 3 g/t Au, but as there are only four pairs of data with grades between 3 and 9 g/t Au, the result is inconclusive. At Sulfuro (88 gold assay pairs), there is 88.6% of the data within a precision of +/- 30%, and good correlation is observed in the data.

11.12 Summary of Quality Control Data

The key points from the quality control review include:

a. Commercial standards/and certified in-house standards have been submitted by all operators at Don Nicolás, with the available standards indicating that acceptable accuracy was achieved.

b. Field duplicates and laboratory duplicates indicate acceptable levels of precision were achieved in assaying by ALS Chemex, SGS and ACME.

c. Coffey Mining could not validate the data for the 2003 drill program by RYSA, as no data was available. It is assumed that Yamana procedures were carried on for RYSA (as the company was a joint venture with Yamana).

The data validation methods used by all operators at the Don Nicolás Project, is considered of industry standard, and is considered appropriate for use in the mineral resource estimation. The quality control data indicates that acceptable levels of accuracy and precision were achieved in sampling and assaying.

Coffey Mining recommending that umpire laboratory checking and blind pulp sample submissions are re-instated for all future drill campaigns at the Don Nicolás Project.

11.13 Bulk Density Determination

The Don Nicolás database contains 161 dry in-situ bulk density measurements sampled at various locations at the Sulfuro deposit and Martinetas region. Tables 11-1 and 11-2 summarize the location and result of the bulk densities samples used for the resource estimate, separated by mineralization/geology and oxidation state.

Table 11-1: Bulk Densities - Martinetas

	Mineralized (> 1g/t Au)[1]		Low Grade (0.3 to 1g/t Au)[1]		Background (< 0.3g/t Au)[1]	
	No. of Data	Mean BD[2]	No. of Data	Mean BD[2]	No. of Data	Mean BD[2]
Oxide	18	2.37	6	2.32	9	2.2
Transitional	22	2.52	4	2.38	3	2.31
Fresh	Assumed	2.56	8	2.41	5	2.35

1. Au mineralization threshold used.
2. Average dry in-situ bulk density units are t/m³.

Table 11-2: Bulk Densities – Paloma (Sulfuro)

	Wall-Rock		Stock Work		LG Vein		HG Vein	
	No. of Data	Mean BD[1]	No. of Data	Mean BD[1]	No. of Data	Mean BD[1]	No. of Data	Mean BD[1]
Oxide	8	2.2	Assumed	2.2	Assumed	2.3	Assumed	2.4
Transitional	8	2.3	2	2.3	Assumed	2.4	2	2.5
Fresh	50	2.4	7	2.4	4	2.5	5	2.6

1. Average dry in-situ bulk density units are t/m³

The Sulfuro deposit results were used for all deposits in the La Paloma region. As the current bulk density database is still being improved, there are some areas that do not contain samples and assumed values have been used.

All bulk density measurements were completed used the paraffin coated, water-immersion (Archimedean) technique on dried drill core sample billets.

The sample billets used were approximately 10 cm long, half-core samples. The drillhole name and downhole distance was recorded for each sample, and was used to determine the spatial location.

The measurements have been carried out by CIMM Peru S.A.C. CIMM Peru has the System of Quality Management ISO 9001:2008 certification “System Management Quality” and is accredited with NTP-ISO/IEC 17025:2006 certification “General Requirements for the Competence of Testing and Calibration Laboratories”, for the preparation and assay of geochemical and metallurgical samples.

The results represent a 5 to 10% decrease in the density values used within the mineralized zones as previously reported (Runge, 2009); which was based on pycnometer determinations. Although there is limited information recorded to date, Coffey Mining is confident that the new bulk density data better represents the true in-situ bulk density. Coffey Mining recommends that more bulk density sampling be conducted within all identified geological and oxidation divisions, to increase confidence in the estimate

11.14 ADEQUATE OF PROCEDURES

There is no identified drilling, sampling, or recovery factors that materially impact the accuracy and reliability of the results of the drilling programs in place at Don Nicolás Project. Infill drilling by successive drill programs, have also confirmed the mineralized intercepts spatially.

All samples are considered to be representative of their respective interval. There are no other known factors which could introduce significant bias into the sampling.

Coffey Mining considers that the sample preparation, sample security and analytical procedures associated with data generated to date are consistent with current industry practise and are considered entirely appropriate and acceptable for the style of mineralization identified and appropriate for use in mineral resource estimation.

12.0 DATA VERIFICATION

MIRL has no formal database for the collation of data. A series of Excel spreadsheets are used to store the data. MIRL is in the process of implementing a commercial database for the storage of a validated geological/analytical data.

In the meantime, Coffey Mining has validated all of the data at the Don Nicolás Project against all original certificates (assay and survey), into a central database. This validated database will be merged into the MIRL database when operational.

Data verification steps undertaken have included:

a. Checking of original assay certificates against the generated database.
b. QAQC results analysis by Coffey Mining to identify any irregularities.
c. Independent verification of drillhole collar locations by GPS during site visits undertaken by Coffey Mining.
d. Visual verification of altered and mineralized drill core during the site visit undertaken by Coffey Mining.

12.1 DRILLHOLE TWINNING

No specific twin holes have been drilled on the Don Nicolás deposits to date. There are two pairs of Diamond-RC drillholes at Arco Iris and three Diamond-Diamond drillhole pairs at Sulfuro that are sufficiently close to each other within the mineralized zones; within ± 5 m, to be used as an indication of twin hole validity:

Table 12-1: Drillhole Twinning Data

Deposit	Pair Type	Hole 1	Hole 2	Separation
Sulfuro	DD-DD	S-D07-53	S-D08-57	4.6 m
		S-D06-32	S-D08-61	1.8 m
		S-D07-46	S-D07-48	0.8 m
Arco Iris	DD-RC	A-D06-06	PRC-07	3.2 m
		A-D06-14	PRC-06	3.3 m

In all cases, the trace and statistical comparisons indicate that there is good correlation between the mineralized intervals in the drillholes.

Coffey Mining considers the database used in the resource to be globally robust and is appropriate for use in the resource estimation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The Don Nicolás Gold Project will mine and process 972 t/d of gold-silver bearing material.

A number of metallurgical testwork programs have been conducted on sample material from the La Paloma and Las Martinetas deposits, East Santa Cruz, Argentina. The testwork performed from 2007 onwards will be reviewed on the basis that it represents testwork programs conducted on representative sample material from the deposit.

13.2 HISTORICAL TEST PROGRAMS

Table 13-1 gives the chronological list of the test programs conducted since 2007.

Table 13-1 Historical Test Work Programs and Reports

Document or Test Program	Facility or Laboratory	Test Programs Conducted	Date of Publication
Report A10681	Ammtec	crushing and grinding parameters, head assays, mineralogy, flotation and leaching tests	June 2007
Report A10830	Ammtec	assays	July 2007
Report A12879	Ammtec	gravity concentration an leaching tests	September 2010
Report A13483-A	ALS Ammtec	grinding parameters, assays and leaching tests	August 2011
Report A13483-B	ALS Ammtec	gravity concentration and leaching tests and cyanide detoxification tests	September 2011
Report A13097-A	ALS Ammtec	gravity concentration and leaching tests	September 2011
Report A13097-B	ALS Ammtec	gravity concentration and leaching tests	October 2011
Report A13097-C	ALS Ammtec	leaching and filtration tests	October 2011
Report S1828T-B	Outotec	thickener settling tests	September 2011

13.3 MINERALOGY

Subsamples from an oxide and sulphide composite sample were subjected to mineralogical examination to determine the deportment of gold and silver. The results were given in Report A10681.

13.3.1 Oxide Sample Mineralogy

The oxide sample was described as predominantly a goethite concentrate with some pyrite and marcasite. Free gold was observed optically. Gold particles were observed in grain sizes ranging from 3 to 45 µm. The gold occurrence was predominantly associated with goethite, and minor gold was observed to be associated with quartz.

13.3.2 Sulphide Sample Mineralogy

The sulphide composite was determined to be predominantly three sulphide minerals, pyrite, sphalerite and galena.

The pyrite was found to be mainly discrete particles, but was also found in gangue composites.

The zinc minerals identified were as sphalerite (major) and zincian tetrahedrite/ tennantite (accessory and trace). The sphalerite was either discrete, or locked in galena less commonly as composites with other sulphides.

Galena was identified as the lead mineral present, and this occasionally occurred as discrete, or also, but more frequently locked with sphalerite.

Copper minerals occurred rarely as discrete particles, and the minerals identified are chalcopyrite, tennantite/tetrahedrite, covellite and bornite.

A presence of visible gold was reported in the plus 1 mm size of the sulphide composite fraction with grain sizes between 5 and 40 µm locked with pyrite, quartz and kaolin. Visible gold was not reported in the size minus 1 mm fraction of the sulphide composite.

13.4 Head Grade Analyses

Report No. A10681 characterized 18 samples that were classified by rock type as sulphide samples, oxide samples and oxide-sulphide samples. The samples were composited for use in the metallurgical test program and a summary of the head assay results is given in**Error! Reference source not found.**Table 13-2.

Table 13-2 Calculated Sample Feed Grades - A10681

Sample Identification	Number of Samples	Assay Range		Arithmetic Average Assay Value	
		Au, g/t	Ag, g/t	Au, g/t	Ag, g/t
Oxide	7	1.8 - 38.2	1.9 - 23.9	12.81	10.22
Sulphide	6	3.4 - 14.9	11.3 - 34.0	6.30	19.01
Oxide-sulphide	5	2.5 - 12.2	14.0 - 52.0	8.37	27.57

The individual samples were combined into one sulphide and one oxide composite. The sulphide-oxide samples were combined with the sulphide samples to form the sulphide composite sample. Table 13-3**Error! Reference source not found.** presents the assayed feed grades of the composite samples:

Table 13-3 Composite Sample Feed Grades - A10681

Sample Identification	Assay Au, g/t	Assay Ag, g/t
Oxide Composite	10.75	8.84
Sulphide Composite	7.25	21.23

A multi-element analysis was also conducted on the composite samples and a summary of the results is presented in Table 13-4. The possible presence of coarse gold may explain this difference.

Table 13-4 Composite Multi-Element Analysis - A10681

Sample Identification	Element						
	Au, g/t	Ag, g/t	Cu, ppm	Zn, ppm	S2-, %	As, ppm	Pb, ppm
Oxide Composite	15.4	12.0	28	58	0.09	123	< 5
Sulphide Composite	6.26	20.0	482	8110	3.39	213	4785

Note that the gold and silver assay values reported in Table 13-4**Error! Reference source not found.** vary significantly from the results for both of the composite samples assays reported in Table 13-3, but are comparable with the calculated averages shown in Table 13-2.

Analytical head assays were performed on samples from East Santa Cruz deposit and were reported in Report No. A10830. Three composite samples were prepared, and in addition, two individual samples were submitted for head grade analysis. No further metallurgical testing was done on these samples.

Table 13-5 Head Grade Multi-Element Analysis - A10830

Sample Identification	Comment	Element					
		Au, g/t	Ag, g/t	Cu, ppm	Zn, ppm	S2-, %	Pb, ppm
Oxide 1	Oxide Composite	4.0	9.0	65	82	< 0.02	34
CYN Composite	Sulphide Composite	8.1	38.0	43	99	1.0	20
CO-D06	Individual Composite	6.9	34.0	31	51	0.78	18
CYS-D06	Individual Composite	3.9	88.0	58	18	0.27	30
Sulphide Combined	Sulphide Combined Composite	10.8	44.0	64	67	0.83	25

The results given in Table 13-5 indicate that the samples are predominantly oxide-type since the copper, zinc and lead concentrations are relatively low, while the sulphide sulphur content does not exceed 1% S.

Report No. A13483 included head assays performed on 28 drill core samples from the samples identified as coming from the Las Martinetas area of the East Santa Cruz deposit.

A 36-element head assay ICP scan was performed and the samples were also analyzed using screen fire assayed for gold at a 75 µm fraction. A summary of the results is presented in Table 13-6.

Table 13-6 Summary of ICP and Fire Assay Results, Martinetas - A13483

Element	Units	Maximum Value	Minimum Value	Average Value
Au	g/t	11.6	0.37	3.29
Ag	g/t	75	0.3	10
Cu	ppm	178	4	28
Zn	ppm	62	4	23
S^{2-}	%	320	< 20	60
As	ppm	35	5	12
Pb	ppm	0.58	< 0.02	0.3

Major variation is expected when comparing the head assay results from the individual samples with those samples used in the testwork. However, the following observations relate to the major elements of interest as given in Table 13-6.

a. Gold: the average Martinetas value of 3.29 g/t Au is significantly lower than the head grade value of the sulphide composite sample.

b. Silver: the average Martinetas value of 10 g/t Ag approximates the oxide composite head grade; although it is significantly lower that the sulphide composite head grade value.

c. Copper: the average Martinetas value of 28 ppm Cu is identical with the oxide value, but is significantly lower than the sulphide composite value.

d. Zinc: the zinc content is seen to be significantly lower in the Martinetas samples than that of the oxide composite sample, and particularly the sulphide composite sample.

e. Sulphide Sulphur: the sulphide sulphur values are generally significantly lower than the values reported in Table 13-4.

f. Lead: the lead content of the Martinetas samples was higher than for the oxide sample tested, but significantly lower than for the sulphide sample.

Report No. A13097-A reported the gold and silver assays for sulphide samples from 30 drill samples identified as the Sulfuro area of the deposit. The gold head values varied from 0.64 to 29.6 g/t Au, with an average value of 7.4 g/t Au. The average silver head value was 27.8 g/t Ag.

Report No. A13097-B was a continuation of the test program reported in Report No. A13097-A, and was conducted on two composite samples prepared from the combination of the individual drill core samples, and identified at Sulfuro Met-01 and Sulfuro Met-02. Table 13-7 summarizes the head assay values obtained for these two composite samples.

Table 13-7 Head Assay Results; Sulfuro Composite Samples - A13097-B

Sample Identification	Element						
	Au, g/t	Ag, g/t	Cu, ppm	Zn, ppm	S_T, %	As, ppm	Pb, ppm
Sulfuro Met-01	6.04	26	385	2,770	3.54	160	1,060
Sulfuro Met-02	7.97	26	1,645	13,900	4.40	220	5,860

As can be seen from the tables, the Sulfuro composite samples contain significantly higher values of copper, lead and zinc than previous samples. This type of material will, amongst other effects, increase the reagent consumption during the leaching process.

Report No. A13097-C was the third part of the extended metallurgical test program. However, this used a separate LG bulk composite sample identified as sample SD-10. The composite assay for this sample is shown in Table 13-8.

Table 13-8 Head Assay Results; SD-10 Composite Sample - A13097-C

Sample Identification	Element			
	Au, g/t	Au, g/t SFA*	Ag, g/t	Cu, ppm
SD-10-Bulk	0.87	0.99	3.0	70

**SFA = Screen Fraction Analysis.*

A complete test program was carried out on this sample labelled SD-10. However, considering the head assay results of this sample, namely 0.99 g/t Au, and 3.0 g/t Ag, this is not considered to be representative of the material proposed for treatment in the future mine, and therefore the results obtained will not be used for design purposes.

13.5 Comminution Testwork

Comminution testwork was performed according to the Bond standardized test procedures.

13.5.1 Unconfined Compressive Strength Test – Report No. A10681

The Unconfined Compressive Strength (UCS) test was performed on individual sulphide and oxide-sulphide samples as per industry standards. The oxide sample was not tested by this method. Table 13-9 presents a summary of UCS results obtained for the sulphide and oxide-sulphide samples tested.

Table 13-9 UCS Test Results

Sample	Rock Type	UCS, MPa
S-D06-09	Sulphide	8.6
S-D06-12		18.3
S-D06-13		13.4
S-D06-04		24.0
S-D06-10		6.5
S-D06-10		15.7
Average of Sulphide Samples		14.4
CYN-D06-01	Oxide-Sulphide	0.3
CYN-D06-03		4.8
CYN-D06-06		6.0
Average of Oxide-Sulphide Samples		3.7

The UCS value results indicate that the sulphide samples are consistent with weak to medium strength rock type samples while the oxide-sulphide samples are consistent with very weak strength rock type samples.

13.5.2 Crushing Work Index-Test – Report No. A10681

The Crushing Work Index (CWi) test was also performed according to the Bond standardized procedures.

The sulphide samples were tested in duplicate and the CWi was found to vary between 7 and 20 kWh/t. The average CWi for sulfide samples was 11.4 kWh/t.

The oxide-sulphide samples were also tested in duplicate. The results from the tests showed discordance between duplicate values. No explanation is given nor were tests repeated. The CWi values ranged from 17.6 to 3.6 kWh/t with an average CWi for the oxide-sulphide samples of 8.0 kWh/t.

The specific gravity was also measured and was found to be 2.72.

13.5.3 Bond Mill Work Index Tests and Abrasion Index Test

The Abrasion Index (Ai), the Rod Mill Work Index (RWi) and Ball Mill Work Index (BWi) tests were performed on oxide and sulphide samples throughout the various test programs. The tests used the standard procedure developed by F.C. Bond.

Table 13-10 summarizes the results obtained from the comminution testwork for Report No. A10681 and indicated that the sample material tested was moderately hard and abrasive.

Table 13-10 Comminution Test Results Summary

Test	Units	Closing Screen Setting, µm	Sample Type: Sulphide	Oxide	Oxide-sulphide
Ai	g		0.50	-	-
RWi	kWh/t		21.8	-	-
BWi	kWh/t	82	17.6	17.2	-

Report No. A13483 described the results obtained from a comminution test. This test was to determine the BWi conducted on a sulphide sample using a closing screen size of 106 µm. The BWi reported was 19.7 kWh/t which was significantly higher than the previously obtained value of 17.6 kWh/t.

Report No. A13097-C presented the results of a Bond BWi determination. The result obtained was a BWi value of 14.9 kWh/t using 106 µm as the closing screen size.

The grinding circuit was designed using the BWi of 17.6 kWh/t

13.6 Leach Testing

Testwork programs followed a path of various leaching options. The types of leach programs followed were:

a. Gravity concentration followed by leaching of the gravity tailings gravity tests were aimed to report any free gold and silver amenable to recovery by gravity separation. Subsequently, leaching tests were performed on the gravity tailings in order to compare gravity tailings extraction kinetics versus whole ore leaching kinetics.

b. Whole Ore Leaching.

c. Coarse Ore Leaching.

d. Sulphide flotation followed by leaching of the flotation tailings.

13.6.1 Gravity Concentration Methodology

The gravity recovery gold content of each composite was determined using a combined Knelson concentrator–amalgamation process at a primary grind particle size of P_{80} passing 75 µm. The milled pulp was fed through the Knelson concentrator to produce a gravity concentrate. The gravity concentrate was removed and amalgamated with mercury. The obtained amalgam was submitted for assay in its entirety. The amalgam tailings were mixed with the Knelson tailings to form the gravity tailings sample that was subsequently used for leach testing.

Although termed gravity recoverable gold (GRG), the technique adopted for the testwork by Ammtec is not the standard GRG procedure utilized in North American laboratories. The results obtained can therefore only be used as a general guide.

For purposes of this report, the Ammtec gravity recoverable gold will be referred to as “gold recovered by gravity concentration” and will be written as GRGC in this report to differentiate this from the North American term GRG as defined by Laplante.

13.6.2 LEACHING METHODOLOGY

Leaching of the test samples was carried out under similar test conditions with the feed source the only change in parameter. Leach feed could be as new feed, gravity tailings or flotation tailings. In one instance coarse feed was used as an alternative parameter.

Standard leach conditions applied to all the tests as follows:

a. Particle Size of P80 < 75 µm.
b. Pulp Density 40% (w/w).
c. Initial pH 10.5.
d. Sodium Cyanide Concentration: 500 ppm.
e. Sampling was performed at 2, 4, 6, 8, 24 and 48 hours.

13.6.3 GRAVITY CONCENTRATION RESULTS

Gravity concentration and leaching test were carried out on the sulphide and oxide composite samples and reported in Report No. A10681.

The results obtained from testing gravity testing are summarized in Table 13-11.

Table 13-11 Gravity Concentration Test Results - A10681

Composite Sample	Gold Recovery, %	Silver Recovery, %
Sulphide	4.52	0.94
Oxide	38.64	8.12

Very low gravity recovery is shown in the sulphide sample partially due to the methodology used for gravity separation.

Gravity only results were also reported in Report No. A12879 for sulphide material tested. The results obtained are given in Table 13-12.

Table 13-12 Concentration Test Results; Sulphide Material - A12879

Sample	Gold Recovery, %	Silver Recovery, %
Sulphide Composite	3.8	2.4
S-D06 1-13	0.7	0.6
S-D06 4-10	0.9	1.7

The results obtained for the sulphide samples tested confirmed the low gold and silver recoveries obtained previously. In these tests, the sulphide composite recorded a gold recovery of 3.8% (compared with 4.5% obtained previously). The two other samples tested recovered negligible gold recovery values of 0.7 and 0.9% respectively.

Report No. A13483-B presented specific GRGC results at three grind sizes for the oxide, sulphide/primary, and transitional samples. These results are summarized in Table 13-13.

Table 13-13 GRGC Recoveries versus Grind Particle Size - A13483-B

Sample	Head Assay,	GRGC Extraction at P_{80} Grind Size		
	g/t Au	125 µm	100 µm	75 µm
Oxide	4.20	70.0	73.8	71.1
Sulphide	1.10	19.5	26.6	34.6
Transitional	3.09	17.9	25.3	30.2

High GRGC values between 70 and 74% were reported for the oxide sample. The sulphide and transitional samples reported between 18 and 35% gold recovery.

13.6.4 Gravity Concentration and Tailings Leach Tests vs Whole Ore Leaching

The gold and silver recovery, as well as the reagent consumptions for the combined gravity plus tailings leach tests versus the whole ore leach tests are summarized and presented in Table 13-14 as reported in test Report No. A10681.

Table 13-14 Summary of Gravity plus Leach vs Whole Ore Leach Test Results - A10681

Composite	Test Parameter	Head Grade (Calculated)		8h Leach Recovery, %		48h Leach Recovery, %		Reagent Consumption, kg/t	
		Au, g/t	Ag, g/t	Au	Ag	Au	Ag	Lime	NaCN
Oxide	Gravity + Tailings Leach	13.9	9	93	78	98	94	2.8	0.50
	Whole Ore Leach	19.9	10	81	73	99	93	1.5	0.75
Sulphide	Gravity + Tailings Leach	7.3	20	78	49	84	64	0.57	0.93
	Whole Ore Leach	7.5	19	77	50	84	59	0.68	0.90

For the sulphide composite, whole ore leaching and gravity recovery followed by gravity tailings leaching presented similar results for gold and silver recoveries. An overall gold extraction of 84% was realized using either test procedure. Gold extraction appears to occur at the same rate in either case. A diagnostic leach analysis of the sulphide composite on the gravity-leach tailings residue determined that over 86% of the contained gold in the leach residue was locked in sulphide minerals. This is reported in a subsequent section.

For the oxide composite, the reported oxide gold extraction by whole ore leaching was 98.8% and the reported gold recovery by gravity concentration followed by gravity tailings leaching was 97.7%. Once again, the results are similar to those previously obtained. However, the gravity tailings leach at a faster rate.

The lime consumption for the oxide sample of gravity tailings leach sample is significantly higher than the whole ore leach sample for undetermined reasons related to the test procedure possibly related to the washing out of soluble oxidized material. The cyanide consumption for both samples and the lime consumption for the sulphide samples are relatively consistent.

13.6.5 Gravity Concentration and Tailings Leach Tests

For a majority of the test programs, the gravity extraction followed by cyanide leaching was not followed by whole ore leaching for comparison purposes.

Report No. A13483-B included the GRGC extraction followed by cyanidation of the gravity tailings. Results vary depending on the feed source. Finer grinding enhanced the gravity recovery in the sulphide samples. However, the overall final recovery after cyanidation does not show the same amount of recovery increase.

Table 13-15 GRGC Recoveries vs Grind Particle Size including Gravity Tailings Leaching - A13483-B

Sample	Head Assay, g/t Au	GRGC Extraction at P_{80} Grind Size			Gravity +Tailings Gold Extraction, %
		125 µm	100 µm	75 µm	
Oxide	4.20	70.0	73.8	71.1	98-99
Sulphide	1.10	19.5	26.6	34.6	87-91
Transitional	3.09	17.9	25.3	30.2	93-96

Report No. A12879 reported the sulphide sample gravity testing results. The results obtained from the leaching tests are given in Table 13-16.

Table 13-16 Summary of Gravity Tailings Leaching Tests - A12879

Composite	Head Grade (Calculated)		Total Recovery 8 hrs, %		Total Recovery 48 hrs, %		Reagent Consumption, kg/t	
	Au, g/t	Ag, g/t	Au	Ag	Au	Ag	Lime	NaCN
Sulphide Composite	7.2	17.7	83	51	85	58	0.65	0.94
S-D06 1-13	5.5	11.6	74	44	77	51	0.79	0.83
S-D06 4-10	7.2	17.8	83	38	87	48	0.66	0.90

The results indicate that gold recoveries between 77 and 87% can be obtained from this material tested. The silver recoveries were found to vary between 48 and 58%. Reagent consumption is relatively low at about 0.8 kg/t for lime, and 0.9 kg/t for sodium cyanide. The leaching kinetics for both gold and silver indicate very rapid dissolution with the reactions being essentially completed within 8 hours of leaching, although the silver dissolution process was still on-going at a very slow rate after 48 hours of leaching.

Report No. A13483-A presented the results from gravity concentration and subsequent leaching tests. The target grind P_{80} of 150 µm was not attained, and the leached residues were reground, retreated to gravity concentration and subsequently releached. The results obtained generally conformed with previous results although some samples recorded very high gravity gold recovery values of up to 73%. The overall gold recoveries varied between 81 and 99%. Of interest is that the gravity concentration recovery for silver was found to range between 3.1 and 4.9% with an overall silver recovery between 65 and 71%. Lime and sodium cyanide consumption values were consistent with previously reported results.

Report No. A13483-B reported additional gravity concentration and tailings leaching tests using the staged grinding procedure as was inadvertently used in Report No. A13483-A. In this investigation, three rock types were tested, namely and oxide sample, a sulphide (primary) sample, and a transition material sample. The results obtained have been summarized in Table 13-17.

Table 13-17 Summary of Staged Grinding and Leaching Results - A13483-B

Test Stage	Recovery, %					
	Oxide		Transitional		Primary Sulphide	
	Au	Ag	Au	Ag	Au	Ag
Gravity Concentration, P_{80}=300 µm	55.5	2.4	12.9	1.7	20.4	4.1
Cyanidation on Gravity Tailings P_{80}=300 µm	36.3	70.6	70.7	65.9	56.5	59.5
Gravity Concentration, P_{80}=150 µm	1.9	0.2	0.9	0.1	2.1	0.3
Cyanidation on Gravity tailings P_{80}=150 µm	4.1	15.5	7.6	19.5	7.2	9.9
Total	97.8	88.8	92.1	87.2	86.2	73.8

The results confirm the generally high gold recovery values attainable using gravity concentration followed by leaching. The results also confirm the very low gravity concentration recoveries for silver. Lime and cyanide consumption values were consistent at generally below 0.5 kg/t for both reagents. The results also clearly indicate that there is no advantage to implementing a staged grinding and recovery process, which, as mentioned earlier in the section, was the incorrect procedure adopted for the tests.

Subsequent leaching of the gravity tailings samples was reported in Report No. A13485-B and presented results which are consistent with previously reported information. The finest grind size in all cases presented the highest gold recovery for all three samples tested.

The gravity concentration and cyanide leaching results have been summarized in Table 13-18.

Table 13-18 Summary of Gravity and Cyanidation Test Results – A13485-B

Test Stage	Recovery, %				Average Reagent Consumption	
	Oxide		Primary		NaCN, kg/t	Lime, kg/t
	Au	Ag	Au	Ag		
Gravity Concentration, P_{80}=75 µm	39.0	2.0	19.2	5.8	-	-
Cyanidation on Gravity Tailings, P_{80}=75 µm	60.0	90.5	72.7	88.9	0.82	0.50
Total	**99.0**	**92.5**	**91.9**	**94.7**	**-**	**-**

The cyanide consumption was higher than for the previous results, while the silver recoveries appear to be uncharacteristically high at 93 to 95%.

A set of Sulfuro composite samples was subjected to the standard gravity concentration and gravity tailings leaching tests. The results from this series of tests was reported in Report No. A13097-A. These results gave an average gold dissolution value of 78% with the highest value equal to 93%. The GRGC value was reported to be 5.9%. Relatively high reagent consumption values of 0.94 kg/t for lime, and 1.6 kg/t for cyanide, were also reported.

Report A13097-B recorded the results obtained from the gravity concentration and leaching testwork conducted on composite samples prepared from the Sulfuro material, and in this case the Sulfuro Met-01 composite sample. A P_{80} grind size of 150 µm was used for the gravity concentration testwork followed by leaching tests on the gravity tailings at various particle regrind sizes. The results obtained confirmed the reagent consumption values as previously obtained. However, the gold dissolution values were lower ranging between about 78 and 88% which is consistent with a sulphidic type material. The results also confirmed that the regrind size influences the dissolution of gold. In this case, the highest recovery attained was 88% at a regrind P_{80} size of 53 µm, and the lowest recovery was 78% for a P_{80} size of 150 µm. Leaching kinetics were again rapid and consistent with those of earlier results obtained.

Gravity concentration and gravity tailings leaching tests were also conducted on the Sulfuro Met-02 composite sample. A grind size of P_{80} of 100 µm was used for this series of tests. The GRGC testwork recorded a value of 2.1%, consistent with the treatment of sulphidic material. Varying cyanide concentrations and pH values were tested in the subsequent leaching tests. The cyanide concentration was varied between 1,000 and 3,000 mg/L, while the pH was varied between 9.5 and 11.5. Apart from improving the leach kinetics at the start of the tests, essentially no discernible difference in the results was obtained with the gold recoveries reported at about 75%. However, cyanide consumption values recorded were significantly higher at 2.2 kg/t (or more for the higher cyanide tests) although the lime consumption remained relatively consistent at 1.5 kg/t. Releaching of the leached residue resulted in a further gold recovery of 2.6% which is consistent with the results obtained for the diagnostic leach tests conducted on the Sulfuro Met-01 sample.

Report No. A13097-C reported the outcomes of a sequential series of tests commencing with leaching tests (no gravity concentration) followed by settling and filtration tests, and conducting testwork to determine carbon circuit design parameters. The sample used was a bulk sample identified as SD-10. The results obtained from the leaching tests resulted in a maximum gold recovery value of 46% being obtained at a grind P_{80} of 75 µm. A grind P_{80} of 125 µm gave a gold recovery value of only 23%. Silver recovery was found to be unaffected by grind size over this range, and the silver recovery value obtained was 65%. Lime and cyanide consumption values were relatively high at 1.4 and 1.7 kg/t respectively. Subsequently settling and filtration tests were conducted and the results are reported in the appropriate section.

As mentioned previously, it is considered that this test program using the sample identified as SD-10 was not conducted with a representative sample since the head grade of the sample used was 0.99 g/t Au, and 3.0 g/t Ag which is significantly lower than the material proposed for processing in the future mine. The testwork conducted for the settling and thickening tests will be considered to be valid since the origin of this sample material is proximal to the material which will constitute the feed to the plant.

13.6.6 Coarse Ore Leach Tests

The concept of coarse ore leaching was also tested to determine the amenability of this material to heap leaching. Although not completely relevant to the present design of the plant, the results will be mentioned as testwork which was undertaken. The results have been recorded in Report No. A13483. A total of 17 samples from Martinetas were selected with a feed size of 12.5 mm. The leaching time was 120 hours. The leach residue was screened and each size fraction assayed for gold, silver and copper.

As expected, gold recoveries varied widely from 12 to 90%, with about 70% of the gold being leached within 24 hours. The low sulphide sulphur (0.02% S) gave gold recoveries which varied between 22 and 90%, while the higher sulphide sulphur gold recoveries averaged only 30%. Lime and sodium cyanide consumption was low at up to 0.50 kg/t and up to 0.40 kg/t respectively. The size fraction analysis confirmed that the majority of the samples required comminution to < 2.5 mm for dissolution values of 75% or better to be attained.

Report No. A13097-A reported the cyanide leach tests for the Sulfuro samples. In direct cyanidation leach tests conducted using the standard procedure; gold dissolution was found to vary from 10 to 97% with and average value of 74%. The average silver dissolution was found to be 55%. The reagent consumptions were higher for these tests than were recorded previously, namely 1.5 kg/t for lime and 1.3 kg.t for sodium cyanide. The major amount of the gold lost in the residue, namely 55%, was in the finest size fraction, minus 38 µm. (The minus 38 µm fraction of the leach residue contained 55% of the gold still remaining in the tailings while 10% of the gold still remained in the coarsest fraction at plus 75 µm.) Nearly 10% of the gold lost in the residue was in the coarsest size fraction, namely plus 75 µm.

A further set of samples was subjected to the standard gravity concentration and gravity tailings leaching tests with the results reported in Report No. A13097-A. These results gave an average gold dissolution value of 78% with the highest value equal to 93%. The GRGC value was reported to be 5.9%. Relatively high reagent consumption values of 0.94 kg/t for lime, and 1.6 kg/t for cyanide, were also reported.

13.7 Sulphide Sample CIP Tests (Carbon Kinetics)

Sequential CIP carbon adsorption rate tests were conducted on leached ore slurry samples to determine the Fleming parameters of ‘k’ and ‘n’ required for the design of the carbon circuit.

Figure 13-1 shows the gold carbon loading versus time results and Table 13-19 presents a summary of a chemical analysis performed on the absorbed carbon as reported in Report No. A13097-B.

Figure 13-1 Carbon Adsorption Tests

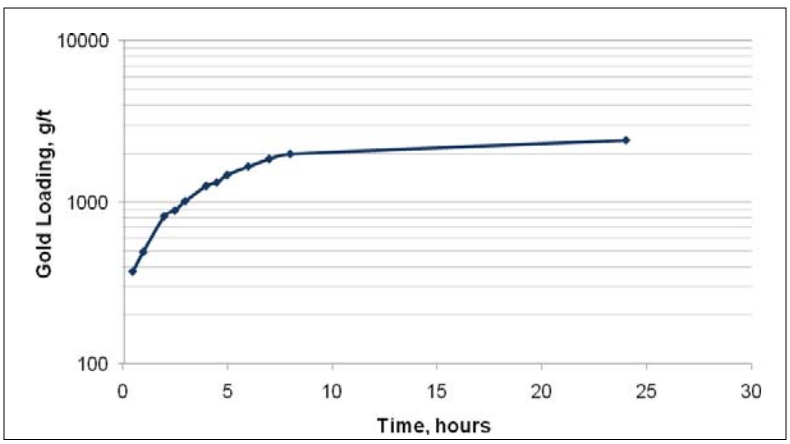


Table 13-19 Chemical Analysis of Loaded Carbon Sample – A13097-B

Element	Units	Maximum Value
Au	g/t	2436
Ag	g/t	3286
Cu	ppm	416
Al	ppm	1,239
Ca	ppm	3,219
Fe	ppm	1,853
Na	ppm	2,690
Ni	ppm	438
Pb	ppm	93
Zn	ppm	304
Mg	ppm	315

For gold, the 'n' value was determined to be 0.58, while the 'k' value was determined to be 81 (hr^{-1}).

