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Preliminary Economic Assessment Technical Report on the Romero Project, Dominican Republic

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Prepared for:

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for GoldQuest Mining Corp. The quality of information, conclusions and estimates contained herein is 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.

GoldQuest Mining Corp. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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1 Executive Summary

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by GoldQuest Mining Corp. (GoldQuest) to carry out a Preliminary Economic Assessment (PEA or 2015 PEA) and technical report for the Romero Project, a resource development gold and copper project owned 100% by GoldQuest located in the Province of San Juan in the Dominican Republic.

Two previous technical reports were prepared for the GoldQuest project pursuant to Canadian Securities Administrators' National Instrument 43-101 Standards for Disclosure for Mineral Projects and Form 43-101F1 - Technical Report (collectively, NI 43-101) that documented a resource estimate in 2013 and a PEA in 2014. All technical reports were filed on SEDAR.

This technical report summarizes the results of the 2015 PEA study and was prepared following the guidelines of NI 43-101.

1.1.1 Project Concept

The project concept in this PEA is to develop the Romero deposit as an underground mine utilizing longhole and drift & fill mining methods with cemented paste backfill. The mined mineralized rock would be trucked to surface and fed to a nominal 2,500 tonne per day (tpd or t/d) milling and conventional flotation plant capable of producing a copper concentrate also containing gold and silver.

The total planned mine life is approximately 10 years with approximately 8 Mt of mineralized material mined and processed. Tailings will be stored in a dry stack facility approximately 2 km from the Romero deposit, near the Romero South deposit. Romero South is not planned to be mined in this PEA but remains as a significant mineral resource.

Life of mine (LOM) concentrate production is estimated to be 303 kt (dry) of a bulk Cu-Au-Ag concentrate and will be shipped through the port of Haina near Santo Domingo, Dominican Republic for smelting and refining off-shore.

Electrical power for the project is proposed to be provided by the provincial grid.

1.2 Project Description and Ownership

The Romero deposits on the Tireo property are located in the Province of San Juan, Dominican Republic, on the island of Hispaniola in the Greater Antilles of the Caribbean Sea. The deposits are 165 km west-northwest of Santo Domingo, the capital of the Dominican Republic, at geographical coordinates 19° 07' 00" north, 71° 17' 30" west.

GoldQuest owns a 100% interest in the Tireo property and Romero project through its wholly-owned Dominican subsidiary, GoldQuest Dominicana), via GoldQuest Mining (BVI) Corp., a British Virgin Islands company. The Romero project is located within the La Escandalosa exploration concession of the Tireo property which has an area of 3,997.0 hectares (ha).

The concession was granted to GoldQuest on November 9, 2010 and was applied for on May 14, 2010 to replace a previous exploration concession called Las Tres Palmas which was granted on May 30, 2005 and expired on May 30, 2010, shortly after the Phase 3 drill program was completed. There are six granted concessions and 15 concession applications on the Tireo property.

Concession taxes are RD$0.20 (the current exchange rate is approximately RD$45.00 to US$1.00) per hectare per six-month period, equivalent to US$20 per year for La Escandalosa. An exploitation concession may be requested at any time during the exploration stage and is granted for 75 years.

Exploitation properties are subject to annual surface fees and a net smelter return (NSR) royalty of 5%. A 5% net profits interest (NPI) is also payable to the municipality in which mining occurs as an environmental consideration. The 5% NSR is deductible from income tax and is assessed on concentrates, but not smelted or refined products. Income tax payable is a minimum of 1.5% of gross annual proceeds. Value added tax is 18%. The La Escandalosa concession is also subject to a 1.25% NSR royalty in favour of Gold Fields Limited (Gold Fields).

1.3 History, Exploration and Drilling

Mitsubishi Metals Co. Ltd. of Japan carried out regional exploration of the whole Central Cordillera for copper from 1965 to 1971, although there is no record or evidence of any work in the La Escandalosa concession area (Watanabe, 1972; Watanabe et al., 1974).

Exploration & Discovery Latin America (Panama) Inc. (EDLA) formed a joint venture with Gold Fields on June 1, 2003 to carry out a regional exploration program for gold in the Tireo Formation of the Central Cordillera of the Dominican Republic, with EDLA as the initial operator. A regional stream sediment exploration program was carried out between June, 2003 and April, 2004. This program and the preliminary results are described in a paper by Redwood et al. (2006). GoldQuest became the owner of EDLA in April, 2004.

GoldQuest has completed eight phases of drilling from 2006 to 2015 totaling 164 holes and 40,035 m on the Romero Trend. Holes details can be found in Table 10.2.

1.4 Geology and Mineralization

Romero is located on the south side of the Central Cordillera of Hispaniola and is hosted by the Cretaceous-age Tireo Formation volcanic rocks and limestones, which formed in an island arc environment. The deposit geology is a relatively flat lying sequence of intercalated subaqueous, intermediate to felsic volcanic and volcaniclastic rocks and limestones on the west side of thick rhyolite flows or domes. Mineralization is relatively stratabound and flat lying and is mainly hosted by a dacite breccia tuff.

Mineralization outcrops in a number of places were eroded by rivers and streams, and continuity under barren cap rock has been demonstrated by drilling. Hydrothermal alteration and gold mineralization can be traced for over 2,200 m from Romero to Romero South and beyond to the South. The thickness of the altered dacite tuff breccia horizon is up to about 65 m at Romero South and up to more than 200 m (open) at Hondo Valle and Romero. The mineralized horizon is capped by limestone or dacite to andesite lavas, and underlain by rhyolite or limestone.

Mineralization is intermediate sulphidation epithermal in style. The mineralization is associated with quartz-pyrite, quartz-illite-pyrite and illite-chlorite-pyrite alteration. Alteration is generally strongest in the upper part of the mineralized zone and decreases in intensity with depth. Gold mineralization is associated with disseminated to semi-massive sulphides, sulphide veinlets and quartz-sulphides. The sulphides comprise pyrite with sphalerite, chalcopyrite and galena. Oxidation is shallow, to a depth of 10 m to 15 m.

1.5 Metallurgical Testing and Mineral Processing

Metallurgical test programs were completed in 2011, 2013, 2014 and 2015 by ALS Metallurgical Laboratories, Kamloops, B.C. (ALS) on metallurgical composites selected by GoldQuest. The most recent 2015 tests, KM4601, focused on a finer primary grind utilizing gravity separation, reagent dosage optimization, flotation kinetics and other parameters to produce a saleable copper concentrate with gold and silver credits.

The results indicate a 20% copper concentrate grade with a 96.8% copper recovery can be achieved for Romero. The gold and silver recovery with gravity is approximately 75% and 49.8% respectively.

This technical report is based predominantly on the results from program KM4601, although results from relevant earlier work have been utilized where appropriate to develop the design criteria for the operating plant.

The results of the bench scale test work were used to plot best-fit grade recovery curves for each metal. The resulting curves were used to predict the grade and recoveries of copper, gold and silver at the LOM average head grades.

Table 1.1: Projected Metallurgical Balance

Product	Wt (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu Rec (%)	Ag Rec (%)	Au Rec (%)
Copper Concentrate	3.92	20	54	76.9	96.8	49.8	75.0
Tailings	96.1	0.03	2.1	1.0	3.2	50.2	25.0
Feed Material	100.0	0.81	4.25	4.02	100.0	100.0	100.0

Source: JDS 2015

1.6 Mineral Resource Estimates

The mineral resource estimates for the Romero and Romero South deposits on which the PEA is based were most recently reported by Micon in the NI 43-101 Technical Report issued on December 13, 2013.

The mineral resources as estimated by Micon at Romero and Romero South are summarized in Table 1.2.

Table 1.2: Romero Project Mineral Resources

		Tonnes (x 1,000)	Au (g/t)	Cu (%)	Zn (%)	Ag (g/t)	AuEq (g/t)	Au Ounces (x1,000)	AuEq Ounces (x1,000)
Indicated	Romero	17,310	2.55	0.68	0.3	4.0	3.81	1,419	2,123
	Romero South	2,110	3.33	0.23	0.17	1.5	3.80	226	258
Total Indicated Resources		**19,420**	**2.63**	**0.63**	**0.29**	**3.7**	**3.81**	**1,645**	**2,381**
Inferred	Romero	8,520	1.59	0.39	0.46	4.0	2.47	437	678
	Romero South	1,500	1.92	0.19	0.18	2.3	2.33	92	112
Total Inferred Resources		**10,020**	**1.64**	**0.36**	**0.42**	**3.8**	**2.45**	**529**	**790**

Note: AuEq g/t = (Au g/t)+(Ag g/t)/62.222)+(Cu %)/0.642)+(Zn %)/2.1491).

Source: Micon 2013

The present report and mineral resource estimates are based on exploration results and interpretation current as of October 10, 2013. The effective date of the mineral resource estimate is October 29, 2013.

It is Micon's opinion that there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues which exist that would adversely affect the mineral resources presented above. However, the mineral resources presented herein are not mineral reserves as they have not been subject to adequate economic studies to demonstrate their economic viability. They represent in-situ tonnes and grades, and have not been adjusted for mining losses or dilution.

1.7 Mining

Romero is proposed to be mined as an underground operation using a combination of longhole stoping (LH) and drift and fill (DF) underground mining methods with paste backfill to reach a target production rate of 2,500 tonnes per day (t/d) over a mine life of ten years and extract 7.7 Mt of mineralized material. LH stoping will account for about 30% of total production and the remaining 70% will come from DF. The Romero deposit will be accessed from surface via a spiral decline and all mineralized material and waste rock will be trucked out of the mine via this decline. Three ventilation raises will be required in addition to the spiral decline to circulate the required amount of air through the Romero underground workings.

The Romero deposit will be accessed via a spiral decline. A decline was selected over a shaft to provide early access to the mineralized zones and to reduce initial capital. The decline will be used to haul mineralized material and waste and as general access. The decline will also be used as an exhaust airway.

The decline will descend to a final depth of approximately 415 m below surface (685 masl) and will break through on surface directly above the Romero deposit and close to the proposed mill location to minimize mine to mill haulage.

The size of the decline was selected according to required clearances for the chosen mobile equipment and required ventilation during development and production. It was determined that a 4.5 m wide by 5.0 m high profile would be suitable for a 30 t haul truck. The decline will be driven at a -15% gradient. Level access crosscuts and attack ramps are planned to be developed off the decline at a 4.5 m by 5.0 m profile.

Level access crosscuts are designed to be located every 25 m vertically along the spiral decline to provide access to the potentially mineable resource. Attack ramps would provide the access from the access drifts directly to each mining level and would have a maximum gradient of +/- 15 %. Once a given level has been completely mined and backfilled, the back of the attack ramp access is planned to be slashed down and a ramp would be constructed with the slashed rock to access the next cut above.

The mine production schedule is shown in Table 1.3.

Table 1.3: Mine Production Schedule

Year	TOTAL	Year										
		-1	1	2	3	4	5	6	7	8	9	10
Mineralized Material Mined												
Tonnes (kt)	7,737		614	913	913	913	913	913	913	913	540	196
Avg Au Grade (g/t)	4.02		4.21	3.75	5.33	5.21	4.63	3.97	3.20	2.97	3.06	1.80
Avg Ag Grade (g/t)	4.25		3.12	3.36	4.81	4.66	4.45	4.89	3.73	4.10	4.98	4.56
Avg Cu Grade (%t)	0.81		0.70	0.87	0.82	0.91	0.86	0.73	0.76	0.74	0.90	0.87
Ounces Au (koz)	1,000		83	110	156	153	136	116	94	87	53	11
Ounces Ag (koz)	1,056		62	99	141	137	130	143	109	120	86	29
Tonnes Cu (kt)	63		4	8	8	8	8	7	7	7	5	2
Waste Mined												
Tonnes (kt)	903	40	413	98	15	10	32	38	84	56	83	35
Backfill Placed												
Tonnes (kt)	4,158		146	511	521	511	307	350	548	679	375	210

Source: JDS 2015

1.8 Recovery Methods

The concentrator plant will include standard crushing and grinding unit operations and conventional froth flotation to recover mineral concentrates of chalcopyrite (copper sulphide) from the ground mineralized material.

The concentrate will be transported to designated smelters worldwide for subsequent reduction into copper metal. Mill throughput is designed to be approximately 2,500 dry tonnes per day (dt/d). Total annual concentrate production will be approximately 36,000 t.

The mineral processing facility will be located in the north-west area of the Romero mine site. Listed below are the major process unit operations at Romero:

- Primary jaw crusher;
- Crushed stockpile (live capacity 1,000 tonnes);
- Conveyance of material from the crusher building to the stockpile and onto the main process facility;
- Mill building will contain:
  - Semi-autogenous grinding and ball mills and gravity concentration within closed circuit cyclone classification;
  - Copper flotation and concentrate regrinding via stirred mill;
  - Copper concentrate dewatering through thickening and filtration;
  - Process water, fire water, potable water distribution;
  - Reclaim water distribution;
  - Utility air distribution;
  - Tailings dewatering through thickening and filtration;
  - Concentrate load-out; and
  - Reagent storage and reagent mixing.

The primary jaw crusher will be located near the Romero portal. Mineralized material will be delivered by truck from underground and deposited into a dump pocket feeding a jaw crusher. Feed will be crushed to a nominal product size of 80% passing (P_{80}) 150 mm and conveyed to a 1,000 t live stockpile.