The Fleming parameters for silver adsorption were found to be 'n' as 0.50, with the 'k' value 52 (hr^{-1}).

These 'k' values were found to be particularly low, indicating a less than desirable rate of precious metal loaded over a period of time. The low adsorption rate may be caused by carbon adsorption of elements other than gold and silver. At this time, no specific reason for this low adsorption rate is apparent.

Report No. A13097 also presented the results obtained to determine the Fleming 'k' and 'n' parameters. The tests were conducted using material from the Sulfro Met-01 composite sample.

In preparation of the sequential CIP test to determine the Fleming 'k' and 'n' values, the following additional and incidental metallurgical results were also obtained and have been reported here as part of the overall sequential carbon test program:

a. GRGC value of 3.8%.
b. Overall gold recovery after 48 hours leaching time was 85.5%.
c. Lime consumption was 0.87 kg/t, and cyanide consumption 1.3 kg/t.
d. The Fleming parameters obtained were: the 'n' value as 0.616, and the 'k' value of 99.9 (hr^{-1}).

An equilibrium loading test for carbon was also conducted using this material. In this case, similar GRGC, gold recovery and reagent consumption results were obtained as described for the determination of the Fleming parameters. The Freundlich isotherm parameters generated resulted in the following values:

a. ‘k’ value of 3.64 and

b. an ‘m’ value of 0.347.

Report No. A13097-C also reported the outcomes of the carbon parameters which are required in the design of the carbon circuit. The ‘k’ and ‘n’ values were found to be as follows:

a. An ‘n’ value of 0.742 and

b. the ‘k’ value of 168.2 (hr^{-1}).

In addition, the equilibrium loading tests resulted in the following Freundlich isotherm parameters of ‘k’= 2.926 and ‘m’ = 0.28 for the sample tested.

As mentioned before, the SD-10 sample tested cannot be considered representative of the material proposed for mining since the head grade, the gold dissolution, and the reagent consumption values do not represent typical process data for the Don Nicolás deposit.

13.8 Sulphide Sample Flotation and Flotation Tailings Leach Tests

The sulphide composite sample was subjected to a bulk rougher flotation test to produce a gold/silver concentrate followed by the leaching of the flotation tailings. The objective was to use flotation as a pre-concentration step in order to allow further treatment for gold and silver recovery to be performed on a smaller mass fraction of the material. These results were reported in Report No. A13483.

The sample was ground to a particle size of P_{80} of 75 µm for the bulk flotation test. Flotation was performed at a pH between 6 and 6.7, with a pulp density of 34% and using copper sulphate as an activator and potassium amyl xanthate (PAX) as the collector reagent, and with MIBC as the frother reagent.

The results obtained indicated that up to 75% of the gold and 85% of the silver present in the sample can be recovered into the flotation concentrate.

The resulting tailings from the flotation tests were subjected to cyanide leaching in order to improve the recovery of the gold and silver. The leaching conditions were based on the standard procedure described earlier. Table 13-20 summarizes the results obtained for flotation followed by the leaching of the flotation tailings.

Table 13-20 Sulphide Flotation and Tailings Leaching Test Results – A13483

Element	Head, Calculated g/t	Flotation Recovery, % Au	Tailings Leach Recovery, % Au	Total Combined Recovery, %
Gold	6.42	75.0	80.8	95.2
Silver	17.0	85.4	75.5	96.4

The lime consumption for the leach test was found to be 0.73 kg/t while the sodium cyanide used was 1.2 kg/t. The results show that a combined flotation and flotation tailings leaching step will present a combined gold and silver recovery of 95% and 96% respectively. However, the extraction of the gold and silver from the flotation concentrate has not been characterized.

13.9 Diagnostic Leach Test

On completion of the testing, as reported in Report No. A10681, a diagnostic leach analysis of the sulphide composite on the gravity-leach tailings residue determined that over 86% of the contained gold in the leach residue was locked in sulphide minerals.

Report No. A13097 reported results of a basic diagnostic leach procedure test conducted on the leach residue from a sample from the Sulfuro Met-01 composite material. Table 13-21 summarizes the results obtained.

Table 13-21 Diagnostic Leaching Test Results - Leach Residue, Sulfuro Met-01 – A13097

Gold Deportment	Distribution, %
Cyanidable	21.9
Acid digestible	61.0
Silicate encapsulated	17.1
Total	**100.0**

These results indicated that the standard cyanide leach was incomplete. It also indicated that the major proportion of the gold loss occurs as a result of the gold being non-liberated and/or exposed, and present within the sulphide minerals such as pyrite, galena and sphalerite.

13.10 Settling Tests and Filtration

Report No. A13483-A tested a composite sample to determine the type and dosage rate required for use for thickening/dewatering. The sample tested was ground to a particle size P_{80} of 100 µm.

The flocculant with the trade name Magnafloc (MF336) was added at a dosage rate of 25 g/t and the sample was allowed to settle for three days. The results obtained are given in Table 13-22.

Table 13-22 Flocculation and Settling Testwork Results – A13483-A

Sample	Solids SG	Pulp Density, % solids		Flocculant Dosage, g/t	Supernatant Clarity Description
		Initial	Final		
Comminution Composite	2.61	9.42	55	25	Slightly cloudy

The results obtained resulted in a matter of concern which was that a clear solution was not obtained after a settling time of up to three days.

Further dewatering tests were subsequently conducted. The leach residues from the SD-10 sample were filtered and the results reported in Report No. A13097. The moisture contents obtained for all three grind sizes were consistent at 24 to 25%. However, since dry-stack tailings is no longer being considered in the design of the plant, this aspect will not be discussed any further.

Additional settling tests were also conducted at three grind sizes using the SD-10 sample material. A settled density of 65% was obtained with the resulting supernatant solution being clear.

13.10.1 OUTOTEC (THICKENING TESTWORK)

Outotec (Australia) Pty Ltd performed settling tests on selected samples used for leaching testwork done at Ammtec. Settling tests on leach residue samples from Martinetas (primary and oxide) and Sulfuro were performed according to the Outotec procedure using dynamic bench scale testing apparatus. The detoxified and non-detoxified samples of the Martinetas samples were tested while only detoxified samples of Sulfuro were investigated. Tests were done on the basis of a 45 t/h treatment rate. The flocculant used during testing was Magnafloc 342. Results from the non-detoxified samples are the more relevant results as this reflects the process that was used in the design. A summary of the relevant results is given located in Table 13-23.

Table 13-23 Outotec Test Results - September 2011

Sample	Flux Rate, $t/m^2.h$	Pulp Density, % Solids		Flocculant Dosage, g/t	Supernatant Clarity, ppm	Thickener Diameter, m
		Initial	Final			
Martinetas Oxide Non-Detoxified	0.20	12.9	58.5	20	< 100	17
Martinetas Primary Non-Detoxified	0.19	12.8	57.4	41	< 100	17
	0.49	12.8	57.3	41	123	11
Sulfuro Detoxified	0.20	8.9	50.2	40	<100	17
	0.50	8.8	48.1	40	118	11
	0.20	8.9	52.7	81	< 100	17
	0.50	8.8	50.0	81	152	11

Results show that the flux rate has a dramatic impact on the diameter of thickener required. Acceptable suspended solids levels are obtained in all cases at the 11-m sizing. Good overflow quality is obtained on the 17-m diameter thickener. Flocculant requirement varies dramatically between feed samples with up to 80 g/t required for reasonable clarity with acceptable underflow density for the Sulfuro sample. Outotec states that a full-size thickener based on their test results will have a 2 to 3% increase in underflow density.

Outotec did not indicate the reasons for diluting the feed material to a density value as low as 8.8% solids; whereas typically this thickener feed density value is 12 to 15% solids. Based on the results obtained, a tailings thickener diameter of 13 m was selected for the design of the plant.

13.11 CYANIDE DETOXIFICATION

Oxide and sulphide/primary samples were subjected to cyanide detoxification tests and reported in Report No. A13483-B. The gravity concentration step was initially conducted, followed by cyanidation. The design grind size P_{80} of 75 µm was used for this testwork program.

The final slurry was utilized for cyanide detoxification testwork using the SO_2/air test procedure with a target weak acid dissociable (CN_{WAD}) cyanide level of below 5 ppm.

Table 13-24 gives a summary of the results obtained, while Table 13-25 summarizes the test conditions.

Table 13-24 Cyanide Detoxification Results – A13483-B

Element/Compound ppm	Oxide Sample		Primary Sample	
	Leached slurry (feed)	Detoxified Effluent	Leached slurry (feed)	Detoxified Effluent
CN_{free}	159	< 5	133	< 5
CN_{WAD}	221	0.6	155	0.8
CN_{WAD} related Metals:				
• Cu	24.00	2.80	6.60	7.82
• Ni	0.15	0.20	0.10	0.20
• Zn	2.50	0.04	0.84	<0.02
CN_{SAD} related Metals:				
• Ag	0.06	1.56	0.04	0.12
• Fe	5.70	1.50	2.30	3.20
CN_{Total} (calculated)	**237**	**10.5 (average)**	**161**	**9.1 (average)**

Table 13-25 Summary of Detoxification Test Conditions

Condition/Reagent	Oxide Sample	Primary Sample
Duration, minutes	56	55
pH	10.5	10.5
SO_2 g/g CN_{WAD}	2.94	3.38
Cu^{2+}, mg/l	12.5	57.4
Lime g/g CN_{WAD}	1.42	1.90

The most significant conclusions include the following:

a. A duration of 55 minutes is adequate to reduce the cyanide concentration to permissible concentrations.

b. The primary/sulphide sample requires comparatively more reagent addition than the oxide sample.

The detoxification circuit design has allowed for a retention time of 60 minutes.

Report No. A13097 recorded additional testwork results of cyanide detoxification tests conducted on material from the Sulfuro Met-01 composite sample.

Again, as before, and in order to generate sample material for the detoxification tests, the sample was initially subjected to gravity concentration and gravity tailings leaching. The results will also be recorded in this section for completeness.

The results obtained from the gravity and leaching tests were consistent with the previously obtained results, namely a GRGC value of 3.7%, an overall gold leach recovery of 86.3% after 48 hours, and lime and cyanide consumption values of 0.87

and 1.36 kg/t respectively. The subsequent detoxification testwork resulted in the following data as given in Tables 13-26 and 13-27.

Table 13-26 Cyanide Detoxification Results, Sulfuro Met-01 Sample – A13097

Element/Compound ppm	Leached slurry (feed)	Detoxified Effluent
CN_{free}		
CN_{WAD}	349	2-9
CN_{WAD} related Metals:		
• Cu	129.0	8-24
• Ni	0.25	0.05
• Zn	22.6	0.02-0.41
CN_{SAD} related Metals:		
• Ag	nd	nd
• Fe	3.0	0.1-0.4
CN_{Total} (calculated)	**357.5**	**2.5-9.0**

The target CN_{WAD} value < 5 ppm was not attained in all the tests. It is possible that reaction duration and increased SMBS would have improved the results. It is also possible that the copper content in solution was too low to ensure a higher level of reaction of the cyanide detoxification process since copper was not added and only residual copper was present. It is also noticeable that increased amounts of SMBS was added when compared with the earlier tests.

Table 13-27 Summary of Detoxification Test Conditions-Sulfuro Met-01

Condition/Reagent	Met-01 Sample
Duration, minutes	49-50
pH	8.5-9.5
SO_2 g/g CN_{WAD}	4.2 (average)
Cu^{2+}, mg/l	0
Lime g/g CN_{WAD}	0.93 (average)

14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION AND LOCATION OF RESOURCES

Coffey Mining has estimated mineral resources for Don Nicolás with data available prior to the 15th of August 2011. The mineral resource has an effective date of February 14, 2012. There has been no new information collected between the 15th of August and the effective date of this mineral resource.

The Measured and Indicated mineral resource estimates focus on seven separate vein systems within two separate areas of the Don Nicolás property:

a. La Paloma area: Sulfuro and Arco Iris deposits.

b. Martinetas area: Cerro Oro, Lucia, Calafate, Coyote, and Armadillo deposits.

The Don Nicolás FS is based on Measured and Indicated mineral resources for only four of these deposits, as follows:

a. Sulfuro (La Paloma area).

b. Cerro Oro, Coyote, and Armadillo (Martinetas area).

14.2 BASIS OF THE MINERAL RESOURCE ESTIMATES

Grade estimation of the Don Nicolás mineral resource has been carried out by Coffey Mining using a combination of Ordinary Kriging (OK) and Multiple Indicator Kriging (MIK) as follows (Table 14-1):

Table 14-1: Summary of Estimation Methods

Area	Deposit	Estimation Method	Elements Estimated
La Paloma	Sulfuro	OK	Au, Ag, As, Cu, Sb
	Arco Iris	OK	Au, Ag
Martinetas	Cerro Oro	MIK/OK	Au (MIK), Ag (whole block OK)
	Coyote	MIK/OK	Au (MIK), Ag (whole block OK)
	Lucia	MIK/OK	Au(MIK), Ag (whole block OK)
	Calafate	OK (Accumulation)	Au, Ag
	Armadillo	OK	Au, Ag

The Coffey Mining estimates take into consideration several factors including the quantity and spacing of available data, the interpreted controls on mineralization, and the style of mineralization. The estimations were constrained with appropriate geological and mineralization interpretations.

The following sections provide a brief description of the design criteria and estimation basis used by Coffey Mining in estimating the mineral resources.

14.3 SULFURO DEPOSIT

14.3.1 MINERAL RESOURCE DATABASE AND VALIDATION

The drillhole database in the vicinity of Sulfuro contains a total of 213 resource holes drilled between 2006 and 2010. The database contains 148 diamond-holes for 20,039 m and 3-diamond with RC pre-collars for 725 m (440 m pre-collar). The database also contains 67 surface channel and trenches.

The database used in the estimation study contained gold and silver grade information from 7,087 intervals. Arsenic, copper and antimony were also modelled within the gold/silver mineralized zones.

Prior to loading data into the database, Coffey Mining completed several industry standard validations such as cross-checks of hole depth, assay interval, dates, coordinate limits, sample identification, survey deviation data, lithology codes, QAQC data, etc. Prior to geological modelling, various checks were performed on the database including visual checks of drillhole traces.

One drillhole (S-D10-73) was excluded from the estimation database, due to inconsistencies in the spatial relationship of the geology and assays for this hole compared to neighbouring drillholes. No other significant validation errors were detected in the database.

14.3.2 GEOLOGICAL MODELLING PROCEDURE

Geological modelling is based on grade information, geological observations, and oxidation and mineralized domain boundaries. The resource estimation is constrained using wireframes interpreted and constructed by Coffey Mining. Interpretation and digitising of all constraining boundaries have been undertaken on cross sections generally orientated at 065º SW-NE (drill line orientation).

The resultant digitised boundaries have been used to construct wireframe surfaces or solids defining the three-dimensional (3-D) geometry of each interpreted feature. The interpretation and wireframe models have been developed using the Datamine mine planning software package.

14.3.3 GEOLOGICAL INTERPRETATION

Modelling of the HG Sulfuro mineralized structure indicates that it consists of a generally steeply southwest dipping narrow arcuate tabular gold vein. HG mineralization is enclosed by a wall rock-hosted low grade mineralized domain comprised of stockwork structures. The wall rocks are predominately variously altered andesites. The principal gold vein mineralization is characterized by quartz vein, breccia lode or silica breccia lode structures.

HG mineralization is preferentially located within the quartz vein structures. Three mineralized zones were interpreted:

a. Two HG Quartz Veins (Zone 3); based on geological control and a nominal 5 g/t gold cutoff grade.
b. Low-Grade (LG) halo mineralization (Zone 2); surrounds the HG veins and is delimited by a nominal 1 g/t gold cutoff grade.
c. Stockwork (Zone 1); surrounds the HG and LG domains characterized by geological codes identifying stockwork structures.

The downhole thickness of HG veins ranged from 0.4 to 8.5 m, with an average of 1.84 m, the LG halo ranged from 0.4 to 13.4 m, with an average of 2.68 m and the stockwork zones ranged from 0.3 to 26.65 m with an average of 4.9 m. A total of three mineralized units were created that had strike extents up to approximately 820 m.

Extrapolation of the interpreted mineralization was limited to 20 m along strike from known drilling and no more than 40 m downdip from drill intercepts. Mineralization has not been properly closed off at depth or along strike by current drilling.

Most of the Don Nicolás Project is covered by thin to non-existent alluvium covers, which is poorly vegetated. Weathering profiles for completely oxidised, transitional and fresh material were generated. The drillhole recording of weathering codes was very poor; profiles were generated derived from the profiles previously generated by Runge (2009). The average drillhole thicknesses of the weathered material are shown below:

a. Oxide: from 1 to 69.4 m with average of 18 m.
b. Transitional: from 7.1 to 41.7 m with average of 23 m.
c. Fresh: from 3.35 to 259 m with average of 84.8 m.

14.3.4 STATISTICAL ANALYSIS

The lengths of the samples were statistically assessed prior to selecting an appropriate composite length for undertaking statistical analyses, variography and grade estimation. The data captured within the mineralization wireframes was composited to a regular 1 m downhole composite length.

Statistical analysis was carried out on the composited data for each unit to determine the need for appropriate cutting levels for erratic high grade values for gold and silver and the deleterious elements, arsenic, copper and antimony. Statistical assessment of cutting levels considered grade distribution histograms together with changes in coefficient of variation (CV) and Sichel mean with progressive removal of HG samples. Coffey Mining determined cutting statistics as follows (Table 14-2 and Table 14-3):

Table 14-2: Cutting Levels Sulfuro (Paloma) Deposit

Zone	Domain	Au (g/t)	Ag (ppm)	As (ppm)	Cu (ppm)	Sb (ppm)
3	HG Veins	80	330	1100	3500	325
2	LG Halo	30	130	1100	3500	200
1	Stockwork	2	10	850	950	65

Table 14-3: Cutting Statistics for Au (g/t) and Ag (ppm) – Sulfuro Deposit

Zone	Domain	Element	Cutting Statistics				
			Upper Cut	Mean	Std. Dev.	CV	Data Cut
1	HG Veins	Au	80	12.26	15.68	1.28	3
2	LG Halo		30	1.73	3.39	1.96	4
3	Stockwork		2	0.16	0.24	1.58	6
1	HG Veins	Ag	330	34.02	56.52	1.66	2
2	LG Halo		130	9.06	17.06	1.88	3
3	Stockwork		10	34.02	56.52	1.66	26

14.3.5 BULK DENSITY DATA

A total of 86 dry in-situ bulk density readings were taken from various diamond holes drilled within the Sulfuro deposit. These readings were grouped by lithology type (vein, wall rock or stockwork) and weathering profile (fresh, transitional and oxide), and an average bulk density reading was calculated for each sub-domain. Where no data exists an assumed value was used. Table 14-4 summarizes the density data.

Table 14-4: In-Situ Bulk Densities, Sulfuro Vein

	Wall Rock		Stockwork		Low-Grade Vein		High-Grade Vein	
	Samples	Bulk Density (t/m³)	Samples	Bulk Density (t/m³)	Samples	Bulk Density (t/m³)	Samples	Bulk Density (t/m³)
Oxide	8	2.2	Assumed	2.2	Assumed	2.3	Assumed	2.4
Transitional	8	2.3	2	2.3	Assumed	2.4	2	2.5
Fresh	50	2.4	7	2.4	4	2.5	5	2.6

14.3.6 VARIOGRAPHY

The Isatis geostatistical software package was used to analyze the variography for the Sulfuro deposit. Traditional semi-variograms were initially used to analyze the spatial variability of the gold, silver, arsenic, copper and antimony 1-m composites. In order to obtain a better structured variography, correlograms were also modelled.

Variogram analysis using Isatis software on the separate domains resulted in poorly structured variograms. Consequently, variograms for combined domains were generated and produced less poorly structured variograms. However, as these were still less than optimal omni-direction correlograms were adopted for the final variogram.

To facilitate the use of a single search ellipse orientation for the arcuate vein structure, the search ellipsoids for the OK estimate were oriented to lie perpendicular to the overall dip and dip direction of the modelled mineralization, with the major axis along the plunge directions and the ellipsoids flattened perpendicular to the plane of the mineralization.

14.3.7 BLOCK MODEL

A 3-D block model was constructed for the Sulfuro deposit, covering all the interpreted mineralization zones and including suitable additional waste material to allow for pit optimization design.

Using Datamine software, a sub-block model was created in a rotated local grid, orientated along the principal strike direction of the Sulfuro vein (335°). Block coding was completed on the basis of the block centroid.

The parent block size was selected on the basis of the average drill spacing (20 m section spacing) and the variogram models. A parent block size of 2 mE x 10 mN x 10 mRL was selected as appropriate. Sub-blocking to a 0.1 mE x 0.25 mN x 1.0 mRL size was completed to ensure adequate volume representation.

Bulk density was coded to the block model based on the weathering profile and lithology type.

14.3.8 GRADE ESTIMATION AND VALIDATION

Gold, silver, arsenic, copper and antimony grades were estimated into the block model using OK after a statistical cutting had been applied to the original 1 m composite data.

A variety of visual validation checks were done on the data prior to estimation to ensure that composite values and locations matched the original data in the database.

Numerous displays of the resource block model were viewed in cross section, long section and plan views and compared with the original composites data used for estimation.

14.4 Arco Iris Deposit

14.4.1 Mineral Resource Database and Validation

The drillhole database in the vicinity of Arco Iris contains a total of 42 resource holes drilled between 1996 and 2010. The database contains 37 diamond holes for 3,359 m and 5 RC holes for 605 m. The database also contains 23 surface trenches.

The database used in the estimation study contained gold and silver grade information from 2,586 intervals. Prior to loading data into the database normal industry standard checks were undertaken similar to those described in the previous section for the Sulfuro deposit; no significant validation errors were detected in the database.

14.4.2 Geological Modelling Procedure

The geological sequence at Arco Iris is a series of parallel to sub-parallel generally steeply northwest dipping narrow tabular gold veins. The wall rocks are predominately variously altered andesites. Surface mapping has identified two main orientations for the principal veins; striking 040° and 010°, with the main SW-NE veins (040°) offset approximately 100 m to the east and 20 m to the north by a WSW-ENE fault.

14.4.3 Mineralization Interpretation

Modelling of the Arco Iris mineralization consists of a series of parallel to sub-parallel generally steeply northwest dipping narrow tabular gold veins. The wall rocks are predominately variously altered andesites. The principal gold mineralization is characterized by silica breccia lode and mineralized andesites.

Eight mineralized domains were interpreted, with a nominal 0.5 g/t gold lower cutoff grade and a 1 m minimum sample length. Extrapolation of the interpreted mineralization was limited to 20 m along strike from known drilling and no more than 40 m downdip from drill intercepts. Mineralization has not been properly closed off at depth or along strike by current drilling in any domain/zone.

14.4.4 Statistical Analysis

The lengths of the samples were statistically assessed prior to selecting an appropriate composite length for undertaking statistical analyses, variography and grade estimation. The data captured within the mineralization wireframes was composited to a regular 1 m downhole composite length.

Statistical analysis was carried out on the composited data for each unit to determine the need for appropriate cutting levels for erratic high grade values for gold and silver. Statistical assessment of cutting levels considered grade distribution histograms together with changes in CV and Sichel mean with progressive removal of HG samples. Coffey Mining determined cutting statistics on an individual mineralized domain basis, summary statistics for the combined are listed in Table 14-5.

Table 14-5: Cutting Statistics for Au (g/t) and Ag (ppm) –Arco Iris Deposit

Zone	Domain	Element	Cutting Statistics				
			Upper Cut	Mean	Std. Dev.	CV	Data Cut
1-8	Combined	**Au**	17.5	2.31	3.57	1.55	5
1-8	Combined	**Ag**	15	2.73	2.68	0.98	6

14.4.5 BULK DENSITY DATA

No bulk density data was available for the Arco Iris Deposit, so the data from Sulfuro was used. In order to be conservative, the bulk density of the “wall rock” material was used. Bulk density values of 2.2 t/m^3, 2.3 t/m^3 and 2.4 t/m^3 were used for the oxide, transitional and fresh material, respectively, throughout the deposit.

14.4.6 VARIOGRAPHY

The Isatis geostatistical software package was used to analyze the variography for the Arco Iris deposit. Traditional semi-variograms were initially used to analyze the spatial variability of the gold and silver 1-m composites. In order to obtain a better structured variography, correlograms were also modelled.

Variogram analysis using Isatis software on the separate domains resulted in poorly structured variograms; therefore, variograms for combined domains were generated and produced less poorly structured variograms. As these were still less than optimal, omni-direction correlograms were adopted for the final variogram.

14.4.7 BLOCK MODEL

A 3-D block model was constructed for the Arco Iris deposit, covering all the interpreted mineralization zones and including suitable additional waste material to allow for pit optimization design.

Using Datamine software, a sub-block model was created in a rotated local grid, orientated along the principal strike direction of the Arco Iris veining (40º). Block coding was completed on the basis of the block centroid.

The parent block size was selected on the basis of the average drill spacing 40 m section spacing) and the variogram models. A parent block size of 2 mE x 10 mN x 10 mRL was selected as appropriate. Sub-blocking to a

0.25 mE x 2 mN x 2 mRL size was completed to ensure adequate volume representation.

Bulk density was coded to the block model based on the weathering profile and lithology type.

14.4.8 GRADE ESTIMATION AND VALIDATION

Gold and silver, grades were estimated into the block model using OK after a statistical cutting had been applied to the original 1 m composite data.

A variety of visual validation checks were done on the data prior to estimation to ensure that composite values and locations matched the original data in the database.

Numerous displays of the resource block model were viewed in cross section, long section and plan views and compared with the original composites data used for estimation.

Extra drilling at the Arco Iris deposit has resulted in a significant reduction in gold ounces in the current estimate, compared to the previous reported mineral resource (Runge 2009).

14.5 MARTINETAS CENTRAL AREA

14.5.1 MINERAL RESOURCE DATABASE AND VALIDATION

The drillhole database for the Martinetas Central area deposits (Cerro Oro, Lucia, Coyote and Calafate) contains a total of 117 diamond holes for 12,098 m and 161 RC holes for 10,245 m. The database also contains 79 surface trenches for 6,207 m which were also used for the mineral resource estimate.

The database used in the estimation study contained gold and silver grade information from 19,990 intervals. Prior to loading data into the database normal industry standard checks were undertaken, similar to those described in the previous section for the Sulfuro deposit; no significant validation errors were detected in the database.

14.5.2 GEOLOGICAL MODELLING PROCEDURE

The geological models for the Martinetas Central area deposits are based on grade information, interpretation of geological domain boundaries, and oxidation domain boundaries.

Wireframes have been modelled to constrain the resource estimates. For the purpose of mineral resource estimation in the Martinetas Central Area, four mineralized zones were interpreted by Coffey Mining:

a. Zone 1: Cerro Oro (notional 0.2 g/t Au cutoff grade).
b. Zone 2: Lucia (notional 0.2 g/t Au cutoff grade).
c. Zone 3: Coyote (notional 0.2 g/t Au cutoff grade).
d. Zone 4: Calafate (notional 1.0 g/t Au cutoff grade).

Extrapolation of the interpreted mineralization was limited to 20 m along strike from known drilling and no more than 40 m downdip from drill intercepts. Mineralization has not been properly closed off at depth or along strike by current drilling.

Within Zone 1, two domains have been identified. Due to the nearby influence of the "Falla Blanca" north-south fault system, the orientation of the veinlets within Zone 1 change from a more westerly strike in the east, to a more northerly direction in the west. The two domains were created to allow a change in the variography direction, for the grade estimation using a soft boundary contact.

14.5.3 *Statistical Analysis*

Sample lengths were statistically assessed prior to selecting an appropriate composite length for statistical analyses, variography and grade estimation, as follows:

a. Zones 1, 2 and 3: 2 m downhole composites.
b. Zone 4: downhole widths converted to estimated true horizontal width (EHW), for use in the OK thickness accumulation calculations.

The global effect of the compositing, summarized by Zone and Domain, is shown in the following tables (Table 14-6 and Table 14-7).

Table 14-6: Summary Statistics Au g/t – Composite Data

Zone	Description	Count	Min	Max	Mean	Std. Dev.	SD	CV
1	Domain 1	1615	0.0025	29.22	**0.44**	1.61	2.60	3.68
	Domain 2	2313	0.0025	36.81	**0.59**	2.00	4.02	3.42
2	-	547	0.0038	40.85	**0.72**	2.72	7.38	3.79
3	-	4184	0.0025	682.81	**1.07**	12.22	149.36	11.44

CV = Coefficient of Variance; SD = Standard Deviation

Table 14-7: Summary Statistics Ag g/t – Composite Data

Zone	Description	Count	Min	Max	Mean	Std. Dev.	SD	CV
1	Domain 1	1615	0.1	223.50	4.30	10.67	113.73	2.48
	Domain 2	2312	0.1	72.11	1.73	3.98	15.86	2.30
2	-	547	0.1	232.08	1.54	10.67	113.82	6.93
3	-	4184	0.1	872.40	2.92	15.91	153.15	5.45

Statistical analysis was carried out on the composited data for each unit to determine appropriate cutting levels to apply to HG erratic values. A list of the determined upper cuts applied and their impact is set out in Table 14-8.

Table 14-8: Cutting Statistics, Martinetas

Zone	Domain	Element	Cutting Statistics				
			Upper Cut	Mean	Std. Dev.	CV	Data Cut
1	1	Au	30	0.44	1.61	3.67	0
	2		30	0.58	1.86	3.23	4
2	-		30	0.69	2.39	3.45	2
3	-		40	0.76	3.16	4.14	13
4	-		NC	6.13	6.16	1.00	0
1	1	Ag	100	4.19	9.03	2.16	3
	2		100	1.73	3.98	2.30	0
2	-		100	1.30	5.84	4.50	1
3	-		80	2.62	6.66	2.54	11
4	-		NC	11.57	7.03	0.61	0

Conditional statistics for data within each zone to be estimated by Multiple Indicator Kriging (Zones 1 to 3) are listed in Tables 14-9 to 14-11.

Table 14-9: Indicator Class Means – Zone 1

Bin	Probability Threshold	Grade Threshold (Au g/t)	Class Mean (Au g/t)
1	0.4976	0.11	0.04
2	0.5966	0.17	0.14
3	0.7016	0.27	0.22
4	0.7935	0.45	0.35
5	0.8505	0.68	0.56
6	0.8999	1.08	0.84
7	0.9358	1.63	1.33
8	0.9605	2.51	2.08
9	0.9773	3.80	3.13
10	0.9880	6.45	4.76
11	0.9947	10.02	7.75
12	0.9980	20.40	13.65
	Max	Max	30.98

Table 14-10: Indicator Class Means – Zone 2

Bin	Probability Threshold	Grade Threshold (Au g/t)	Class Mean (Au g/t)
1	0.4863	0.11	0.05
2	0.5923	0.17	0.14
3	0.7002	0.28	0.22
4	0.7971	0.57	0.40
5	0.8483	0.83	0.68
6	0.8995	1.28	1.01
7	0.9196	1.73	1.54
8	0.9470	2.71	2.09
9	0.9671	4.12	3.24
10	0.9799	5.97	4.90
11	0.9909	10.87	8.05
12	0.9945	14.40	12.24
	Max	Max	30.79

Table 14-11: Indicator Class Means – Zone 3

Bin	Probability Threshold	Grade Threshold (Au g/t)	Class Mean (Au g/t)
1	0.5860	0.13	0.04
2	0.6986	0.22	0.17
3	0.8004	0.45	0.32
4	0.8542	0.73	0.56
5	0.9006	1.21	0.94
6	0.9395	2.37	1.63
7	0.9579	3.68	2.98
8	0.9723	5.55	4.54
9	0.9845	10.55	7.53
10	0.9900	13.40	12.08
11	0.9940	19.20	16.93
12	0.9969	38.50	27.30
	Max	Max	138.15

It should be noted that, while gold grades are not cut or capped for the purposes of MIK estimation, the use of cut grades is employed for variography and the change of support process; HG cuts have been implemented prior to the calculation of the intra class statistics.

14.5.4 BULK DENSITY DATA

Seventy-Five (75) dry in-situ bulk density readings were taken from various diamond holes drilled within the Martinetas central zone. These readings were grouped by mineralization and weathering profile. Average bulk density reading was calculated for each sub-domain; Table 14-12 summarizes the density readings:

Table 14-12: Bulk Densities, Martinetas

	Mineralized (> 1 g/t Au)[1]		Low-Grade (0.3 to 1 g/t Au)[1]		Background (< 0.3 g/t Au)[1]	
	Data	Mean BD[2]	Data	Mean BD[2]	Data	Mean BD[2]
Oxide	18	2.37	6	2.32	9	2.2
Transitional	22	2.52	4	2.38	3	2.31
Fresh	Assumed	2.56	8	2.41	5	2.35

1. *Au mineralization threshold used.*
2. *Average dry in-situ bulk density units are t/m³.*

14.5.5 Variography

Several types of variogram calculations are employed to determine the directions of the continuity of the mineralization:

a. Traditional variograms are calculated from the raw assay values.
b. Gaussian variograms are based on the results after declustering and a transformation to a normal distribution.

Correlograms are 'standardized' by the variance calculated from the sample values that contribute to each lag.

MIK Variography

Grade and indicator variography were generated to enable grade estimation via MIK and change of support analysis to be completed. In addition, Gaussian variograms were also examined as part of the change of support process. Five out of the 12 indicator thresholds for Zones 1 to 3 had variograms modelled. Interpreted anisotropy directions correspond well with the modelled geology and overall geometry of the interpreted zones.

OK Variography (Thickness Accumulation)

Grade variography (to be used for OK estimation for gold and silver) for Zone 4 generally shows less well defined structure than the other zones. This can be attributed to the lesser number of samples contained within the zone (only 19 intercepts are available to estimate the narrow Calafate vein). Correlograms have been calculated and modelled for grade accumulations and thickness.

14.5.6 Block Model

A 3-D block model was constructed for the Martinetas Central area deposit, covering all the interpreted mineralization zones and including suitable additional waste material to allow later pit optimization studies.

The parent block size was selected on the basis of the average drill spacing (30 m section spacing) and the variogram models. A parent block size of 20 mE x 5 mN x 10 mRL was selected as appropriate. Sub-blocking to a 1.0 mE x 0.5 mN x 0.25 mRL size (1/20th parent block) was completed to ensure adequate volume representation.

The attributes coded into the block models included the weathering and mineralization models. Bulk density was coded to the model based on the weathering profile and lithology type. A visual review of the wireframe solids and the block model indicates robust flagging of the block model.

14.5.7 Grade Estimation

Mineral resource estimation for the Martinetas Central area mineralization was completed as follows:

a. Zone 1 to 3: MIK for gold; OK for silver.
b. Zone 4: using OK on the 2D accumulation parameter (grade times thickness for both gold and silver).

All grade estimation was carried out using Datamine mine planning software.

14.5.8 Selective Mining Units

Calculation of selective mining unit estimates was undertaken using macros developed by Coffey Mining.

Applying the modelled variography, variance adjustment factors were calculated for Zones 1 to 3 (inclusive) for a 5 mE x 2 mN x 2.5 mRL selective mining unit (SMU) via the indirect lognormal change of support. Whole block and SMU grades were estimated.

An 'information effect' factor is commonly applied to the originally derived panel-to-block variance ratios to determine the final variance adjustment ratio. The goal of incorporating information effect is to calculate results taking into account that mining takes place based on grade control information.