The primary grinding will consist of one SAG mill with pebble crusher followed by primary screening. The secondary grinding circuit will consist of a ball mill and gravity concentrator operating in closed circuit with the cyclones.

The cyclone overflow, at approximately 31% solids, and a particle size of (P_{80}) 75 microns, will flow by gravity to the flotation circuit. Copper concentrate will be produced with conventional froth flotation in a typical rougher and cleaner configuration.

The copper rougher concentrate will be reground in a stirred mill to produce a particle size of (P_{80}) 23 microns.

The flotation concentrate and gravity concentrate products will be combined and dewatered in high rate thickeners with the under flow feeding a filter feed stock tank. A dedicated pressure filter will dewater the concentrate to a moisture content of approximately 8%.

The copper concentrate will be loaded into trucks by front end loader and transported to a port for shipment to off-shore smelters/refineries for further processing.

The tailings will be thickened and filtered for either deposition as dry stack tailings or paste backfill underground.

The process plant will operate with 100% reclaim water from the thickener overflows to meet the process water requirements. Fresh water will be required for gland seal and reagent mixing.

1.9 Infrastructure

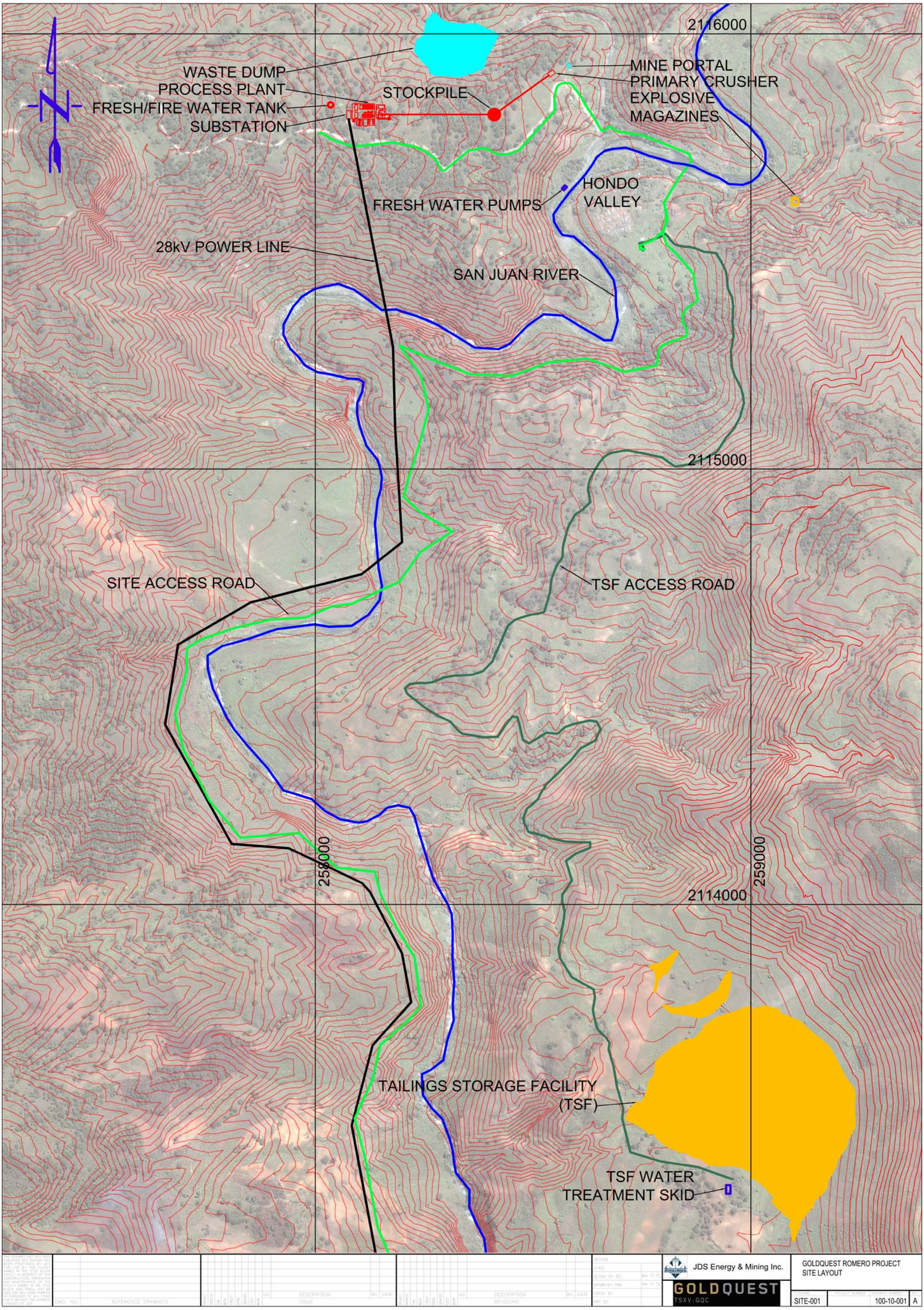
The Romero mine site will be accessed with a new 13.3 km access road. In addition, 4.5 km of existing road will be upgraded to accommodate increased traffic. Asite facility area of approximately 135,000 m^2 will need to be prepared for the substation, water storage, process plant, stockpile, primary crusher, dry stack tailings storage facility (TSF) and all associated conveyors. Electrical power will be supplied via a new 15 km long 28 kV overhead power line from the Sabaneta dam.

Major building installations will include a 2,625 m^2 process plant, a 170 m^2 maintenance shop warehouse, a 225 m^2 truck shop, and bulk explosives storage. A 1,360,000 L combination fresh/firewater tank will supply sufficient fire protection and fresh water to the plant. Potable water and waste water treatment systems will be installed.

Other major infrastructure items include:

- 3.5 km TSF access road;
- 75,000 L diesel storage tank with dispensing unit;
- Dry stack tailings storage facility;
- Emergency backup power generator;
- Communications systems;
- Sewage treatment plant;
- Fresh water pumps;
- Process water tank;
- On-site substation; and
- Upgraded Sabaneta substation.

The site layout is shown in Figure 1.1.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WASTE DUMP
PROCESS PLANT
FRESH/FIRE WATER TANK
SUBSTATION
STOCKPILE
MINE PORTAL
PRIMARY CRUSHER
EXPLOSIVE
MAGAZINES
HONDO
VALLEY
FRESH WATER PUMPS
28kV POWER LINE
SAN JUAN RIVER
2115000
SITE ACCESS ROAD
TSF ACCESS ROAD
258000
259000
2114000
TAILINGS STORAGE FACILITY
(TSF)
TSF WATER
TREATMENT SKID
JDS Energy & Mining Inc.
GOLDQUEST
TSXV:GQC
GOLDQUEST ROMERO PROJECT
SITE LAYOUT
SITE-001
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1.10 Environment and Permitting

Initial baseline environmental studies began in 2013. The project is in close proximity to two National Parks, José del Carmen Ramírez National Park and Armando Bermudez National Park. The project will develop facilities in a manner that does not impact the parks.

The Romero Project is also located on the San Juan and La Guama Rivers, upstream of the Sabaneta reservoir that provides irrigation to downstream agricultural lands. At least three small villages use the San Juan River downstream of the project. Water and waste management planning will need to protect the San Juan River watershed flows and water quality for the surrounding villages and the Sabaneta reservoir users.

The project proposed in this PEA is not expected to require any resettlement. Some land acquisitions will likely be necessary for the proposed tailings facility, mill site, and ancillary facilities.

Permitting of a new mine carries some risk due to the the proximity of the project to a national park and the San Juan and La Guama Rivers. As project plans progress, it will be important to not encroach on the park, to complete thorough and scientifically defensible baseline environmental studies and to conduct an effective engagement and consultation program from the community to the national level.

1.11 Operating and Capital Cost Estimates

The capital cost estimate was prepared using first principles, applying project experience and avoiding the use of general industry factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study. Given that assumptions have been made due to a lack of available engineering information, the accuracy of the estimate and/or ultimate construction costs arising from the engineering work cannot be guaranteed. The target accuracy of the estimate is ±30%.

Table 1.4: Summary of Capital Cost Estimate

Description	Pre-Production (US$M)	Sustaining/Closure (US$M)	Total (US$M)
Mining	14.9	61.5	76.4
Site Development	9.7	0	9.7
Ore Crushing & Handling	7.1	0	7.1
Process Plant	35.6	0	35.6
On-Site Infrastructure	26.1	0	26.1
Tailings & Waste Rock Management	2.6	6.6	9.2
Project Indirects	9.9	0	9.9
Engineering & EPCM	12.7	0	12.7
Owner's Costs	3.1	0	3.1
Closure	0	19	19
Subtotal	**121.7**	**87.1**	**208.8**
Contingency (20%)	21.4	5.1	26.5
Total Capital Costs	**143.0**	**92.3**	**235.3**

Source: JDS 2015

Table 1.5: Summary of Operating Cost Estimate

Operating Costs	$/t milled	Life of Mine (US$M)
Mining	29.60	229.0
Processing	15.53	120.2
Tailings	2.64	20.5
G&A	5.00	38.7
Total	**52.78**	**408.3**

Source: JDS 2015

1.12 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities to the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted that the tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations to represent an indicative value of the after-tax cash flows of the Romero project.

1.12.1 Main Assumptions

Main economic and smelter return assumptions are summarized in Tables 1.6 and 1.7.

Table 1.6: Economic Assumptions

Item	Unit	Value
NPV Discount Rate	%	6
Corporate Income Tax Rate	%	27
Asset Tax	%	0.5
Export Withholding Tax	%	5
Local Community Tax	%	5
Declining Balance Depreciation Rate	%	15
Equity Finance	%	0
Capital Contingency (Overall)	%	20

Source: JDS 2015

Table 1.7 Net Smelter Return Assumptions

NSR Parameters	Unit	Cu Concentrate
Smelter Payables		
Cu Payable	%	96.5
Au Payable	%	97.5
Ag Payable	%	90
Minimum Deduction in Conc	%	1
Au Minimum Deduction	g/t	0.6
Ag Minimum Deduction	g/t	20
TC/RCs		
Treatment Charge	$/dmt conc	0.085
Cu Refining Charge	US $/lb	0.09
Au Refining Charge	US $/oz	6
Ag Refining Charge	US $/oz	0.5
Transport Costs		
Moisture Content	%	8
Transport to Port	US$/wmt conc	$100.00
Total	**US$/wmt conc**	**$100.00**
	US$/dmt conc	**$108.70**

Source: JDS 2015

1.12.2 Results

This preliminary economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Economic Results are shown in Table 1.8.

Table 1.8 Economic Results*

Results	Unit	Value
Gross Revenues	US$M	1,174
LOM Pre-Tax Free Cash Flow	US$M	530
Average Annual Pre-Tax Free Cash Flow	US$M/yr	58
Pre-Tax $NPV_{6\%}$	US$M	355
Pre-Tax IRR	%	46
Pre-Tax Payback	Years	2.3
NPV to Pre-Production CAPEX	times	2.5
Taxes	US$M	187
LOM After-Tax Free Cash Flow	US$M	343
Average Annual After-Tax Free Cash Flow	US$M/yr	37
After-Tax $NPV_{6\%}$	US$M	219
After-Tax IRR	%	34
After-Tax Payabck	Years	2.7
Break-Even Au Price‡	US$/Au oz	628
Cash Cost*	US$/Au oz	813
Cash Cost Net of By-Products**	US$/Au oz	572

(‡) Based on constant Cu price of US$2.90/lb

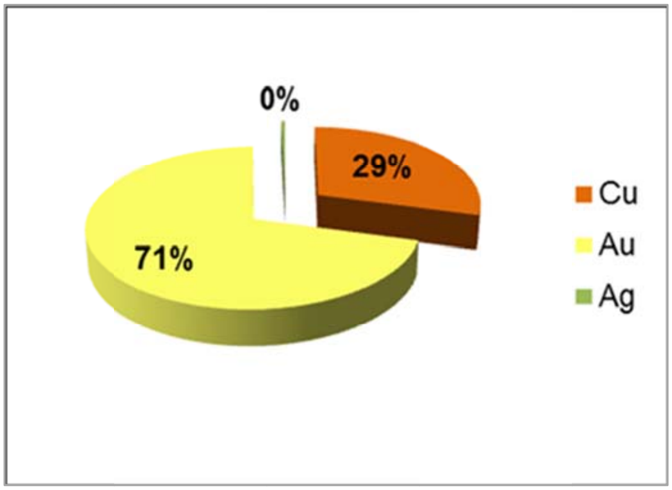
(*) Cash Cost = (Treatment Charges + Refining Charges + Royalties + Operating Costs + Sustaining & Closure Capital Costs)/Payable Au oz

(**) Cash Cost Net of By Products = ((Treatment Charges + Refining Charges + Operating Costs + Sustaining & Closure Capital Costs) – (Payable Cu lbs * 2.90/lb) – (Payable Ag oz * $17/oz)) / Payable Au oz

Source: JDS 2015

The contribution by metal to the project economics are shown in Figure 1.2.

Figure 1.2: Life of Mine Payable Metal by Value



Source: JDS 2015

1.12.3 Sensitivities

Sensitivity analyses were performed using metal prices, mill head grade, CAPEX, and OPEX as variables. The value of each variable was changed plus and minus 10% independently while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1.9.