14.5.9 Estimation Parameters

Zones 1 to 3:

a. Au: MIK
b. Ag: OK

Zone 4:

a. Au: OK (on grade-thickness accumulation).
b. Ag: OK (on grade-thickness accumulation).

The MIK gold estimates are based on indicator correlogram models and a set of ancillary parameters controlling the source and selection of composite data. The sample search parameters were defined based on the variography and the data spacing, and a series of sample search tests performed in Isatis geostatistical software; a total of 12 indicator thresholds were estimated for Zones 1 to 3.

The Zone 4 OK estimates are based on grade correlogram models and a set of ancillary parameters controlling the source and selection of composite, accumulation and thickness data (for both gold and silver).

Hard domain boundaries were applied to all zones and for both Au and Ag estimation, except Zone 1 where a soft boundary was used to distinguish Domains 1 and 2 (west and east) in order to allow for a change in orientation in the zone.

Alternative estimates were also completed to test the sensitivity of the reported model to the selected MIK and OK interpolation parameters (E-type estimate comparisons for gold and ID2 for silver). An insignificant amount of variation in overall grade was noted in the alternate estimations.

14.6 ARMADILLO

14.6.1 *MINERAL RESOURCE DATABASE AND VALIDATION*

The drillhole database in the vicinity of Armadillo contains a total of 41 resource holes drilled between 1996 and 2011. The database contains 20 diamond holes for 1,722.1 m and 21 RC holes for 1314 m. The database also contains 10 surface trenches.

The database used in the estimation study contained gold and silver grade information from 2,964 intervals.

Prior to loading data into the database, several industry standard validations and checks were carried out similar to those at Sulfuro and Martinetas; no significant validation errors were detected in the database.

14.6.2 *GEOLOGICAL MODELLING PROCEDURES*

Based on grade information and geological observations, oxidation and mineralized domain boundaries were interpreted and wireframes modelled to constrain the mineral resource estimation for the Armadillo deposit. Interpretation and digitising of all constraining boundaries has been undertaken on cross sections generally orientated north-south (drill line orientation).

The resultant digitised boundaries have been used to construct wireframe surfaces or solids defining the 3-D geometry of each interpreted feature. The interpretation and wireframe models have been developed using the Datamine mine planning software package.

The geological sequence at Armadillo consists of six sub-parallel ESE (110º) striking and sub-vertically dipping narrow tabular gold/silver mineralized structures, associated with silicified quartz veins within a flatly dipping rhyolitic sequence.

Modelling of the Armadillo mineralization consists of a generally steep northerly dipping narrow tabular gold vein mineralization. The wall rocks are predominately flat dipping rhyolites. The principal gold/silver vein mineralization is characterized by quartz vein structures.

High grade mineralization is preferentially located within the quartz vein.

The downhole thickness of HG veins ranged from 0.6 to 10 m, with an average of 1.87 m. A total of six mineralized units were created that have a strike extent up to approximately 260 m.

Extrapolation of the interpreted mineralization was limited to 20 m along strike from known drilling and no more than 20 m downdip from drill intercepts. Mineralization extends to a depth of approximately 20 to 60 m below surface.

14.6.3 STATISTICAL ANALYSIS

The lengths of the samples were statistically assessed prior to selecting an appropriate composite length for undertaking statistical analyses, variography and grade estimation. The data captured within the mineralization wireframes was composited to a 2 m downhole composite length, residual lengths less than 1 m were combined with the adjacent composite.

Statistical analysis was carried out on the composited data for each unit to determine the need for appropriate cutting levels for erratic high grade values for gold and silver. Statistical assessment of cutting levels considered grade distribution histograms together with changes in CV and Sichel mean with progressive removal of HG samples. Coffey Mining determined cutting statistics on an individual mineralized domain basis; summary statistics are as follows (Table 14-13 and 14-14).

Table 14-13: Summary Cut Statistics Au g/t – Composite Data

MZone	Description	Count	Min	Upper Cut	Mean	Std. Dev.	CV	Data Cut
10 to 60	All HG Zones	311	0.003	30	**4.10**	5.72	1.40	6
99	LG Halo	1188	0.003	0.9	**0.20**	0.21	1.09	9
0	Background	1465	0.003	0.5	**0.04**	0.08	1.88	11

Table 14-14: Summary Cut Statistics Ag g/t – Composite Data

MZone	Description	Count	Min	Upper Cut	Mean	Std. Dev.	CV	Data Cut
10 to 60	All HG Zones	311	0.1	30	**5.84**	6.44	1.10	7
99	LG Halo	1188	0.1	9	**1.40**	1.57	1.13	5
0	Background	1465	0.1	4	**0.67**	0.73	1.08	6

14.6.4 BULK DENSITY DATA

No actual bulk density measurements were taken from the Armadillo deposit, data from the Martinetas central zone, located approximately 1.5 km to the northwest, was used to represent this area. These readings were grouped by mineralization and weathering profile (fresh, transitional and oxide) and an average bulk density reading was calculated for each sub-domain.

14.6.5 VARIOGRAPHY

The Isatis geostatistical software package was used to analyze the Armadillo Deposit (MZone 10 to 60 combined, MZone 99 and MZone 0) variography. Traditional semi-variograms were initially used to analyze the spatial variability of the gold and silver 2-m top-cut composites. In order to obtain a better structured variography, correlograms were also modelled.

14.6.6 BLOCK MODEL

A 3-D block model was constructed for the Armadillo deposit, covering all the interpreted mineralization zones and including suitable additional waste material to allow later pit optimization studies.

A sub-block model was created in a rotated local grid, orientated along the principal strike direction of the Armadillo deposit, using Datamine mine planning software. Block coding was completed on the basis of the block centroid, wherein a centroid falling within any wireframe was coded with the wireframe solid attribute.

The parent block size was selected on the basis of the average drill spacing (15-m section spacing) and the variogram models. A parent block size of 15 mE x 5 mN x 10 mRL was selected as appropriate. Sub-blocking to a 0.75 mE x 0.25 mN x 0.5 mRL size was completed to ensure adequate volume representation. The attributes coded into the block models included the weathering and mineralization models. A visual review of the wireframe solids and the block model indicates robust flagging of the block model.

Bulk density has been coded to the block model based on the weathering profile and mineralization type.

14.6.7 GRADE ESTIMATION AND VALIDATION

Mineral resource estimation for the Armadillo mineralization was completed using OK within all mineralized domains for gold and silver assays.

Gold and silver grades were estimated into the block model using OK after top-cuts had been applied to the original 2-m composite data. Variography for the HG veins was combined (MZone 10 to 60). However, for the estimation, only data for each respective zone was used.

Neighbourhood testing was carried out for this estimation. Second pass with two times multiplier for the search radii was applied, although the majority of the blocks were estimated in the first pass. .

A variety of visual validation checks were done on the data prior to estimation to ensure that composite values and locations matched the original data in the database.

Numerous displays of the resource block model were viewed in cross section, long section and plan views and compared with the original composites data used for estimation.

14.7 Mineral Resource Classification

The grade estimates have been classified as Measured, Indicated and Inferred in accordance with NI 43-101 guidelines based on the confidence levels of the key criteria that were considered during the resource estimation. The key criteria and associated confidence considered for each of the seven deposits consisted of:

a. Drilling/Sampling Techniques.
b. Logging.
c. Drill Sample Recovery.
d. Sub-Sampling Techniques and Sample Preparation.
e. Quality of Assay Data.
f. Verification of Sampling and Assaying.
g. Location of Sampling Points.
h. Data Density and Distribution.
i. Any Previous Audits or Reviews.
j. Database Integrity.
k. Geological Interpretation.
l. In-Situ Dry Bulk Density Sampling.
m. Estimation and Modelling Technique.
n. And any Mining Factors or Assumptions.

The criteria used to distinguish between each category are as follows:

MEASURED MINERAL RESOURCE:

a. Based on regions with strong geological understanding and continuous mineralization.

b. Other factors considered include block variance and slope of regression as measures of estimate quality.

c. Consideration is primarily given to those blocks where the distance to the nearest composite used in the estimate is less than 20 m (at Sulfuro).

INDICATED MINERAL RESOURCE

a. Based on regions with strong geological understanding and continuous mineralization.

b. Other factors considered include block variance and slope of regression as measures of estimate quality.

Consideration is primarily given to those blocks where the distance to the nearest composite used in the estimate is less than 30 to 40 m depending on deposit.

INFERRED MINERAL RESOURCE

a. Based on regions that lie within interpreted regions proximal to known mineralization.

b. Other factors considered include block variance and slope of regression as measures of estimate quality.

c. Meet the estimation criteria for each of the deposits.

Assignment of mineral resource categories into the block model was achieved by the use of a block model manipulation macro in conjunction with wireframes constructed to delineate Measured, Indicated and Inferred mineral resources for each deposit, where applicable.

14.8 MINERAL RESOURCE SUMMARY TABLE

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Table 14-1 provides a summary of Measured and Indicated mineral resources estimated for all deposits found on the Don Nicolás property.

Note that the FS is based only on four mineral resource areas; Sulfuro, Cerro Oro, Coyote and Armadillo deposits (shown in bold text in Table 14-15).

Additional mineral resources in the Inferred category are listed in Table 14-16.

Table 14-15: Don Nicolás Measured and Indicated Mineral Resources

			Measured Mineral Resource					Indicated Mineral Resource				
	Deposit	Cutoff Au, g/t	kt	Au		Ag		kt	Au		Ag	
				(g/t)	(kOz)	(g/t)	(kOz)		(g/t)	(kOz)	(g/t)	(kOz)
La Paloma	**Sulfuro**	**0.3**	**270.9**	**7.2**	**62.5**	**24.0**	**208.9**	**921.4**	**3.7**	**109.3**	**13.8**	**408.4**
		1.6	**168.8**	**10.8**	**58.7**	**33.4**	**181.3**	**329.6**	**8.4**	**88.6**	**25.4**	**268.8**
	Arco Iris	0.3						36.8	1.7	2.1	2.2	2.6
		1.6						18.0	2.4	1.4	2.8	1.6
Martinetas	**Cerro Oro**	**0.3**						**2,528.5**	**1.1**	**85.6**	**3.9**	**316.5**
		1.6						**378.3**	**3.3**	**39.9**	**6.1**	**73.8**
	Lucia	0.3						94.1	1.3	4.1	0.8	2.3
		1.6						18.3	3.9	2.3	0.7	0.4
	Coyote	**0.3**						**1,603.4**	**1.9**	**99.7**	**3.5**	**179.5**
		1.6						**440.8**	**5.1**	**72.4**	**5.8**	**82.5**
	Calafate	0.3						4.0	3.2	0.4	10.8	1.4
		1.6						4.0	3.2	0.4	10.8	1.4
	Armadillo	**0.3**						**179.0**	**3.1**	**17.6**	**4.7**	**27.0**
		1.6						**102.7**	**4.9**	**16.1**	**6.2**	**20.5**
Total All Deposits		**0.3**	**270.9**	**7.2**	**62.5**	**24.0**	**208.9**	**5,367.1**	**1.8**	**318.9**	**5.4**	**937.5**
		1.6	**168.8**	**10.8**	**58.7**	**33.4**	**181.3**	**1,291.7**	**5.3**	**221.1**	**10.8**	**449.0**

Table 14-16: Don Nicolás Inferred Mineral Resources

	Deposit	Cutoff Au, g/t	Inferred Mineral Resource				
			kt	Au		Ag	
				(g/t)	(kOz)	(g/t)	(kOz)
La Paloma	**Sulfuro**	**0.3**	**535.0**	**1.2**	**20.6**	**5.4**	**92.5**
		1.6	**47.3**	**7.0**	**10.7**	**18.7**	**28.4**
	Ramal	0.3	134.8	1.9	8.3		
		1.6	58.5	2.7	5.1		
	Rocio	0.3	89.2	4.1	11.9		
		1.6	89.2	4.1	11.9		
	Arco Iris	0.3	262.4	2.3	19.4	2.1	17.5
		1.6	164.0	3.0	15.7	2.5	13.2
Martinetas	**Cerro Oro**	**0.3**	**995.8**	**1.0**	**32.9**	**4.1**	**130.7**
		1.6	**144.4**	**3.4**	**15.9**	**7.0**	**32.7**
	Lucia	0.3	225.5	1.1	7.9	2.1	15.3
		1.6	38.1	3.4	4.1	4.4	5.4
	Coyote	**0.3**	**612.6**	**1.6**	**30.5**	**3.1**	**60.9**
		1.6	**132.6**	**4.7**	**20.2**	**5.6**	**23.8**
	Calafate	0.3	3.4	5.8	0.6	11.7	1.3
		1.6	3.4	5.8	0.6	11.7	1.3
	Armadillo	**0.3**	**209.7**	**1.9**	**12.6**	**4.2**	**28.4**
		1.6	**66.0**	**5.0**	**10.6**	**6.9**	**14.6**
Total		**0.3**	**3,068.5**	**1.5**	**144.8**	**3.5**	**346.6**
		1.6	**743.5**	**4.0**	**94.9**	**5.0**	**119.4**

Notes:
Rocio and Ramal were reviewed but not re-estimated by Coffey Mining, as no new sampling exists. The grades of these Inferred Mineral Resources (gold only) are the same as previously reported by Hidefield in 2009 as estimated by Runge mining consultants (Runge). Tonnages for Rocio and Ramal have been adjusted to match the newly collected bulk density data from the nearby Sulfuro deposit.

Material reported above the 1.6 g/t Au cutoff is inclusive in the material reported above the 0.3 g/t Au cutoff.

14.9 Reasonable Prospects for Economic Extraction

Based on the mineralization continuity, shape and distribution, the deposits investigated in this study are expected to be mined using traditional open-cut mining methods.

It is also considered appropriate to report the resource above a 0.3g/t Au and a 1.6g/t Au cutoff grade, where the material between 0.3 and 1.6g/t Au cutoff could be used for a heap leach extraction, similar to the nearby working mine Cerro Vanguardia.

15.0 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVE SUMMARY

Table 15-1 sets out the Proven and Probable Mineral Reserves for the Don Nicolás Project as of December 2011 based on metal prices of US$1,100/oz Au and US$25/oz Ag.

Table 15-1: Mineral Reserve Summary, Don Nicolás Project

Deposit/Category		Tonnes	Au g/t	Ag g/t
La Paloma	Proven	137,624	10.74	33.55
	Portable	257,406	4.96	14.84
	Total	395,030	6.97	21.36
Martinetas	Proven	-	-	-
	Portable	680,803	4.30	5.07
	Total	680,803	4.30	5.07
Armadillo	Proven	-	-	-
	Portable	128,275	3.34	4.60
	Total	128,275	3.34	4.60
S/Totals	Proven	137,624	10.74	33.55
	Probable	1,066,484	4.34	7.37
Total		**1,204,108**	**5.08**	**10.36**

Note: Mineral Reserves are associated with an ore/waste strip ratio of 11.9/1. In this ratio, waste includes 2.1 Mt of LG mineralization between 0.3 and 1.5 g/t.

These Mineral Reserves have been generated from a resource base consisting of the Measured and Indicated Mineral Resources (Section 14.0) together with an appropriate combination of open pit resource-reserve conversion parameters, as will be described in the following sections of the report.

Location of the mining areas is shown in the following figures:

Figure 15-1: Martinetas Mining Area

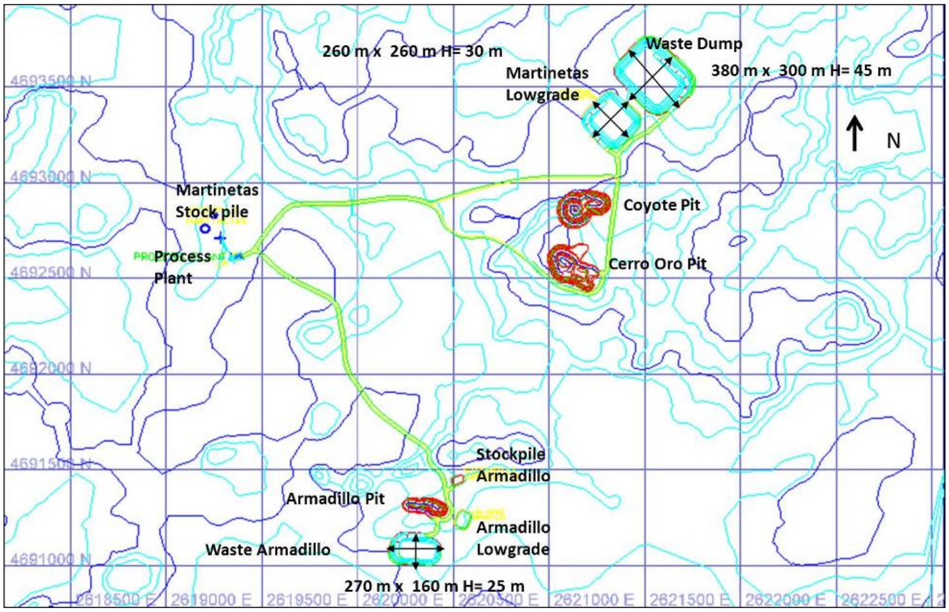
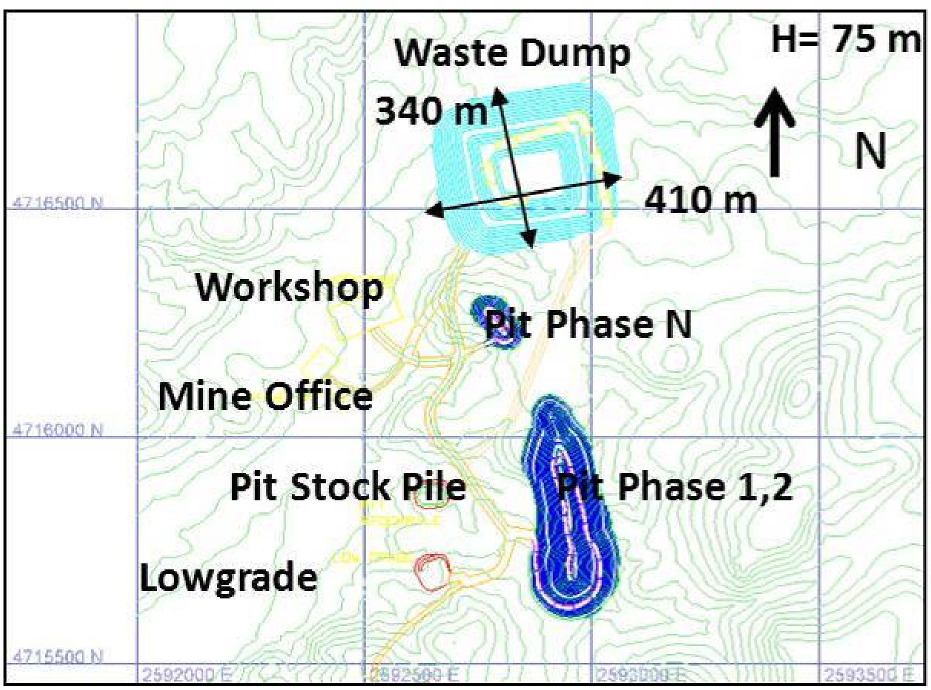


Figure 15-2: La Paloma Mining Area



15.2 Pit Optimization Parameters

The principal technical and economic parameters applied in generating the Mineral Reserves are listed in Table 15-2.

Table 15-2: Pit Optimization Parameters

Parameter	Unit	Value
Au Price	$/oz	$1,100
Ag Price	$/oz	$25
Royalties	%	Boca Mina (Mine Mouth): 3% on operating profit
		Export Tax: 5% over total revenue
Mining Costs: La Paloma	US$/t	2.80 US$/t of material (5.0 US$/t of ore will be added for hauling to Martinetas)
Martinetas	US$/t	2.80 US$/t of material
Process Costs: CIL	$/t milled	35 US$/t for all materials
G&A	$/t milled	7.10
Offsite Refining & Freight	$/oz Au	8.00
	$/oz Ag	0.50
Overall Slopes		Initial 60 m (62°), Second 50 m (52°) deeper (45°) overall

The pit optimizations incorporate metallurgical recovery formulas applied to each block in the resource model. Most of the metallurgical testwork has been carried out under MIRLP's supervision at the ALS Ammtek laboratory in Perth, Australia. The following table sets out the formulae included in the pit optimization routines.

Table 15-3: Metallurgical Recovery Parameters

Deposit		Parameters
La Paloma	Oxides	Rec Au=100*(1-(0.00072 Cu + 0.0056 Sb + 0.0305 Au + 0.0066 Ag + 0.1 S +0.015)/Au)
	Traditional	Rec Ag=100*(1-(0.0091 Cu + 0.374 Ag + 0.075)/Ag)
	Fresh	Rec Au=100*(1-(0.00072 Cu + 0.0056 Sb + 0.0305 Au + 0.0066 Ag + 0.1 S +0.015)/Au)
Martinetas & Armadillo	Oxides	Rec Au=100*(1- (0.025 x Au + 0.075)/Au). For <0.2 g/t assume a recovery of 40%.
	Traditional	Rec Ag=100*(1-(0.49 x Ag + 0.115)/Ag). For <0.3 g/t assume a recovery of 5%.
	Fresh	Rec Au=100*(1- 0.025 x Au + 0.075)/Au). For <0.2 g/t assume a recovery of 40%.

The results from the initial pit optimizations applying the parameters above are shown in Table 15-4.

Table 15-4: Don Nicolás Pit Optimization Results

Optimized Pit	Ore kt	Au g/t	Ag g/t	Waste kt	Rock kt	Waste/Ore	Au oz in Oz'000	Ag oz in Oz'000
La Paloma	385	7.63	23.05	9,604	9,998	25.0	94	285
Martinetas	648	4.47	5,07	3,370	4,018	5.2	93	106
Armadillo	86	5.06	6.31	678	764	7.9	14	17
Total	**1,118**	**5.60**	**11.35**	**13,651**	**14,770**	**12.2**	**201**	**486**

Note: Volumetrics for the Whittle optimized shells have been estimated over the original models.

15.3 CUTOFF GRADE ANALYSIS

Using the technical and economic parameters shown in above, cutoff grades (COG) were estimated for each pit using the following formula:

$$COG = \frac{C_m + C_p + G\&A}{(P_{Au} - C_s) \times R_{Au}}$$

Where:

- Cm: Mining Cost
- Cp: Process Cost
- G&A: General and Administration Cost
- PAu: Gold Price
- Cs: Selling Cost
- Rau: Gold recovery

The following royalties were also applied:

a. Boca Mina: 3% on operating profit

b. Export Tax: 5% over total revenues

The following table summarizes the cutoff grades calculated for each deposit.

Table 15-5: Don Nicolás Cutoff Grades

Pit	COG (g/t Au)
La Paloma	1.60
Coyote	1.50
Cerro Oro	1.50
Armadillo	1.50

15.4 Dilution and Ore Loss

For both the La Paloma and Armadillo deposits, dilution was estimated based on NCL experience and agreed with MIRL. The criteria applied at both these locations assume an operational over-break as a function of the open pit equipment selectivity and that over-breaking will be 0.30 m each side of the vein. The Au and Ag diluting grades were taken from the resource block model information.

For the Cerro Oro and Coyote deposits in the Martinetas area, the Multiple Indicator Kriging resource model estimated by Coffey Mining (Section 14.0) includes dilution.

No ore loss factors have been included in any of mine designs which is consistent with open pit mining protocol.

15.5 General Pit Design Parameters

The general design parameters used in the detailed pit design, including the geotechnical data described above are as follows:

a. Bench Height, Single-Bench Mining: 5 m

b. Bench Face Angle: 80º

c. Berm Width: 2.5 m

d. Total Width Allowance Final Roads: 10 m

e. Maximum Grade for Ramps: 12%

15.6 Geotechnical Parameters

For the FS, MIRL engaged Golder Associates Argentina S.A. (Golder) to provide geotechnical slope design for the Project.

A geotechnical investigation was undertaken at the Don Nicolás Project comprising and initial site reconnaissance in March 2011 and a detailed drilling, trial pitting and in-situ testing campaign between June and September 2011. Soil and rock core samples were recovered for laboratory testing. The drilling works comprised:

a. Eight oriented boreholes were drilled in Martinetas open pit sites from which five boreholes were tested for hydraulic conductivity;

b. Five oriented boreholes were drilled in Sulfuro open pit sites from which four boreholes were tested for hydraulic conductivity;

c. Two oriented boreholes were drilled in Armadillo open pit and both were tested for hydraulic conductivity; and

d. Two vertical boreholes were drilled in each processing plant site and both were tested for hydraulic conductivity. These boreholes were not oriented.

Core orientation was carried out with ACT II Reflex device.

Based on the fieldwork and laboratory testing and subsequent analyses, geotechnical domains based on lithological groupings have been defined for this study. Table15-6 summarizes the occurrence of each lithological unit at the different pits according to data gathered during the field campaign.

Table 15-6 Geotechnical Domains and Occurrences by Pit

Lithology	Martinetas Norte	Martinetas Sur	Armadillo	Sulfuro
Andesite				X
Breccia	X		X	X
Decite			X	
Fault		X	X	
Granite	X	X		
Ignimbrite	X	X	X	
Rhyolite	X		X	
Saprolite	X			X
Soil		X		
Tuff	X	X	X	X
Argillized Tuff	X		X	
Vein		X		

Based on RMR classification, most of the lithological units were classified as Fair Rock except for the Andesite (RMR=68) and Granite (RMR=63) which were classified as Good Rock and the “overburden” units (Soil and Saprolite) which are classified as Poor Rock. For those units occurring in more than one pit, it could be observed that their rock mechanical properties and therefore their quality were similar from one pit to another. Therefore, geotechnical domains can be defined per lithological unit, which means each lithological unit corresponds to a geotechnical domain. Only Saprolite and Soil material may be grouped in order to simplify the analysis and taking into account that both units have poor to very Poor Rock quality.

Table 15-7 Table 15-7presents the average parameters assumed for this feasibility assessment for each domain

Table 15-7 Geotechnical Domains Properties Adopted for this Study

Geotechnical Domain	Lithology	UCS (MPa) (Rounded)	RMR (Rounded)	Specific Weight (KN/m^3) (assumed value)
D And	Andesite	20	69	27
D Bx	Breccia	12	56	25
D Da	Dacite	52	59	27
D Ft	Fault	20	20	23
D Gr	Granite	133	65	27
D Ig	Ignimbrite	29	57	24
D Rh	Rhyolite	24	47	25
D Ov	Overburden (Soil + Saprolite)	6	28	18
D T	Tuff	17	42	23
D Wt	Argillized Tuff	15	44	20
D V	Vein	220	58	27
D And	Andesite	20	69	27

The nominal pit slope configurations for the stability analysis by Golder are presented in the following table.

Table 15-8: Overall Slope Recommendations, Golder

Zone	Overall Slope Angle
La Paloma	48°
Martinetas	53°
Armadillo	45°

15.7 PIT DESIGN AND MINING PHASES

Mineable pit phases were designed based on optimized nested pit shell guidance, gold grade, strip ratio and access. Ramps in final walls have a design width of 10 m and a gradient of 12%. A nominal minimum mining width of 30 m was used for phase design.

Physical parameters of the different pits can be summarized as follows:

a. Cerro Oro has a top elevation of 180 m above sea level (masl) and has a bottom elevation of 110 m masl.

b. Coyote has a top elevation of 180 m masl and has a bottom elevation of 85 m above sea level and is located 200 m to the south of Coyote.

c. Armadillo has a top elevation of 150 m masl and has a bottom elevation of 105 m above sea level and is located 1,200 m to the south of Cerro Oro.

d. The La Paloma pit has been subdivided into three mining phases: Sur 1 phase has a top elevation of 165 masl and a bottom elevation of 120 masl.

Sur 2 phase has the same top elevation as Sur 1 and has a bottom elevation of 45 masl. La Paloma Norte phase starts in the same bench and finishes in the 120 m bench.

Final pit design for La Paloma is presented in Figures 15-3 to 15-5.

Figure 15-3: La Paloma Final Pit Design

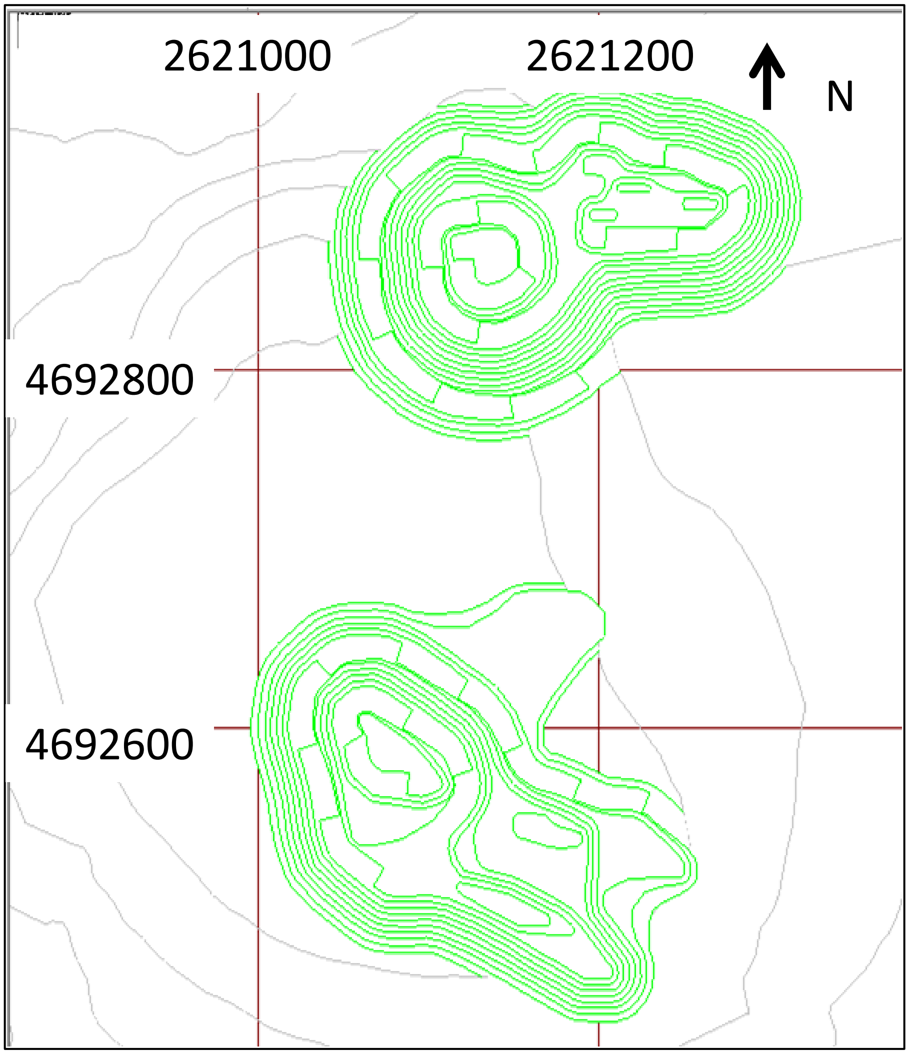


Figure 15-4: Martinetas Final Pit

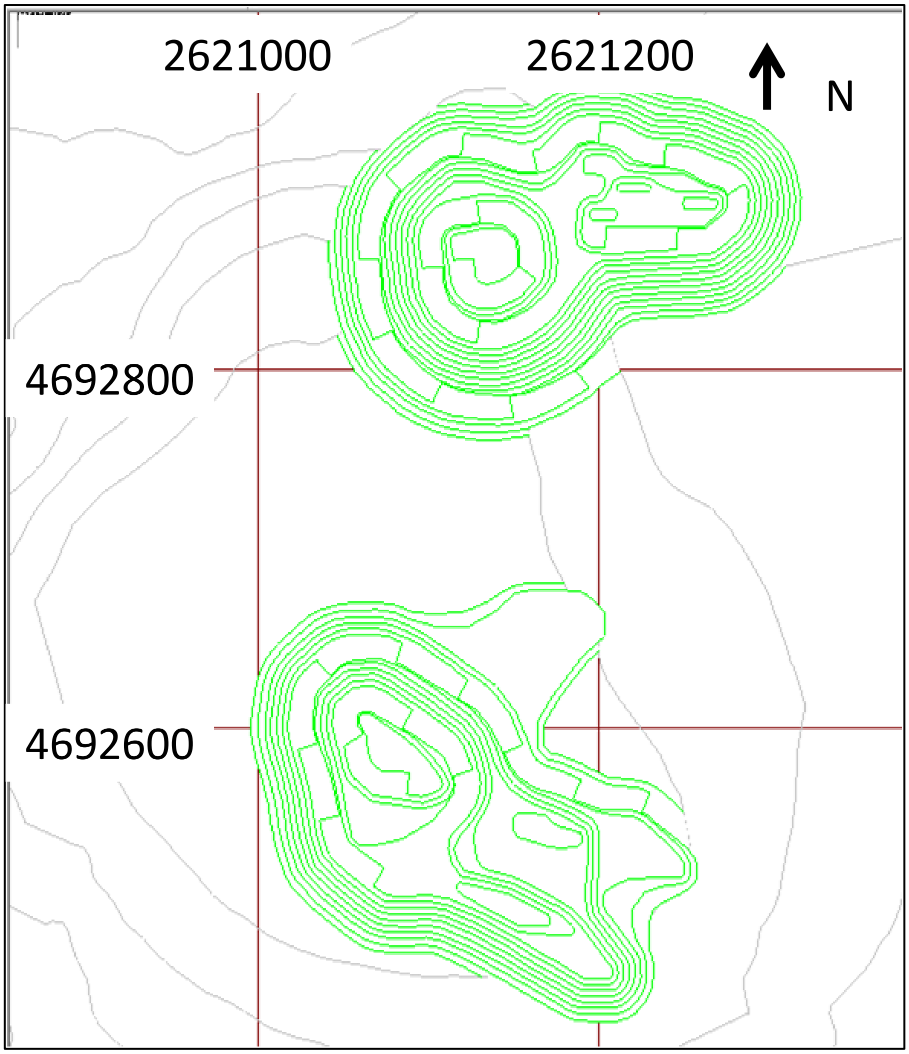
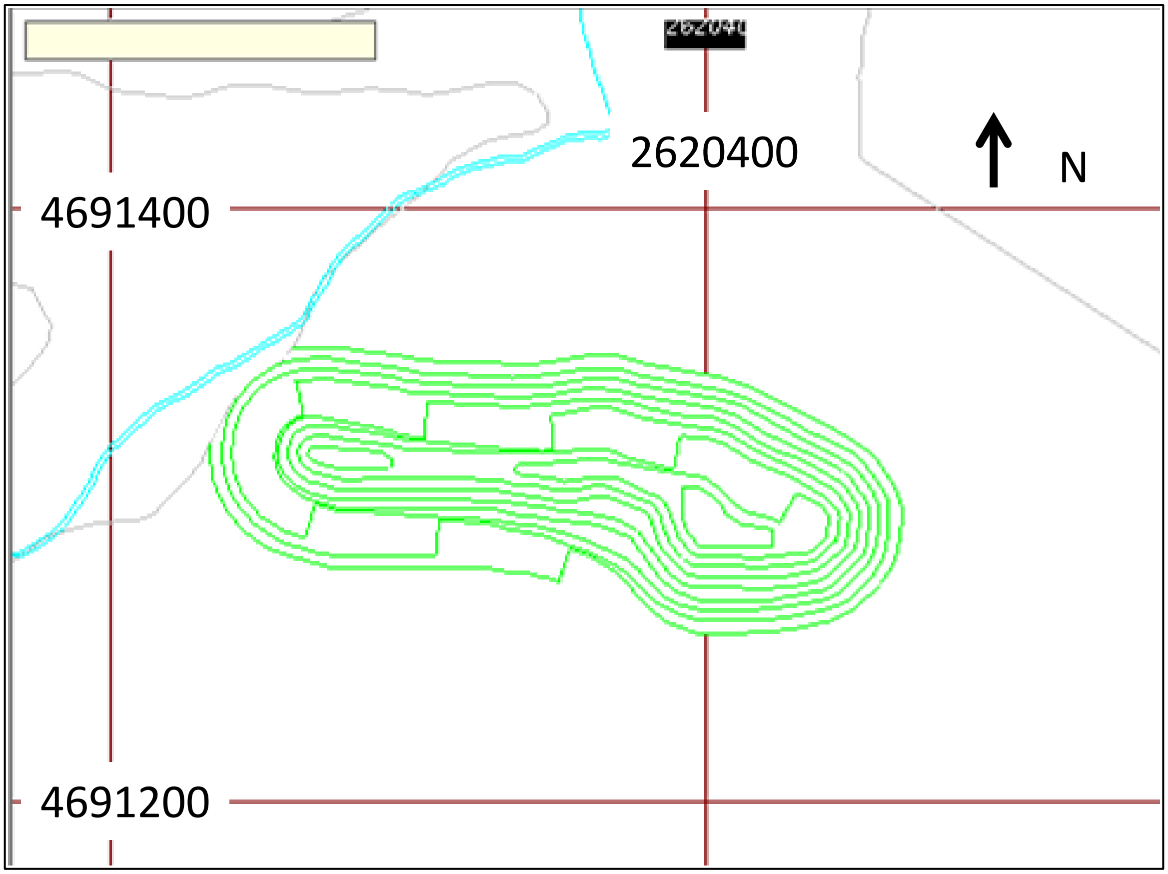


Figure 15-5: Armadillo Pit



The phase's volumetrics and development index for each one is as follows.