Table 1.9: Sensitivities Analysis on After-Tax Results

After-Tax $NPV_{6\%}$ (US$M)			
Variable	**90%**	**100%**	**110%**
Metal Price (Combined)	156.7	219.1	281.6
Cu Price	201.1	219.1	237.2
Au Price	175	219.1	263.3
Head Grade	160.3	219.1	278
OPEX	239.5	219.1	198.8
CAPEX	235.7	219.1	202.6

Source: JDS 2015

1.13 Interpretation and Conclusions

Industry standard mining and processing methods were used in this PEA. Sufficient information and data was available to the QPs for a PEA-level study and the goal of producing a NI 43-101 compliant PEA study was achieved.

The preliminary economic results, based on the assumptions highlighted in this report, show a positive outcome.

It is important to note that this result is only preliminary and could change significantly as more information is gathered and market conditions change. This assessment includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

The QPs of this report recommend that the Romero project be advanced to a preliminary feasibility study level (PFS).

1.14 Recommendations

It is recommended that Romero proceed to the preliminary feasibility study stage in line with GoldQuest's desire to advance the project. It is also recommended that environmental and permitting continue as needed to support Romero project development plans.

It is estimated that a PFS and supporting field work would cost approximately $3.9 M. A breakdown of the key components of the next study phase is as follows in Table 1.10.

Table 1.10: Cost Estimate to Advance Romero to PFS Stage

Component	Estimated Cost (US$M)	Comment
Resource Drilling & Updated Resource	0.6	Conversion of inferred resources to indicated within and immediately adjacent to the proposed mine. Drilling will include holes for combined resource, geotech and metallurgical purposes.
Metallurgical Testing	0.2	Variability test work including expanded comminution, grinding, flotation and filtration testwork as well as multi-element ICP tailings and concentrate analysis.
Access Road	0.1	Reconnaissance, test pitting, borrow source indentification and road design
Backfill Testing	0.1	Paste backfill testing including tailings characterization, rheology, strength tests
Geotechnical/ Hydrology/Hydrogeology	0.5	Mine and surface facilities geotechnical investigations (logging, test pitting, sampling, lab tests, etc.)
Engineering & Design	1.5	PFS-level mine, infrastructure, tailings storage, paste backfill & process design, cost estimation, scheduling & economic analysis
Environment	0.1	Other investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology
Total	**3.1**	Excludes corporate overheads and future permitting activities

Source: JDS 2015

2 Introduction

2.1 Basis of Technical Report

This Technical Report was compiled by JDS for GoldQuest. This technical report summarizes the results of the 2015 PEA study and was prepared following the guidelines of NI 43-101.

2.2 Scope of Work

This report summarizes the work carried out by the consultants and the scope of work for each company is listed below, and combined, makes up the total project scope.

JDS scope of work included:

- Compile the technical report which includes the data and information provided by other consulting companies;
- Underground mine design and planning;
- Design required site infrastructure, identify proper sites, plant facilities and other ancillary facilities;
- Implement and supervise 2015 metallurgical testing program;
- Develop a conceptual flowsheet, specifications and selection of process equipment;
- Establish recovery values based on metallurgical testing results;
- Design processing to realize the predicted recoveries;
- Estimate mining, process plant and infrastructure OPEX and CAPEX for the Project;
- Prepare a financial model and conduct an economic evaluation including sensitivity and project risk analysis; and
- Interpret the results and make conclusions that lead to recommendations to improve value, reduce risks.

Micon scope of work included:

- Project setting, history and geology description;
- Sample preparation and data verification; and
- Mineral resource estimate.

2.3 Qualifications, Responsibilities and Site Visits

The results of this PEA are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between GoldQuest and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional associations. The QPs are responsible for the specific report sections as follows:

Table 2.1: QP Responsibilities

QP	Company	Report Section(s)	Site Visits
Michael Makarenko, P.Eng.	JDS Energy & Mining Inc.	1, 2, 3, 15, 16, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27, 28, 29	April 6-18, 2015
Kelly McLeod., P.Eng.	JDS Energy & Mining Inc.	13, 17	Did not visit site
B. Terrence Hennessey, P.Geo.	Micon International Limited	4, 5, 6, 7, 8, 9, 10, 11, 12, 14	January 9-12, 2013

Source: JDS 2015

The Romero project is in an exploration stage and a site visit by Kelly McLeod, P. Eng. was not necessary to complete this PEA. Ms. McLeod relied on information and knowledge from GoldQuest and JDS.

2.4 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or "metric" except for Imperial units that are commonly used in industry (e.g., ounces (oz.) and pounds (lb.) for the mass of precious and base metals).

All dollar figures quoted in this report refer to United States (US$ or $) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 29. This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

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

2.5 Sources of Information

The sources of information include data and reports supplied by GoldQuest personnel as well as documents cited throughout the report and referenced in Section 28. In particular, background Property information was directly taken from the 2013 Mineral Resource Estimate and 2014 Micon PEA.

All tables and figures are sourced from JDS, unless otherwise indicated.

3 Reliance on Other Experts

The Qualified Person's opinions contained herein are based on information provided by GoldQuest and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

The tailings management facility sub-section 18.18 was provided by SRK Consulting (Canada) Inc. (SRK). Michael Makarenko, P. Eng., reviewed this sub-section and assumed responsibility for its content.

The Qualified Person's used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending.

The various agreements under which GoldQuest holds title to the mineral lands for this project have not been thoroughly investigated or confirmed by the authors and no opinion is offered as to the validity of the mineral title claimed. The descriptions were provided by GoldQuest.

The description of the property is presented here for general information purposes only, as required by NI 43-101. The authors are not qualified to provide professional opinion on issues related to mining and exploration lands title or tenure, royalties, permitting and legal and environmental matters. Accordingly, the authors have relied upon the representations of the issuer, GoldQuest, for Section 4 of this report, and have not verified the information presented therein.

4 Property Description and Location

This section was taken from the 2014 Micon PEA Report (Preliminary Economic Assessment for the Romero Project, Tireo Proerty, Province of San Juan, Dominican Republic – May 27, 2014).

4.1 Property Location

The Tireo property, and the contained Romero project, is located in the Province of San Juan, Dominican Republic, on the island of Hispaniola in the Greater Antilles of the Caribbean Sea. Romero is 165 km west-northwest of Santo Domingo, the capital of the Republic, and 35 km north of San Juan de la Maguana, the capital of the Province (Figure 4.1). The geographical coordinates of GoldQuest's Hondo Valle Camp servicing the Romero project are 19° 07' 00" north, 71° 17' 30" west, and the Universal Transverse Mercator (UTM) coordinates are 258,730 east, 2,115,543 north (North American Datum 1927 (NAD 27) Conus (Continental USA), Zone 19Q).

Figure 4.1: Location Map of the Romero Project and La Escandalosa Concession

Source: GoldQuest (2010)

4.2 Property Description

4.2.1 Property Status

GoldQuest owns a 100% interest in the Tireo property and Romero project through its wholly owned Dominican subsidiary INEX Ingeniería y Exploración, S.R.L. (INEX). INEX, now called GoldQuest Dominicana, is owned by GoldQuest Mining (BVI) Corp., a British Virgin Islands company, which is, in turn, wholly owned by GoldQuest. The Romero and Romero South deposits are located on the La Escandalosa exploration concession which has an area of 3,997.0 ha and is shown on a map in Figure 4.2. It was granted on November 9, 2010. The concession was applied for on May 14, 2010 to replace a previous exploration concession called Las Tres Palmas which expired on May 30, 2010, shortly after the Phase 3 drill program was completed. Under Dominican mining law it is permitted to re-apply for an exploration concession between 30 and 1 day(s) before the expiry of an existing concession.

The concession is part of the Tireo property in San Juan owned by GoldQuest. It comprises of 15 exploration concessions or applications: La Escandalosa, Loma Los Comios (formerly called Los Comios), Descansadero (formerly called Los Chicharrones), Los Lechones (formerly called La Bestia), Aguita Fria (formerly called Jengibre), Loma El Cachimbo (formerly called Loma Viejo Pedro), Los Gajitos (formerly called El Crucero), Valentin (formerly called El Barrero), La Tachuela (formerly callled La Fortuna), Tocon de Pino, Las Tres Veredas, Patricio, Piedra Dura, Toribio and La Pelada. (See Table 4.1 and Figure 4.3).

Table 4.1: Description of Tireo Property Exploration Concessions

Name	Status	Area (ha)	Application Date	Title Date	Mining Registry Date	Resolution Number	Expiry Date
Las Tres Palmas / La Escandalosa	Granted (40494)	3,997	14-May-10	9-Nov-10	12-Nov-10	IV-10	9-Nov-15
Los Comios / Loma Los Comios	Granted (41579)	2,028	1-Oct-12	1-Dec-13	11-Nov-13	VI-13	1-Nov-18
La Bestia/ Los Lechones	Granted	550	5-Jul-13	30-Dec-14	15-Jan-15	II-15	30-Dec-19
Jengibre / Aguita Fria	Under Reapplication	1,384	5-Jul-13				
Loma Viejo Pedro / Loma El Cachimbo	Under Reapplication	3,514	21-Dec-09				
Los Chicharrones / Descansadero	Granted (41621)	725	25-Oct-12	13-Dec-13	8-Jan-14	II-14	13-Dec-18
El Crucero / Los Gajitos	Granted	370	1-Oct-12	15-Oct-19	7-Nov-14	III-14	15-Oct-19
El Barrero/Bartola/ Valentin	Under Reapplication	300	25-Oct-12				
Tocón de Pino	Under Application	744	17-Nov-08				
Las Tres Veredas	Granted	790	20-Jun-12	1-Dec-2014	8-Jan-2014	I-15.	1-Dec-2019
Patricio / La Guinea	Under Application	2,768	12-Feb-14				
Piedra Dura	Under application	362	21-Apr-14				
La Tachuela/ La Fortuna	Under application	330.25	21-Apr-14				
Toribio	Under application	2,351.45	29-May-14				
La Pelada	Under application	631	29-May-14				
Total		**20,884.7**					

Source: GoldQuest, 2015

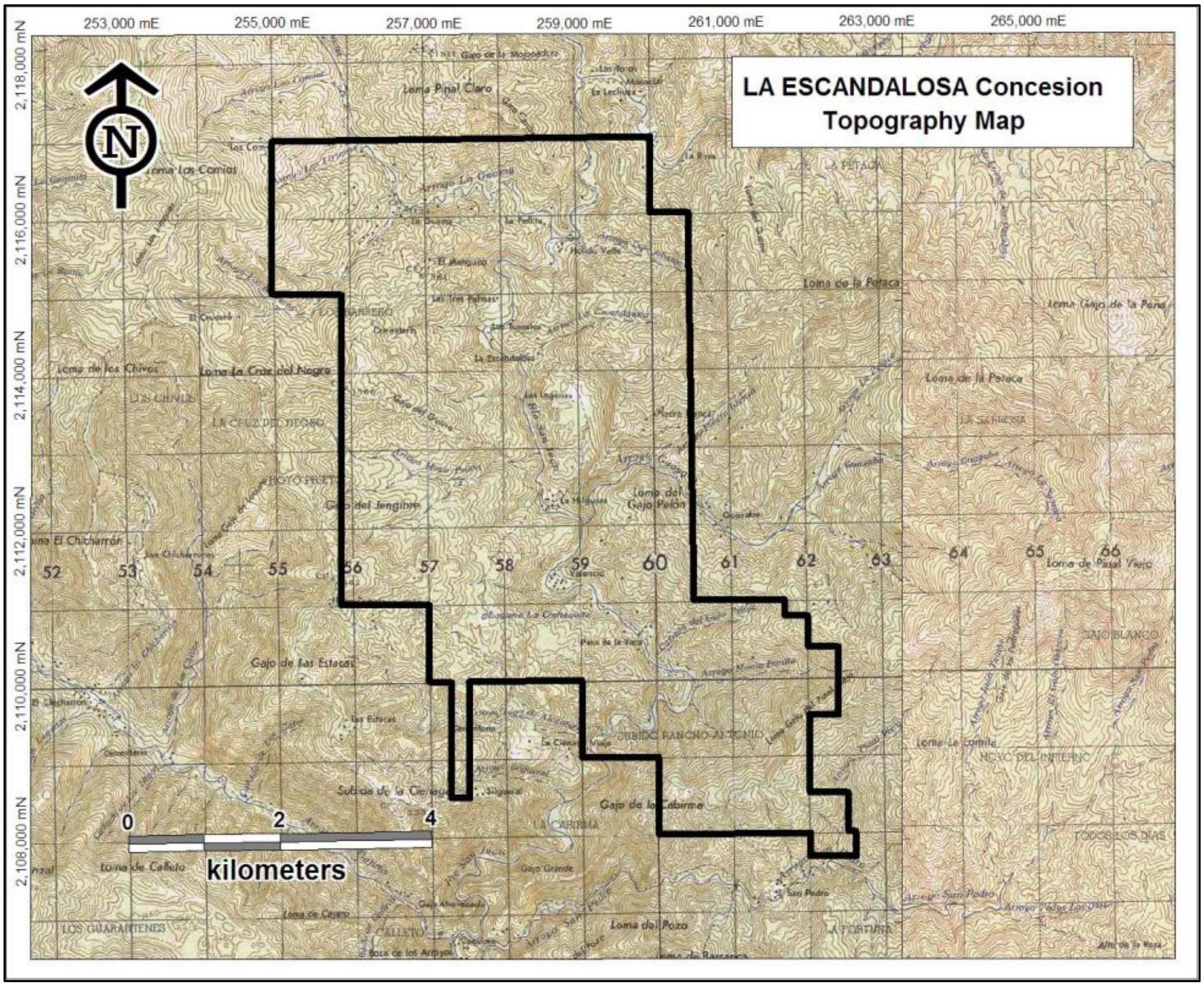
Concession taxes are RD$0.20 (twenty Dominican centavos equal to about US$0.0044 or 0.44 US cents at the current exchange rate of RD$45 to US$1.00) per hectare per 6-month period, equivalent to about US$20 per year for La Escandalosa. An exploitation concession may be requested at any time during the exploration stage and is granted for 75 years.

Exploitation properties are subject to annual surface fees and a net smelter return royalty of 5%. A 5% net profits interest is also payable to the municipality in which mining occurs as an environmental consideration.

The 5% NSR is deductible from income tax and is assessed on concentrates, but not smelted or refined products. Income tax payable is a minimum of 1.5% of gross annual proceeds. The value added tax is 18%.

The concession is also subject to a 1.25% NSR royalty in favour of Gold Fields Limited. More detail on taxes and royalties is provided below.

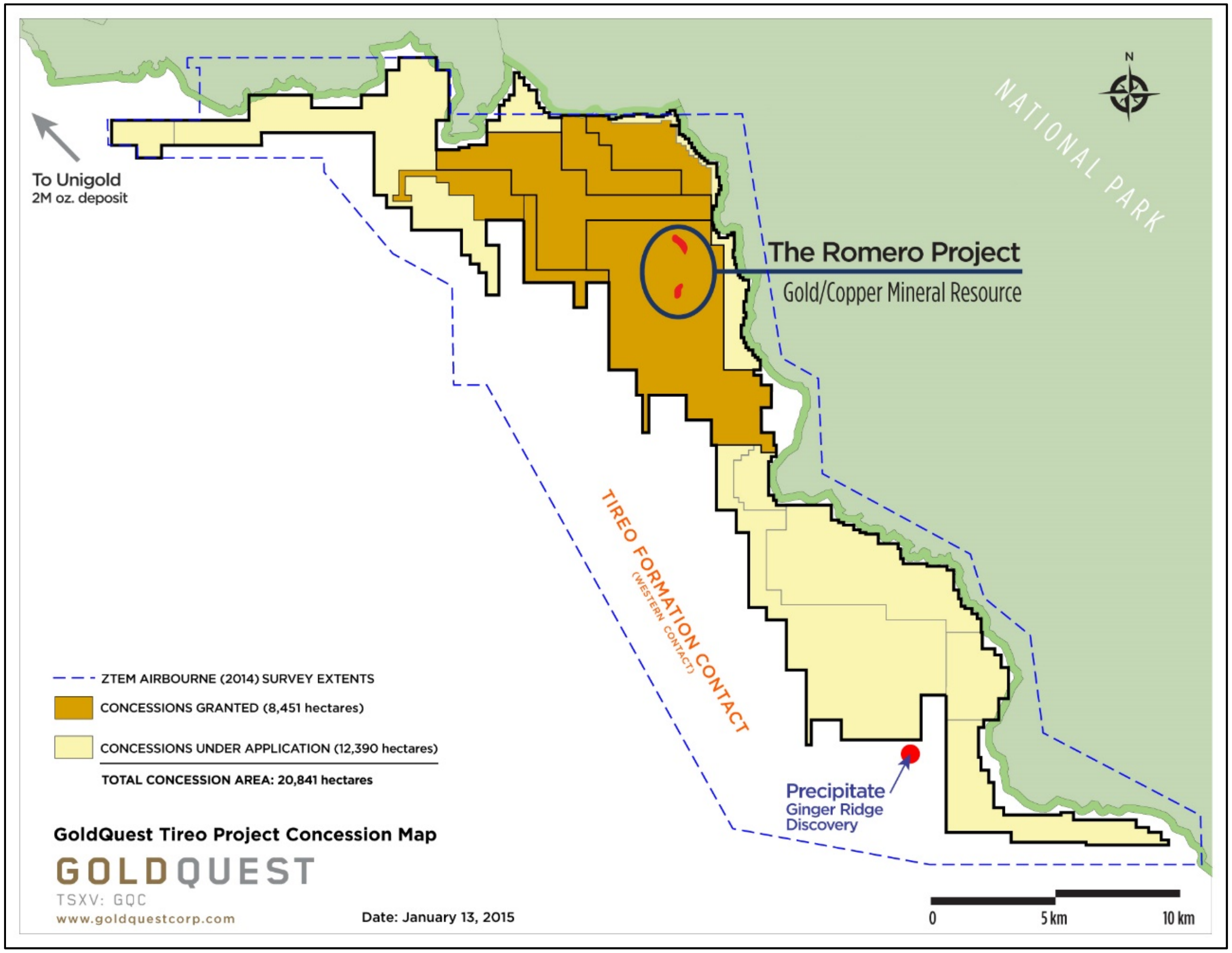
Figure 4.2: Map of La Escandalosa Exploration Concession



(1:50,000 topographic map, 1 km grid squares); , grid is UTM NAD27 Conus

Source: GoldQuest (2010)

Figure 4.3: Map of the Tireo Property, Including La Escandalosa Concession



Source: GoldQuest (2015)grid is UTM NAD27 Conus

4.2.2 Property Legal History

GoldQuest's subsidiary company Exploration and Discovery Latin America (Panama) Inc. (EDLA), a private company registered in Panama, started exploring for gold in the Dominican Republic in 2001, through its subsidiary INEX. Later in 2001, EDLA was acquired by MinMet plc (MinMet), a company registered in Dublin, Ireland, and whose shares were traded on the Irish Venture Exchange and, later, also on the Alternative Investment Market (AIM) of the London Stock Exchange. In 2004, MinMet spun off EDLA and its Dominican Republic assets into Wellington Cove Explorations Ltd., a company registered in Canada, by means of a reverse takeover with a name change to GoldQuest Mining Corp. This was followed by an application to list the shares for trading on the TSX Venture Exchange (TSXV) of the Toronto Stock Exchange (TSX).

EDLA formed a joint venture with Gold Fields on June 1, 2003 to carry out a regional exploration program for gold in the Tireo Formation of the Central Cordillera of the Dominican Republic, with EDLA as the initial operator. This program led to the discovery of mineralization at La Escandalosa (now known as the Romero South deposit) in late 2003.

The Las Tres Palmas exploration concession was staked by INEX on December 13, 2003 and a formal application was made on May 18, 2004. Title was granted on May 30, 2005 and was valid for three years until May 30, 2008, with two extensions of one year each being granted which extended the title up to May 30, 2010. The concession was originally held in the name of Minera Duarte S.A., a Dominican corporation which was also owned by GoldQuest, and it was transferred to INEX in November, 2006 as part of an internal corporate reorganization.

On January 31, 2006 GoldQuest entered into a Joint-Venture Letter of Intent (LOI) with Gold Fields to explore certain properties in the Dominican Republic, including Las Tres Palmas, Los Comios, Los Chicharrones, La Bestia, El Crucero, Loma Viejo Pedro and Jengibre. The LOI superseded all prior agreements with Gold Fields. The terms of the LOI were formalized in a Mining Venture Agreement which was signed in March, 2007 with an immediate effective date.

Under the terms of the agreement, Gold Fields had the right to earn a 60% interest in the selected projects held by GoldQuest in the Dominican Republic by expending US$5 million over three years. Gold Fields assumed direct project management on May 31, 2007.

Subsequent to vesting its 60%, Gold Fields had the right to choose up to four projects whereby it could earn an additional 15% by expending a further US$5 million on each. GoldQuest had the right to maintain a 40% interest in one of the designated projects of its choice by fully funding its share of expenditures up to bankable feasibility study. At GoldQuest's election, upon completion of the additional 15% earn-in, Gold Fields would arrange funding of GoldQuest's proportionate share of subsequent development and construction expenditures. In return, Gold Fields would be granted an additional 5% interest in the specific project (to 80%) and the funding would be deemed a loan, payable out of 90% of GoldQuest's profits from production. In the case of GoldQuest contributing on one project to bankable feasibility study, Gold Fields could earn an extra 5% (i.e. to 65%) by arranging funding of GoldQuest's proportionate share of the subsequent bankable feasibility study.

Development and construction expenditures and the funding would be deemed a loan, payable out of 90% of GoldQuest's profits from production.

On November 26, 2008, Gold Fields advised GoldQuest that it had completed its US$5 million expenditure requirement and had earned a 60% interest in the properties. Gold Fields also informed GoldQuest that it had chosen not to proceed with any further exploration in the Dominican Republic.

On August 5, 2009, GoldQuest entered into a purchase agreement with Gold Fields Dominican Republic BVI Limited to purchase Gold Fields' 60% interest of the Dominican Joint Venture and thereby regain 100% ownership of the properties. The purchase price was the issue of 8.6 million shares in GoldQuest from treasury, representing approximately 12.3% of the issued and outstanding common share capital of GoldQuest at that date, and the grant of a 1.25% NSR royalty on the properties. The transaction was closed on November 18, 2009.

In 2009, GoldQuest reorganized its subsidiaries through a new British Virgin Islands (BVI) company, GoldQuest Mining (BVI) Corp. (GQC-BVI), which became the owner of INEX. The Panamanian subsidiaries EDLA and GoldQuest (Panama) Inc. were subsequently wound up. In 2010 INEX changed from a Public Limited Company (Sociedad Anónima or S.A.), INEX, Ingeniería y Exploración, S.A., to a Limited Liability Company (Sociedad de Responsibilidad Limitada or S.R.L.), INEX, Ingeniería y Exploración, S.R.L. On August 15th, 2014, INEX changed its name to GoldQuest Dominicana.

The Las Tres Palmas concession expired on May 30, 2010, shortly after the Phase 3 drill program was completed. INEX (now known as GoldQuest Dominicana) applied for the La Escandalosa exploration concession to replace Las Tres Palmas on May 14, 2010. It was granted on November 9, 2010.

4.3 Dominican Republic's Mining Law

Mining in the Dominican Republic is governed by the General Mining Law No. 146 of June 4, 1971, and Regulation No. 207-98 of June 3, 1998. The mining authority is the General Mining Directorate (Dirección General de Minería - DGM) which is part of the Ministry of Energy and Mines as of July 30th, 2013 governed by law 100-13.

The properties are simply known and recorded in their respective property name under a Licence of Metallic Exploration Concession. Title is valid for three years. Two separate one-year extensions are allowed. After five years the concessions may be reapplied for giving the concessions a further three to five years. Concession taxes are 20 Dominican centavos (RD$ 0.20) per hectare, per six-month period for concessions between 1,000 and 5,000 ha in size, equivalent to about US$0.0044 per hectare per year (at the current exchange rate of RD$45 to US$1.00). The taxes are paid every six months during the first weeks of January and June. Due to the small amounts involved, the full yearly amount is paid at the start of the year. A report has to be submitted to the DGM every six months summarizing the work completed during the previous six months, work plans and budget for the next six months, and any geochemical data. There is no specified level of work commitment per concession.

The concessions have not been surveyed, however, the claim owner, GoldQuest Dominicana, has erected a reference monument centrally within the property, as required in the claim staking process, and this is surveyed by the DGM. A detailed description of the staking procedure follows:

- The claim system revolves around one principal survey Departure Point (Punto de Partida or PP), as opposed to staking all corner points with a physical stake as would be done in Canada;
- Three types of survey points need to be calculated, a Departure Point (PP), a Reference Point (Punto de Referencia or PR) and three visually recognizable Visual Points (Visuales, V1, V2 and V3);
- The PP point is a visual point from which the proposed claim boundary point can be clearly seen by line of sight. The PP point is usually a topographic high with a distance to the proposed claim boundary greater than 100 m;
- From the PP point a second point, the PR is selected. The PR point is usually another topographic high or a distinctive topographic feature such as river confluence or a road/trail junction. The bearing and distance between the PP and PR points are calculated and tabulated;
- From the PR point three separate visually identifiable points, V1, V2 and V3, are selected, usually distinctive topographic feature such as confluences of rivers or road/ trail junctions. The bearing and distances between the PR point and three visual points, V1, V2 and V3, are calculated and tabulated;
- From the PP point the distance to the proposed claim boundary a north-south or east-west line of not less than 100 m is calculated. The corner points of the claim are calculated from the point at which this line intersects the claim boundary. The corner points (Puntos de connección) are defined by north-south or east-west lines from the point at which the line intersects the boundary and then from each other until the boundary is completed. There is no limit to the number of points that can be used and no minimum size of claim;
- A government surveyor is sent out to review all survey points in the field after legal and fiscal verification of the claim application by the mines department;

The exploration concession grants its holder the right to carry out activities above or below the earth's surface in order to define the areas containing mineral deposits by using any technical and scientific methods. For such purposes the holder may construct buildings, install machinery, communication lines and any other equipment that the exploration work requires. No additional permitting is required until the drilling stage, which requires an environmental permit;

An exploitation concession may be requested at any time during the exploration stage, and this grants the right to prepare and extract all mineral substances found in the area, allowing the beneficiary to exploit, smelt and use the extracted materials for any business purpose. This type of concession is granted for a period of 75 years.

Exploitation properties in the Dominican Republic are subject to annual surface fees and a net smelter return of 5%. A 5% net profits interest is also payable to the municipality in which mining occurs as an environmental consideration. The value added tax is 18%.