Table 15-9: Phase Mining Sequence

Phase	Ore			Waste	Contained Equiv.Gold	Stripping Ratio	Total Tonnage/ k oz Au eq)
	Tonnage (kt)	Au (g/t)	Ag (g/t)	Tonnage (kt)	Oz´000		
La Paloma Sur 1	62	4,99	12,18	723	10,54	11,6	74,5
La Paloma Sur 2	301	7,68	24,63	6.408	79,71	21,3	84,2
La Paloma Norte	32	4,17	8,40	382	4,49	11,9	92,3
Coyote	422	5,04	5,61	2.176	70,11	5,2	37,1
Cerro Oro	259	3,10	4,18	1.607	26,58	6,2	70,2
Armadillo	128	3,34	4,60	702	14,19	5,5	58,5

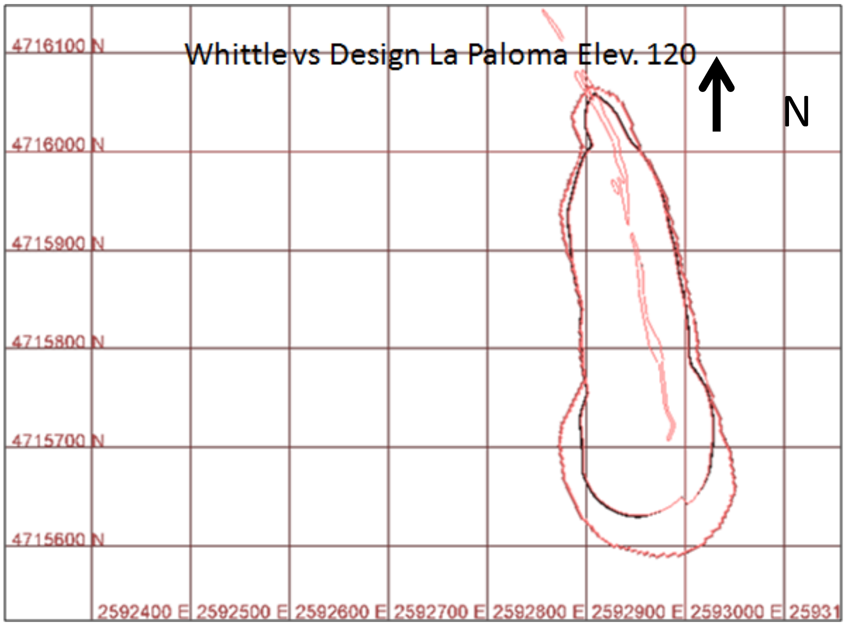
Note: COG for La Paloma was 1.60 g/t Au and for the other sectors was 1.50 g/t Au.

The optimized pit shells and ultimate pit designs volumetrics are compared in Table 15-9. The two shapes are shown graphically in Figure 15-6. The ultimate pit designs although slightly larger than the pit shells fit reasonably well to the optimized pit shell. Total material has decreased by 10.6%, strongly influenced by the lower material moved at La Paloma. The additional waste moved at Martinetas was accumulated in the smoothing of the pit bottoms and the inclusion of the ramps. Contained metal are very similar. These variations are considered to be within acceptable limits for that kind of veins.

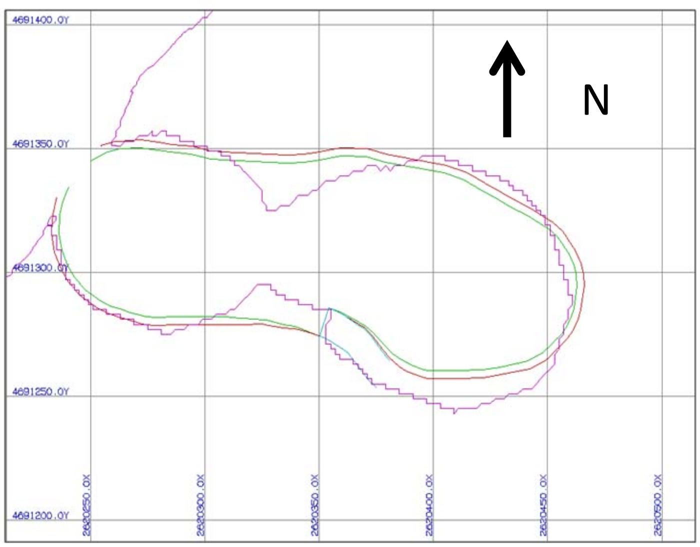
Table 15-10: Material Contained in Optimized Pit Shell and Ultimate Pit Design

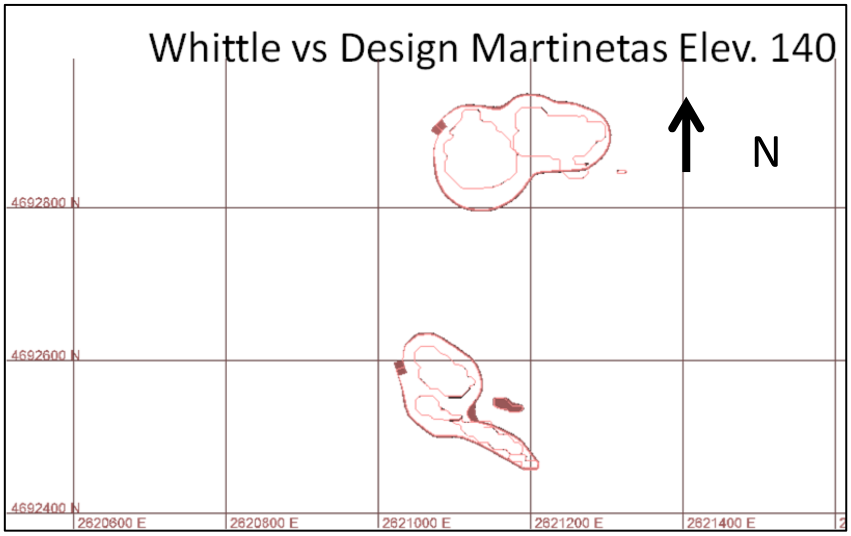
Item	La Paloma		
	Ultimate Shell	**Mine Design**	**Difference (%)**
Ore Processed (kt)	385	395	2,7%
Gold Grade (g/t)	7,63	6,97	-8,6%
Contained Metal (k troy oz)	94,28	88,52	-6,1%
Waste Mined (kt)	9.604	7.513	-21,8%
Total Mined (kt)	9.988	7.908	-20,8%
Stripping Ratio	25,0	19,0	-23,8%
Item	**Martinetas**		
	Ultimate Shell	**Mine Design**	**Difference (%)**
Ore Processed (kt)	648	681	5,1%
Gold Grade (g/t)	4,47	4,30	-3,8%
Contained Metal (k troy oz)	93,09	94,15	1,1%
Waste Mined (kt)	3.370	3.783	12,3%
Total Mined (kt)	4.018	4.464	11,1%
Stripping Ratio	5,2	5,6	6,8%
Item	**Armadillo**		
	Ultimate Shell	**Mine Design**	**Difference (%)**
Ore Processed (kt)	86	128	48,7%
Gold Grade (g/t)	5,064	3,34	-34,0%
Contained Metal (k troy oz)	14,02	13,75	-2,0%
Waste Mined (kt)	678	702	3,5%
Total Mined (kt)	764	830	8,6%
Stripping Ratio	7,9	5,5	-30,4%

Figure 15-6: Plots of Ultimate Pit Shells Compared to Mine Designs



Whittle vs Design Armadillo Elev. 145





15.8 Mineral Reserves

NCL used the resources block models developed by Coffey for mine planning. The in-pit mineral reserve was estimated after applying the mining selectivity, mining dilution and ore loss adjustments. This reserve is the basis for the mine design and production schedule. Table 15-11 summarizes the in-pit resource by classification.

Table 15-11: Resource by Classification inside the Ultimate Design Pit*

Deposit/Category		Tonnes	Au g/t	Ag g/t
La Paloma	Proven	137,624	10.74	33.55
	Portable	257,406	4.96	14.84
	Total	395,030	6.97	21.36
Martinetas	Proven	-	-	-
	Portable	680,803	4.30	5.07
	Total	680,803	4.30	5.07
Armadillo	Proven	-	-	-
	Portable	128,275	3.34	4.60
	Total	128,275	3.34	4.60
S/Totals	Proven	137,624	10.74	33.55
	Probable	1,066,484	4.34	7.37
Total		**1,204,108**	**5.08**	**10.36**

* *Includes 178,600 t of dilution material classified to probable from La Paloma and Armadillo.*

Ore and waste determination was based on the estimation of a net economic benefit that considered metal prices, recoveries, and operating and selling costs. Cutoff grade estimates were based on a zero profit value using the economic and technical parameters included in Section 3.

Cutoff grades estimates were 1.50 g/t Au for Martinetas and Armadillo and 1.60 g/t Au for La Paloma. According to this, all material included below the cutoff grade in the Selective Mining Unit (SMU) was considered as dilution.

As was explained before, for Martinetas the Multiple Indicator Kriging method was considered more appropriate to estimate resources. This method includes dilution for an assumed mining scenario and SMU. The SMU block in this case was 5 mE x 2 mN x 2.5 mRL.

Dilution included in each phase is the following.

Table 15-12: Dilution per Area

Phase	Total Reserves			Dilution			
	Tonnage (t)	Au (g/t)	Ag (g/t)	Tonnage (t)	Au (g/t)	Ag (g/t)	%
La Paloma Ph1	62.176	4,99	12,18	20.505	0,58	3,05	33%
La Paloma Ph2	300.886	7,68	24,63	96.112	0,70	4,44	32%
La Paloma PhN	31.968	4,17	8,40	10.410	1,16	4,16	33%
Cerro Oro	258.645	3,10	4,18	-	0,00	0,00	*
Coyote	422.158	5,04	5,61	-	0,00	0,00	*
Armadillo	128.275	3,34	4,60	51.673	0,79	2,08	40%
Total	**1.204.108**	**5,08**	**10,36**	**178.700**	**0,74**	**3,58**	

*Dilution for Cerro Oro and Coyote is included in the MIK model

16.0 MINING METHODS

16.1 PRODUCTION DESIGN CRITERIA

Mine scheduling for the Don Nicolás FS was undertaken by NCL Ltda. (NCL) based on resource models for each area with a mine schedule developed to meet the following criteria:

a. Nominal plant capacity of 1,000 t/d (350,000 t/a).

b. Three hundred and sixty (360) operating d/y.

c. The production plan developed on a monthly basis using the whole bench as minimum mining unit.

d. Waste production will meet requirements for construction materials for roads, platforms, and the tailings dam, mainly during pre-stripping.

e. Maximum vertical advance per phase per month is one 5-m bench.

f. LG material (0.3 to 1.5 g/t Au) stored in separate stockpile areas at each pit.

g. Paloma plant ore production (>1.6 g/t Au) stockpiled close to the pit for subsequent transport to the Martinetas process plant some 50 km distant.

h. Plant ore production from Martinetas and Armadillo feeds directly to the crusher or stockpile areas adjacent to the crushing facility.

As explained in an earlier section, for Martinetas the Multiple Indicator Kriging method was considered more appropriate for estimating resources. This method includes dilution for an assumed mining scenario and SMU. The SMU block in this case was 2 mE x2 mN x 2.5 mRL.

At Paloma the SMU size is 2.5 mE x 2 mN x 5 mRL andin Armadillo the SMU size is 2.0 mE x 2.0 mN x 2.5 mRL.

16.2 MINE PLAN AND PRODUCTION SCHEDULE

The mine production plan was developed by NCL. The process is described below:

a. With the objective of achieving the best possible production profile, the mine plan is run for the life of the mine at the phase-bench level in a yearly basis. At this stage, the schedule is based on tonnage–grade curves for each phase and bench. Ore and waste are assumed to be equally distributed within the bench. From this exercise, the number of working phases per period, their geometries, and the requirements of loading units are assessed.

b. Preproduction has been defined as a 4-month period prior to the completion of mill commissioning.

c. All scheduling was carried out on 5-m benches. The schedule incorporates long-term ore stockpiles. LG ore stockpiles were established in the same area as waste dumps.

d. Different mine plans were prepared varying total rock movement per year, grades feeding to the process plant and ore sources. In order to verify the economical merit of each plan, a simplified cash flows statement spreadsheet was used using the economic and technical parameters described in Section 3.

Based on the preceding information, the resulting production plan is summarized by year in Table 16-1:

Table 16-1: Summary Mine Production Plan by Year

	Ore >1.5 g/t Au			LG 0.3-1.5 g/t Au			Waste Moved	Total Material Moved
	kt	Au g/t	Ag g/t	kt	Au g/t	Ag g/t	kt	**kt**
Period								
1 (PP)	42	4,870	5,931	68	0,735	1,886	713	**823**
2	303	5,089	7,464	524	0,738	2,965	3,850	**4,677**
3	372	6,254	16,653	304	0,812	4,031	4,511	**5,187**
4	348	4,501	8,782	1,132	0,683	2,862	2,279	**3,759**
5	140	3,406	5,176	120	0,643	4,507	643	**903**
Total	**1,204**	**5,076**	**10,362**	**2,149**	**0,714**	**3,114**	**11,997**	**15,350**

Note: PP = Pre-Production period of 4 months prior to completion of plant commissioning.

16.3 Pit Phase

Highlights of the pit phases and mining schedule for both ore and waste are summarized below:

a. Pre-production: total pre-stripping of 823 kt starting with Paloma Sur Phase 1 and Coyote.

b. Paloma Sur Phase 1 is finished during the first year of production; rock movement at Paloma Phase 2 begins in this period.

c. During Year 03 of production, rock movement begins in the Paloma North, Armadillo and Cerro Oro phases.

d. Maximum production per phase occurs in Paloma Sur Phase 2 with 11.8 kt/d moved during Year 02 of production.

e. The initial phases are generally mined at a lower rate with the intent of keeping as many alternative areas open as possible in each phase.

f. In the production years, there are two to three active phases in any period.

Table 16-2 summarizes the active phases per year showing total material movements in thousands of tonnes:

Table 16-2: Total Material Movement by Phase (kt)

Period	Martinetas			La Paloma			Total
	Cerro Oro	Coyote	Armadillo	South 1	South 2	North	
1 (PP)		419		405			**823**
2		2,089		389	2,200		**4,677**
3		943			4,245		**5,187**
4	2,947		89		308	416	**3,759**
5	108		794			2	**903**
Total	**3,055**	**3450**	**882**	**793**	**6,752**	**417**	**15,350**

Note: It can be observed that at least two phases are simultaneously active in every year.

Table 16-3 summarizes the phased schedule for mining ore supplied to the process plant on a yearly basis:

Table 16-3: Total Yearly Ore Mining by Phase (kt)

Period	Martinetas			La Paloma			Total (kt)
	Cerro Oro	Coyote	Armadillo	South 1	South 2	North	
1 (PP)		32		10			**42**
2		240		52	11		**303**
3		150			221		**372**
4	239		10		69	30	**348**
5	20		118			2	**140**
Total	**259**	**422**	**128**	**62**	**301**	**32**	**1,204**

Table 16-4 summarizes the pre-production phase of the mine operation:

Table 16-4: Pre-Production Phase (4 months)

Month	Coyote			La Paloma Phase Sur 1			Total		
	Ore kt	Waste kt	Total kt	Ore kt	Waste kt	Total kt	Ore kt	Waste kt	Total kt
9	0.001	0.40	0.4	0.3	25.6	26.1	**0.3**	**26.0**	**26.5**
10	1.7	44.3	48.3	1.3	84.4	86.1	**3.0**	**128.7**	**134.4**
11	9.8	106.2	134.7	2.7	136.2	139.9	**12.6**	**242.3**	**274.6**
12	20.1	171.4	235.3	6.0	144.8	152.4	**26.1**	**316.2**	**387.7**
Total	**31.7**	**322.2**	**418.7**	**10.2**	**391.0**	**404.5**	**41.9**	**713.3**	**823.2**

16.4 Mine Equipment Requirements

16.4.1 General Considerations

The FS has assumed that an owner's fleet of equipment will be used in the Don Nicolás mine operations.

Supplier names and equipment types are provided for orientation purposes only. The selection of example machines is based on a preliminary technical and economic assessment of feasibility level estimates provided by suppliers and their associated dealers. Reference to machine types does not reflect a final recommendation of equipment supply. Further analysis will be carried out at the engineering and procurement stages of the Project.

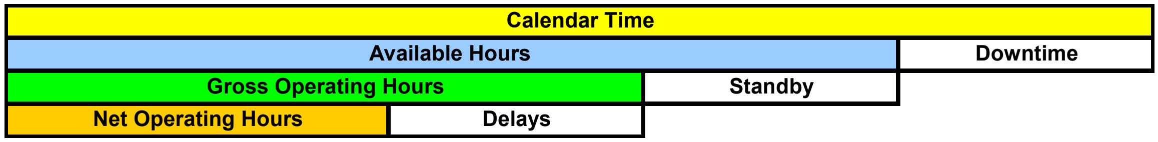
The Don Nicolás mine design specifies 5-m benches and has adequate phase geometry to achieve the required productions. The design calls for annual mill throughput of 350,000 t of ore. Mining will advance simultaneously in two different pits with two to three pit phases generally active at any time. The vertical advance rate of the pits will be relatively low, with a maximum of eleven 5-m benches per phase per year.

The Martinetas deposits (Cerro Oro, Coyote, and Armadillo) lie close to the main process plant while the La Paloma deposit (principally the Sulfuro vein) is located some 50 km to the northwest. Mine equipment units have been estimated on the basis of a common operating fleet shared between the different operating areas. The drilling and loading equipment will therefore combine high productivity, mobility, and flexibility at relatively low cost, and will be sized for efficient mine selectivity.

Time cycles, work regime structure, and standard standby and delay parameters have been applied in estimating the mine equipment requirements.

The time model used for calculating equipment hours is shown as follows:

Figure 16-1: Time Model Structure



16.4.2 MINE EQUIPMENT REQUIREMENTS

The required yearly equipment fleet is shown in Table 16-5.

Table 16-5: Mine Equipment Requirements by Year

Area	Unit	Y01	Y02	Y03	Y04	Y05
Load Fleet	FEL CAT 966	**1**	**2**	**2**	**2**	**1**
	Excavator CAT 336DL	**1**	**2**	**2**	**2**	**1**
Hauling Fleet	Haul Truck M.Benz Actros 3344	**4**	**8**	**10**	**7**	**3**
Drill Fleet	Diesel Drill Sandvik DX800	**1**	**2**	**2**	**2**	**1**
Auxiliary						
	Bulldozer D6	-	1	1	1	1
	Bulldozer D8	1	1	1	1	1
	Grader 140 M	1	1	1	1	1
	Water Truck	1	1	1	1	1
Auxiliary Fleet		**3**	**4**	**4**	**4**	**4**
Support						
	Backhoe/Hammer	1	1	1	1	1
	Fuel Truck	1	1	1	1	1
	Support Truck	1	1	1	1	1
	Lowboy Truck	1	1	1	1	1
	Lightning Plant	3	6	6	6	3
Support Fleet		**7**	**10**	**10**	**10**	**7**

The equipment requirements have been estimated using the following material properties.

Table 16-6: Material Parameters for Equipment Sizing

		Ore La Paloma	Ore Martinetas	Waste La Paloma	Waste Martinetas
Dry density (in-situ)	t/m^3	2.43	2.35	2.32	2.29
Swell	%	40	40	40	40
Moisture	%	2.0	2.0	2.0	2.0
Dry density (swell)	t/m^3	1.74	1.68	1.66	1.64
Wet density (in-situ)	t/m^3	2.48	2.40	2.37	2.34
Wet density (swell)	t/m^3	1.77	1.71	1.69	1.67

16.4.3 MINE LABOUR

Two areas will report to the mine superintendent:

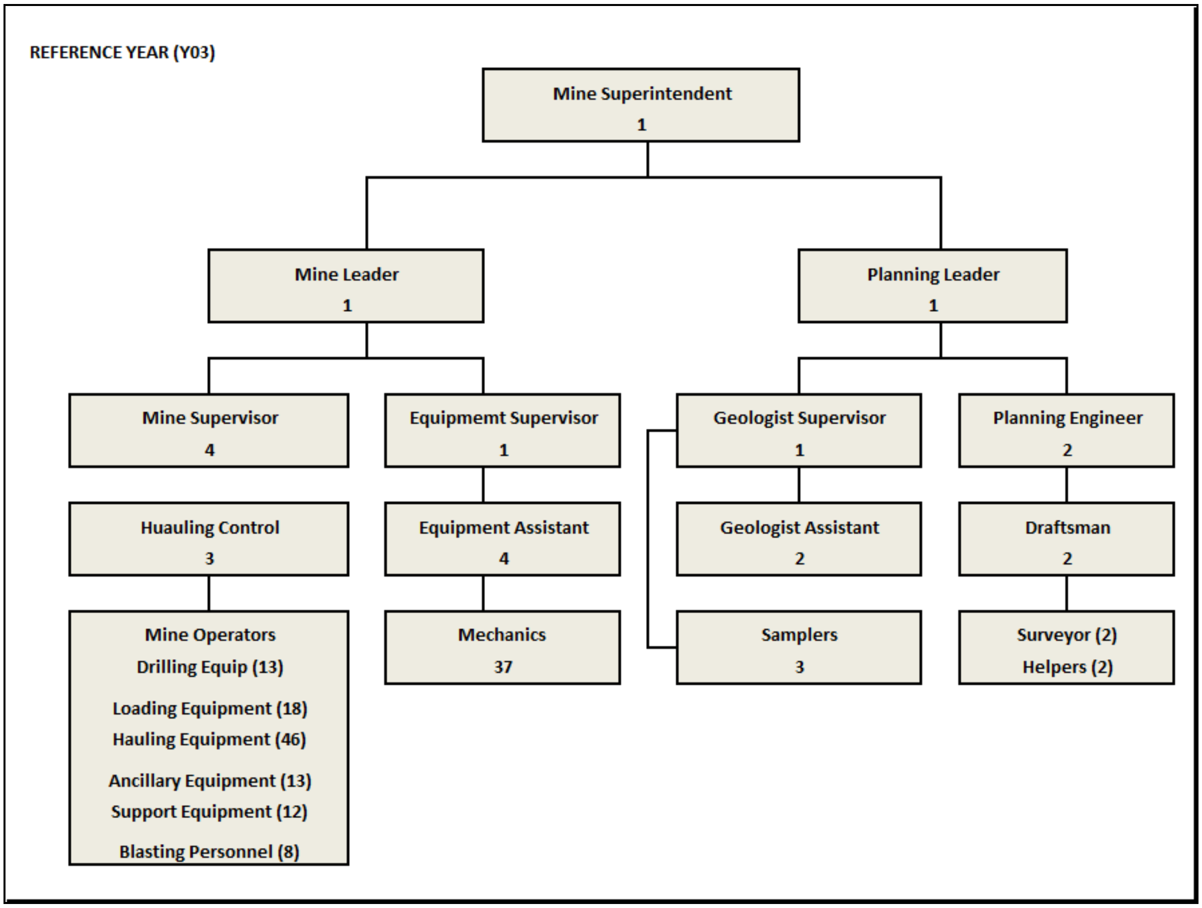
a. Mine Operations.
b. Technical Services.

Each of these areas will have three different personnel levels:

a. Seniors and Supervisors.
b. Technicians.
c. Operators.

The organizational structure is shown in Figure 16-2. Boxes include number of people for Year 03.

Figure 16-2: Mine Organization Chart



Under the direction of the mine operations superintendent, the mine operations department will be responsible for the open pit operation. This will include drilling, blasting, loading, and hauling of ore and waste, dump and haul road construction/maintenance, pioneering work, and mine dewatering. The number of operators is based on the annual equipment requirements and the crew's schedule, ensuring that sufficient operators are available on each crew to operate the equipment.

The mine maintenance area will report to the mine leader. Maintenance crews will work the same shift schedule as mine operations crews.

The planning area will provide daily, weekly, and monthly plans for operations working on full rotation. Under the direction of the chief geologist, the geology area will be responsible for updating the resource model, calculating ore reserves, and grade control.

Number of people per period is presented in Table 16-7.

Table 16-7: Total Mine Labour per Period

Labour		Y01	Y02	Y03	Y04	Y05
Loading		**9**	**16**	**18**	**13**	**8**
• FEL	Operators	4	7	10	7	3
• Excavator	Operators	5	9	8	6	5
Hauling	Operators	**16**	**34**	**46**	**31**	**13**
Drilling	Operators	**7**	**11**	**13**	**11**	**7**
Blasting	Operators	**6**	**8**	**8**	**8**	**6**
Auxiliary		**9**	**13**	**13**	**13**	**11**
• Bulldozer 1	Operators	-	4	4	4	2
• Bulldozer 2	Operators	3	3	3	3	3
• Motorgrader	Operators	3	3	3	3	3
• Water Truck	Operators	3	3	3	3	3
Support		**12**	**12**	**12**	**12**	**12**
• Backhoe/Hammer	Operators	3	3	3	3	3
• Fuel Truck	Operators	3	3	3	3	3
• Support Truck	Operators	3	3	3	3	3
• Lowboy Truck	Operators	3	3	3	3	3
Mechanics	#	**20**	**33**	**37**	**31**	**20**
Mine Adm	#	19	29	29	29	20
• Mine Management	#	5	9	9	9	6
• Mine Maintenance	#	3	5	5	5	3
• Technical Services	#	11	15	15	15	11
Total	#	**99**	**156**	**176**	**148**	**97**

A maximum of 176 people will be required on Year 03 for the Don Nicolás operations.

17.0 RECOVERY METHODS

17.1 INTRODUCTION

The Don Nicolás processing facilities will be designed to process a nominal 350,000 t/a, or 972 t/d, of gold-silver bearing material from an open pit operation. The concentrator will be designed to produce gold-silver doré product.

17.2 SUMMARY

The unit processes selected were based on the results of metallurgical testing performed at Ammtec, Australia, along with project-related parameters provided by MIRL. The metallurgical processing procedures selected for the design will produce gold-silver doré.

The process flowsheet follows conventional crushing and ball mill grinding and cyclone classification. The gravity concentration circuit in the grinding circuit includes the recovery of coarse and liberated gold using a centrifugal concentrator followed by the tabling of the gravity-gold product to up-grade the concentrate prior to smelting. The ball mill cyclone overflow will be treated in an 8-stage CIL circuit to recover gold from the feed material using activated carbon. Loaded carbon will be transferred from the head CIL tank to the elution circuit on a daily basis, while regenerated and/or fresh carbon will be brought from the carbon plant for adding to the CIL circuit. The loaded carbon will initially be acid-washed to remove calcium and other impurities, followed by the elution, or stripping, process. The gold will be recovered by electrowinning. The eluted carbon will be regenerated in a kiln prior to screening for the removal of carbon fines. The regenerated carbon will subsequently be returned to the adsorption circuit. The CIL tailings will be discharged to the tailings thickener. Tailings thickener underflow will be pumped to the cyanide detoxification tank where cyanide levels will be chemically reduced to acceptable environmental levels prior to disposal to the tailings storage pond. This thickening stage will allow for greater control of water management and enable some of the cyanide present in the water to be re-circulated for re-use in the plant.

Process water will be recycled from the tailings thickener overflow, and this will be supplemented with process water recovered from the tailings dam. Fresh water will be used for gland service, reagent preparation and gravity circuit fluidisation, as well as for water make-up purposes, as required.

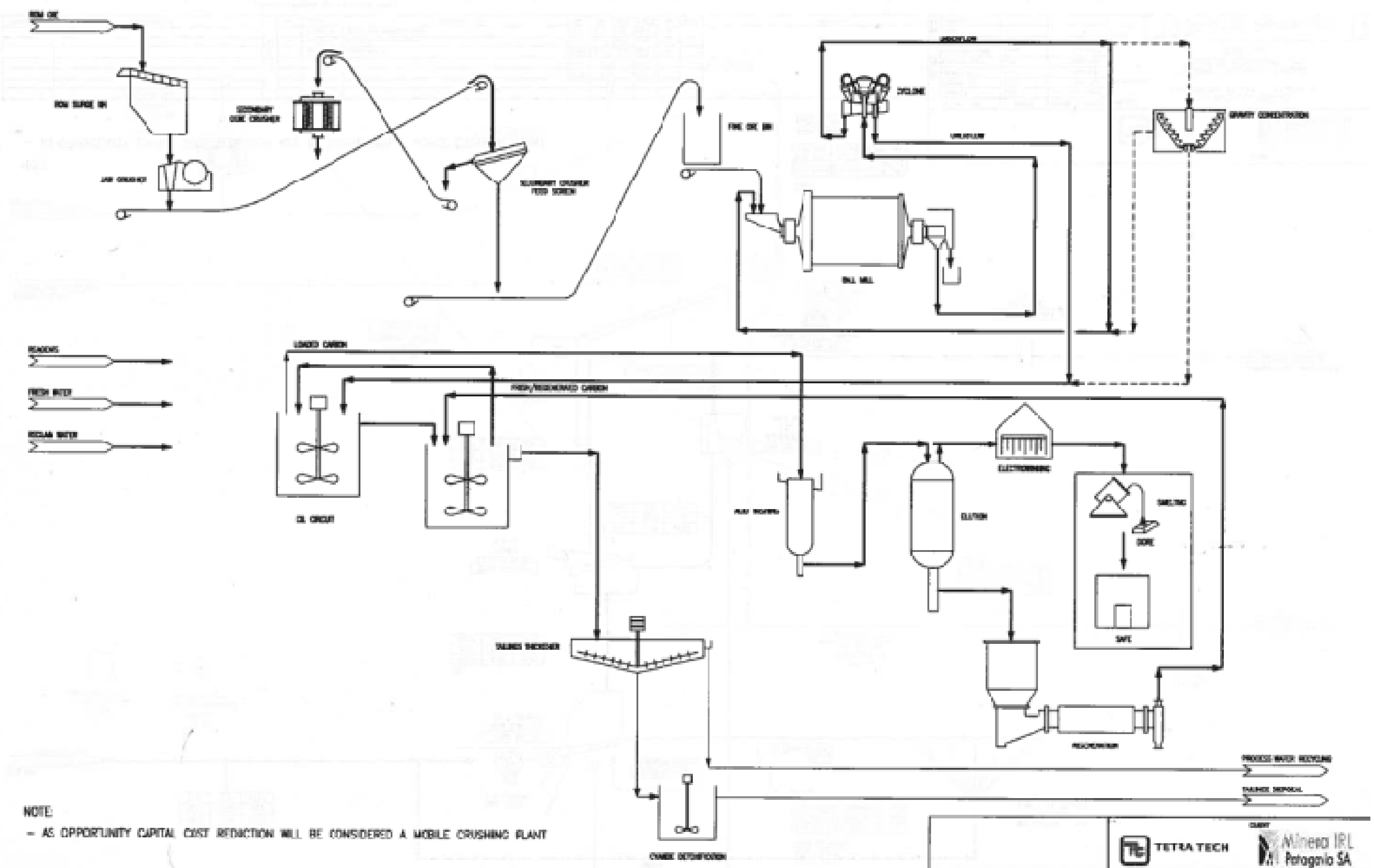
The Don Nicolás processing facility will consist of the following unit operations:

a. Two-stage crushing and screening.
b. Fine Ore Bin Storage and Reclaim for Grinding.
c. Ball Mill Grinding and Classification.

d. Gravity Recovery Circuit.
e. CIL Leaching.
f. Carbon Handling and Treatment.
g. Electrowinning and Smelting (gold refining).
h. Tailings Thickening.
i. Cyanide Detoxification.
j. Thickened Tailings Deposition.
k. Process Water Reclamation.

The simplified flowsheet is shown in Figure 17-1.

Figure 17-1 Simplified Process Flowsheet



17.3 PLANT DESIGN

17.3.1 MAJOR DESIGN CRITERIA

The process plant has been designed to treat gold-bearing material at the rate of 972 t/d, equivalent to 350,000 t/a. The major criteria used in the design are outlined in Table 17-1.

Table 17-1 Major Design Criteria

Criteria	Unit	Value
Operating Year	d	360
Crushing Circuit Utilization	%	60
Grinding, CIL and Carbon Circuits Utilization	%	92
Crushing Circuit Throughput Rate	t/h	67.5
Grinding and Leaching Process Rate	t/h	44.0
Ball Mill Feed Size, 80% Passing	µm	6,500
Ball Mill Product Size, 80% Passing	µm	75
Ball Mill Circulating Load	%	300
Bond Ball Mill Work Index, design	kWh/t	17.6
Bond Abrasion Index, design	g	0.50
Specific Gravity Ore		2.70
Moisture Content Ore	%	2.0
Leach Circuit Retention Time	h	36
Head Grade, Design	Au, g/t	6.27
Head Grade, Design	Ag, g/t	16.7
Anticipated Recovery, Design	Au, %	94.4
Anticipated Recovery, Design	Ag, %	48.5

The design parameters selected are based on testwork results obtained from Ammtec, Australia, from a number of metallurgical test programs performed over the period from 2007 to 2011.

The grinding mill was sized based on the Bond Work Index data obtained from the testwork. The leach circuit was sized based on the optimum leach retention times as determined during the laboratory leaching tests. Typical plant design parameters have been used in the design of the leach and carbon circuit.

17.3.2 OPERATING SCHEDULE AND AVAILABILITY

The crushing and processing plants was designed to operate on the basis of 24 hr/d, for 360 d/a.

The crusher utilization will be 60% and the ball mill grinding, CIL, and carbon circuit utilization will be 92%. These utilizations will allow for a potential increase in crushing rate, and will allow sufficient downtime for the scheduled and unscheduled maintenance of the crushing and process plant equipment.