The NSR is deductible from income tax and is assessed on concentrates, but not smelted or refined product. Income tax payable is a minimum of 1.5% of annual gross proceeds (Pellerano and Herrera, 2001).

4.4 Environmental Regulations and Liabilities

The environment is governed by the General Law of the Environment and Natural Resources No. 64-00 of August 18, 2000. The environmental authority is the Vice-Minister of Environmental Affairs of the Ministry of the Environment and Natural Resources (formerly called the Subsecretary of Environmental Affairs of the Secretary of State of the Environment and Natural Resources until 2010).

An environmental permit is required for trenching and drilling. The main steps in the procedure to obtain this are as follows:

- Complete the Prior Analysis Form with the project data including name of the project, name of the company, location on a 1:50,000 scale map and name of the legal representative;
- Present a description of the planned work including type of equipment to be used, size of the drill platforms, amount of water that will be required, environmental management plans for fuel, oil and grease, and recirculation of water;
- Obtain authorization of the land owners with copy of property title;
- Pay a tax of RD$5,000.00 (about US$118, using a US$0.02:RD$1 F/X rate);
- Obtain a copy of the Resolution of the exploration concession title; and
- Provide UTM coordinates of the vertices of the exploration concession.

GoldQuest obtained the required permits for the different phases of trenching and drilling at the La Escandalosa concession.

Water Management Consultants Ltda., of Santiago, Chile carried out a hydrological and hydrochemical baseline survey at La Escandalosa in 2006 (Water Management Consultants, 2006). Currently the company is working with AMEC to monitor on-going baseline studies.

GoldQuest carried out trenching by hand. The trenches were back filled and re-vegetated. The company used man-portable drill rigs for all drilling phases. No access roads were made. The rigs were moved using existing roads, and then by hand on footpaths to the drill sites. Drill platforms were cut by hand where necessary, and were back filled and re-vegetated after drilling was finished. Sumps were dug by hand to allow settling of rock cuttings and drill mud from returned drill water, and were subsequently filled in and re-vegetated.

An archaeological survey has not been carried out.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

This section was taken form the 2014 Micon PEA Report.

5.1 Accessibility

The Romero and Romero South deposits are located on GoldQuest's Tireo property in the Province of San Juan, Dominican Republic. The property is situated 165 km west-northwest of Santo Domingo, the capital of the Republic, and 35 km north of San Juan de la Maguana, the capital of the Province and nearest large town (urban population 145,885 in 2008, see Figure 4.1). The geographical coordinates of GoldQuest's field camp at the village of Hondo Valle on the La Escandalosa concession are 19° 07' 00" north, 71° 17' 30" west, and the Universal Transverse Mercator coordinates are 258,730 east, 2,115,543 north (datum NAD 27 Conus, Zone 19Q).

The total distance by road from Santo Domingo to Hondo Valle is 240 km and takes five to six hours by four-wheel drive vehicle. The route is summarized in Table 5.1 and is described in the following paragraphs.

Table 5.1: Summary of the Road Access to the Romero Project

From	To	Road Type	Distance (km)	Time (hours)
Santo Domingo	San Cristóbal	Route 6, multi-lane, paved	28	0 h 30 m
San Cristóbal	Cruce de Azua	Route 2, Sánchez Highway, multi- and 2 lane, paved	99	1 h 10 m
Cruce de Azua	San Juan	2 lane, paved	64	0 h 45 m
San Juan	Sabaneta	Minor, paved	20	0 h 30 m
Sabaneta	Boca de los Arroyos	Minor, unpaved	12.7	0 h 30 m
Boca de los Arroyos	Hondo Valle	Track, unpaved	16.3	1 h 35 m
Total			240	5 h 0 m

Source: Micon 2014

Flying time to the project, by helicopter from Santo Domingo, is one hour and helicopters can land at Hondo Valle and other points in the project area.

Access from Santo Domingo is by multi-lane highway to San Cristóbal (Route 6, 28 km, 30 minutes), then the two-lane highway (Route 2 or the Sánchez Highway) via Baní (32 km, 30 minutes); Azua de Compostela (52 km, 40 minutes) and the Cruce de Azua (Azua Turning – 15 km, 10 minutes), and from there to San Juan de la Maguana (64 km, 45 minutes).

From San Juan, a minor paved road goes north through the villages of Juan de Herrera, La Maguana and Hato Nuevo to Sabaneta (20 km, 30 minutes) at the Sabaneta Dam.

From there an unsurfaced road in generally poor condition is taken along the west side of the reservoir through the communities of Ingeñito and La Lima to Boca de los Arroyos (12.7 km, 30 minutes), which is the end of the useable road for most trucks.

From Boca de los Arroyos an unsurfaced dirt road in very poor condition goes north to Hondo Valle (16.3 km, 1-hour plus) and is only passable by four-wheel drive vehicles when dry. This road has very steep grades and climbs over 1,000 m up to 1,712 m altitude on the ridge of Subida de la Ciénaga, including a 663 m climb in a 2.0 km distance (average 1 in 3-grade). The road then proceeds along the ridges of Gajo de las Estacas (1,606 m altitude), Hoyo Prieto (1,562 m altitude), Gajo del Jenjibre and Loma La Cruz del Negro (1,712 m altitude).

The ridges are covered in saprolite and the ridge-top road becomes very slippery to impassable when heavy rains occur. The road from Boca de los Arroyos to Hondo Valle was built in 2000 and was reopened by GoldQuest in 2004. It requires continual maintenance to keep open. A 2.9 km branch from this road was later completed from the Subida de la Ciénaga to La Higuera village, but this route still has the very steep initial climb from Boca de los Arroyos. A 5-km section of road was recently completed by the Catholic church, from Hondo Valle directly to La Higuera on the east side of the San Juan river, creating a complete circle route. This road can be used to access both the Romero and Romero South deposits. There are no other roads in the concession area and access is by foot or mule. Figure 5.1 shows the village of Hondo Valle, GoldQuest's field camp and core storage area (yellow arrow) and a red ellipse outlining the approximate location of the Romero deposit. The San Juan River flows through the foreground.

Figure 5.1: Hondo Valle Camp and Village, Looking North

Source: GoldQuest
Red ellipse shows approximate location of Romero deposit. Yellow arrow shows camp.

The Romero South deposit is located approximately 950 m south of Romero under a small plateau on the east side of the San Juan River. A view of the landscape around Romero South can be seen in Figure 5.2. The canyon of the San Juan River lies beyond the plateau.

Figure 5.2: View of Las Lagunas Plateau Looking Southwest

The drill rig is on hole LTP-24, blue spot under the yellow arrow.
The red ellipse shows the approximate location of the Romero South deposit.

Source: GoldQuest

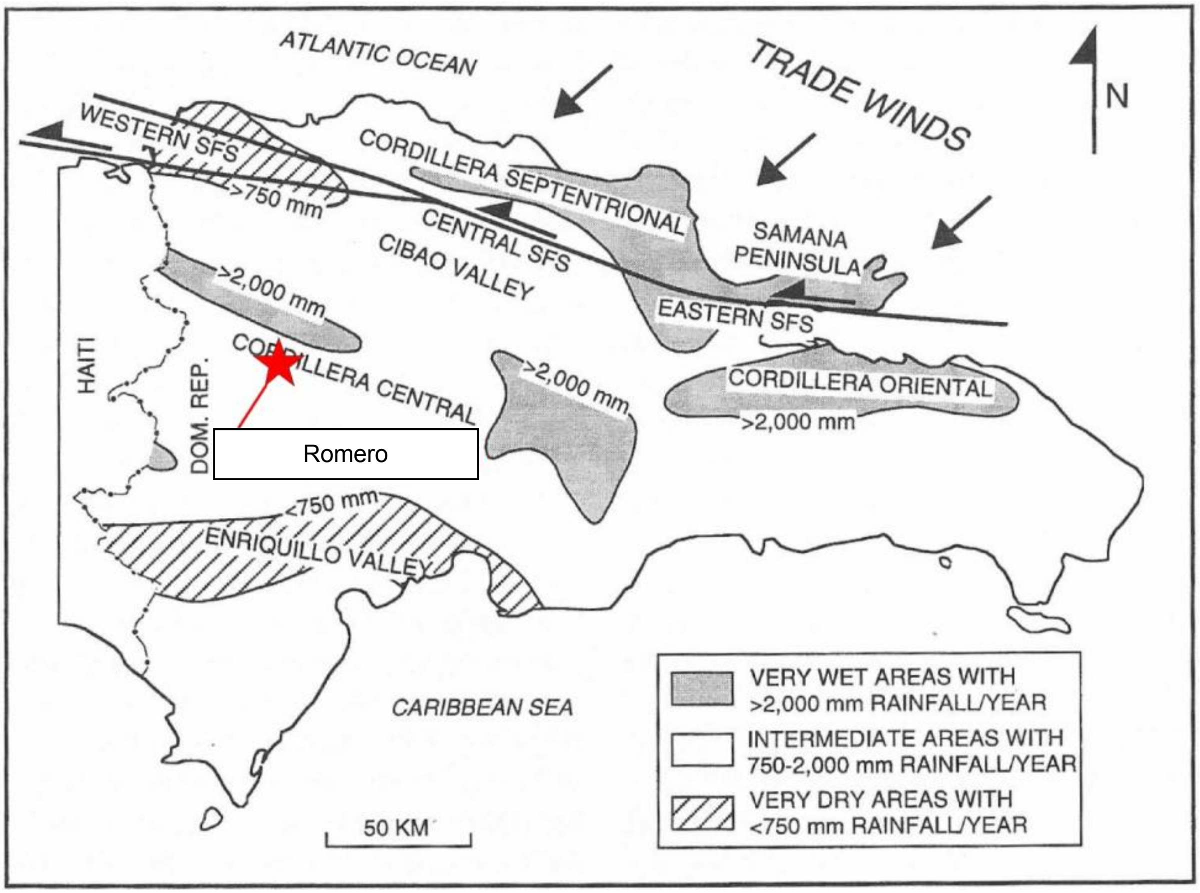
5.2 Climate

The climate in the Romero area is temperate to hot at lower elevations (below 1,000 masl). Northeast trade winds from the Atlantic Ocean bring moisture to the island with the highest rainfall on the northeast side of the Central Cordillera and a rain shadow in the San Juan Valley (see Figure 5.3). The nearest climatic data available are for San Juan, 25 km to the south at a lower altitude of 400 m. The average annual rainfall there is 961 mm with 91.5 days of rain per year mostly between May and October, and an average temperature of 24.9°C. There is a dry season from December to March and a rainy season from April to November (García and Harms, 1988). The climate at Hondo Valle is wetter and cooler. Precipitation increases from south to north in the Central Cordillera from 970 to 1,800 mm per year, with a corresponding temperature decrease from 24°C to 18°C related to increasing altitude (Bernárdez and Soler, 2004).

As part of a baseline monitoring program, GoldQuest has recently established a weather station at Hondo Valle and is gathering more detailed data (wind velocity, precipitation, temperature and atmospheric pressure).

The country is prone to hurricanes with September being the peak month. The worst hurricanes in recent years were Georges in 1998 (Category 3 on the Saffir-Simpson Hurricane Wind Scale of 1 to 5, with 5 being the most intense), and David in 1979 (Category 5).

Figure 5.3: Annual Rainfall in the Dominican Republic



Source: Mann et al., 1998

The Romero project is located on the southern side of the Central Cordillera; (Mann et al., 1998).

The life zone is neotropical montane forest, zoned by altitude, with subtropical wet forest below 800 m, lower montane wet forest at 800 m to 2,100 m in the project area and upper montane wet forest above this. The lower montane forest is a broadleaf forest and pine forest, the latter dominated by the native Hispaniolan pine (Pinus occidentalis, also called Haitian or Criollo pine). These occur in pure stands in the upper montane forest. Much of the forest in the region has been cut and burned for agriculture, but remnants exist on some ridges and peaks. The forest is preserved intact within the José del Carmen Ramírez National Park (764 km^2), created in 1958, which borders the east side of the La Escandalosa concession, and the Armando Bermúdez National Park (766 km^2), created in 1956, on the north and east sides of GoldQuest's San Juan claims (Figure 4.3).

The steep valley sides in the project area are cultivated, with regular burning to clear old crops, while the upper land is now mostly open grassland. Agricultural commodities in the valley are black beans (habichuela) and pigeon peas (guandulies), which are important cash crops and give three harvests a year. Maize, yuca, plantain, bananas and coffee are also grown. Cattle, goats and pigs are raised, oxen are used for ploughing and wild pigs are hunted.

Land ownership is in large tracts of both private and government land, few of which have well defined boundaries or clear legal title. GoldQuest has made a map of land owners in the main areas of interest of the project for the purposes of negotiating access agreements.

5.3 Local Resources and Infrastructure

The nearest large town to the project is San Juan de la Maguana, 40 km to the south. There are three villages within the concession area at Hondo Valle (population about 80), La Higuera (population about 200) and La Ciénaga Vieja (population about 100), although their population varies seasonally. Hondo Valle was built by relief aid following Hurricane Georges in 1998 for displaced people, and previously had only a few houses. There are no longer any villages upriver of Hondo Valle. All local transport is by mule and horse. There are primary schools in the villages, but no health centres, electricity supply, phone or other basic services. The population is Dominican of mixed Taino Indian, African and Spanish-European descent, with seasonally migrant Haitian labour of African origin.