17.4 Process Plant Description

17.4.1 Crushing Circuit

The crushing circuit will reduce the mined material size from a nominal top size of 500 mm to a product size P_{80} of 6.5 mm in preparation for the grinding process. The crushing circuit facility will contain the following main items of equipment:

a. Stationary Grizzly.
b. ROM Surge Bin.
c. Vibrating feeder to the jaw crusher.
d. Jaw Crusher, C98 or equivalent.
e. Conveyor Belts.
f. Belt Magnet and Metal Detector.
g. Sizing Screens.
h. Cone Crusher, HP200 or equivalent.
i. Belt Scale.
j. Fine Ore Bin.

Haulage trucks with a nominal capacity of 28 t will bring run-of-mine [ROM] material to the dry crushing plant. The material will be dumped directly from the trucks for crushing, although provision has been made for material to be dumped onto a temporary emergency ROM stockpile in the event of unscheduled crusher plant stoppages or breakdowns. The ROM surge stockpile will also serve as a blending facility to mix the material coming from different sources prior to feeding the plant.

The trucks will dump the ROM material onto a stationary grizzly. This grizzly will prevent oversize rocks from entering the ROM surge bin. The surge bin will have a nominal capacity of twice a truck load capacity. The surge bin will be equipped with a vibrating feeder which will feed the ROM material to the primary jaw crusher. The jaw crusher will reduce the feed size from minus 500 mm to a size less than 100 mm with a closed side setting of 70 mm. The crushed material will be deposited onto the conveyor belts together with the vibrating feeder undersize material to the sizing screen. A belt magnet will be installed to remove tramp iron, followed by a metal detector which will activate an alarm to stop the belt in the event that metal by-passes the detector. The crusher plant utilization will be 60% and the design throughput will be 68 t/h.

The sizing screen will receive the crushed material from the jaw crusher and the grizzly feeder fines, or undersize. The sizing screen will be a double-deck vibrating screen with a final product size P_{80} of 6.5 mm. The screen oversize will be conveyed to a cone crusher which will have a closed side setting of 15 mm. The crushed product material from the cone crusher will be returned by conveyor belt to the sizing screen feed conveyor. The screen undersize minus 7 mm material will be discharged onto a conveyor, which will transport the fine crushed material into the fine ore bin, or mill feed bin. The fine ore bin will have a live capacity of 1,000 t, equivalent to a full day's operation

17.4.2 Grinding Circuit Operation

The grinding circuit will reduce the size of the crushed material to a final product size with a product P_{80} of 75 µm, suitable for the subsequent gold recovery by gravity concentration and cyanide leaching processes. The grinding process will be a single-stage operation with the ball mill in closed circuit with classifying cyclones. The grinding circuit will have the following main items of equipment:

a. Vibratory Feeders.
b. Conveyor Belt.
c. Conveyor Belt Weigh Scale.
d. Ball Mill, 3,960 mm diameter x 5,780 mm long.
e. Mill Discharge Pumpbox.
f. Cyclone Feed Slurry Pumps.
g. Classification Cyclone Cluster.
h. Vibrating Trash Screen.

The material in the mill feed bin will be drawn from the mill feed bin under controlled feed rate conditions using the vibrating feeders. These feeders will discharge the material onto a conveyor belt feeding the ball mill. A belt scale will control the feed to the ball mill. The mill will be fed at the rate of 44 dry t/h new feed. The cyclone underflow and the gravity circuit tailings will also constitute part of the feed to the mill. Process water will be added as required to maintain the slurry density of the ball mill at 72% solids. The ball mill will operate at a speed which will be 75% of critical.

The ball mill will have a classification circuit consisting of a mill discharge pumpbox, cyclone feed pumps and a classifying cyclone cluster. The discharge from the ball mill will be directed into a mill discharge pumpbox where dilution water will be added as required to adjust the slurry density for cyclone classification. The slurry in the mill discharge pumpbox will be pumped to a cyclone cluster for classification. The cut size for the cyclones will be at a particle size of P_{80} of 75 µm, and the circulating load will be 300%. The cyclone underflow will be returned to the ball mill together with feed material for further grinding. A 30% split of the cyclone underflow stream will be directed to the gravity concentration circuit. The mill feed rate of 44 t/h of new feed will constitute the feed to the carbon-in-leach (CIL) circuit.

The cyclone overflow from the classification circuit will be directed to a vibrating trash screen. Process water will be used to spray the deck of the screen to wash off any oversize tramp material or trash. This will be collected in a tote which will be emptied as required.

The trash screen underflow will be discharged by gravity flow into the CIL feed distributor box ahead of the leaching and adsorption process. The pulp density of the screen underflow slurry will be approximately 44% solids.

Provision will be made for the addition of lime to the ball mill for the adjustment of the pH of the slurry in the grinding circuit prior to the leaching process.

Grinding media will be added to the mill in order to maintain the grinding efficiency and power. Steel balls will be added periodically using a ball charging kibble.

A weightometer located on the mill feed conveyor will control the feeding rate of the mill feed bin reclaim feeders to the mill so as to maintain the required grinding throughput rate.

17.4.3 Gravity Concentration and Concentrate Tabling Operation

The gravity concentration circuit will produce a concentrate containing coarse gold from the grinding circuit, and will treat the gravity concentrate on a shaking table to upgrade the concentrate suitable for its smelting at the refinery.

The main items of equipment in this circuit will be the following:

a. Feed Preparation Sizing Screen.
b. Centrifugal Gravity Concentrator.
c. Concentrate Holding Tank.
d. Gemeni Shaking Table.

A nominal 30% portion of the cyclone underflow in the grinding circuit will be directed to the gravity circuit as gravity circuit feed. The gravity circuit feed will initially be screened over a vibrating screen in order to remove oversize and grit particles which are greater than 2 mm in size. The screen oversize material will be returned to the grinding circuit for further grinding.

The screen undersize will be the feed to the centrifugal concentrator. The concentrator will operate continuously on a batch basis and will be flushed twice every hour to remove the concentrate collected in the unit. The concentrator flush will be deposited into the concentrate holding tank, which will collect all the concentrate from the centrifugal concentrator over a whole day of production. Gravity tailings will be discharged from the concentrator and returned to the grinding circuit.

The gravity concentrate will be stored until a sufficient amount has been collected for the tabling process. Tabling of the gravity concentrate will normally be conducted as a batch process on a daily basis. When tabling is complete, the table tailings will be returned to the ball mill circuit for further treatment. The table concentrate will be collected, dried and transferred to the furnace for smelting.

17.4.4 CIL Circuit – Carbon Circuit

The gold and silver will be leached with cyanide and adsorbed onto activated carbon in this section of the processing circuit. Loaded carbon will be recovered periodically, nominally on a daily basis, for eluting/stripping to recover the gold and silver, and reactivated carbon will be added to replenish the eluted carbon in the circuit. The main items of equipment will be the following:

a. CIL Feed Distributor.
b. CIL Tanks.
c. Carbon Transfer Air-Lifts.
d. Interstage Screens.
e. Loaded Carbon Screen
f. Carbon Safety Screen.

The slurry from the grinding circuit will be fed into the first of eight CIL tanks. The pH will be controlled by the addition of lime which will be added as required. Cyanide solution will be added to maintain the necessary concentration required for leaching. These additions will be made to the CIL feed distributor, and the first CIL tank, with provision made for the addition of lime and/or cyanide further down the train of CIL tanks. The cyanide concentration required in the CIL circuit will be dictated by operational conditions, but will be between 300 and 500 mg/L as sodium cyanide, and will depend on the type of material being treated in the plant. Each CIL tank will be equipped with a downcomer, baffles, and an agitator. Each CIL tank will also be equipped with air injection facilities to maintain the dissolved oxygen concentration at an acceptable concentration of > 5 mg/L. The leaching of the gold and silver will proceed down the train of tanks with the overall CIL circuit retention time being 36 hours.

Each CIL tank will be equipped with an interstage wedge wire screen and an air-lift carbon/slurry transfer system. The screens will retain the carbon in the respective CIL tank while permitting the pulp to flow through the screen to the next CIL tank in the circuit. The interstage screen aperture size will be 1,000 µm. The carbon concentration in each tank will be maintained at 20 g/L. On a daily basis, an amount of 1.4 t loaded carbon will be transferred for elution from the head tank in the CIL circuit which will require the transfer of a slurry volume of 70 m^3/d in the CIL train. Also, an amount of 1.4 t of regenerated and/or fresh carbon will be added to the final tank in the CIL circuit every day. Air-lifts will be used to transfer this slurry on a daily basis. The slurry transfer cycle will be set by the plant operations, but could be done sequentially starting with the tail CIL tank or could be done by initially transferring slurry from all the even-numbered tanks (or odd-numbered tanks). Generally, the inter-tank carbon transfer will be achieved by the semi-continuous use of air-lifts fitted to each CIL tank. Transfer durations will be conducted as required in order to maintain the carbon concentration at the nominal 20 g/L.

As mentioned, loaded carbon will be transferred from the first CIL tank to the loaded carbon screen by means of an air-lift. The loaded carbon screen will be a vibrating screen equipped with spray water nozzles to wash the ore particles off the carbon. The loaded carbon will be transferred to the carbon circuit for acid treatment, elution and regeneration. The screen underflow will contain the slurry and wash water and will be returned to the first CIL tank.

The leached slurry will exit the CIL circuit via the interstage screen on the last CIL tank and will be discharged over the carbon safety screen. This vibrating screen will also be equipped with spray water nozzles and will prevent any carbon particles, which may have by-passed the last interstage screen or been lost through a holed interstage screen, from reporting to the residue. Any carbon which reports to the carbon safety screen will be collected in a bin and will be returned to the CIL circuit. The safety screen undersize slurry will be transferred to the tailings thickener.

17.4.5 TAILINGS THICKENER

The plant tailings will be thickened to maximize the recovery of water and cyanide prior to cyanide detoxification and final deposition. The tailings circuit will include the following equipment:

a. Thickener Feed Well.
b. Thickener Unit, 13.0 m Diameter.
c. Thickener Underflow Pumps.

Flocculant reagent will be added to aid the settling of the tailings in the thickener. The flocculant will be added as a dilute solution of 0.02% strength at an addition rate of 25 g/t.

The thickener overflow solution, containing re-usable cyanide, will be directed to a standpipe which will direct the overflow solution to the process water tank for re-use in the plant. The solids will be thickened to a pulp density of 60% solids and will then be pumped to the cyanide detoxification circuit.

17.4.6 CYANIDE DETOXIFICATION AND TAILINGS DEPOSITION

The cyanide content of the slurry will be reduced to concentration levels lower than the regulated cyanide limits in the cyanide detoxification tank prior to the slurry being discharged as plant tailings to the tailings storage facility.

The main item of equipment will be:

a. Cyanide Detoxification Tank with Agitator.
b. Air Generation and Supply System.
c. Reagent Supply Systems.

The tailings thickener underflow will be pumped to a detoxification tank to reduce the cyanide concentration to an acceptable environmental level prior to tailings disposal. The thickened tailings slurry will be pumped into the detoxification tank where dilution water and reagents and air will be added to reduce the cyanide concentration. The reagents that will be added will include copper sulphate, sodium metabisulphite (SMBS) which will generate the sulphur dioxide required for the cyanide detoxification reaction, and lime which will be added as required to maintain an alkaline pH for optimum cyanide detoxification. The addition of process water will reduce the density to the required design density of 50% solids which optimises the transfer of oxygen required for the reactions to proceed. The cyanide detoxification tank will be equipped with air addition points, as well as with an agitator to enable the air and the reagents to be thoroughly mixed with the tailings slurry.

The slurry will have a minimum residence time in the tank of one hour. This was determined by testwork to be the acceptable period of time required in order for the cyanide concentration to be reduced to acceptable environmental limits. The overflow from the cyanide detoxification tank will be directed to the tailings dam standpipe, and then to the tailings storage facility. Although designed for a retention time of one hour, the detoxification plant will actually only treat about 50% of the tailings slurry produced thereby allowing a greater recovery of cyanide from the process solutions while still meeting the environmental standards

17.4.7 Carbon Circuit

Acid Washing

Loaded carbon will enter the acid wash tank from the loaded carbon screen. Under normal operating conditions, the loaded carbon will first be acid treated, then neutralized with caustic solution, or thoroughly rinsed, followed by the subsequent elution stage.

The acid wash circuit will have the following equipment:

a. Loaded Carbon Screen.
b. Acid Wash Tank.
c. Carbon Transfer Pump.
d. Acid Solution Pumps.
e. Acid Wash Pumpbox.

The acid wash tank will receive a batch of 1.4 t of loaded carbon for acid washing prior to eluting. A 3% hydrochloric acid strength solution will be re-circulated through the bed of carbon in the acid wash tank. This acid washing treatment will remove scale build-up and other inorganic contaminants which will inhibit gold adsorption onto the carbon. The duration of the acid wash will be about four hours. This will be followed by a water rinse to remove the remaining traces of hydrochloric acid.

The acid-washing step can be by-passed, if operational circumstances permit, with the loaded carbon being transferred directly to the elution column.

Loaded Carbon Elution/Desorption/Stripping

The gold and silver adsorbed onto the carbon will be desorbed/eluted in the elution column. The elution circuit will have the following equipment:

a. Elution Column.
b. Carbon Transfer Pumps.
c. Solution Pumps.
d. Hot Water Boiler with Heat Exchangers.
e. Solution Tanks.
f. Solution Samplers.

After acid washing, the loaded carbon will be pumped to the elution column. The elution solution will be heated using a combination of plate and frame heat exchangers and a diesel-fired hot water heater.

After reaching the elution/stripping temperature, the solution will be pumped upward through the elution column. The elution column/strip vessel will be designed to treat a 2-t batch of loaded carbon although planned the production rate will be about 1.4 t depending on operational conditions. The elution/stripping of gold and silver from the loaded carbon will be accomplished using a modified Zadra based elution process. The gold will be eluted from the carbon under temperature and pressure conditions to form a pregnant solution. The gold-bearing solution will exit the elution vessel, and will then flow through the two cool-down heat exchangers to the electrowinning circuit. The two cool-down heat exchangers will cool the pregnant solution before this catholite solution is transferred to the electrowinning circuit. The barren catholite solution leaving the electrowinning cells will be re-circulated via the barren solution tank.

After completion of the elution cycle, the carbon will be transferred to the regeneration kiln storage bin via a dewatering screen

Electrowinning

The gold and silver will be recovered by electrowinning from the catholite solution. The electrowinning circuit will have the following equipment:

a. Electrowinning cell with anodes and cathodes.
b. Sludge/Cathode Washing Tank.
c. Filter Press.
d. Filter Feed pump.

The catholite solution will enter the electrowinning cell from the elution column. The electrowinning will be undertaken in one 4.0 m^3 electrolytic cell. The gold and silver will be electro-plated onto stainless steel wool cathodes. The total daily metal recovery will average 14 kg gold and silver.

The flowrate through the electrowinning cells will be about 7 m^3/h. The solution leaving the electrowinning cells will flow by gravity to the barren solution tank for making up to strength with caustic and cyanide for the following elution/stripping cycle. On completion of the electrowinning process, the electrowinning cell will be drained and the deposited metal will be washed off the cathodes into the cathode wash tank. The cathodes can also be lifted out of the cells and placed inside the cathode wash tank for the thorough washing and removal of deposited metal, if necessary. The cell sludge and the metal washed off the cathodes will be washed into the cathode wash tank as required. The sludge and metal will then be discharged to the electrowinning sludge filter press to remove the bulk of the solution which will be re-used in the electrowinning circuit, or recycled to the leach circuit. The filter press will be cleaned out at the end of the filtration cycle. The metal and sludge will be placed into trays for drying in the drying oven followed by smelting.

A typical elution cycle time will be about 18 hours in total. The elution and electrowinning circuit design of 2 t per elution will allow for the treatment of HG material, or maintenance demands

REFINERY

The gold and silver will be smelted into doré bars. The refinery will have the following equipment.

a. Drying Oven.
b. Flux Mixer.
c. Diesel Smelting Furnace.
d. Doré Safe and Vault.

When the metal and sludge has been dried, the material will be mixed with fluxes which are typically a combination of borax, nitre if required, and silica sand. Smelting will take place in a diesel-fired tilting crucible furnace. The furnace will be equipped with gear driven tilt mechanism for the pouring of the molten gold-silver melt product into moulds. A cascading mould system will be used. The doré bars will be cleaned to remove adhering slag and will be stored in the doré safe until the bars are despatched to their final destination. The smelting will be dictated by operations, but will probably be performed every one to two weeks

CARBON MANAGEMENT

The carbon handling circuit will include all the components necessary to move, store, add, reactivate, and remove carbon in the carbon system. Carbon will be transferred between the various unit operations in the plant by a screw type pump (high clearance for low degradation) and by pressurization in the elution column. Carbon transfer in the adsorption circuit will be by air-lifts. The valves will be operated locally. Carbon regeneration, or reactivation, will be conducted in a diesel-fired rotary kiln. Fresh carbon will be added to the quench tank in the carbon circuit which will subsequently be discharged over a sizing screen to remove undersize carbon particles from the fresh, or reactivated, carbon prior to its return to the adsorption circuit.

Eluted carbon will be reactivated prior to its return to the CIL circuit. The reactivation circuit will have the following equipment:

a. Eluted Carbon Dewatering/Sizing Screen.
b. Kiln Feed Bin.
c. Kiln Screw Feed Conveyor.
d. Reactivation Kiln.
e. Quench Tank.
f. Reactivation Carbon Sizing/Dewatering Screen.
g. Carbon Transfer Pumps.

The eluted carbon will be transferred hydraulically to the reactivation kiln feed bin via a vibrating dewatering screen. The screen underflow will be collected in the quench tank. An inclined screw conveyor will feed the carbon to the rotary kiln while also allowing for some additional drainage of water from the carbon to take place.

Reactivation (regeneration) of the eluted carbon includes the removal of adsorbates which have accumulated during the adsorption process. This process will restore the porous structure of the activated carbon. This reactivation process will be done using a rotary kiln. The carbon entering the kiln will initially be wet and the kiln will first dry the carbon. The steam-carbon gasification reaction subsequently oxidises the carbonised adsorbate residues as part of the reactivation process. The throughput design capacity of the kiln will be 83 kg/hour of carbon with the kiln design temperature of 680ºC.

The reactivated carbon will exit the kiln and drop directly into the quench tank. The carbon will be dewatered and sized over the reactivated carbon sizing/dewatering screen. The sizing/dewatering screen will be equipped with water spray bars to ensure that fine carbon particles will be removed from the reactivated carbon before this reactivated carbon is returned to the CIL circuit. The fine carbon particles will be transferred to the tailings thickener

17.4.8 Reagent Preparation

The reagent preparation section will prepare the reagents for use in the various parts of the processing circuit. The main items of equipment will be the following:

a. Bag Breakers.
b. Exhaust Fans.
c. Screw Conveyors.
d. Mixing and Holding Tanks.
e. Transfer and Metering Pumps.

The reagents will be prepared reasonably close to the point of usage. The reagent preparation section will be under a roof to protect the reagents and equipment. Most reagents will be received in bulk as in palletized bags, chemtainers, drums or bulk bags. The reagent preparation section will contain strategically located safety showers and eyewash stations. Each reagent preparation area will be bunded to contain any spillage which may arise during the preparation stage with each bunded area served by a sump pump for the cleaning up and control of any spillage arising.

Lime

The design is based on using hydrated lime as the pH modifier for the various unit processes. Hydrated lime will be delivered in 40-t trucks. It will be added to the grinding circuit, CIL circuit for protective alkalinity for the cyanide, and also to the cyanide detoxification tank. The hydrated lime will be off-loaded into a storage silo. A screw conveyor will add the required amount of hydrate lime to the hydrated lime tank. The hydrated lime slurry strength will be 20%. The hydrated lime will then be distributed to the addition points via a closed loop piping system

Sodium Cyanide

The sodium cyanide will be delivered in bulk boxes as small briquettes. Sodium hydroxide, (caustic), will initially be added to the mixing tank to ensure that the solution will be alkaline. The cyanide will be dissolved in water to the required concentration strength of 20% in a mix tank. The cyanide solution will then be transferred to the holding tank from where it will be distributed to the points of usage.

The cyanide preparation area will be isolated and only approved personnel will be allowed to enter the preparation area. This area will be equipped with a hydrogen cyanide monitor to provide warning in the event that HCN gas is present.

Flocculant

Flocculant will be used in the tailings thickener as an aid in the settling process. The flocculant will be prepared at the required concentration in a proprietary vendor-supplied flocculant preparation facility. Flocculant will be delivered in bulk bags. A screw conveyor will deliver the correct amount of dry flocculant powder to be mixed with water prior to delivery into the flocculant mix tank. The flocculant will be allowed to hydrate in the mix tank before being transferred to the holding tank where it will be made up to the required dosing strength. A metering pump will transfer the required amount of flocculant from the holding tank to the point of addition at the tailings thickener.

Hydrochloric Acid

Hydrochloric acid will be used for the dissolution of acid-soluble contaminants in the carbon acid washing process, typically calcium which has precipitated as calcium carbonate in the pores of the activated carbon. The hydrochloric acid will be delivered as concentrated acid. A sufficient amount of acid will be pumped from the acid containers directly to the water-containing acid wash pumpbox where the concentration will be adjusted to approximately 3% acid strength.

Sodium Hydroxide

The sodium hydroxide will be delivered in bulk bags. It will be mixed with water to make up batches of caustic solution at the required solution strength of 20%. The caustic solution will then be transferred to the caustic storage tank from where it will be pumped to the elution circuit as required.

Sodium Metabisulphite (SMBS)

SMBS will be supplied as a solid material in bulk bags. It will be mixed with water to a 20% solution strength in the mixing tank. This will be pumped from the mix tank to the holding tank. The solution will then be pumped to the cyanide detoxification tank at the required dosage rate.

Copper Sulphate

Copper sulphate will be supplied in the pentahydrate form as a solid crystalline material which will be shipped in bulk bags. It will be mixed with water to a 20% solution strength in the copper sulphate conditioning tank. The solution will be mixed using air agitation to assist the dissolution process. The copper sulphate solution will be pumped to the cyanide detoxification tank at the required dosage rate.

Activated Carbon

Activated carbon will be delivered in bulk bags. It will be added to the quench tank as required and pumped to the carbon sizing screen to remove any fines prior to entering the CIL circuit.

17.4.9 Water Circuit

The water circuit will provide the amount and type of water required for the processing in different areas of the Plant. The main items of equipment will be the water tanks for the different water circuits and pumps.

The water system for the process plant consists of the following circuits:

a. Fresh Water Supply System.
b. Fire Water Supply System.
c. Gland Service Water Supply System.
d. Process Water Supply System.

Fresh Water

Fresh water will be supplied by boreholes. The fresh water will be pumped to the fresh/fire water tank for distribution to the gland service circuit, and the reagent preparation section, and to the plant water system as make-up water, if required

Process Water

The tailings thickener overflow solution will make up the bulk of the process water with the remainder coming from the tailings storage facility. This process water will be used for grinding and in other parts of the plant as required.

17.4.10 Air Supply

Air will be required for process use and instrumentation purposes. The main items of equipment required will be the following:

a. Air Compressors.
b. Plant and Instrument Air Filter, Drier and Receiver.
c. Process Air Blowers.

Two air compressors will supply the required plant and instrument air to the process plant utilizing one common distribution system. The air from the compressors will be fed to an air receiver and will pass through an air filter to remove remnant grease or oil, and will be dried prior to use.

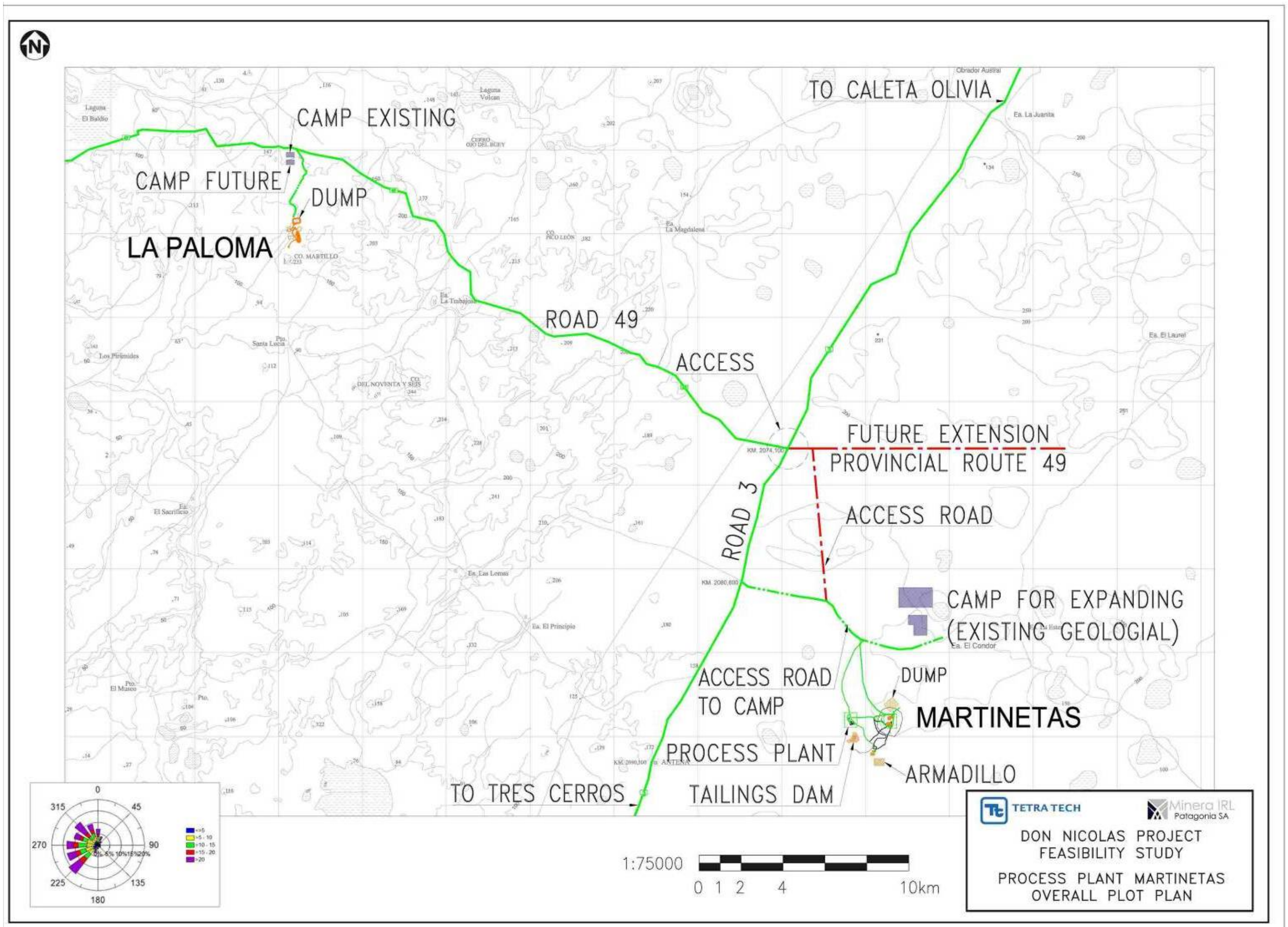
Two centrifugal air blowers will provide the air required for the preparation of the copper sulphate, as well as the CIL circuit, and the cyanide detoxification circuit.

18.0 PROJECT INFRASTRUCTURE

18.1 Location of Principal Infrastructure

The following figure illustrates location of the principal infrastructure of the Don Nicolás Project:

Figure 18-1: Don Nicolás Infrastructure



18.2 Access Roads

Over the life of the Don Nicolás Project, as currently defined, some 395,000 t of ore will be transported from the La Paloma mining area to the central plant at Martinetas, a distance of approximately 50 km.

The proposed routing from Paloma will follow an existing secondary highway (Ruta 49) for some 33 km to the junction with the north-south national highway, Ruta No. 3. At this point, the 27-t trucks carrying the Paloma ore will cross Ruta No. 3, continue eastwards for approximately 1 km before turning south towards the Martinetas plant area, an additional approximate16-km distant. The initial section of this final segment will require new construction while other sections of the 50-km distance will only require upgrading of existing roads. Design criteria for the new and upgraded sections will be consistent with a contractor style of construction and the low traffic demand over the limited mine life.

Several costs estimates have been carried out based on various design assumptions with final construction estimates obtained from local contractors currently working in this area of Santa Cruz province.

It is expected that only two to four 28-t trucks per hour will be required for the transportation of La Paloma ore with a peak occurring in the second year of plant production.

Topography is not currently available for all sections of the proposed access road linking the two mine sites. However, the overall topography of the area is not challenging and it is anticipated that selection of a suitable route for the road will not be unduly difficult.

18.2.1 GEOTECHNICAL

This section has been prepared by Golder, San Juan, Argentina. A geotechnical investigation was undertaken at the Don Nicolás Project comprising an initial site reconnaissance in March 2011 and a detailed drilling, trial pitting and in-situ testing campaign between June and September 2011. Soil and rock core samples were recovered for laboratory testing. The drilling for the plant site comprised:

a. Two vertical boreholes were drilled in each processing plant site and both were tested for hydraulic conductivity. These boreholes were not oriented.

A total of 139 trial pits were excavated and mapped in Don Nicolás Project, 37 in March 2011 and 102 between July and September 2011.

Table 18-1 Trial Pits Summary (Tailing Storage Facility (TSF); Waste Dump (WD); Heap Leaching (HL); South Plant (SP))

Location	No. of Trial Pits
La Paloma	37
Southwest Plant Option	11
North Plant Option	13
TSF Option 1	29
TSF Option 12	22

WD + HL + SP	27

Geotechnical conditions for the plant site foundations and earthworks have been assessed based on the results of the drilling, trial pits and laboratory testing. The plant site structures will be founded in rock which is close to surface or outcropping. No materials were identified that would cause problems with design and construction of foundations.

18.3 Tailings Storage

This section of the report has been prepared Golder, San Juan, Argentina. It summarizes the feasibility level design of the TSF for the Project, originally completed in November 2011 and updated in January 2012 for inclusion in the NI 43-101 Technical Report.

The Don Nicolás Project site is located in a region of relative tectonic stability with a very low historic earthquake occurrence in the interior of the South America tectonic plate. Site-specific seismic hazard analysis confirms that the site is located in an area classified as a low seismic hazard. A design coefficient of 0.1 g has been taken as the peak ground acceleration for an earthquake return period of 1:10,000 years.

A total of 29 trial pits were excavated (to maximum depths of 3 m) to ascertain the geological conditions of the TSF area; however, no geotechnical drilling has been undertaken.

Three stratigraphic horizons were defined visually and are as follows:

a. Topsoil between depths of 0.0 to 0.2 m.
b. Sand/gravel, weathered and fractured rock between a depth of 0.2 m. and 3.0 m.
c. Intact rock below a depth of 3.0 m.

An options study was undertaken to evaluate five potential TSF locations and four deposition methods: slurry (50 Wt % solids), thickened slurry (65 Wt % solids), paste (75 Wt % solids) and filter cake (85 Wt % solids). Based on this study a slurry deposition system has been adopted. Tailings will be pumped with a 50% solids content by weight from the cyanide detoxification plant to the TSF. The preferred site was chosen due to its available storage capacity, its close proximity to the mineral processing plant and beneficial pipeline topography.

The TSF impoundment area will be contained by the construction of four earthwork dams which will be constructed to an initial elevation of 139 m asl (Phase 1). These dams will then be subsequently raised to a final height of 144 m asl during a second construction period (Phase 2).

All dams will be constructed of rock fill material obtained from suitable waste rock from the Martinetas pits and will be constructed directly on top of the in-situ superficial deposits. All dams will be built with a slope angle of 1(v):2(h) on both the upstream and downstream sides and have a crest width of 8 m.

Stability analyses were carried out for an embankment 10 m high with slope angles of 1(v):2(h). The dams will be stable provided that the rock fill material has a shear strength friction angle of greater than 30º.

A filter material which is coarser than the tailings but finer than the rock fill material will be placed on the downstream face of the dams. The filter material will be screened and processed from either Patagonia gravels or from waste rock from Martinetas.

A low permeability barrier will be incorporated into the tailings storage basin. This will comprise a 1-mm smooth surface HDPE geomembrane for the majority of the lining works, including the basal area and natural slopes of the TSF basin. The upstream face of the tailings dams will be lined with a 1-mm HDPE geomembrane textured on both sides.

On the natural side slopes of the TSF a 100-mm thick layer of fine grained material will be placed to act as a protective layer between the in-situ ground and geomembrane. The basin area of the TSF will be prepared for the installation of geomembrane by proof rolling and profiling the existing in-situ superficial deposits, and removing oversized sharp stones. On the embankments the geomembrane will be placed directly over the fine filter material.

The first phase of construction provides a total tailings storage volume of 415,520 m^3 or 1.58 years plus an allowance for a 1-m freeboard. A total of 51,691 m^3 of rock fill is required to form the embankments. The total geomembrane lining area for this first phase is 129,405 m^2.

Phase 2 provides a further tailings storage volume of 652,515 m^3 or 2.48 years plus an allowance for a 1-m freeboard. A total of 159,765 m^3 of rock fill is required to raise the embankments. The additional area of geomembrane lining required for phase 2 is 33,180 m^2. If required in the future, the TSF capacity could be increased by additional raises of the embankments.

A water balance of the TSF and associated catchments was carried out using climate data presented in the environmental baseline report. The results of the water balance indicate that the available water for return by month ranges from 660 m^3/d to 1,030 m^3/d with an average of 835 m^3/d (based on a normal rainfall year). In addition, a sensitivity analyses was carried out which suggests that the return water available by month could be as low as 605 m^3/d during a dry year and as high as 1,140 m^3/d during a wet year.

A 3-D hydrogeological model was developed to evaluate seepage losses from the TSF, based on losses through assumed defects in the geomembrane liner. The results indicate that the seepage rate through the base of the TSF will be approximately 45 m^3/d (1.4 m^3/d/ha). The results also indicated that the seepage losses through the face of the tailings dams are negligible.

The geomembrane will be held in place by a system of anchor trenches. Additional ballasting will be provided by a layer of fine grained soils placed on top of the geomembrane to provide additional weight. Large ballast bags will be placed on the exposed side slopes to provide additional surcharge. An above liner drainage system of geotextile wrapped, perforated PVC pipework and covered in gravel, will be laid in a radial pattern to a sump at the lowest part of the basin. Water will be pumped from this system using a small submersible pump to assist consolidation and densification of the tailings, and to reduce the hydraulic head on the liner system.

Surface water run-off will form temporary ponds against the downstream toe of each dam. A protection layer of rock boulders (Rip-Rap) will be placed on the downstream slopes of the tailings dams to prevent erosion. This water will be pumped back to the Plant's raw water system. The volume of water within the TSF impoundment area has been evaluated to ensure that there will be sufficient freeboard in the event of a 1:100 year storm event so that overtopping of the tailings dams does not occur. Water stored in the tailings pond will be returned to the process water tank located at the plant using pumps mounted on a floating barge system.

The return water pipeline will be a 125 mm nominal diameter PE pipe with an SDR of 15.5 (110 psi).The return water pump will be 3" centrifugal pump with a 2.4 kW motor.

The tailings pipeline will be a 125-mm nominal diameter HDPE pipe with a SDR of 15.5 (110 psi). Tailings will be discharged into the TSF via a series of spigots. Dust generation from the surface of the tailings will be controlled by regular cycling of the point of discharge to ensure the surface of the tailings is maintained damp at all times. The tailings pump will be a 6-in centrifugal slurry pump with a 30 kW motor.