GoldQuest built a small field camp at Hondo Valle (1,086 masl) in November, 2006, comprising wooden huts with cement floors and lower walls, core shack, secure core storage and a gasoline generator. Previously the company rented small houses in the village. Communication is managed via a VSAT (Very Small Aperture Terminal) system which comprises a 2.4 m satellite dish installed at the camp. Hand-held satellite phone can also be used. A cell phone signal can be obtained on the high parts of the access road and on some high ridges.

The San Juan River is dammed 15 km south of Hondo Valle at Sabaneta to form the Sabaneta Reservoir (Presa de Sabaneta), built in 1975 to 1981, at 584 m altitude at the edge of the Central Cordillera. This has 6.3 megawatts (MW) of hydroelectricity generation capacity, and also provides irrigation for the San Juan valley. The average annual rainfall at the Sabaneta Reservoir is 1,086 mm. The average flow is 8.13 cubic metres per second (m^3/s), and varies from 4.0 m^3/s in March to 16.82 m^3/s in September (ACQ & Asociados, 2006).

5.4 Physiography

The Romero project is located in the Central Cordillera which is up to 3,087 masl on Pico Duarte, 32 km east of the project, the highest mountain in the Caribbean. The concession lies on the west side of Loma de la Petaca mountain (altitude 1,972 m) and is traversed by the San Juan River, which flows south into the San Juan Valley. Altitudes in the concession vary from 700 m to 1,789 m.

The Romero and Romero South deposits are located in the valley of the south-flowing San Juan River. The relief within the project area is over 1,000 m with steep slopes. There are three geomorphological zones:

- Ridges: defined by remnant ridge crests with red clay lateritic tops on the east and west sides of the valley at between 1,300 to over 1,712 masl, and interpreted to be a remnant plateau. The road from Boca de los Arroyos to Hondo Valle runs along the ridge top on the west side of the valley;
- Valleys: defined by a wide valley with a plateau on the east side at an altitude of 1,100 to 1,200 masl at Los Tomates, and 1,120 to 1,150 masl at Las Lagunas, south of Romero South; and
- Canyons: the actual course of the San Juan River is a series of alternating canyons and broad meanders. The river drops from 1,080 to 900 masl with a gradient of 180 m over 3,200 m (5.6%) from Hondo Valle to La Higuera. The canyons are 100 to 160 m deep and are often inaccessible. The meandering course is unusual for mountainous terrain. Large meanders with broad terraces or old river channels have formed on outcrops of soft limestone and hydrothermal alteration, and the canyons in harder volcanic rocks, especially rhyolites.

6 History

This section was taken from the 2014 Micon PEA Report.

6.1 Historical Mining

Hispaniola was first occupied by Taino Indians and divided into five chiefdoms (cacicazgos) ruled by chiefs (caciques), including that of Maguana in the central part. The Indians were of the Arauca group which migrated from northeastern Venezuela through the Lesser Antilles and into the Greater Antilles starting from about 4,000 BC. The Taino Indians arrived in Hispaniola in about 800 AD (Lara and Aybar, 2002). The Taino collected alluvial gold by picking nuggets from the streams, rather than mining or panning it, and had no knowledge of refining or smelting. They created gold artifacts by hammering, few of which have survived.

Alluvial gold is still washed occasionally by locals in Arroyo La Guama, above Hondo Valle, but it is a very limited artisanal activity.

The discovery of Hispaniola by Columbus in 1492 was followed by a Spanish gold rush between 1493 and 1519. San Juan de la Maguana, founded in about 1506, was an important gold mining area (Guitar, 1998). Place names near the south end of the La Escandalosa concession are toponymic evidence of early gold mining, such as Arroyo del Oro (Gold Stream), Loma Los Mineros (Miner's Ridge), La Fortuna (The Fortune) and Loma del Pozo (Mine Shaft Ridge). There is no physical evidence of any historical mining in these areas now. The Spanish mines were of three types: alluvial in rivers, alluvial in dry paleochannels, and underground or pit mines (Guitar, 1998).

San Juan de la Maguana was founded in about 1506 by Captain Diego Velázquez during the second wave of colonization of the island which spread westwards from Santo Domingo in the period 1502 to 1509, following the first wave of colonization from the northwest coast to Santo Domingo (Lara and Aybar, 2002; Moya Pons, 2002). The town was named for Saint John and the Taino chiefdom of Maguana. San Juan was an important early Spanish gold mining area and included important mine owners such as Christopher Columbus' son, Hernando Colón. Indian labour was organized from 1503 under the native encomienda allocation scheme of tribute labour (Guitar, 1999). In 1514 there was a redistribution of Taino labour, and 45 Spaniards at San Juan de la Maguana received a total of 2,067 Indians. African slaves were introduced from 1505 as supervisors and technicians, rather than labourers, bringing their experience of mining, smelting, refining and gold smithing from west Africa (Guitar, 1998). In 1519, all gold mining on the island ended with the exhaustion of the deposits and the near extinction of the Indian labour. That same year San Juan de la Maguana was the scene of the first indigenous revolt in the Americas.

Following the demise of gold mining, San Juan became a centre for sugar cane and cattle production, but was abandoned in 1605 to 1606 during the “Devastations” when the Spaniards withdrew from all of the western and northern parts of the island due to their inability to hold them against attacks by maroons (escaped slaves and Indians) and pirates. The area was later occupied by the French, leading to the present day division of the island of Hispaniola into the Republic of Haiti, founded in 1804, and the Dominican Republic, which became independent in 1844. San Juan de la Maguana was refounded in 1733 in the frontier area and was largely populated with settlers from the Canary Islands.

6.2 Exploration in the 1960s and 1970s

Mitsubishi Metals Co. Ltd. of Japan carried out regional exploration of the whole Central Cordillera for copper from 1965 to 1971, although there is no record or evidence of any work in the La Escandalosa concession area (Watanabe, 1972; Watanabe et al., 1974).

A claim post exists at Hondo Valle marked “Marinos XIV” and dated 16 May 1973. No information has been found about this.

6.3 SYSMIN Regional Surveys in the 2000s

The Romero area is covered by the 1:50,000 geological map sheets and memoirs for Arroyo Limon (No. 5973-III; Bernardez and Soler, 2004) and Lamedero (Sheet No. 5973-II; Joubert, 2004), mapped by the European Union funded SYSMIN Program in 2002 to 2004. SYSMIN also carried out a stream sediment sampling program and aeromagnetic and radiometric surveys of the Central Cordillera.

6.4 Exploration by GoldQuest

Exploration & Discovery Latin America (Panama) Inc. (EDLA) formed a joint venture with Gold Fields on June 1, 2003 to carry out a regional exploration program for gold in the Tireo Formation of the Central Cordillera of the Dominican Republic, with EDLA as the initial operator. A regional stream sediment exploration program was carried out between June, 2003 and April, 2004. This program and the preliminary results are described in a paper by Redwood et al. (2006). GoldQuest became the owner of EDLA in April, 2004.

Gold mineralization was discovered in the Romero area in late 2003 by the EDLA-Gold Fields joint venture regional stream sediment exploration program. Stream sediment samples gave anomalies of 42 ppb, 36 ppb and 12 ppb Au in Escandalosa Creek, and 21 ppb and 11 ppb Au in Los Jibaros Creek at Hondo Valle, while outcrop samples gave up to 5.62 g/t Au from Hondo Valle and up to 2.2 g/t Au from Escandalosa Creek. The Las Tres Palmas exploration concession was applied for on December 18, 2003 and title was granted on May 30, 2005 for five years. A new exploration application was submitted on May 14, 2010, and the concession was granted for another five years on November 9, 2010 according to Dominican Mining Law. The project was operated by GoldQuest between 2003 and 2007, by Gold Fields from May 31, 2007 until November, 2009 and since then by GoldQuest.

6.5 Historical Resource Estimates and Production

There are no known historical resource estimates for the property and no known production of base or precious metals beyond the undocumented production of small amounts of placer gold from streams by the local inhabitants.

In 2012, GoldQuest announced a mineral resource in accordance with NI-43 101, for the Escandalosa deposit (Steedman and Gowans, 2012), which is now known as Romero South. That mineral resource has been superseded by the estimate presented in this report (Hennessey et al, 2013).

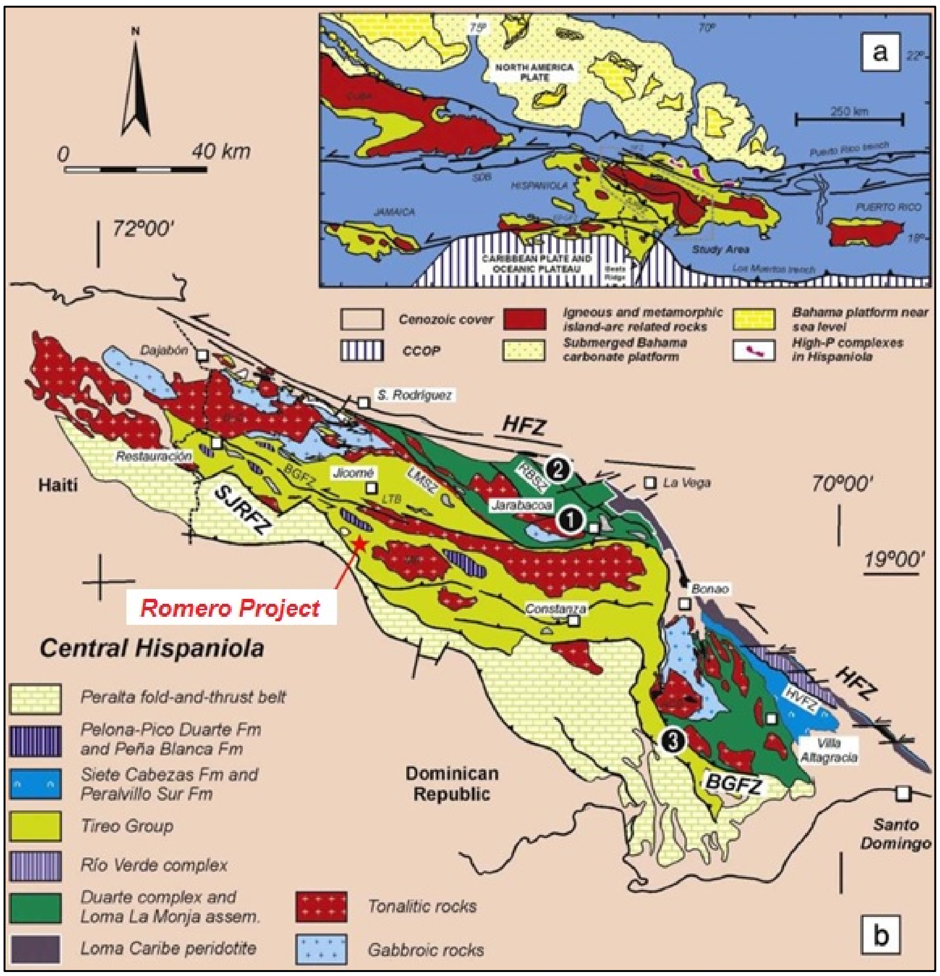
7 Geological Setting and Mineralization

This section was taken from the 2014 Micon PEA Report.

7.1 Regional Geology

The Romero project is located on the south side of the Central Cordillera of the island of Hispaniola which is a composite of oceanic derived accreted terrains bounded by left-lateral strike slip fault zones, and is part of the Early Cretaceous to Paleogene Greater Antilles island arc (Figure 7.1).

Figure 7.1: Regional Geological Map



Source: Map from Escuder Viruete et al., 2008, Fig. 1a) Plate Tectonic Setting of Hispaniola. (b) Regional Geology Map of the Central Cordillera of Hispaniola showing the Location of the Romero project.

Hispaniola is located on the northern margin of the Caribbean plate which is a left-lateral transform plate boundary. The tectonic collage is the result of west-southwest- to southwest-directed oblique convergence of the continental margin of the North American plate with the Greater Antilles island arc, which began in the Eocene to Early Miocene and continues today (Escuder Viruete et al., 2008).