An emergency tailings storage pond has been included at the low spot between the plant and the TSF with a capacity for approximately half a day's tailings production plus an allowance for a suitable freeboard. This pond will allow tailings within the delivery pipeline to be stored outside of the delivery pipeline in the event of a breakdown, thereby preventing the settlement of solids within the pipe causing a potential blockage.

Monitoring instrumentation in the form of survey monoliths and ground water piezometers will be installed at the TSF site so that developing changes can be monitored and remediation action taken if required.

18.4 POWER SUPPLY

18.4.1 POWER PLANT

Electrical power will be provided by rented and contractor-operated 4-stroke diesel generator sets. The decision to use site generated power was made on the basis that at present the option of connecting to the Santa Cruz grid system requires additional study and time, for example to investigate environmental permitting requirements and easement costs.

The power station will consist of four 1.4 MW diesel generators, with one unit on standby. The power requirement has been calculated as follows:

a. Process plant nominal installed power = 4 MW.

b. Estimated average plant consumed power = 3 MW

c. Three 1.4 MW generator sets (operating at a typical 75 % load) will produce 3 x 1.4 x 0.75 = 3.15 MW

18.4.2 POWER DISTRIBUTION

The power distribution system will be at 6.6kV voltage extending to each of the facilities where secondary transformers will be installed as required.

The main power load areas will consist of crushing plant, grinding and CIL, elution, refinery and reagents, the truck maintenance shop, and warehouses buildings. The administrative area will include the offices, clinic, truck scale, and security buildings.

The industrial area will be supplied with power via cables installed in duct banks from the main distribution switchgear.

18.5 WATER SUPPLY

This section has been prepared by Tetra Tech based on exploration field work and analysis of the resulting data completed by Hidroar S.A., at the Project site over several months in 2010 and 2011. Hidroar is a well-known consulting group with substantial experience of hydrologic exploration and modelling in the Santa Cruz district or Argentina.

Investigations of the surface water and groundwater hydrology of the Project area were principally designed to identify a viable process water supply of nominal capacity of 1,000 m^3/d, to investigate sources for camp water (90 m^3/d), and to evaluate mine dewatering requirements. In addition, the hydrogeological investigations provided data and information to support environmental characterization and permitting requirements.

Subsequent design of the tailings facility indicates that significant return water to the process plant will be available, and so the net demand from well water will be considerably lower than 1,000 m^3/d once the plant and tailings facilities are at full capacity.

18.5.1 GENERAL CONSIDERATIONS

The hydrology of southeastern Patagonia, including the Project area, is controlled by the semi-arid climate and bedrock geology of the region. Precipitation averages approximately 168 mm annually with a notable dry spring season and mean annual humidity of 54.2%. Temperature varies from mean monthly averages of 1.3°C in

June, to 14.8°C in January. High winds characterize the region with average annual speed of 26.3 km/h.

Precipitation accumulates as surface water run-off or infiltrates into the groundwater system which is controlled by the bedrock geology. Groundwater occurs within Mesozoic volcanic rocks and sediments associated with regional northwest-southeast trending graben structures.

In the La Paloma area, basaltic and andesitic volcanic rocks host quartz veins with gold mineralization. Secondary fractures in bedrock are, in part, associated with variably well-developed structures including the most important local structure, the northeast-southwest trending Mill Fault. At Martinetas, layered rhyolitic rocks are fractured and cut by the north-south trending, steeply dipping Blanca Fault and numerous northwest-southeast and northeast-southwest faults; widespread fracturing is also present.

Regional physiographic landforms were shaped by the interaction of wind and rainwater run-off on the bedrock and are characterized by rounded hills and small plateaus, ephemeral drainages and basins, fluvial terraces, and regional peneplains. Perennial surface drainage in the region is limited to the Rio Deseado; flow in ephemeral channels occurs only in response to large-scale precipitation events.

Regionally, groundwater flows from west to east including through and beyond the Project area with local shallow discharge to springs and deeper regional flux ultimately discharging to the Atlantic Ocean. Groundwater resides primarily within interconnected fractures in bedrock and moves preferentially along pathways constrained by interconnected fault zones of enhanced permeability. Recharge to the aquifer occurs through infiltration of an estimated 6% of precipitation into ephemeral channels and bedrock fractures and joints.

18.5.2 HYDROGEOLOGICAL EXPLORATION

To characterize the groundwater systems in the La Paloma and Martinetas areas, 26 exploratory wells were drilled and evaluated. Wells were sited in areas of potential enhanced permeability identified based on geologic and geomorphological analysis supported by electrical resistivity geophysical surveys. Testing included constant rate pumping over periods of one to seven days to estimate hydraulic properties and long-term production sustainability.

Hydraulic properties were estimated based on 11 slugs, and 13 pumping tests on 13 wells, most within the vicinity of the proposed mine pits. Hydraulic conductivities (K) were estimated to range from 10^{-5} to 10^{-7} m/d, and are characteristic of fractured rock aquifers (Freeze and Cherry, 1979), along with storage (S) from 4.5E-04 to 8.5E-04. Laboratory analysis indicates the groundwater is suitable in quality for mill usage.

Analysis of pumping tests indicates that groundwater is recoverable from the aquifers at the La Paloma and Martinetas areas through strategically located wells. At La Paloma, wells will supply water for mine support dust control and general usage. At Martinetas, wells will supply raw water for the mill and dust control; some wells will be strategically located to support pit dewatering. Wells at El Cóndor camp will provide water for personal and operational use. Pending final water balance estimates, six wells are anticipated for water supply and/or dewatering support at Martinetas and the camp area, including:

Table 18-2: Initial Water Supply Well Requirements

Area	Wells	Depth (m)	Diameter (mm)	Production (m^3/h)	Total Production (m^3/d)
Martinetas	3	150	150	6	1,150
Armadillo	2	150	150	15	
Camp	1	60	150	1 – 11	90

Pumps will be optimally selected based on estimated well capacity and total dynamic heads. At Martinetas, water will be pumped from the wells and piped to local collection tanks then transferred to the fresh/fire water tank at the process plant. Wells will be operated at pumping rates keyed to mill make-up requirements which are anticipated to fluctuate based on operational considerations and seasonal conditions.

Additional wells are allowed for as sustaining capital in both the Martinetas and Paloma areas to cover any dewatering needs encountered (in addition to sump and pump costs include in the mine costs).

The hydrologic studies at Don Nicolás indicate that adequate water supply is available to support mine operation. In addition, the water supply will be sufficiently distant from regional users that no interruption of current usage is likely.

18.5.3 WATER FOR HUMAN CONSUMPTION

Water consumed at the main Martinetas/El Cóndor camp shall come from the well located at El Cóndor camp, where it will be pumped to a collection tank and distributed by a piped pressure pump system. It will be used for showering, washing and general cleaning.

Water consumed at La Paloma camp shall come from wells located near the proposed office and camp facilities, where it will be pumped to a collection tank and distributed by a piped pressure pump system, to be used for showering, washing and general cleaning.

Drinking water shall come in the form of distilled water provided in 20-L dispensing stations that will be allocated as needed around the Martinetas, La Paloma and processing plant sites.

18.5.4 Waste Water Treatment

There shall be two effluent Treatment Plants located at each campsite. The Martinetas plant shall have a treatment capacity of 28.6 m^3/d at a BOD <50-mg/L discharge. The plant shall receive effluent and grey water from the 130-person camp including wash house, clinic and canteen.

The La Paloma plant shall have a treatment capacity of 5.5 m^3/d at a BOD <50-mg/L discharge. It shall also receive effluent and grey water from the 40-person camp including wash house, clinic and canteen.

Individual toilet and wash areas located the processing plant site for the Administration Building, Laboratory and Truckshop shall have individual septic tank systems, where the separated liquid solution will drain to earth. The solids that accumulate over time will be removed via vacuum pump and trucked to the Martinetas/El Cóndor effluent treatment plant where it will be discharged for treatment.

Treated water will be used to complement requirements for dust control on the mine roads.

18.5.5 Drainage

Drainage channels will be installed around all plant and mine areas which will channel natural rain water run-off into the nearest drainage system which in the in case of local topography will likely find its way into clay-pan depressions.

18.6 Waste and Ore Dumps

Geotechnical analyses were performed by Golder on preliminary waste dumps designs and not on the final designs that were completed later. However, slope angle analysis is based on the same rock type and principles as earlier studies. The following waste rock dump geometry was assessed:

a. Natural angle of repose for this type of materials (volcanic rocks), and therefore, waste rock dump lift angle: 36°.
b. Lift height: 20m.
c. Berm width: 10 m.
d. Number of lifts: 2.
e. Final height: 40 m.

This geometric configuration results on a slope of about 1.4H: 1V for each lift. The overall H:V ratio may vary depending on the total amount of lifts necessary to place the total waste material volume.

Stability analysis of waste rock dump design was performed using Slide 6.012 software from Rocscience Inc. Both Mohr Coulomb and Generalized Hoek and Brown strength type models were used. Four different analyses were included:

1) Simple analysis. No extra load considered.
2) A 45-t truck load applied at 1 m from the edge of the lift. Maximum load for each axle of around 10 t (20 t in total) at 3 m of distance from each other.
3) A 45-t truck load applied at 3 m from the edge of the lift. Maximum load for each axle of around 10 t (20 t in total) at 3 m of distance from each other.
4) Seismic load. Horizontal acceleration of 0.1g.

The Table 18-3 presents the Factor of Safety (FOS) values obtained from the analyses:

Table 18-3 Waste Dump Stability Analysis

Analysis Performed	Mohr-Coulomb	Hoek & Brown
Case 1	1.255	1.869
Case 2	1.240	1.866
Case 3	1.255	1.865
Case 4	1.024	1.529

In mining, a FOS equal or greater than 1.2 is commonly accepted. Taking into consideration the type of mining infrastructure to be assessed (waste dump), values in Table 18-3 show that most of the analyses fulfill the acceptance criteria. Depending on the type of strength selected, FOS values significantly change. Nevertheless, during mine operation, regular geotechnical inspections and laboratory and in-situ tests must be performed in order to re-validate the input parameters and, therefore, the stability analysis.

The waste rock and stockpile facilities will be built from the bottom up in 20-m lifts including security berms on the edge. It is necessary to build a perimeter drainage channel located a few meters away from the toe of the waste rock dump in order to avoid stagnation of water and therefore, gulling of the toe due to water erosion.

Waste dump maximum design dimensions and capacities are:

Table 18-4: Waste Dump Capacities

Sector	Length (m)	Width (m)	Height (m)	Capacity Thous m^3
La Paloma	410	340	75	4,410
Martinetas	380	300	45	2,935
Armadillo	270	160	25	528

The plant and LG stockpile capacities are:

Table 18-5: Stockpile Capacities, Plant and LG Material

Area		kt	Capacity Thous m^3
La Paloma	Plant Ore	33	18
	LG	55	30
Martinetas	Plant Ore	2.185	1.214
	LG	76	42
Armadillo	Plant Ore	3	2
	LG	52	29

18.7 BUILDINGS

The permanent mine buildings will be designed and constructed by local Argentine contractors and construction will be programmed so that these buildings are erected as early as possible to allow their use during construction. The architecture of the facilities will allow for the use of local building materials and methods to be compatible with the surrounding infrastructure.

The Workshop/Warehouse (240 m^2) will house an electrical workshop, offices and storage space for maintenance items.

The owner's Truckshop and adjacent maintenance areas (864 m^2 combined) will have capacity for two truck bays, single light vehicle bay and oil lubricant storage area. General repair areas, welding shop, an instrument workshop, tool storage, a security store have also been included in this complex.

The Administration Building (312 m^2) will be a single storey building and will include general areas for engineering, geology and administration personnel plus individual offices for management personnel.

The combined assay laboratory and change house building (160 m^2) will house the assay laboratory, assay office, metallurgists' offices, separate washrooms for male and female personnel, and a storage room for laboratory supplies. The assay laboratory has been sized to process approximately 219 samples per day and will include sample preparation, acid digestion, atomic absorption (AA) finish, fire assay and a wet laboratory.

The main operations camp with capacity for 130 people will be located immediately to the north of the existing El Cóndor exploration camp; a smaller 40-man camp will be located at La Paloma mine site for mine operations personnel. The camp facilities will be provided with a kitchen, laundry and a health clinic at both camp sites.

18.8 Fuel Supply, Storage and Distribution

18.8.1 Consumption

During operation, the diesel consumption for the mining fleet (including auxiliary and light vehicles) is anticipated to range between 0.5 M and over 2 M L/y, with an average of about 1.5 M L/y. Fuel use for process plant equipment (strip solution heater, carbon regeneration kiln and barring furnace) will be relatively steady at about 120,000 L/y.

The power station will consume approximately 7.5 M L/y of diesel.

Gasoline use will be minor, as required for a few light vehicles used on the public roads, and will be satisfied by purchase from local retail suppliers.

18.8.2 Fuel Storage and Quality

Diesel fuel will be sourced from major suppliers in Santa Cruz or Chubut provinces able to provide certification as to its quality to ensure minimization of environmental impact. It will be delivered in bulk tankers every alternate day.

18.8.3 Storage and Distribution

The facility will consist of two 150-m^3 tanks for a total capacity of 300 m^3 which allows for 15-day storage at nominal plant consumption in case of any disruption to supply.

A skid mounted fuel supply mechanical/piping package with controls and interlocks will contain tanker unloading pump, fuel transfer pumps, strainers and filters. A light and a heavy vehicle bowser, on opposing sides to separate traffic, will complete the installation. The complete facility will be located within a bunded area able to contain the required volume of the tanks with a sump draining to the adjacent vehicle wash down facility hydrocarbon sump. It will be located 25 m clear from the nearest building. Fire suppression sprays will be permanently piped into the firewater distribution system.

Fuel for use in the process plant will be intermittently pumped to a 1,000-L tank within the process plant building; from whence it will gravity feed to all the demand points.

18.9 SECURITY

A security fence will prevent access to all, but the loading point and fuel dispensing around the Processing Plant. An electronic card system incorporated with metallic turnstiles shall be utilized where restricted access is required.

An electronic card system will be installed for fuel dispensing, preventing fuel withdrawal by unauthorized personnel and recording use against each card.

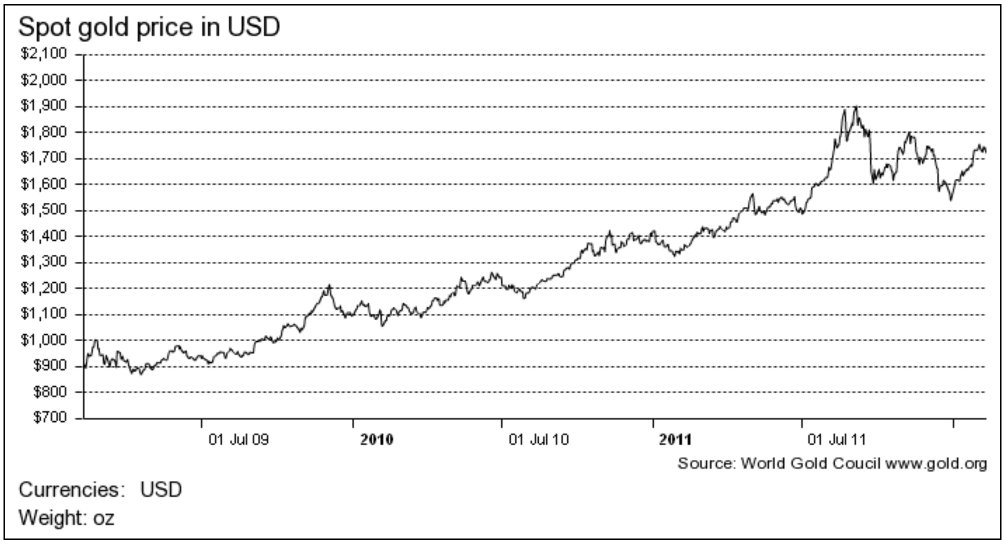
19.0 MARKET STUDIES

19.1 METAL PRICES

Over the past three years, gold prices have varied from less than US$800/oz to US$1,900/oz as shown in the Figure 19-1. At the time of writing this report (mid-February 2012), the metal is trading in the US$1,725 to 1,760/oz range.

It is impossible to accurately forecast the future price of gold. For the purposes of this study, a gold price of US$1,250 per troy ounce has been assumed, a forecast that is consistent with Tetra Tech's price recommendations for carrying out scoping, prefeasibility, and feasibility studies.

Figure 19-1: 3-Year Gold Prices



Don Nicolás will have a modest silver credit equivalent to only approximately 5% of total revenue, and so is of much less economic impact compared to the dominant gold revenues.

Silver prices have been volatile over the past five years and are currently trading in the range of US$30 to $35/oz. The FS assumes a silver price of US$25/oz that is at the upper limit of Tetra Tech's expectations for studies is not of concern considering the metal's very low contribution to the revenue profile of Don Nicolás.

19.2 DORÉ REFINING TERMS

Based on typical rates from refining companies, the following terms should be achievable for deliveries to major European gold refiners:

a. Gold payment would be based on 99.9% of the London gold price and silver payment would be 99% of the London silver price.

b. A treatment charge of US$0.50 per troy ounce of doré is anticipated based on a delivery point of London, Paris, Zurich or Frankfurt airport.

19.3 Doré Transport

A recognized security company will be responsible for transporting the doré from the mine site to a regional airport for shipment to a refiner, probably in the USA or Europe. The major refiners will normally make arrangements for the shipment from a major airport to the final destination. The doré will initially be transported by road from the mine site to Rio Gallegos, Comodoro Rivadavia, or Puerto Deseado.

The Company plans to make shipments on a bi-monthly basis. Transport costs, including insurance, are anticipated to be US$4,000 per shipment plus US$0.70 per US$1000 of shipment value.

19.4 Gold and Silver Sales

All-in realization costs (including transport above) are estimated at US$2.43/oz of doré.

For the FS, a total refining cost of $1.10 per troy ounce (99.5%) was used for gold and $0.05 per troy ounce (99%) for silver. It was assumed that the doré bars will be 95% pure.

There are several large gold refineries in North America that have a long history of service to the mining industry. Although MIRLP has not explicitly contracted any of these companies, the primary refineries that will likely be considered are as follows:

a. Johnson Matthey – Salt Lake City, Utah or Brampton, Ontario.

b. Canadian Mint - Ottawa, Ontario.

c. Metalor – Marin, Switzerland).

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL STUDIES

Vector Argentina S.A. (now Ausenco Vector) carried out the baseline studies for the Project.

The baseline study was carried out during two separate periods: the initial study was done between 2007 and 2008 during which a regional analysis was carried out, and a detailed complementary baseline study has been carried out starting in 2010.

Both campaigns have addressed the following disciplines: Physical Aspects – Geology and Geomorphology, Climate, Air Quality, Soil, Hydrology, Maximum Storm (flooding) Events, Hydrogeology, Water Quality, Soil Usage, Seismology; Biological Aspects – Flora, Fauna, Limnology, Ecosystem Characterization; Socioeconomic and Cultural Aspects – Socioeconomic Report, Archaeology and Paleontology, Protected Natural Areas, Opinion Study, Vehicular Traffic. The following disciplines are still being developed: Particulate Matter Modeling, Cost-Benefit, Landscape, and the characterization of the Acid Rock Drainage (ARD) Potential.

Studies on each discipline were carried out by means of field surveys, laboratory assessments, and office preparation of the corresponding reports and maps according to methodologies appropriate for compliance with Argentine regulations.

20.1.1 LOCATION AND ACCESSES

The Project is located in the Deseado Department, in the northeast of Santa Cruz Province. The concession granted by the Mining Provincial Agency (DPM) to MIRL for exploration activities covers a total area of 127,000 ha. In this area, the two sectors of interest to the Project are called La Paloma and Martinetas which collectively have an area of 700 ha. The towns closest to the Estancia El Cóndor campsite are: Tres Cerros (38 km southwest); Fitz Roy (93 km northeast); and Jaramillo (82 km northeast), all within the Province of Santa Cruz. The nearest large town is Puerto Deseado, which is100 km to the east of the Project.

This town will provide the Project with equipment and supplies. The town of Puerto San Julián, 170 km to the south of the Project, is expected to become the preferred accommodation centre, because of the availability of services there.

The region that has been studied is centered on parallel 48ºS and meridian 67.5ºW. It is a wide and diverse region generally characterized as a low-shrub steppe, with little topsoil.

20.1.2 PHYSICAL ASPECTS

Geologically, east of the Andean Cordillera are two large volcanic massifs separated by much younger, sediment filled valleys. The northern volcanic block is the Somuncura Massif. The southern Deseado Massif is host to the Don Nicolás Project and is an uplifted fault block with a surface area of approximately 60,000 km^2. The Deseado Massif is dominated by rhyolitic and andesitic volcanic and volcaniclastic lithologies of Middle to Upper Jurassic age. These lithologies contain a number of Au and Ag-Au mineralized epithermal deposits hosted in veins and quartz breccias, including those in the La Paloma and Martinetas areas, emplaced in the Jurassic rocks of the Bajo Pobre formation, in the case of La Paloma; and in the Chon Aike formation in the Martinetas section.

The Don Nicolás Project area is defined as one with very low seismic activity (Zone 0, according to INPRES-CIRSOC 103). Due to its extension, estimations were made for the two specific locations, La Paloma and Martinetas, which returned similar results to the regional ones. For the design level corresponding to an earthquake with a 475-year return period, the rock/firm soil ground acceleration (Type I) was 0.013 g.

The Project area is located on the eastern Patagonian plains and is generally characterized by flat to gently undulating landforms dissected occasionally by incised valleys. Some areas within the Project area exhibit hilly terrain. Elevation ranges from 130 to 220 m amsl. The Project area remains covered by a fine layer of volcanic ash arising from the 1991 eruption of Cerro Hudson, located approximately 450 km northeast in Chile. Prior to this eruption, cattle and sheep grazing predominated; however, the pastoral industry has not recovered from the effects of the ash blanket and the area is now largely uninhabited.

The Project area is located in a vast arheic section which lacks any remarkable fluvial system. In this area there are countless closed basins of different sizes, especially in the Deseado Massif region, towards which short sections of centripetal temporary water courses converge. The Project area; however, owes its current physiognomy to the relief caused by water and wind action on acid volcanic rocks. In the Western area, the drainage is connected to endorheic dendritic networks. In turn, the eastern section presents several isolated ponds. Depressions are a distinctive feature of the general Patagonian geomorphology.

The soils were mainly formed by volcanic sediments, transported and later deposited by wind or other factors. The predominance of sandy materials created shallow soils, with little or no structure; they are hard and compact when dry, and have little water storage capacity and poor drainage.

The Patagonian plains of southern Argentina endure strong westerly winds that persist throughout most of the year, and particularly during the summer months. The predominant climate is “arid cold plateau” where average monthly temperatures above 10ºC generally occur between November and March. Average monthly temperatures below 5ºC generally occur from June through August.
Annual precipitation averages 200 mm, with occasional heavy snow falls in the winter. The thin coastal strip experiences more rainfall than these annual values, especially to the north. Rainfall distribution shows that there is a winter concentration.

At Estancia El Cóndor, there is a Vantage-type Davis weather station. The July 2008 to December 2010 period was analyzed; however statistical results obtained cannot be deemed climatology-worthy due to the brief period during which measurements were taken. The mean wind speed recorded was 14.8 km/h, with a maximum of 88.5 km/h.

During that period, the air mean temperature was 10.1ºC, with extreme absolute values ranging from - 9.9ºC to 34.9ºC. The mean relative humidity recorded was of 52.7%, with extreme values between 6 and 100%. Total annual precipitation for 2009 was 116.2 mm and 130 mm for 2010.

The environmental concentrations of suspended particulate matter (PM10), nitrogen dioxide, sulfur dioxide, carbon monoxide, ozone and photochemical oxidizers, hydrogen sulfide, and lead were assessed by the Air Quality Monitoring and extrapolated to daily equivalent concentrations (USEPA, 2005). Results did not exceed the maximum concentrations set forth by the Air Quality Reference Levels defined in National Act No. 24585, Annex IV, Table No. 8.

Hydrogeological studies were performed in the La Paloma and Martinetas areas. The Arco Iris vein in the La Paloma area has transmissivity (T) and permeability (K) values in the order of 7 to 40 m^2/day and 0.2-1.2 m/day, respectively. The Sulfuro vein area has mean values of 2 m^2/day and 0.04 m/day. The storage factor was calculated to be between 1.5 and 7.3 x 10-3, which indicates the presence of a confined to semi-confined aquifer.

At Martinetas, the Coyote Norte and Cerro Oro sections bear certain similarities regarding the T and K ranges, with results varying from 1 to 5 m^2/day and 0.01 to 0.11 m/day, respectively. Data interpretation indicates that both areas, as well as the Calafate section, share a strong hydraulic connection. The three sections have a storage factor of about 1 x 10-4, which shows the medium’s confined features. The Armadillo section has the best hydraulic features of the Martinetas area; there is a clear increase of T and K when compared to the analyzed sections, with values of 5 to 8 m^2/day and 0.11 m/day, respectively. The storage factor is higher than in the northern section, with an average value of 1.4 x 10-03.

Regionally, electrical conductivity values exceed 1,500 µmho/cm, except for a section including the Bema and La Golodrina Estancias, where values are below 1,500 µmho/cm. This may be construed as corresponding to mature waters (chloride and chloride-sulfate) with specific contributions of immature waters (bicarbonate), favored by the recharge condition operating through fissure systems.

There is little water usage nowadays in the region, mainly due to the low population density; it is possible to highlight six locations of interest regarding the Project's potential direct and indirect influence. In the analyzed locations, there is the El Cóndor campsite, with a current water consumption of 20 m^3/day; the La Cabaña facility, with a maximum water requirement of 2.5 m^3/day; and Tres Cerros, where water consumption levels amount to 30 m^3/day. In these three places, water is supplied by variable-depth water wells, which range from 60 to 150 m, where water electrical conductivity fluctuates between 1,440 and 3,600 µS/cm. Hydro-geological exploration programs carried out in 2011 successfully discovered water sources in the general Martinetas area sufficient for process plant and other infrastructure requirements; see Section 18.0 of this report for a summary of the 2011 water exploration program.

Water management for the Project is covered in Sections 18.4.2 and 20.2.3.

20.1.3 *Biological Aspects*

Vegetation is sparse and dominated by grasses and low shrubs. The local flora is mostly represented by Nassauvia sub-shrub steppes which, together with the Nassauvia semi-deserts, cover 79% of the total surveyed area. The low coverage values of the herbaceous layer, the virtual lack of palatable species, and the considerable proportion of naked soil suggest a very low forage capacity for sheep. The next relevant vegetation unit, regarding coverage, is the herbaceous shrub-like steppe, which accounts for 13 % of the total surveyed area. Ponds and depressions cover 2.5%, and only some of them have topsoil.

The year-round fauna survey results have shown significant seasonal variations with regards to diversity, abundance, and makeup of the fauna specimens that are within or make use of the area. Throughout the year, the wild macrofauna was mostly represented by guanacos, accounting for about 95% of observed mammals. As for birds, choiques (lesser rheas) accounted for 25%; chingolos (rufous-collared sparrows), for 18%; and yal negros (mourning sierra-finches), for 7%. Altogether, they make up almost 50% of all birds observed. These species were the most abundant ones, and have occupied almost every surveyed environment.

The baseline study makes a comprehensive analysis of the whole sampling year, and of all the sampled seasons. Four field sampling campaigns were carried out, one for each season, namely; spring 2010, and summer, autumn and winter 2011.

Aquatic bird species deserve special attention in this project. While the study area has two small "mallínes" located in the southeast sector are of the Project, there are periods of precipitation that cause the formation of temporary water bodies. Thus, after significant periods of rain during spring, the presence of several species of water birds, including plovers and ducks can be observed. During the winter campaign, also after rains, the presence of hundreds of waterbirds included flamingos, ducks, plovers, and cauquenes among others. This observation was repeated for many bodies of water, including the L2 lagoon. These temporary bodies of water are therefore considered to be important in the area, resting places, feeding grounds and migratory bird refuges.

The lakes are environments of particular importance for migratory birds. This was evident at Don Nicolás during spring and particularly during the winter. The winter baseline survey coincided with a period of high rainfall in the 20 to 30 days prior to the campaign. A large number of aquatic birds were noted in the L2 lagoon and in the numerous temporary bodies of water. The simultaneity of the rains with the onset of the "biological spring" is what gives a unique value from the point of view of conservation of the lagoons, since they become migratory end points and temporary stays during the migratory processes.

The following species that have a variety of conservation status have been identified in the study area:

a. *Tinamotis ingoufi, Patagonian Tinamou;* Conservation status: CARPFS[7]-R: Rare. IUCN: least concern.
b. *Phoenicopterus chilensis*, Flamingo; Conservation status: CARPFS-NA: Not threatened. UICN-NT: Near threatened, UINCN Red List
c. *Conepatus humboldti*, Patagonian skunk; Conservation status: CARPFS-NA: Not threatened. CITES II: Appendix II according to CITES (Convention on International Trade in Endangered Species), CITES Appendices. SAREM-LR-nt: Least risk - Near threatened according to the SAREM[8] Red Book of Threatened Mammals of Argentina.
d. *Dolichotis patagona*, Patagonian Mara; Conservation status: CARPFS-I: undetermined. SAREM-VU: Vulnerable - Red Book of Threatened Mammals of Argentina.

Appropriate management plans should be developed for these species if the Project's impact assessment identifies that as a need.

A limnology study was conducted, which described the following environmental parameters; permanent and temporary water bodies' temperatures, dissolved oxygen, pH, and conductivity. This study concluded that abundant zooplankton is not abnormal, reaching its lowest point in winter, and progressively increasing in spring-summer; abundant phytoplankton throughout the study year, peaking in autumn, is not abnormal either. The abundant phytobenthos increased in spring and summer, and peaked in autumn. The abundance of microinvertebrates is not abnormal, with a minimum in winter and progressively increasing in spring.

[7] CARPFS: Patagonian Wildlife Advisory Council
[8] Argentine Society for the Study of Mammals

20.1.4 Social Aspects

The local towns of Jaramillo and Fitz Roy have a water supply system that uses large-diameter wells and smaller perforations, the contributions of which are of limited quality and quantity. Therefore, they need to complement their supply by importing water, in water trucks, from neighboring areas, as well as using reverse osmosis equipment to treat it.

The land has been historically used for sheep breeding activities. It takes 7 to 10 ha to maintain one sheep's average annual requirements. Naturally arid conditions, plus high numbers of sheep, have caused deep erosion of the soils. Fodder species are proportionately low, and a large area is covered by erosion pavement, due to topsoil loss.

The nearest populations within the area of direct influence are those in Fitz Roy and Jaramillo to the north. Tres Cerros to the south and Puerto San Julián are within the area of indirect influence.

The town of Jaramillo had a population projection for 2010 of 222 people; Fitz Roy, of 178; and San Julián, of 7,112.

There are no culturally or historically valuable locations in the Project's area of indirect influence. 25km to the northeast of the La Paloma area, there is a protected natural area, the Petrified Forest National Monument.

Archaeological research shows that, due to the frequency of findings, intra-site material densities (in both sections) and the diversity-quality of raw materials, the Project area is considered to be archaeologically sensitive, especially in the areas related to exposures on the edges of gullies, wetland meadows and ponds.

The paleontological survey of the Martinetas and La Paloma areas has shown the predominant development of soil and vegetation in Quaternary alluvial deposits, which cover the underlying rocks. No fossils were observed in the water-eroded alluvial profile exposures. Moreover, some exposures were identified in the Chon Aike formation, in which it is improbable that fossil remains will be found.

It is not expected that any people will be displaced. In the Martinetas area, the inhabitants of Estancia El Cóndor, both managers and employees, withdrew when the estancia was acquired by the mining company previous to its purchase by MIRL. In the La Paloma area, the purchase is being negotiated and the owners have requested to remain on the site because they run it as a campsite for visitors to the Petrified Forest Park.

20.2 Waste and Tailings Disposal, Site Monitoring, and Water Management

20.2.1 Waste and Tailings Disposal

Waste Disposal

Based on the present mineral resource and the proposed mining plan, three barren material waste dumps will be built with the following capacities:

Sector	Length (m)	Width (m)	Height (m)	Capacity (m^3 x 1000)
La Paloma	410	340	75	4,410
Martinetas	380	300	45	2,935
Armadillo	270	160	25	528

Acid Base Accounting (ABA) tests performed in the La Paloma section on barren material to be stored there indicate that it is a potential Acid Rock Drainage (ARD) generator.

For Martinetas, the ABA tests performed indicate that of the 41 samples analyzed, 98% of them have an ***uncertain potential for acid drainage generation***, and the remaining 2% have a ***high potential*** of doing so.

The implications of potential ARD and associated metals leaching on the environment needs to be considered in the design of these waste rock facilities and for their final closure. Kinetic testing is being carried out on a selection of the waste rock from the La Paloma area In order to understand and quantify the acid-generating and metals leaching potentials over time. Laboratory results are available through cycle (week) 12 of a minimum 20-cycle program. At the end of this program, a report will be issued. The recommendations coming out of this kinetic testing program, as well as on any kinetic testing program carried out subsequently on the waste rock material from the Martinetas section and tailings material should be integrated into the final waste dump and tailings dam and related infrastructure designs, and related management plans in the EIA document.

Tailings disposal

The tailings impoundment is designed to store 3.6 years' production of tailings materials, but this period could be expanded if necessary. The tailings impoundment will have a final area of approximately13 ha. For capital and operational costs optimization reasons, the impoundment will be constructed in two stages. The walls will be built with stripping or barren material, and have an approximate 1:2, (v):(h) gradient. A closed basin will be built on the low-permeability argillaceous natural riverbed. Nevertheless, it shall be lined with an HDPE geomembrane, in order to prevent seepage into the subsoil. Its effectiveness will be checked using piezometers downstream of the dam, to monitor the groundwater quality.

Tailings will be transported through a 650-m long pipe capable of dealing with up to 800 m^3/day, and distributed pursuant to a design ensuring the creation of only one central pond. Water will be returned to the Mill Plant.

No ARD data is available for the tailings material. This should be tested, as soon as material becomes available from the metallurgical testing programme, using both static and kinetic methodologies. The tailings dam design should be conservative with respect to effluent containment, both to surface water and ground water, until sufficient data becomes available.

20.2.2 SITE MONITORING

The baseline studies include monitoring activities performed on the air, water, noise, flora, and fauna quality, and also state checkpoints, both upstream and downstream of the Project. Once the location of all the facilities has been defined, the checkpoints' network will be expanded, so as to cover all potential emission sites.

Environmental Management Plans (EMPs) will be developed as part of the production stage Environmental Impact Report (EIR). These will include EMPs for all effluent-generating facilities. A monitoring plan will be prepared for surface and groundwater quality, air quality, flora and fauna, etc., for all the locations stated in the EIR.

EMPs will provisionally define prevention, control, minimization, and mitigation measures for the negative impacts on environmental and social factors, and also the strengthening of positive impacts resulting from the Project's construction, operation, and closure.

The EMP will develop the following programs: Safety and Occupational Health Program, Contingency Plan, Community Relations Plan, Monitoring, Control, and Follow-up Program, Environmental Training Program, and Affected Areas Recovery Program.

Control, prevention and mitigation measures will include atmospheric emissions, noise emissions, erosion and settlement, physical and chemical stability of the Project components (open pits, underground mines, and waste dumps), water resources, hazardous materials transportation, storage, and handling, soil and soil use, landscape value, cultural heritage and biodiversity.

20.2.3 WATER MANAGEMENT

See Section 18.4.2 for information on the hydrology, water supply, and dewatering requirements for the Project.

Regarding the Project's water balance, a conservative 6% value – if compared to precipitation (168 mm) – is considered for groundwater recharge; this is an approximately 10 mm/year water layer height. Based on this estimation, a net recharge towards the aquifer systems (porous and fractured media) is defined at about 60 Hm^3/year, for a 6,017 km^2 area, including the basins within the area of influence.

In this context, the Project's water requirement, estimated at 1,000 m^3/day (0.365 Hm^3/year) would represent the water volume coming into a recharge area of about 36 km^2 although, for practical and conservative reasons, the recharge area is set at about 50 km^2.

The results of the total groundwater reserve (confined and geological reserves) estimation show a 6.2 Hm^3 volume for the abovementioned area.

The Project plans to process some 1,000 tons of ore a day; this volume will vary depending on the Plant's operation, which entails having a permanent water supply. The 1,000 m^3/day expected water supply will be met by water returned from the tailings storage facility, supplemented with water extraction from the mine's dewatering wells in the Martinetas section. First, two wells with an individual capacity of 15 m^3/h in the Armadillo section shall be used; then, five wells with a 6 m^3/h capacity each in the Cerro Oro/Coyote section shall be used.

The human consumption requirement will be focused on the Estancia El Cóndor mining campsite, which will be 90 m^3/day. The campsite will be supplied by four boreholes, all of them near the estancia entrance, each with a 0.1 to 5 m^3/h capacity.

20.3 PERMITTING

Mining projects in Argentina must comply with an environmental permit procedure, pursuant to the provisions of National Act No. 24585, on the Environmental Protection for Mining Activities, which the province of Santa Cruz adheres to. This procedure includes having the Mining State Secretariat and other participating entities with applicable jurisdiction – like the Cultural Heritage Bureau, which reports to the Culture Undersecretariat, the Provincial Agricultural Board, etc. – to review/approve the documents.

These permits must be renewed by the enforcement authority every two years.

At present MIRL Patagonia S.A. has the following permits:

a. Environmental Impact Statement (EIS), (Resolution 287), "Martinetas" area Exploration Stage, made up of files, with approval date on November 21, 2011:

   i. No. 406.200/H/02 "Armadillo" Mine;
   ii. No. 406.196/CMP/97, "Gol I" Mine;
   iii. No. 406.197/CMP/97, "Gol II" Mine;
   iv. No. 415.232/CMP/96, "Mar III" Mine;
   v. No. 410.767/CMP/99, "Mar IV" Mine;
   vi. No. 411.825/CMP/95, "Micro I" Mine;

vii. No. 411.826/CMP/96 “Micro II” Mine; and

viii. No. 405.498/H/, MD “Mara”.

The company also has, with approval date on November 21, 2011.

b. Environmental Impact Statement (EIS), (Resolution 288), La Paloma area Exploration Stage, made up of files:

i. No. 415.448/H/07, MD “La Lechuza I”;

ii. No. 403.975/YAM/05, MD “Syrah”;

iii. No. 413.218/H/06, MD “La Paloma II”; and

iv. No. 404.392/P/02, “La Paloma I” Mine.

Table 20-1 contains the dates on which the EIR’s documents need to be submitted, the Project status, the feedback from the enforcement authorities, and the permits’ expiration dates.

Table 20-1 Environmental Impact Reports Deadlines

Company	Project Status	Area	Approval Date	Expiration Date
MIRLP	Exploration	La Paloma	11/21/2011	11/21/2013
		Martinetas	11/21/2011	11/21/2013

Some ancillary facilities must also comply with permit procedures to be installed and operated, such as explosives magazines before the RENAR (National Weapons Registry), fuel deposits before the Energy Secretariat, hazardous waste disposal locations, etc.

20.4 Social or Community Requirements

National Act No. 54585 does not require the Project to be submitted to a public hearing. Citizen engagement is addressed by the abovementioned review mechanism, in which the various authorities participate.

20.5 Mine Closure

No Comprehensive Closure Plan (CCP) has yet been prepared, and its costs have not yet been estimated. The EIR will include a Conceptual Closure Plan (CCP), for all its components for the Project construction, operation and closure stages. The main objective of the Conceptual Closure Plan is to ensure that all areas where mining or ore processing activities took place are restored in such a way that they provide adequate public safety, and a use similar to the original one to the ground affected by mining activities.

This CCP will be gradual and concurrent with the termination of economic activities, and remain active for at least five years after the total closure of activities.

As the facilities are gradually shut down, all the signs related to mining activities should be removed and replaced by signs warning about the remaining dangers and the closure activities.

Facilities that may be used by locals for agricultural and livestock and/or tourism activities, such as access roads, bridges, culverts, water wells, etc., may be transferred to their future users.

20.5.1 Closure of Tailings Storage Facility

The CCP should consider the following activities, for TSF closure:

a. Removal and restoration of disturbed areas including structure footprints, access roads, pipelines, etc.
b. Stabilization, shaping, contouring, capping and re-vegetation of disturbed surfaces.
c. Post Closure Water Management.
d. Monitoring activities to confirm the design assumptions adopted for closure, etc.

If acid drainage is a possibility, monitoring activities should be arranged to control the water quality, and treatment should be implemented (if applicable) to uphold the standards set out in Act No. 24585, Annex IV, Water Quality Reference Levels; Table 1: Water sources for human consumption; Table 2: Water sources for aquatic life protection in surface fresh water; Table 5: Water sources for irrigation; and Table 6: Water sources for livestock consumption.

Passive or active treatment systems may be applied accordingly.

20.5.2 Waste Rock Management Facilities Closure

The CCP should consider the following measures, amongst others, when closing the WRF:

a. Surfaces should be profiled with stable slopes, generating centrifugal gradients to prevent water accumulation.
b. Surfaces should be covered with organic soil recovered during construction, and waste rock dumps' surfaces shall be revegetated.
c. Surface runoff perimeter diversion canals should be maintained, as well as groundwater quality monitoring wells.

Should acid drainage be a possibility, the abovementioned measures should be implemented.

20.5.3 PIT

Once the mining activities are over, access roads should be closed, surface runoff perimeter diversion channels or berms maintained, and surfaces stabilized. If a pond forms, monitoring activities should be arranged to control the water quality, and treatment should be implemented (if applicable) to uphold the standards set out in Act No. 24585, Annex IV, Water Quality Reference Levels; Table 2: For aquatic life protection in surface fresh water.

Passive or active treatments may be applied accordingly.

20.5.4 PROCESS PLANT

All buildings and infrastructures in the process plant should be dismantled and/or demolished, and debris then removed from the location. Equipment should be taken off site for reuse. Recyclable materials should be processed for reuse or transformation. All materials should be disposed of pursuant to applicable laws. Pollutant materials should be stored and safely disposed of. Once all equipment and facilities have been removed, surfaces should be re-profiled with stable slopes and centrifugal gradients, to prevent water accumulation. Foundations should be covered with organic soil, and the whole surface should be revegetated.

20.5.5 ACCESS ROAD

At this stage of the Project, the access roads have not yet been identified. The rest of the roads and accesses' surfaces should be scarified, machine borders, bridges and culverts removed, and the shape of natural water courses and runoff crossings recovered. Reclaimed areas should be revegetated.

Roads and accesses needed for agricultural and livestock activities may be transferred to local users.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST

21.1.1 CAPITAL COST SUMMARY

The following is a summary of the Initial and Sustaining capital cost estimates for the Don Nicolás Gold Project:

Table 21-1: Capital Cost Summary, Life-Of Mine (LOM)

Area/WBS	Initial Capital Cost US$k	LOM Sustaining Captial Cost US$k
Direct Costs		
• Mine Equipment & Facilities	$4,983	$3,056
• Process Plant	$26,246	$0
• Tailings Storage Facility	$2,289	$1,239
• Water Supply	$976	$872
• Access Roads	$755	$0
• Other Infrastructure	3291	$846
Sub-Total Direct Costs	**$38,540**	**$6,013**
Indirect Costs		
• EPCM	$3,258	
• Freight	$1,400	
• Other	$1,025	$582
Sub-Total Indirects	**$5,683**	**$582**
Owner's Cost		
• Owner's Cost	$4,521	
• Pre-Production Mining	$2,359	
Sub-Total Owner's Cost	$6,880	$0
Contingency, 10%[9]	$4,222	$660
Total Project Cost	**$55,526**	**$7,255**

[9] No Contingency applied to Owner's Cost or Pre-Production mining (stripping)

21.1.2 RESPONSIBILITIES

Tetra Tech has compiled the capital cost estimates for the Don Nicolás gold project based on inputs from various sources, as follows:

a. Pre-Production Mining: NCL Ltda., Santiago, Chile
b. Mine Equipment: NCL Ltda., Santiago, Chile
c. Tetra Tech support on quotes, etc.
d. Process Plant: Tetra Tech, Santiago and Vancouver
e. Tailings Facility Golder, Argentina
f. Water Supply Tetra Tech based on Hidroar study
g. Access Roads: Penta Sur, Argentina reviewed by Tetra Tech
h. Other Infrastructure Penta Sur, Argentina, reviewed by Tetra Tech
i. Owner's Cost Minera IRL Patagonia SA
j. Initial Fills: Minera IRL Patagonia SA

21.1.3 BASIS OF ESTIMATE

The estimate was prepared using project information contained in, but not limited to the following documents:

a. Process Flow Diagrams.
b. Design Criteria Reports.
c. General Arrangement Drawings.
d. Plot Plans and Layout Drawings.
e. Equipment Lists.
f. Preliminary Engineering Drawings and Sketches.
g. Electrical Single Line Diagrams.
h. Topographic Maps (1-m intervals, provided by MIRLP).
i. Geotechnical reports containing recommended geotechnical design bases for earthworks and foundations (provided by Golder, MIRLP's geotechnical consultant).
j. The Project Implementation Plan (PIP), including the project schedule. The PIP is based on an EPCM project execution methodology.

The labour component of the capital cost estimate is based on rates relevant to the Santa Cruz and Patagonia regions of Argentina. No allowance has been included for labour rate increases caused by future variations in the local market conditions.

In addition, several costs included in the CAPEX have been provided by MIRLP, specifically Owner's Costs and First Fill estimates based on input from Tetra Tech.

21.1.4 CURRENCY, ESTIMATE BASE DATE, FOREIGN CURRENCY

All project capital costs are expressed in United States Dollars (US$) with the following provisions:

a. Costs are based on fourth quarter 2011 market conditions with no provision carried in the estimate for inflation or escalation beyond this date.
b. Currency costs have been converted to United States dollars (US$). An Argentine currency exchange rate of 4.25 to 4.30 pesos to the US$ has been applied. No provision has been made for variations in this currency exchange rate.

Pricing information from vendors was generally provided in US$, however, some site specific items have been provided by Argentine source currencies which are then converted to US$ at the corresponding project exchange rates.

In some cases, pricing was received from vendors in more than one source currency. In these cases, if US$ was not one of the currencies, each currency provided was converted to US$ at the applicable exchange rates, and one total was used for CAPEX.

21.1.5 ACCURACY

The capital cost estimate, including contingency, for the mining, process plant and infrastructure has been prepared in accordance with standard industry practices. Sufficient engineering effort was expended to support a capital estimate with an intended level of accuracy of +/- 15%.

21.1.6 CONTINGENCY

Contingency at 10% included in the capital cost estimate is an allowance for expected variations in the cost or quantity for labour, material and equipment, for the given scope of work and for the economic climate existing at the time the estimate was made. The contingency amount is an integral part of the cost estimate.

Contingency does not cover potential scope changes, force majeure, currency fluctuations, and other project or economic risks in Argentina.

No contingency has been applied to either Owner's Cost or Pre-Production mining requirements (pre-stripping).

21.1.7 TAXES

All tax related issues are outside the Tetra Tech scope of work. No Argentine Value Added Tax (IVA) is included in the cost estimate.

21.1.8 OWNER'S COST

Owner's costs are defined and provided by MIRLP.

Owner's costs compromise the following items which will be incurred during the engineering and construction periods for the Project:

a. Owners Project Development Team.
b. Owners Construction Monitoring Team.
c. MIRLP Site Operations during Project Development.
d. MIRLP Site Operations Recruitment and Training of Operating Personnel.
e. Acquisition of Management Information Systems and Telecommunications Systems.
f. Construction all Risk Insurance.
g. Lenders Independent Engineers.
h. Tax and Legal Consultants.
i. Environmental Permitting Costs.
j. Community Relations Programs.

21.1.9 EXCLUSIONS

a. Escalation beyond the estimate base date of four quarter 2011.
b. Escalation due to local market conditions.
c. All impacts of foreign currency exchange rate variations.
d. Allowances for any scope changes or extension of mine life.
e. Allowances for project risks.
f. Allowance for the risks associated with the Argentine political or legal environment.
g. Allowance for the risk of (and costs to comply with) changes to any laws, regulations, rules or policies.
h. Allowance for risks associated with (and costs of) permits, licenses and other authorizations from governmental authorities.
i. Costs associated with royalties and taxes (included in the economic analysis).
j. Costs associated with warehouse inventory over and above spare parts, and first fill of consumables.
k. Working capital and on-going capital (included in economic analysis).
l. Project EIA application and approval process.

m. Sustaining Capital (included in economic analysis).

n. Exploration Drilling.

21.2 Operating Costs

21.2.1 Summary of Operating Costs

Operating costs have been developed for the four years of operation currently projected for the Don Nicolás Project. The estimate covers all site operations, including the costs of production supervision, operating labour and consumables, maintenance supervision, labour and materials for the mine, processing, ore haulage, and general and administration.

Also included below are estimated Off-Site Costs (from MIRLP) and royalties (also from MIRLP).

The summary LOM operating cost estimates are as follows:

Table 21-2: LOM Operating Cost Summary

	LOM OPEX Costs US$k	US$/t Processed	US$/ oz. Au Recovered
Mining	$41,561	$34.52	$230
Processing	$38,100	$31.64	$210
General & Administration	$16,456	$13.67	$91
Ore Transportation	$3,220	$2.67	$18
Total Site Operating Cost	$99,337	$82.50	$549
Off Site Costs	$951	$0.79	$5
Royalties	$9,085	$7.54	$50
Total Cash Cost	$109,373	$90.83	$604
Silver Credit	$4,732	$3.93	$26
Total Production Cost (after Ag Credit)	**$104,641**	**$86.9**	**$578**

21.2.2 Responsibilities

Tetra Tech has compiled the capital cost estimates for the Don Nicolás Project gold project based on inputs from various sources, as follows:

a. Mining: NCL Ltda., Santiago, Chile

b. Process Plant: Tetra Tech, Santiago and Vancouver

c. General & Administration: Tetra Tech with input from Penta Sur, Argentina
d. Ore Transportation: NCL Ltda.
e. Off Site Costs: MIRLP
f. Royalties: MIRLP

21.2.3 Basis of Estimates – General

The estimates have been prepared in US$ and are based on an Argentine exchange rate of AR$4.25-4.30/US$.

The intended level of accuracy of the estimate, based on the unit costs assumed for the Project and current at the time of the study, is +/- 15%. No contingency has been included in the operating cost estimates.

Operating expense is defined as any recurring expenditure that can be expensed in the tax year in which it occurs. The operating cost estimate includes all recurring costs for payroll, contractors, maintenance parts and consumables, reagents, freight, etc. to operate the facilities described in this study.

Operating expenses commence at the end of the four month pre-production period. In general, the detailed operating costs are zero-based estimates built up for each operating area.

21.2.4 Exclusions

Costs to transport the gold-silver doré product from the plant gate to market and other realization costs are not considered as part of site operating costs and are dealt with in the Project economic analysis.

No allowances have been made for variation and escalation of unit operating costs (i.e. the costs are in constant US$ as of the third quarter 2011).

All costs have been estimated excluding taxes unless stated otherwise.

Other exclusions:

a. Corporate Overhead.
b. Exploration and Exploration Drilling.
c. Marketing and Sales Cost (in economic analysis).
d. Depreciation Amortization, Taxes (all in economic analysis).
e. Interest Charges.

21.2.5 SITE OPERATING COSTS BY ELEMENT

The main elements of the site operating costs are:

a. Power.
b. Diesel.
c. Explosives.
d. Process Consumables.
e. Maintenance and Administrative Materials.
f. Labour.
g. Contracted Services.

Table 21-3 summarizes the elements making up the LOM operating costs.

Table 21-3: Site Operating Costs by Element, LOM

	LOM OPEX COSTS US$k
Power	$18,000
Diesel	$7,200
Explosives & Blasting Accessories	$7,400
Mine Maintenance	$16,400
Process Operating Consumables	$9,900
Process Maintenance Materials	$1,700
Camp Catering Costs	$1,000
Other General & Administration	$10,237
Labour	$27,500
Total Site Operating Cost	**$99,337**

21.2.6 KEY ELEMENTS

POWER

The principal source of power will be from contract-operated diesel generators located in a separate building near the process plant area.

DIESEL

Principal demand for diesel will come from the mine operations, and to a lesser extent the process plant, camp requirements, and water supply.

Diesel for mine trucks and other mobile equipment will require an estimated 6.6 M litres over the mine life, or approximately 2 M L/y at full production. In the plant, diesel consumption will average approximately 120,000 L/y.

LABOUR

The Don Nicolás Project will employ over people 273 people in the first year of production, 302 in the second year, and an average of 240 for the remaining 1.5 years of the mine life.

The operations and maintenance personnel will reside in a site camp facility mainly working on two shifts, 12-h/d, and on a rotation of days on by days off.

Manpower loadings include a 12.5 factor to account for statutory holidays, vacation, sick leave and other time off.

Table 21-4 summarizes the manpower requirements in the second production year (four crews).

Table 21-4: Manpower Requirements, Production Year 02

Area	Manpower Production Y02
Mine Operations	85
Mine Support	25
Mine Maintenance	37
Mine Supervision	29
Process Plant, all departments	54
General & Administration	54
Ore Transport	18
Total Y02	**302**

PROCESS CONSUMABLES

Table 21-5 summarizes the principal consumables related to the operation of the CIL process plant.

Table 21-5: Principal Process Consumable Consumptions

Consumable	Consumption kg/year	Average Cost US$/year
Steel Balls	413,300	$454,600
Quicklime	524,500	$132,100
Sodium Cyanide	360,700	$1,262,500
Activated Carbon	17,500	$66,500
Diesel	121,400	$105,600
Sodium Meta-bisulphate	86,400	$81,200
Process Water	98,500	$123,100

EXPLOSIVES

At full production in Years 02 and 03, annual explosives consumption will average approximately 800 to 900 t/y. Relatively dry conditions are expected in the pits; therefore, the bulk will be ANFO with a 10% allowance for emulsions in wetter conditions.

GENERAL AND ADMINISTRATION

This cost element covers the following principal project activities or areas:

a. Administrative, Accounting, Human Resource Staff.

b. Environmental and General site Staff.

c. Camp Catering Costs (and related power costs).

d. General Administrative Expenses such as insurance, Office Supplies, Travel, Safety, Medical, Water Supply and Communications.

e. Maintenance of Miscellaneous Light Vehicles.

22.0 ECONOMIC ANALYSIS

The results of the Financial Analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

Forward-looking information impacting the Financial Analysis could include changes to commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and other risks.

22.1 INTRODUCTION

A financial analysis of the Don Nicolás Project was undertaken using the discounted cash flow analysis approach. Cash flows were projected for LOM, which includes construction, operation and closure phases. The cash inflows were based on projected revenues for the LOM. The projected cash outflows, such as capital costs, operating costs and taxes; were subtracted from the cash inflows to estimate the net cash flows. A financial model (Model) was constructed on a quarterly basis to estimate the net cash flows over the LOM. The net cash flows were summarized on an annual basis. The cash inflows and outflows were assumed to be in constant fourth quarter 2011 US dollar basis.

The Don Nicolás Project was evaluated on a project stand-alone, 100% equity-financed basis. The financial results, including Net Present Value (NPV) and Internal Rate of Return (IRR) do not take past expenditures into account; these are considered to be sunk costs. The analysis was done on a forward-looking basis, with the exception of the sunk costs to date, which were taken into account for tax calculations.

22.2 SUMMARY OF INPUTS AND ASSUMPTIONS FOR FINANCIAL ANALYSIS

The inputs and assumptions that form the basis of the Model include metal prices, mining schedule, mining inventory, processing throughputs, realization costs, operating costs, capital costs, royalties and taxation parameters.

22.2.1 METAL PRICES

The Don Nicolás Project will produce a gold/silver doré. Long-term gold price was selected to reflect current industry standards and is in line with the three-year tailing average gold price of US$1,280/oz and silver price of US$23.91/oz. The base case long-term gold and silver prices are presented in Table 22-1.

Table 22-1: Metal Prices

Metal	Unit	Price
Gold rice	US$/oz	1,250
Silver Price	US$/oz	25.0

The financial evaluation was also undertaken using a long-term gold price of US$1,500/oz to reflect a price closer to the current spot gold price.

22.2.2 TREATMENT AND REFINERY CHARGES

The Project will produce doré bars. The gold and silver content of the doré is estimated to be 95%.

The following estimated refining costs and recoveries are included in the Financial Analysis and are based on an analysis of existing contracts and market practice:

a. Refining costs per ounce for doré: US$1.10/oz doré.

b. Return rate for gold content: 99.90%.

c. Return rate for gold content: 99.50%.

22.2.3 TRANSPORT

It is expected that the doré bars produced at Don Nicolás Project will be transported by a security vehicle to Commodoro Rividavia. From Commodoro Rividavia, the shipment will be air freighted to Buenos Aires for transfer to an international carrier for air freight to the refinery's location.

The refinery charges mentioned above assume that the refinery is responsible for insurance and transport once delivered to Buenos Aires airport.

The transport costs, including insurance and custom broker costs, incorporated in the Financial Analysis are based on industry standards and are estimated to be:

a. Fixed – US$4,000 per shipment; and

b. Variable – US$0.70 per US$1000 of value.

It is estimated that on average there will be two shipments per month.

22.2.4 MINING AND PROCESSING SCHEDULES

The FS mine production schedule is based on mining Measured and Indicated Mineral Resources. Ore mining production is scheduled to commence four months prior to commencing of gold production.

The process plant is designed for a processing capacity rate of 350 kt/a. The process plant is scheduled to ramp up to full capacity within three months of operation. Recoveries are based on the algorithms derived from metallurgical testwork and process design work completed during the FS.

Table 22-2: Don Nicolás Mining and Processing Schedule

		Y-01	Y01	Y02	Y03	Y04	LOM
Mine Production							
• Waste	kt	713	3,851	4,511	2,279	644	11,997
• LG Material	kt	68	524	304	1,132	120	2,149
• Mine Production	kt	42	303	375	348	137	1,204
• Gold Grade	g/t	4.87	5.09	6.22	4.51	3.41	5.08
• Silver Grade	g/t	5.93	7.46	16.52	8.79	5.21	10.36
• Contained Gold	koz	6.6	49.5	75.1	50.3	15.0	196.5
• Contained Silver	koz	8.0	72.7	199.3	98.2	22.9	401.2
Plant Production							
• Plant Feed	kt		320	350	350	184	1,204
• Gold Grade	g/t		5.08	6.30	4.56	3.74	5.08
• Silver Grade	g/t		7.38	16.90	8.91	5.88	10.36
• Contained Gold	koz		52.2	70.9	51.3	22.2	196.5
• Contained Silver	koz		75.9	190.2	100.2	34.8	401.2
• Gold Recovery	%		94.4%	90.2%	91.7%	94.2%	92.1%
• Silver Recovery	%		47.7%	48.5%	44.9%	48.36	47.4%
• Gold Production	koz		49.2	63.9	47.0	20.9	181.0
• Silver Production	koz		36.2	92.2	45.0	16.8	190.2

22.2.5 OPERATING COSTS

The Project operating costs are presented in Section21.2. The total operating costs estimated over the LOM are US$99.34 million. A breakdown of the LOM operating cost is presented in Table 22-3.

Table 22-3: Don Nicolás Operating Costs

Parameter	Cost US$ million	Cost US$/t ore
Mining	41.56	34.5
Mining - La Paloma to Martinetas Cartage	3.23	2.7
Processing	38.10	31.6
G&A	16.45	13.7
Total	**99.34**	**82.5**
Note: Costs are estimated in 1Q 2012 US dollars.		

22.2.6 CAPITAL COSTS

The Project capital costs are presented in Section 21.1. The total Capital Cost (excluding closure costs) estimated over the life of the Project from commencement of construction is US$62.78 million. A breakdown of the LOM capital costs is presented in Table 22-4.

Table 22-4: Don Nicolás Capital Costs

Parameter	Cost US$ million
Pre-Production Capital	
• Directs	
• Mining	4.98
• Processing	26.25
• Tailing Dam	2.29
• Water Supply	0.98
• Infrastructure	4.05
• Indirects	5.68
• Owners Costs	4.52
• Pre-Production	2.36
• Contingency – Directs & Indirects	4.42
Total Pre-Production Capital	**55.53**
Sustaining Capital	
• Mining	3.90
• Tailings Dam	1.24
• Water Supply	0.87
• Indirects	0.58
• Contingency – Directs & Indirects	0.66
Total Sustaining Capital	**7.26**
Total Capital Cost	**62.78**

Note: Costs are estimated in 1Q 2012 US dollars.

22.2.7 CLOSURE COSTS AND SALVAGE VALUE

The end of mine life closure costs is estimated to be US$4.0 million. Due to the relatively short mine life the salvage value of the Project's equipment is expected to be more than offset the end of mine life closure costs. For the purpose of the financial analysis the salvage value is assumed to offset the end of mine closure costs.

22.2.8 INVENTORY AND WORKING CAPITAL

The Financial Analysis has assumed that 1,000 oz of gold and silver production is maintained in inventory over the life of the Project.

Working capital has been built into the Model by keeping track of the liquidities required and is calculated based on the difference between the current assets minus the current liabilities. The calculation is based on the assumption of a one month delay on account receivables and one month delay on account payables.

22.2.9 ROYALTIES

The royalties included in the Model are set out below.

PROVINCIAL ROYALTY

An ad valorem provincial royalty of 3% of run of mine value.

ROYAL GOLD ROYALTY

A 2% net smelter royalty payable to Royal Gold Inc.

YAMANA ROYALTY

A US$3.00/oz gold royalty, with a cap of US$2 M, payable to Yamana.

22.2.10 ARGENTINA TAXES

The details of the Argentina taxes included in the Model are set out below.

INCOME TAX

Argentina corporate income tax is levied at a rate of 35%.

Tax losses can be carried forward for a period of up to five years. With production forecast to commence in fourth quarter 2013, only MIRLP's accumulated tax losses from 2009 to 2011 year have been included in the financial analysis. It is estimated that for the period year 2009 to 2011, MRLP has gross tax losses of ARS$13.0 M or equivalent of US$3.1 M from previous activities in Argentina, which are an allowable deductions for income tax. The estimate excludes any tax losses incurred from 1 January, 2012 to production.

In Argentina, the method of tax depreciating and amortising fixed assets usually allowable is the straight-line method or unit of production. The depreciation and amortization rate included in the Model is based on a unit of production basis.

Depreciation/Amortization and Sunk Costs

In Argentina, the depreciation and amortization of intangibles is allowable deduction for income tax.

The depreciation rate incorporated in the Model is based on 33.33% straight-line.

It is estimated that as at December 31, 2011, MIRLP had sunk costs relating to prospecting, exploration, studies, etc. of ARS$ of ARS$75.8 M or equivalent to US$17.8 M, which are immediately deductible for tax purposes once production commences.

IVA – General Sales Tax

In Argentina, purchases are subject to the payment of the " Impuesto al Valor Agregado" (IVA), which is a general sales tax. The current IVA tax rate is 21%. IVA incurred on the initial project capital cost and is assumed to be recovered once in production. It is estimated that the IVA incurred on the initial project capital cost will be recovered in approximately 1.5 years. Once in production, IVA has been excluded from the operating assumptions due to the activity of the Project. Since the Project involves export of goods, IVA is assumed to be immediately recoverable, consistent with Argentina established practice. MIRLP also has incurred IVA from previous expenditure which can only be recovered once MIRLP is generating revenue. This IVA has not been included in the Model.

Export Duty

The export duty applicable to the export of minerals varies depending on the kind of mineral involved. Exports of minerals like raw gold or raw silver are taxed at a 10% rate. However, in case the minerals to be exported are subject to an industrial procedure, the applicable export duty will be reduced to 5%. All duties are calculated over the mineral's FOB value. The gold and silver to be exported at Don Nicolás has been subject to an industrial procedure in Argentina before its shipping to final customer. As such, the applicable export duty is 5% and has been incorporated in the Model.

Financial Tax

A financial tax of 0.60% rate is incurred on deposits and 0.60% is incurred on withdrawals from the banking accounts of the Company.

22.3 Project Cash Flows and Financial Returns

22.3.1 Cash Flows

The cash inflows are based on projected revenues. The projected cash outflows, such as capital costs, operating costs and taxes; are subtracted from the cash inflows to estimate the net cash flows. A summary of the annual cash flows is presented in Table 22-5.

Table 22-5: Don Nicolás Cash Flow

		Y-01	Y01	Y02	Y03	Y04	LOM
In Flows							
• Net Revenue	US$ M	-	60.9	81.7	59.6	27.6	229.9
Out Flows - Operating							
• Operating Costs	US$ M	-	(28.3)	(31.1)	(28.0)	(12.0)	(99.3)
• Royalties	US$ M	-	(2.4)	(3.4)	(2.3)	(1.1)	(9.1)
• Other Taxes	US$ M	-	(3.8)	(5.1)	(3.7)	(1.7)	(14.3)
• Income Tax	US$ M	-	-	(2.7)	(3.4)	(2.2)	(8.3)
Total Out Flows - Operating	**US$ M**	**-**	**(34.4)**	**(42.2)**	**(374.)**	**(17.0)**	**(130.9)**
Cash Flow from Operation	US$ M	-	26.5	39.5	22.2	10.6	98.9
Out Flows - Investing							
• Initial Capital Costs	US$ M	(55.5)					(55.5)
• Initial Capital Costs - IVA	US$ M	(11.7)	11.7				-
• Sustaining Capital Costs	US$ M		(6.0)	(1.3)			(7.3)
• Closure Costs	US$ M					(4.0)	(4.0)
• Salvage Value	US$ M					4.0	4.0
• Movement in Working Capital	US$ M		(2.6)	(1.6)	3.2	1.0	-
Total Out Flows - Investing	**US$ M**	**(67.2)**	**3.1**	**(2.9)**	**3.2**	**1.0**	**(62.8)**
Net Cash Flow	**US$ M**	**(67.2)**	**29.7**	**36.6**	**25.4**	**11.6**	**36.1**
Net Cash Flow before taxes	**US$ M**	**(67.2)**	**33.5**	**44.4**	**32.5**	**15.5**	**58.7**

Note:

1. *Costs are estimated in 1Q 2012 US dollars.*
2. *Net Revenue is gross revenue less realization costs (transport and refinery charges).*
3. *Net Cash Flow before tax is before other taxes (5% export duty and 0.6% debit & 0.6% credit tax) and Corporate Income Tax of 35%.*

22.3.2 Financial Returns

The Project was evaluated on a project stand-alone, 100% equity-financed basis. The base case gold price used in the Financial Analysis was US$1,250/oz. The financial evaluation was also undertaken using a gold price of US$1,500/oz to show the impact of a higher gold price (also see the Section 22.5 for further sensitivity analysis). The NPV, IRR and Payback Period are presented in Table 22-6. The Project financial returns demonstrate that the Project is financially robust.

Table 22-6: Summary of Don Nicolás Financial Results

Parameter	Units	Base Gold Price US$1,250/oz	Upside Gold Price US$1,500/oz
Net Cash Flow before tax	US$ M	58.7	101.6
NPV @ 5% real (before tax)	US$ M	44.7	82.2
NPV @ 7% real (before tax)	US$ M	39.9	75.6
NPV @ 8% real (before tax)	US$ M	37.6	72.4
IRR (before tax)	%	34.6%	56.3%
Payback (before tax)	Years	1.8	1.5
Net Cash Flow (after tax)	US$ M	36.1	62.2
NPV @ 5% real (after tax)	US$ M	25.1	48.0
NPV @ 7% real (after tax)	US$ M	21.6	43.7
NPV @ 10% real (after tax)	US$ M	19.8	41.4
IRR (after tax)	%	22.8%	38.1%
Payback (after tax)	Years	2.0	1.7

Note:

1. *NPVs as at commencement of construction.*
2. *NPVs are based on mid-period discounting.*
3. *Before tax is before 5% export duty, 0.6% debit & 0.6% credit tax and 35% Income Taxes.*
4. *Post-tax includes tax deduction for prior expenditure and a deduction for allowable prior tax losses.*
5. *Payback starts from the commencement of production.*

22.4 Unit Cost of Production

A summary of the analysis of the LOM average unit cost of production on a per ounce basis is provided in Table 22-7.

Table 22-7: LOM Unit Cost of Production

Parameter	Units	Cost
Mining	US$/oz	248
Processing	US$/oz	210
G&A	US$/oz	91
Site Cash Operating Costs	US$/oz	549
Realization Costs	US$/oz	5
Silver Credit	US$/oz	(26)
Total Cash Operating Costs net of silver credits	**US$/oz**	**528**
Royalties	US$/oz	50
Total Operating Costs net silver credits	**US$/oz**	**578**

Note:
1. *Costs are estimated in 1Q 2012 US dollars.*
2. *Per ounce based on payable gold.*

22.5 FINANCIAL SENSITIVITY ANALYSIS

A Sensitivity Analysis was performed on the Base Case NPV, using a 7% discount rate, and IRR. Positive and negative variations up to 15% in either direction were applied independently to each parameter: gold price, capital cost, operating cost and gold grade. The Sensitivity Analysis is reflected in the Figures 22-1 and 22-2. The results demonstrated that the Project is most sensitive to variation in gold grade and gold price. Operating cost has the least impact on the sensitivity of the NPV.