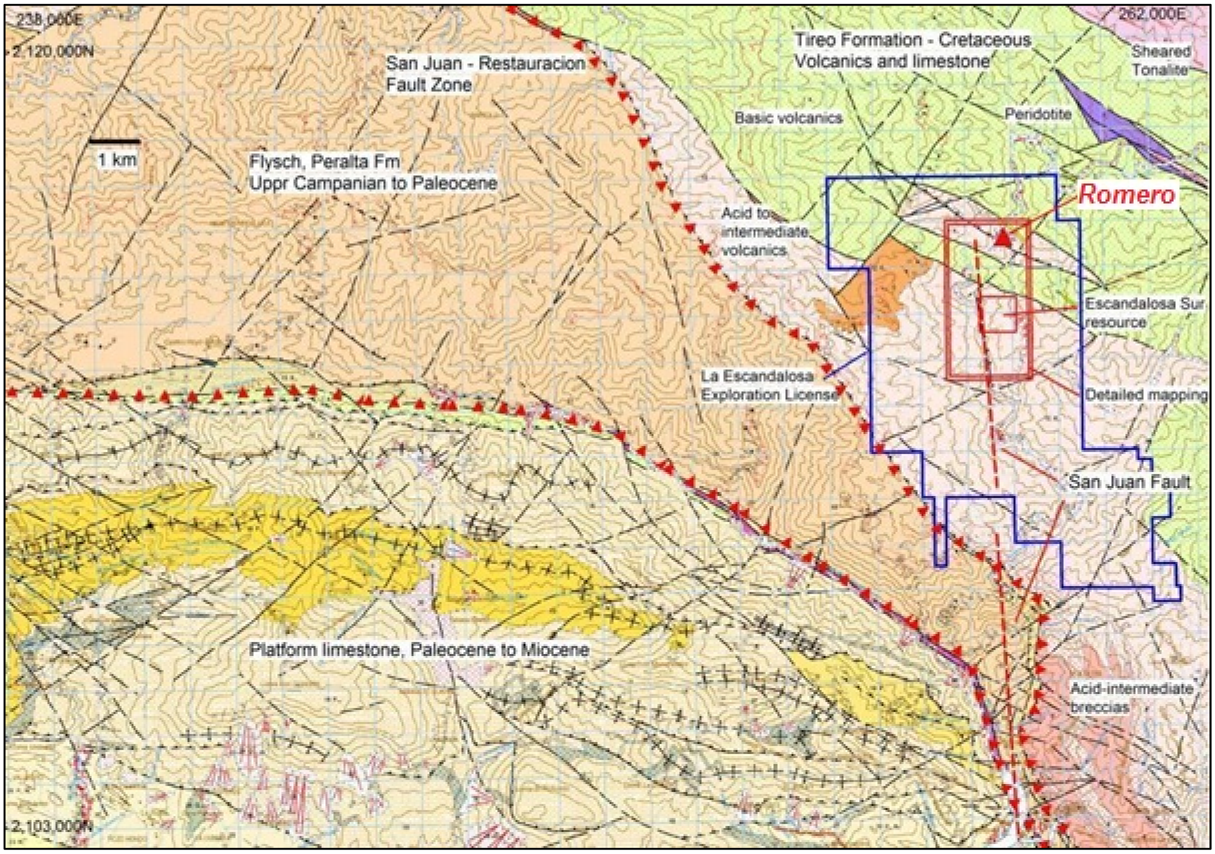
Primitive island arc volcanic rocks of the Early Cretaceous Los Ranchos and Maimón Formations in the Eastern Cordillera are interpreted to be related to northward subduction (Lebron and Perfit, 1994). Cessation of subduction in the mid Cretaceous was marked by accretion of the Loma del Caribe peridotite between the Eastern and Central Cordilleras (Draper et al., 1996) and by early Cretaceous greenstones and intrusions of the Duarte Complex in the Central Cordillera, interpreted to be of metamorphosed ocean island or seamount origin (Draper and Lewis, 1991; Lewis and Jimenez, 1991). This was followed by arc reversal and southward subduction, with formation of calc-alkaline volcanic and sedimentary rocks of the Tireo Formation of late Cretaceous to Eocene age in the Central Cordillera (Lewis et al., 1991). Since then the tectonics of the Central Cordillera have been dominated by a left lateral transpressional strike slip related to the Caribbean-North American plate boundary.

The Romero and Romero South deposits are hosted by Cretaceous-age Tireo Formation volcanic rocks and limestones (Figure 7.2). The Tireo Formation is bounded on the south side by flysch comprising calcareous slates, limestones, sandstones and shales of the Trois Rivieres or Peralta Formation of upper Campanian to Paleogene age. The contact with the Tireo Formation is a northwest-trending, southwest-verging reverse fault, the San Juan-Restauración fault Zone, which represents a transpressional fault bend. South of the Peralta Formation is a block of Paleocene to Miocene marine and platform limestone of the Neiba and Sombrerito Formations forming an antiformal restraining bend structure with reverse faults and folds (Figure 7.2). The Central Cordillera is bounded on the south side of these formations by an east-southeast-trending, south-verging, high angle reverse fault. To the south is the east-southeast-trending San Juan graben with a thick sequence of Oligocene to Quaternary molasse sediments deposited in a marine to lagoon environment, with Quaternary alkaline basalts related to graben extension.

The San Juan Valley is a major north-south-trending lineament and fault (Figure 7.2). This may have played a role in the localization of mineralization at Romero. There is a major deflection in the frontal thrust of the Central Cordillera with further transport south on the east side and a sinistral compressional bend. The Trois Rivieres-Peralta Formation is thinned in the fault zone, indicating that this may also reflect a basin depositional margin.

The tectonic deflection coincides with a major north-northwest-trending aeromagnetic and aero radiometric break which lies 3 km to 5 km west of the mineralization at Romero. On the east there is high amplitude magnetic topography with a general east-southeast ridge texture in the Tireo Formation, tonalites and shear zones, against a magnetic low with smooth textures on the west in the Trois Rivieres Formation.

Figure 7.2: Regional Geology of the Romero Area



Source: 1:50,000 geological map by Bernárdez and Soler, 2004.

The 1:50,000 published geological map shows acid to intermediate volcanic rocks of the Tireo Formation in the south part of the La Escandalosa concession, and basic volcanic rocks of the Tireo Formation in the north part, with a northwest-trending block of acid to intermediate volcanic rocks at Romero (Figure 7.2, Bernárdez and Soler, 2004). The bedding and foliation generally strike northwest and have moderate to steep dips to the northeast. The major structures are northwest-trending faults and thrusts, and north-south- and northeast-trending faults. In contrast, mapping by GoldQuest has shown that the geology comprises felsic to intermediate volcanic rocks and limestones with low to moderate dips.

The nearest intrusive bodies shown on the 1:50,000 published map are 3 km to 7.5 km from Romero and are in the Tireo Formation (Figure 7.2). These comprise a small sheared peridotite and foliated tonalite body, 3 km northeast of Romero; a foliated tonalite pluton at Loma del Tambor (more than 30 km long by 5 km wide) in a west northwest-trending shear zone 5 km northeast of Romero; and the Macutico Batholith tonalite (16 km long by 12 km wide), 7.5 km southeast of Romero, dated at 85 to 92 million years old (Ma) (Late Cretaceous) (Bernárdez and Soler, 2004; Joubert, 2004).

7.2 Project Geology

Geological mapping at Romero has been carried out for GoldQuest at a scale of 1:10,000 (Gonzalez, 2004) and 1:2,000 scale (MacDonald, 2005; Redwood, 2006b, 2006c), with revision and additional mapping by Gold Fields (Dunkley and Gabor, 2008a, 2008b). A geological map at 1:2,000 scale is shown in Figure 7.3. A petrographic study was carried out by Tidy (2006). Infra-red spectrometry (Pima) has been used to aid identification of alteration minerals.

The geology of the Romero area comprises a relatively flat lying sequence of intercalated subaqueous volcanic rocks and limestones which youngs from west to east as a function of erosional level. The oldest rocks are rhyolite flows exposed in the San Juan River on the west side. These are overlain by dacite breccias which contain the gold mineralization. These in turn are overlain by limestones and andesite breccias. The stratigraphy is described from oldest to youngest in this section.

7.2.1 Lithological Units

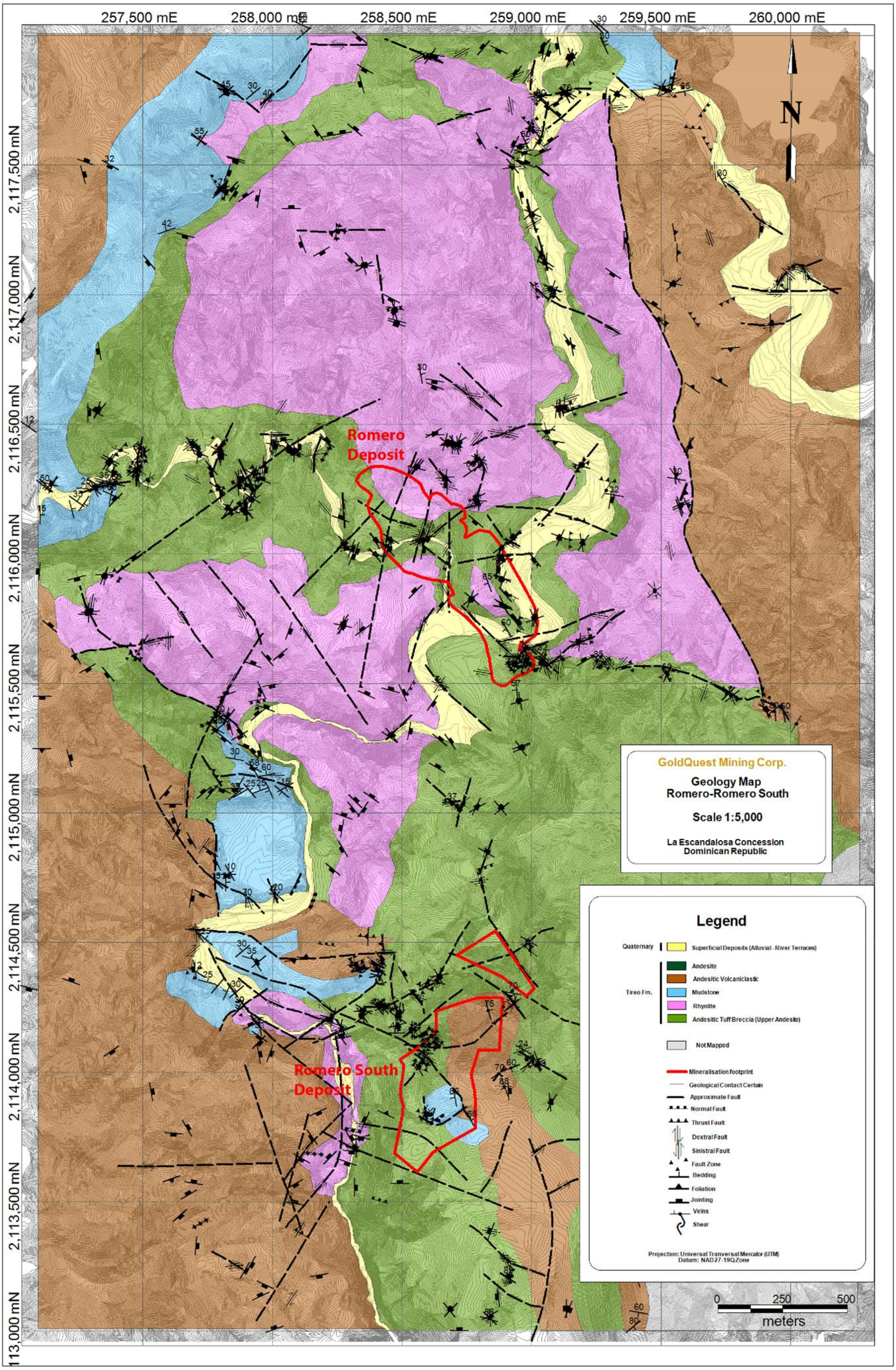
7.2.1.1 Rhyolite

Rhyolite outcrops sporadically for at least 2,000 m of strike length on the west side of the altered horizon from north of Romero to Romero South. There are two apparent rhyolite centres at Romero and Romero South defined by thick rhyolite outcrops, and in between these the flows are thinner with more breccias. The rhyolite is volcanic, rather than intrusive, and has the form of thick flows or lava domes with marginal flows and hyaloclastite breccias. The flows have autobrecciation and flow banding in places. The hyaloclastite tuffs and breccias are intercalated with limestone, andesite and dacite.

The rhyolite is a very siliceous and hard rock with phenocrysts of quartz, plagioclase and green hornblende. The mafic minerals have usually been altered to magnetite and trace pyrite. Petrography shows an andesine composition for plagioclase phenocrysts, with the matrix ones slightly more sodic. The highly siliceous nature is, in part, due to silicification.

7.2.1.2 Dacite

Dacite is most commonly the favourable host horizon for hydrothermal alteration and gold mineralization which can be traced for about 2,200 m from Romero to Romero South on the east side of the San Juan River. The dacitic volcanic rocks overlie rhyolite lavas and are interpreted to be autobreccias and hyaloclastite breccias derived from the rhyolite. The high porosity and permeability of the dacites has evidently made them a receptive host for hydrothermal fluids.


257,500 mE
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2,113,000 mN
N
Romero Deposit
Romero South Deposit
GoldQuest Mining Corp.
Geology Map
Romero-Romero South
Scale 1:5,000
La Escandalosa Concession
Dominican Republic
Legend
Quaternary
Superficial Deposits (Alluvial - River Terraces)
Tireo Fm.
Andesite
Andesitic Volcaniclastic
Mudstone
Rhyolite
Andesitic Tuff Breccia (Upper Andesite)
Not Mapped
Mineralisation footprint
Geological Contact Certain
Approximate Fault
Normal Fault
Thrust Fault
Dextral Fault
Sinistral Fault
Fault Zone
Bedding
Foliation
Jointing
Veins
Shear
Projection: Universal Tranversal Mercator (UTM)
Datum: NAD27-19Q Zone
0
250
500
meters

The dacite is overlain by limestone or by andesite breccia. The altered dacite horizon varies from a thick body between rhyolite and andesite at Romero, to a thinner discrete horizon within less strongly altered dacite at Romero South.

At Romero the dacitic volcanics occur above and east of the rhyolite flow/dome and dip from 40° to 50°E near the base to 15°E at the top contact in Jibaros Creek. They form a body with a vertical thickness of greater than 200 m. The soft altered dacite is susceptible to landslides, and erosion to form river terraces.