Figure 22-1: NPV at 5% real (post-tax) Sensitivity Chart

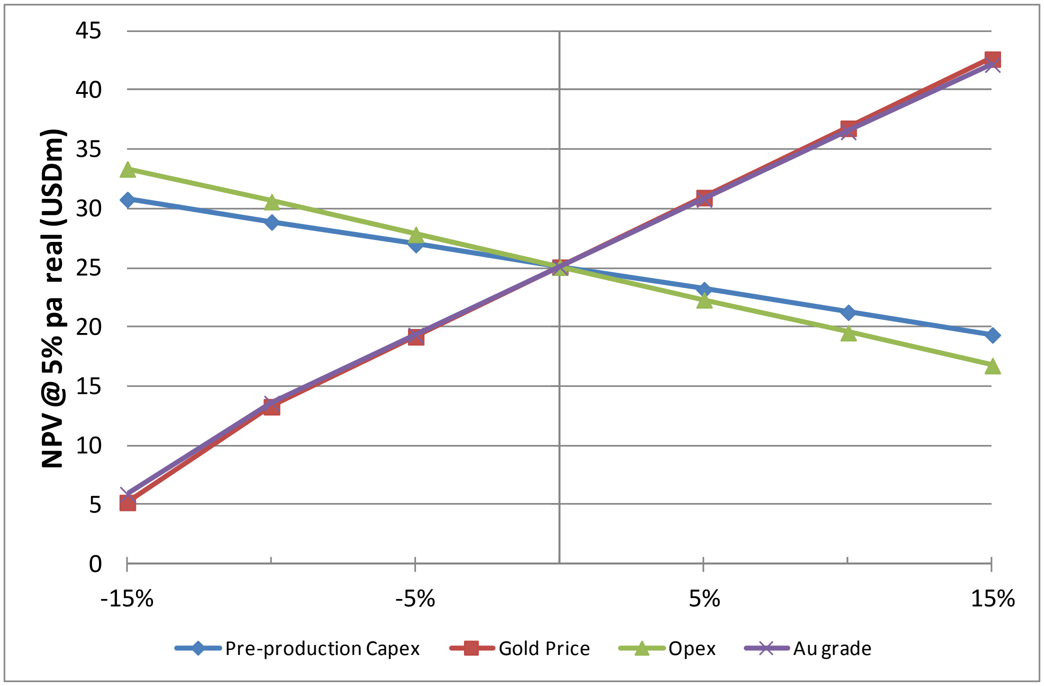
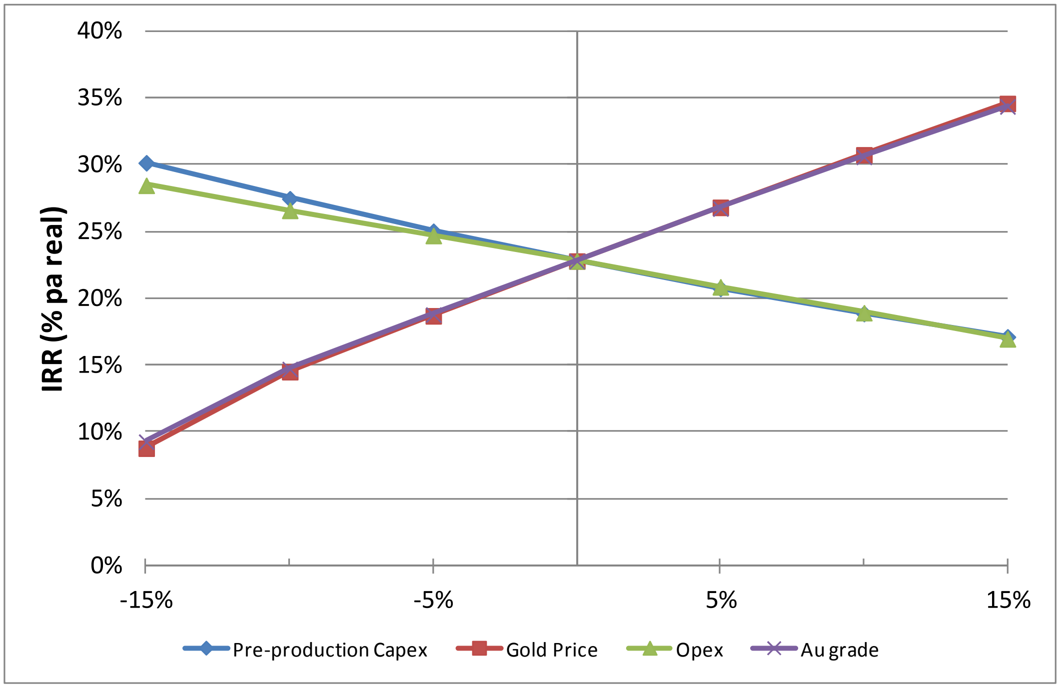


Figure 22-2: IRR real (post-tax) Sensitivity Chart



As with most gold projects, gold price is one of the most sensitive elements of the analysis. The breakeven point of the gold price for the NPV at 5% real (after tax) is approximately US$1,021/oz, whereas the IRR real (after tax) reaches zero when the price of gold is US$974/oz. Table 22-8 shows the impact of different gold prices on Project returns.

Project financial returns demonstrate that the Project is financially robust.

Table 22-8: Sensitivity of Financial Returns versus Gold Price

Gold Price US$/oz	Pre-tax NPV @ 5% real US$ M	Pre-tax IRR (real) %	Pre-tax Payback (years)	Post-tax NPV @ 5% real US$ M	Post-tax IRR (real) %	Post-tax Payback (years)
900	(7.9)	(1.0)%	3.8	(16.9)	(8.3)%	3.8
1,000	7.1	10.1%	2.4	(2.9)	2.8%	3.4
1,100	22.2	20.4%	2.1	11.1	12.9%	2.3
1,200	37.2	30.0%	1.9	20.5	19.6%	2.1
1,300	52.2	39.1%	1.7	29.7	26.0%	1.9
1,400	67.2	47.9%	1.6	38.9	32.1%	1.8
1,500	82.2	56.3%	1.5	48.0	38.1%	1.7
1,600	97.3	64.6%	1.3	57.1	43.8%	1.6
1,700	112.3	72.5%	1.3	66.2	49.3%	1.5

23.0 ADJACENT PROPERTIES

Don Nicolás is located in a region of Argentina that has seen significant exploration over the past few years, particularly for epithermal gold deposits. Several prominent discoveries have been made including Cerro Negro (Goldcorp Inc.), Cerro Moro (Extorre Gold Mines Ltd.), Pinguino (Argentex Mining Corporation), Mina Martha (Coeur Argentina).

In the immediate vicinity of the Don Nicolás property, the Las Calandrias exploration project belonging to Mariana Resources Inc. lies immediately to the north of MIRLP's Escondido property, northeast of the La Paloma claims. According to public information, this property of some 160,000 ha hosts two target areas of interest, Calandrias Norte and Calandrias Sur, the former a vein system and the latter interpreted as a disseminated precious metal mineralized system and a bulk-mining target.

MIRLP's Don Nicolás claims are surrounded by ground held by several other exploration companies including Anglo Ashanti, Fomicruz S.E., Mirasol Resources, and Extorre Gold Mines Ltd., owners of the Cerro Moro advanced exploration project some 60 km to the east-south-east of the Martinetas deposits and proposed plant site.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT IMPLEMENTATION

24.1.1 PERMITTING

The key environmental certificates are issued by the Santa Cruz Department of Environment following a review of the Environmental Impact Assessment (EIA) Report. The permit contains agreed protocols regarding development and construction, operations, mitigations methods, monitoring standards, closure procedures and financial guarantees.

The EIA is currently being prepared with the objective of completing and submitting to the Santa Cruz authorities in the first half of 2012. Initial meetings have already been held with the authorities in 2010 and comments from those meetings have been reflected in the EIA report.

The ensuing permitting process is expected to be completed during the second half of 2012 and development is expected to take approximately 12 months with first production of gold targeted for Q4 2012.

Mining projects in Argentina must comply with an environmental permit procedure, pursuant to the provisions of National Act No. 24585, on the Environmental Protection for Mining Activities, which the province of Santa Cruz adheres to. This procedure includes having the Mining State Secretariat and other participating entities with applicable jurisdiction – like the Cultural Heritage Bureau, which reports to the Culture Undersecretariat, the Provincial Agricultural Board, etc. – to revise the documents.

Further information on permitting and the current status of Don Nicolás permits can be found in Section 20.0 of this report.

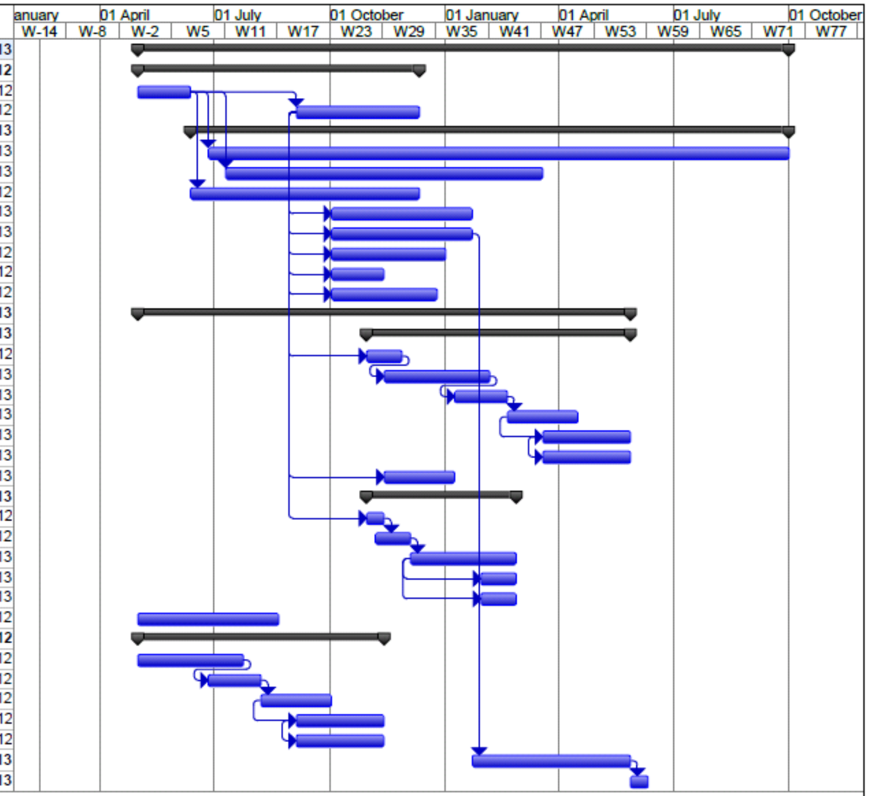
24.1.2 PROJECT SCHEDULE

A project summary schedule is presented in Figure 24-1. The schedule is based on a 3- to 4-month period of detailed engineering followed by plant construction beginning in Q4 2012 and first production by Q4 2013.

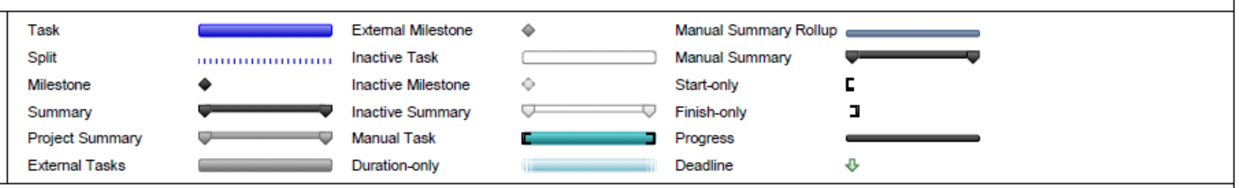
The implementation schedule will be highly sensitive to scheduling the site earthworks to avoid the southern hemisphere winter period (June to August). In order for that to be possible and align with the project release date, significant ongoing preparations need to be completed during the early phase of the implementation contract. These preparations range from access roads to mobilization of the site management team, to having advanced engineering to a point of being ready to take advantage of the first construction season for earthworks and start of critical path concrete works during the summer months of the southern hemisphere starting in approximately the fourth quarter of 2012 and first quarter of 2013.

Figure 24-1: Project Implementation Schedule

ID	Task Name	Duration	Start	Finish
1	**Project**	370 days	Tue 01-05-12	Mon 30-09-13
2	**Engineeering**	160 days	Tue 01-05-12	Mon 10-12-12
3	For LLE	30 days	Tue 01-05-12	Mon 11-06-12
4	Detail engineering	70 days	Tue 04-09-12	Mon 10-12-12
5	**Procurement**	340 days	Tue 12-06-12	Mon 30-09-13
6	Mill	66 wks	Tue 26-06-12	Mon 30-09-13
7	Crusher	36 wks	Tue 10-07-12	Mon 18-03-13
8	Other long lead items	26 wks	Tue 12-06-12	Mon 10-12-12
9	Other normal items	16 wks	Tue 02-10-12	Mon 21-01-13
10	Mining fleet	16 wks	Tue 02-10-12	Mon 21-01-13
11	Camp	65 days	Tue 02-10-12	Mon 31-12-12
12	Warehouse and offices	30 days	Tue 02-10-12	Mon 12-11-12
13	Power generation contract	60 days	Tue 02-10-12	Mon 24-12-12
14	**Construction**	280 days	Tue 01-05-12	Mon 27-05-13
15	**Plant**	150 days	Tue 30-10-12	Mon 27-05-13
16	Earthworks	20 days	Tue 30-10-12	Mon 26-11-12
17	Concrete	60 days	Tue 13-11-12	Mon 04-02-13
18	Structural	30 days	Tue 08-01-13	Mon 18-02-13
19	mechanical	40 days	Tue 19-02-13	Mon 15-04-13
20	Piping	50 days	Tue 19-03-13	Mon 27-05-13
21	Electrical	50 days	Tue 19-03-13	Mon 27-05-13
22	Tailings Dam	40 days	Tue 13-11-12	Mon 07-01-13
23	**Camp**	85 days	Tue 30-10-12	Mon 25-02-13
24	Earthworks	10 days	Tue 30-10-12	Mon 12-11-12
25	Concrete	20 days	Tue 06-11-12	Mon 03-12-12
26	Buildings	60 days	Tue 04-12-12	Mon 25-02-13
27	Piping	20 days	Tue 29-01-13	Mon 25-02-13
28	Electrical	20 days	Tue 29-01-13	Mon 25-02-13
29	Warehouse and offices	80 days	Tue 01-05-12	Mon 20-08-12
30	**Power generation**	140 days	Tue 01-05-12	Mon 12-11-12
31	Concrete	60 days	Tue 01-05-12	Mon 23-07-12
32	Structural	30 days	Tue 26-06-12	Mon 06-08-12
33	mechanical	40 days	Tue 07-08-12	Mon 01-10-12
34	Piping	50 days	Tue 04-09-12	Mon 12-11-12
35	Electrical	50 days	Tue 04-09-12	Mon 12-11-12
36	**Preproduction Mining**	18 wks	Tue 22-01-13	Mon 27-05-13
37	**Commissioning**	10 days	Tue 28-05-13	Mon 10-06-13



Project: Schedule and cash flow
Date: Mon 13-02-12



24.1.3 PROJECT ORGANIZATION

The study assumes that the Don Nicolás Project will be executed by an EPCM contractor working under the administration of the Owner's Team. Key participants in the Project will be:

a. The Owner's Project Staff.

b. The EPCM Contractor.

c. Owner/EPCM Steering Committee.

The EPCM contractor will provide a Project Management Team working under the overall direction of a Project Manager. The EPCM contractor will be responsible for the execution of the Project including adhering to the established schedule, capital cost, safety, and quality targets for the Project.

The Senior Management Team (steering committee) will be drawn from the project management organization. It will be an integrated team composed of senior representatives of the Owner's Project Team, the EPCM contractor, stakeholders, and the Owner's Operations Team. They will be kept informed of cost and progress of the Project and will ensure that the resources of all organizations are available to satisfactorily complete the Project.

A Construction Management Team will be responsible for the construction of the Project and be headed up by the EPCM Project Manager who will, in turn, report to the Owners representative.

24.1.4 PROCUREMENT

Procurement of equipment and materials will be from an appropriate mix of international and local supply, maximizing the utilization of local Argentine fabrication and construction services that can support the project schedule and quality objectives. For significantly large packages, the Project will pre-qualify a range of suitable bidders, confirming both technical and financial suitability in order to facilitate the bidding process.

24.1.5 CONSTRUCTION

Construction of the Don Nicolás Project will be performed by mainly Argentine contractors under the direction and management of the Construction Management Team, reporting to the Project Manager. The Construction Management Team will be an integrated team of Argentine and ex-patriot professionals, optimized for functional effectiveness.

Local availability of qualified and specialty workers, such as electricians, mechanics, carpenters, outfitters and welders is currently in tight supply owing to a very competitive market so there may be a need for recruitment in adjacent provinces of Argentina.

Construction offices and job site facilities, as well as the temporary construction facilities (pipe shop, crushing and concrete centres, form, formwork and built-ins), will be located at the process plant site, as close as possible to the centre of the services to be rendered. Construction worker accommodations will be placed adjacent to the process plant to minimize transportation requirements.

24.1.6 Environmental, Health Safety

A site-specific Environmental, Safety and Health Plan (ES&H Plan) will be developed to address the hazards and precautions associated with engineering design and construction activities. The plan will be written in compliance with North American safety standards and local Argentine regulations, supplemented by site specific procedures developed by the EPCM contractor to insure compliance with the most stringent applicable international requirements.

A hazard and operability analysis (HAZOP) will be conducted by the Project Design Team during the detailed design stage. This analysis will strive, through engineering efforts, to eliminate hazards that are identified during design. The Systematic Team approach will identify hazards associated with operability that require attention in order to eliminate undesirable consequences.

Training will be a key component of communicating health and safety protocol to provide safe and healthy working conditions.

24.2 Risks and Opportunities

24.2.1 Project Implementation Period

Construction and development of the Don Nicolás Project will require a “fast-track” approach in order to meet the schedule mentioned above. A key element will be successful approval of the EIA Report, and timing of construction activities to coincide with the southern hemisphere summer months of November through March/April.

24.2.2 Country and Regulatory Risk

The Project is located in one of the more mining-friendly provinces of Argentina in a region where several other exploration and operating mines have been successfully developed. Prominent examples are Anglo Ashanti’s Cerro Vanguardia open pit operation, now into its twelfth year of production, Manantial Espejo (Pan American Silver) and San Jose (Hochschild), all of which are in production, and Cerro Negro (Goldcorp) and Cerro Morro (Extorre) both in the development phase.

In the past year, some mining (and oil) operations have been disrupted by labour actions; however, these have generally been of short duration and have been solved quickly with provincial government intervention. Disruption to operations over extended periods in this particular region of Argentina can therefore be considered at the low to moderate end of the scale.

24.2.3 CURRENCY RISK

Recent Federal Government decrees following the Federal election point to ongoing intervention to control the dollar exchange rate and a gradual, but controlled devaluation of the Argentine peso.

24.2.4 CONSTRUCTION COST RISK

Costs in the Argentine Patagonia are in general higher than in other regions of the country owing to remote location, climatic conditions, and scarce supply of skilled human resources at all levels of the supply chain.

Official inflation rates published by government agencies are in the order of 10 to 12% per year, although some private economic consultancies quote estimates in the 25-30% range as more realistic. The gradual lifting of various subsidies that began in late 2011 may also impact local costs in Argentine pesos.

24.2.5 UNDERGROUND RESOURCE OPPORTUNITY

Deeper mineral resources at Sulfuro have not been included in the mine plan owing to the high costs associated with their development. Increased tonnage and/or grade in this and other deeper targets may mean that these would become economically attractive under some future combination of cost and metal price.

24.2.6 HEAP LEACH OPPORTUNITY

LG mineral resources totalling 2.15 Mt at 0.71 g/t Au will be stockpiled as part of the mine plan representing material between 0.3 g/t Au and 1.5g/t Au, as follows:

Table 24-1 Low Grade Mineral Resources

	LG 0.3 – 1.6 g/t Au		
	kt	**Au**	**Ag**
La Paloma Sur 1	8	1.04	5.74
La Paloma Sur 2	43	1.17	6.53
La Paloma Norte	3	1.21	5.12
Coyote	852	0.74	3.09
Armadillo	52	0.53	2.48
Cerro Oro	1,190	0.68	3.01
Total	**2,149**	**0.71**	**3.11**

With total contained gold ounces of 49,060 oz this resource could represent upside as an incremental addition to production from the CIL Plant. However, metallurgical testwork is required to investigate the leach characteristics of this LG mineralization.

24.2.7 EXPLORATION OPPORTUNITY

Ongoing exploration around the existing mineral resource centres may lead to additional economic discoveries that could contribute to an extension of the current mine life of four years. MIRLP holds mineral exploration rights totalling some 270,000 ha and so it is reasonable to be optimistic regarding future exploration success.

24.2.8 POWER COST REDUCTION OPPORTUNITY

The Company is investigating the installation of a power line to the Martinetas plant site as an opportunity to significantly reduce the cost incurred by producing power by diesel generation. Initial investigations are encouraging and, if this proves feasible, will have a material beneficial effect on cash operating costs.

25.0 INTERPRETATION AND CONCLUSIONS

The Don Nicolás FS as summarized in this Technical Report dated February 14, 2012 illustrates the technical and financial viability of the project based on fourth quarter 2011 constant dollars.

Initial Capital Costs of US$55.5 M have been estimated to support an annual process rate of 350,000 t over a 3.6 year mine life with a payback period of approximately 2 years; an additional US$7.26 M will be required for sustaining capital.

Based on metal prices of US$1,250/oz gold and US$25/oz silver, the Project generates an IRR pre-tax of 34.6% and 22.8% post-tax.

NPV have been estimated as follows:

Table 25-1: Don Nicolás Net Present Value Estimates

Discount Rate, %	5.0	7.0	8.0
NPV, Pre-Tax, US$M	$44.7	$39.9	$37.6
NPV, Post-Tax, US$M	$25.1	$21.6	$19.8

Other principal conclusions and interpretation of the results from the FS are:

a. The Mineral Resources have been estimated by an experienced geological consultancy, Coffey Mining Limited.

b. The Mineral Reserves and mine plan have been generated by an experienced mining consultancy, NCL Ingeniería y Construcción Limitada based on the Coffey Mining geological block model.

c. The Mineral Resources are largely defined within the Indicated category. Only approximately 11% of the total project resources are Measured and this in turn leads to a correspondingly percentage of Proven Reserve in the mine plan (11% of the total reserve).

d. Close infill drilling and blasting control will be required to avoid dilution along ore-waste boundaries defined by an assay wall cutoff grade.

e. The Project has been designed and costed in line with its short life of 3.6 years. Some areas of the project, particularly plant design, will require additional engineering input at the detailed design stage.

f. The FS assumes that the costs required for closure and remediation of the site after 3.6 years of operation will be balanced by the salvage value obtained from mine, process and other assets; therefore no closure costs as such have been allowed for.

g. The EIA is scheduled for completion within the next few months. There is no guaranteed date when this will subsequently be approved by the Santa Cruz authorities; approval delays would impact the development of the site and plant start-up projected for Q4 2013, and no costs for such delays have been allowed for.

h. Costs are expressed in fourth quarter 2011 US dollars and, where required, have been converted at the Argentine exchange rates valid at that time (ARP4.25-4.30).

i. No allowance has been included for inflation or escalation or other changes resulting from changing economic conditions in Argentina, neither in the cost estimates, nor the economic analysis, nor any other aspect discussed in this report.

j. The Project has some promising opportunities to extend mine life and improve costs. In particular, heap leaching of lower grade gold resource and lower operating costs through connection to the provincial power grid need to be followed up, in addition to ongoing exploration that could provide additional ounces of production.

26.0 RECOMMENDATIONS

The results of the FS demonstrate that Don Nicolás is a viable and feasible project within the terms of reference despite its short 3.6 mine life. Additional recommendations to increase the project's viability are:

a. Complete a scoping study to establish the technical and economic details of heap leaching the lower-grade gold mineralization already defined in the general Martinetas area.

b. Continue exploration at depth and along strike from the existing vein deposits in both the La Paloma and Martinetas areas with the objective of increasing the resource base to extend the mine life.

c. Investigate a blasting strategy to optimize dilution control and fragmentation in the selective mining of veins.

d. Complete detailed engineering.

e. Water levels in pumping and observation wells (e.g. piezometers and monitoring wells) should be recorded and evaluated regularly to build an empirical database on which future water supply management decisions will be based. Also, there should be regular recording of how much water is pumped from wells, and from the pits.

Coffey Mining has recommended the following areas for follow up work:

f. More bulk density sampling required, (at least 50 samples) from each lithological/oxidation unit, plus Bulk Density sampling should be ongoing. A budget of US$10,000 has been estimated for this.

g. More drilling recommended to test known mineralization along strike and down-dip at Martinetas (12,000 m of RC drilling – cost $2.4 M planned to commence in March 2012). More planning is required to understand the controls of mineralization at Arco Iris with the objective of bringing this into the mine plan at a later date.

h. Umpire laboratory/blind pulp submission testing required with all future drilling programs.

Ausenco Vector has recommended the following areas for follow up work:

i. Static and kinetic acid rock drainage (ARD) test work on the tailings material when samples become available out of the metallurgical test work program.

j. Kinetic test work on waste rock from the Martinetas section in order to understand and quantify their potential acid-generating and metals leaching potentials over time.

k. Recommendations coming out of this kinetic testing program, as well as on any kinetic testing program carried out subsequently on the waste rock material from the Martinetas section and tailings material, should be integrated into the final waste dump and tailings dam and related infrastructure designs, and related management plans in the EIA document.

27.0 REFERENCES

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Fernández, R.R. et. al., 2008. Los Depósitos de Oro y Plata vinculados al Magmatismo Jurásico de la Patagonia: Revisión y Perspectivas para la Exploración. Revista de la Asociación Geológica Argentina 63 (4): 665 - 681.

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28.0 CERTIFICATE OF QUALIFIED PERSON

I, Alistair Cadden, of San Juan, Argentina, do hereby certify:

I am a Principal with Golder Associates Argentina S.A. with a business address at Caseros 34(N), San Juan, J5400ERB, Argentina.

This certificate applies to the technical report entitled Technical Report, Don Nicolás Gold Project, Santa Cruz, Argentina, dated February 14, 2012 (the "Technical Report").

I am a graduate of Imperial College University of London, (MSc Soil Mechanics, 1992,). I am a member in good standing of the Institute of Materials Mineral and Mining, UK, and a Charted Engineer registered with the Engineering Council, registration number 569976. My relevant experience is 25 years of civil engineering, mining geotechnical engineering and tailings dam design. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

My most recent personal inspection of the Property was January 14 2011 for 1 day.

I am responsible for Sections 15.6, 18.2 (part), 18.3, 18.6, 21.1(part) and 21.2(part) of the Technical Report.

I am independent of Minera IRL Patagonia S.A. as defined by Section 1.5 of the Instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

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

As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 14th day of February, 2012 at San Juan, Argentina.

Original document dated, signed and sealed by Alistair Cadden, C.Eng.

Alistair Cadden, B.Eng, MSc, DIC, C.Eng, MIMMM, AIEMA.
Principal
Golder Associates Argentina S.A.

I, Marius André de Ruijter, of Delta, British Columbia, do hereby certify:

I am a Senior Metallurgical Engineer with Tetra Tech Company WEI Inc. (Tetra Tech) with a business address at 800-555 West Hastings Street, Vancouver, BC, Canada V6B 1M1.

This certificate applies to the technical report entitled Don Nicolás Gold Project, Santa Cruz, Argentina, February 14, 2012 (the "Technical Report").

I am a graduate of the University of Witwatersrand, Johannesburg, South Africa (B.Sc. Physics, Mathematics, 1970; B.Eng., 1973; M.Eng., 1979). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #31031. My relevant experience with respect to this project includes copper and base metal sulphide mineral flotation projects, and 25 years' experience in the gold industry encompassing laboratory research and development, operational gold plant responsibility, and process plant design. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument"). I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

My most recent personal inspection of the Property was in September 2011 for one day.

I am responsible for Sections 13.0 and 17.0 of the Technical Report.

I am independent of Minera IRL Limited as defined by Section 1.5 of the Instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

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

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report [or the parts of the Technical Report that I am responsible for] contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 14th day of February, at Vancouver, Canada.

Original document dated, signed and sealed by
André de Ruijter, P. Eng.

Marius André de Ruijter
Senior Metallurgical Engineer
Tetra Tech WEI Inc.

I, Anthony Sanford, of Lima, Peru, do hereby certify:

I am Environmental Services Manager with Ausenco Vector (Environment & Sustainability) with a business address at Calle Esquilache 371, Piso 6, San Isidro, Lima 27, Peru.

This certificate applies to the technical report entitled Technical Report, Don Nicolás Gold Project, Santa Cruz, Argentina, dated February 14, 2012 (the "Technical Report").

I graduated with a Bachelor of Science (Geology and Applied Geology) in 1984 from the University of Natal, a Bachelor of Science (Hons.) in Geology in 1985 from the University of Natal, and a Masters in Business Administration in Natural Resource Management from Dundee University in 1998. I am a member in good standing of the South African Council for Natural Scientific Professions (Pr.Sci.Nat.), License # 400089/93. I have worked as a Geologist continuously since 1985 and as an Environmental Scientist since 2004. For the past 7.5 years I have been employed with Ausenco Environment & Sustainability. During this period I have fulfilled roles as Senior Geologist and Environmental Services Manager Ausenco Environment and Sustainability Peru. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

I have not visited the proposed project site for the Don Nicolás Project due to time constraints and expect to make a site within the next 90 days.

I am responsible for Section 20 of the Technical Report.

I am independent of Minera IRL as defined by Section 1.5 of the Instrument.

I have had no prior involvement with the Property that is the subject of the Technical Report.

I have read the Instrument and the section of the technical report that I am responsible for, has been prepared in compliance with the Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the technical report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 14th day of February, 2012 at Lima, Peru

"Original document dated, signed and sealed Anthony Sanford"

Anthony M. Sanford, MBA (Mineral Resources Management), B.Sc (Hons.). Pr.Sci.Nat.
Professional Natural Scientist- South Africa # 400089/93
Environmental Services Manager
Ausenco Vector

I, Callum Grant, of Vancouver, British Columbia, do hereby certify:

I am a Senior Consultant with Tetra Tech WEI Inc. (Tetra Tech) with a business address at 800-555 West Hastings Street, Vancouver, BC, Canada V6B 1M1.

This certificate applies to the technical report entitled Don Nicolás Gold Project, Santa Cruz, Argentina, February 14, 2012 (the “Technical Report”).

I am a graduate of the University of Aberdeen, Scotland (B.Sc. Honours Geology, 1971) and of McGill University, Montreal (M. Eng. Mining, 1977). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #27500). My relevant experience consists of more than 30 years in the minerals industry as geologist, mining engineer, and project manager. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

My most recent personal inspection of the Property was September 2011 for two days.

I am responsible for Sections1.0 to 5.0 inclusive, 18.0, 19.0, 22.0, and 24.0 to 26.0 inclusive of the Technical Report.

I am independent of Minera IRL Limited as defined by Section 1.5 of the Instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

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

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report [or the parts of the Technical Report that I am responsible for] contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 14th day of February, at Vancouver, Canada.

Original document dated, signed and sealed by
Callum Grant, P. Eng.

Callum LB Grant
Senior Consultant
Tetra Tech WEI Inc.

I, Carlos Guzmán, of Santiago, Chile, do hereby certify:

I am a Principal and Project Director with NCL Ingeniería y Construcción Ltda. with a business address at General del Canto 235, Providencia, Santiago Chile.

This certificate applies to the technical report entitled Technical Report Don Nicolás Project, Santa Cruz Province, Argentina, for Minera IRL Limited, dated February 14, 2012 (the "Technical Report").

I am a graduate of the Universidad de Chile, (Mining Engineer, 1995). I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Australasian Institute of Mining and Metallurgy, License #229036 and a Registered Member of the Chilean Mining Commission. I have practiced my profession continuously since 1995. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

My most recent personal inspection of the Property was during November 2011 for 2 days.

I am responsible for Sections 15.0 and 16.0 and parts of Sections1.0 and 26.0 of the Technical Report.

I am independent of Minera IRL Ltd. as defined by Section 1.5 of the Instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

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

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

Signed and dated this 14th day of February, 2012 at Santiago, Chile.

"Original document dated, signed and sealed by Carlos Guzmán"

Carlos Guzmán, Registered Member, Chilean Mining Commission (0119)
Principal and Project Director
NCL Ingeniería y Construcción Ltda.

I, Doug Corley, of Perth, Western Australia, do hereby certify:

I am a Principal Resource Geologist with Coffey Mining Pty Ltd. with a business address at 1162 Hay Street, West Perth, Western Australia, 6005.

This certificate applies to the technical report entitled Technical Report, Don Nicolás Gold Project, Santa Cruz, Argentina, dated February 14, 2012 (the "Technical Report").

I am a graduate of James Cook University, Townsville, Queensland, Australia, and hold a Bachelor of Science degree (with Honours) in Geology (1991). I am a member in good standing of the Australian Institute of Geoscientists (MAIG) and I am also a Registered Professional Geoscientist, (R.P. Geo, licence number 10,109). I have practiced my profession continuously since 1991, as a result of experience and qualifications; I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

My most recent personal inspection of the Property was March 2011 for 5 days.

I am responsible for Sections 6.0 to 12 inclusive and 14.0, 23.0 and part of Section 1.0 of the Technical Report.

I am independent of Minera IRL as defined by Section 1.5 of the Instrument.

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

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

Signed and dated this 14th day of February, 2012 at Perth, Western Australia

"Original document dated, signed and sealed by Doug Corley

Doug Corley, BSc. (Hons.),
MAIG, R.P. Geo
Principal Resource Geologist
Coffey Mining Pty Ltd

I, Hassan Ghaffari, P.Eng., of Vancouver, British Columbia, do hereby certify:

I am a Manager of Metallurgy with Tetra Tech WEI Inc. (Tetra Tech) with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.

This certificate applies to the technical report entitled Technical report, Don Nicolás Project, Santa Cruz, Argentina dated February 14, 2012 (the "Technical Report").

I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #30408. My relevant experience with respect to mineral process engineering includes 22 years of experience in mining and plant operation, project studies, management, and engineering. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

My most recent personal inspection of the Property was September 4, 2011 for one day.

I am responsible for Sections 18 and 21 (Project Infrastructures section and Capital and operating cost section) of the Technical Report.

I am independent of Minera IRL Limited. as defined by Section 1.5 of the Instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

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

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

Signed and dated this 14th day of February, at Vancouver, Canada.

Original document dated, signed and sealed by Hassan Ghaffari, P. Eng.

Hassan Ghaffari, P.Eng.
Manager of Metallurgy
Tetra Tech WEI Inc

I, Stephen A. Osterberg, P.G., PhD, of Golden, Colorado, do hereby certify:

At the time of the site visit and through January 31, 2012, I was a Senior Geologist with Tetra Tech Inc. with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401 USA.

This certificate applies to the technical report entitled "Technical report Don Nicolás Gold Project, Santa Cruz, Argentina" dated February 14, 2012 (the "Technical Report").

I am a graduate of the University of Wisconsin, Oshkosh, (Bachelor of Science degree with high honors in Geology, 1982). I am also a graduate of the University of Minnesota, Duluth (Master of Science degree in Geology, 1985). I am also a graduate of the University of Minnesota in Minneapolis, Minnesota (Doctorate of Philosophy in Geology, 1993). I am a Registered Member in good standing of the Society for Mining, Metallurgy and Exploration (SME # 4103097RM). I am also a licensed Professional Geologist in good standing in the State of Wyoming, registration number PG-3444. My relevant experience is that I have worked as a geologist for a total of twenty-eight years since my graduation from university; I have been an employee of mining companies (Kerr McGee, Noranda, BHP Minerals) and mining-related consulting companies (Knight Piesold, MFG, Mine Mappers, and Tetra Tech), or self-employed in mining consulting for more than 22 years. I have also been involved in University-level research in geology for 8 years. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

My most recent personal inspection of the Property was August 7-9th for three days.

I am responsible for all of Section 18.4 of the Technical Report.

I am independent of Mineral IRL Limited as defined by Section 1.5 of the Instrument.

I have no prior involvement with the Property that is the subject of the Technical Report.

I have read the Instrument and Section 18.4 of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, Section 18.4 of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Original document dated, signed and sealed by
Steven Osterberg, P.G., PhD

Steven A. Osterberg, P.G., PhD
Senior Geologist
Tetra Tech Inc.