South of the La Escandalosa creek and the Escandalosa fault, the mineralized horizon in the dacite is exposed in a trail at the discovery outcrop where there is strong argillic and sericite-quartz alteration with jarosite after pyrite. Trenching there returned high gold grades. Holes LTP-05 and LTP-06 were drilled on the trenches and returned low grade gold values and are interpreted to be in the lower part of the Romero South zone with land-slipped higher grade material from the upper part in the trenches. Hole LTP-07 was drilled higher up slope and intersected the whole width of the mineralized horizon.

To the west of the discovery outcrop, the mineralized horizon outcrops in a cliff on the east side of the San Juan Canyon. The cliff face is a fault plane (strike 355, dip 80°E) with gossan, jarosite and copper carbonate staining of silicified dacite with zones of semi-massive pyrite and abundant sphalerite and chalcopyrite.

There are similar looking outcrops with a low angle of dip on the west side of the San Juan River as well. These are apparently continuous across the canyon with an apparent dip of 10°W, and there does not appear to be any significant displacement across the prominent north to south lineament that forms the San Juan Canyon. However, no disseminated gold mineralization has been found west of the river by reconnaissance soil and rock sampling.

Lithologically the dacite breccias generally have a lapilli grain size with varying proportions of:

- Rounded clasts of siliceous rhyodacite probably derived from the rhyolite flow/dome, and commonly with quartz veinlets and disseminated pyrite. They often have a colour change at the rim. There are variations in phenocrysts and texture;
- Green elongate fiamme-like clasts with quartz and plagioclase phenocrysts, which are locally parallel and may define poor bedding. These are interpreted to be glass with diagenetic or post-alteration flattening and alteration of the glass to green illite-chlorite, and some are pyrite-rich. They are interpreted to be hyaloclastite derived from chilling and shattering of the rhyolite lava on contact with water, rather than pumice clasts of pyroclastic origin;
- Rounded pyrite-rich porphyry clasts. These have very fine grained disseminated to semi-massive pyrite and often have a pyrite-rich or colour-changed rim. They are interpreted to be derived from pyrite mineralization; and
- Fine grained, aphyric siliceous clasts.

The clast distribution is generally polymict, but varies to monomict, which probably indicates an in-situ hyaloclastite breccia. The matrix of the breccia is fine grained. The clast shape varies from angular to rounded, and sorting is usually poor with clast size from <1 mm up to 100 mm. There are also fine grained tuff to ash sized breccias with a curved convex clasts and shards which are hyaloclastites.

Some weakly altered hyaloclastite breccias have a red limestone matrix (e.g. Los Tomates Ridge). It is possible that the control of the favourable horizon within the dacite breccias was a carbonate matrix which was dissolved by hydrothermal fluids, thus enhancing porosity and permeability and fluid flow.

7.2.1.3 Limestone

Two units of limestone have been mapped, Maroon Limestone and Gray Limestone. They have similar lithofacies and are distinguished by colour and outcrop in different areas. The colour difference is interpreted to due to hydrothermal alteration and bleaching.

The Maroon Limestone is a maroon coloured, fine grained micritic limestone, with fine to medium bedding, thin graded beds of volcanic sandstone (probably a resedimented hyaloclastite or autoclastic sandstone) and red chert or jasperoid beds. The dips are low although there are locally high dips due to folding. The Maroon Limestone occurs in several horizons and is intercalated with dacite breccia, rhyolite flows and hyaloclastites.

The Gray Limestone has a similar lithofacies to the Maroon Limestone and forms a well-defined mappable unit at Romero South. It forms a graben-block bounded by northeast- and northwest-trending faults, with stratigraphic contacts on the southeast and southwest sides. Stratigraphically the Gray Limestone lies directly above the altered and mineralized dacite breccias, and is overlain by andesites. The Gray Limestone is finely bedded (10 cm to 15 cm beds), dark grey, locally maroon coloured, micritic limestone, with laminated dacitic volcanic sandstone beds, and black chert beds. In the drill core there are some beds of fine grained pyrite. The limestones have open folds with dips up to 50° to 60°. The vertical outcrop interval is about 110 m.

The Gray Limestones are bounded on the north side by the Escandalosa fault which trends 070° east-northeast with a vertical dip which forms cliffs and can be mapped for 1,200 m. It is interpreted as south-side down. Andesite breccias outcrop on the north side of fault. On the east side the Gray Limestone is in stratigraphic contact with andesite. On the west side the Gray Limestone is bounded against dacite by a fault trending 135° (east-side down) to the north of the Romero South discovery outcrop and holes LTP-05 and LTP-06. The southern contact of the Gray Limestone is the Escandalosa Sur fault which trends 055° with a steep dip (north-side down).

On the southwest side of Romero South the Gray Limestone contact over mineralized dacite is stratigraphic (LTP-08, LTP-09) and is exposed in cliffs in the San Juan Canyon and on the hill top at platform LTP-08. Gray Limestone outcrop in cliffs continues to south of LTP-09 for an undefined distance, and may be terminated or displaced by the inferred southwest continuation of the Escandalosa Sur fault.

7.2.1.4 Andesite

Coarse grained, green, chlorite-altered andesite breccias are well exposed in the Escandalosa creek and its tributaries and form the ridge on the east side of the mapped area of alteration. The andesites outcrop over a vertical interval of about 220 m to the top of the ridge. They overlie dacite breccias from Romero South to Romero and form the hanging wall to the altered unit.

The lithology is a green volcanic conglomerate or breccia. The green colour is chlorite alteration with carbonate and magnetite. The clasts are gravel to block (30 cm) sized and rounded, in a sandy matrix, but there is no bedding except for a weak low angle parting. The composition is andesite to quartz-phyric dacite.

Further south of Romero South, at La Higuera, the andesites comprise a sequence of andesitic to dacitic lavas or volcanic sandstones/ash tuffs, with texture varying from crowded phenocrysts to fine grained aphyric. The phenocrysts include pyroxene, quartz, plagioclase and other mafic minerals with alteration to chlorite, epidote, magnetite and pyrite.

7.2.1.5 Dykes

The only intrusive rock mapped is a single dyke of plagioclase-phyric andesite with a chilled margin cutting andesitic volcanic rocks at La Laguna (Romero South), with a trend of 128° and 85°E dip.

7.2.2 Structure

The principal lineament trends are northeast, northwest and north-south. Faults were mapped in the field. West-northwest-trending faults dominate in the northern part of the area, and northeast-trending faults in the south. The faults are generally steep and show vertical displacement, although it has not been established whether this is normal or reverse movement. However, slickensides often show horizontal to low angle plunge indicating strike slip movement. In places this can also be mapped by lateral offset of units, notably right lateral displacement on the Hondo Valle fault. North-northwest- to northwest-striking low angle reverse faults and thrusts occur at a number of localities in the Romero area, although the scale of thrusting is uncertain.

The thinly bedded limestones have tight folding, and bedding is locally steep or overturned. The hinges dip to the east with reverse faults, shallow east limbs and overturned steep west limbs, indicating west-verging folding and thrusting. The limestones have focused deformation due to low rheological competency, while the more massive limestone beds and volcanic units are not folded.

The structural observations are consistent with the transpressional tectonics that have affected the Central Cordillera since the Eocene. This may include strike slip reactivation of older, steeper normal faults.

7.2.3 Alteration and Mineralization

7.2.3.1 Silicic and Phyllic Alteration

Phyllic and silicic alteration have been mapped as a continuous zone over about 2,200 m of strike length with a general north-south trend from Romero to Romero South. Gold mineralization with anomalous silver, zinc and copper is associated with the phyllic and silicic alteration. Mapping and drilling support a model of stratabound and stratiform alteration of dacite breccias.

The alteration types are pervasive and are quartz-pyrite alteration (silicification), quartz-illite-pyrite alteration (phyllic) and illite-chlorite-pyrite alteration, with gradations between each type. Discrete zones of silicification can be mapped in places, notably at Romero, but it is usually gradational with, or alternates with phyllic alteration and they have generally been mapped together as phyllic alteration. A similar relationship is seen in drill core where phyllic and silicic alteration can be logged separately in some places, and in others alternate every few metres. Silicification varies from intense, giving a very hard, cherty rock, to moderate and weaker intensities with progressive lowering of hardness and rock quality designation (RQD) measurements of core. Quartz forms irregular veining in phyllic alteration.

Silicification and phyllic alteration appear to be strongest in the upper part of the altered horizon where fluid flow may have been focused. Lower down the alteration becomes weaker and is typically pale blue-green illite and chlorite (confirmed by Pima) with disseminated pyrite and no quartz.

The phyllic-silicic alteration zone is marked by an absence of magnetite due to magnetite destruction by sulphidization.

7.2.3.2 Propylitic Alteration

Propylitic alteration occurs in both the hanging wall and the footwall to the phyllic-silicic alteration zone.

The andesite breccia of the hanging wall has pervasive chlorite alteration with trace to 1% disseminated pyrite giving the rock a dark green colour. It is accompanied locally by epidote, calcite veinlets, quartz veinlets, silicification and magnetite.

The footwall dacite breccias and rhyolites also have propylitic alteration with chlorite-magnetite-(epidote-quartz-pyrite) and local silicification. There is up to 5% magnetite, after hornblende, and widespread barite in veinlets and replacement, especially in the lower part of La Escandalosa creek. Magnetite and barite alteration are stronger in the footwall than the hanging wall.

The first appearance of magnetite in the hanging wall and footwall to the phyllic-silicic zone marks the start of the propylitic zone and is sharply defined in core. The magnetite is a combination of primary igneous magnetite and hydrothermal alteration of mafic minerals.

There is a narrow zone of hematite-silica above and below the phyllic-silicic zone in some holes indicating a redox front. The hydrothermal fluid is interpreted to have been reducing with lateral flow in the main phyllic-silicic horizon, changing to oxidizing with vertical flow into the hanging wall and footwall.

7.2.3.3 Hydrothermal Breccias

There are several types of phreatic hydrothermal breccias with sulphides in the phyllic and silicic alteration zones. These are volumetrically small and are only seen in core and not in outcrop. Most of the breccias at Romero South are volcaniclastic.

Three types of phreatic breccia have been identified in core, listed from oldest to youngest based on cross-cutting relationships:

- A black jigsaw breccia with a black matrix of silica, fine grained pyrite and a fine grained, black, non-sulphide mineral (biotite?) in zones of tens of centimetres. It is matrix to clast supported;
- This is cut by quartz-sulphide veinlets which can form a network fracture breccia; and
- A clay-matrix breccia cuts silicified rock and is a jigsaw, clast-supported breccia with angular, milled silicified clasts in a matrix of soft pale grey-green clay-pyrite. It forms irregular breccia veinlets of a few to tens of centimetres width. It is interpreted to be a phreatic breccia rather than a fault breccia due to the matrix of clay (in silicified zones) and pyrite (which does not appeared to be milled), but may in fact be fault breccia.

7.2.3.4 Fault Breccias

Late-stage fault breccias also occur. These have a soft clay matrix when in phyllic alteration zones. Faults in rhyolite form a mylonite of brittle fractured shards. The fault breccias affect and thus postdate alteration and the thick white quartz veins.

7.2.3.5 Barite

White barite is commonly present in veinlets and hydrothermal breccias with quartz and calcite, and in places forms a fine-grained pervasive replacement. It is more abundant in the footwall to the phyllic alteration zone than in the hanging wall. Barium usually does not show in geochemistry due to the insolubility of barite in the acid digestion used for the ICP analyses.

In the San Juan river at Romero South there is a 10-m wide, white barite vein surrounded by a stockwork of barite veinlets, associated with silica and phyllic alteration. Pervasive, very fine-grained white barite occurs with quartz replacing rhyolite in the lower part of the Escandalosa creek.

7.2.3.6 Quartz Veining

There are two types of quartz veining, namely veinlets associated with phyllic alteration, and massive white quartz veins.

The quartz veinlets are white quartz and chalcedony which form irregular veinlets and network veinlet breccias in the phyllic alteration zone. There are also rare straight-sided veinlets. The quartz may have a vuggy texture with a centre line. Quartz is accompanied by white barite, calcite and sulphides. Sulphides may dominate in some veinlets. Minor, late stage quartz veinlets cross-cut quartz-sulphide veinlets.

Massive white quartz veins are locally common in the propylitically altered andesite breccia, especially in the Escandalosa fault zone. The veins are white, massive and multi-directional and may have minor pyrite and chalcopyrite. They are up to at least 2 m wide as shown by abundant river boulders in the Escandalosa creek. Massive white quartz veins can also occur in the phyllic zone, and are distinct from the quartz-chalcedony veinlets described above.

7.2.3.7 Calcite Veining

Calcite veinlets are common in the Maroon and Grey Limestone and are of two types, bedding parallel ptygmatic (strongly deformed), and irregular cross-cutting veinlets with quartz and/or barite. The latter also occur in volcanic rocks.

7.2.3.8 Limestone Bleaching

The Gray Limestone is interpreted as hydrothermally altered and bleached Maroon Limestone based on the restricted outcrop of Gray Limestone in the hanging wall of the phyllic alteration zone. The Gray Limestone has a similar lithofacies to the Maroon Limestone, and has an extensive regional distribution, in contrast to the Maroon Limestone.

It is interpreted that the original colour of the limestone is maroon and that this is indicative of deposition in an oxidizing environment suggesting continental lacustrine rather than submarine conditions. Hydrothermal alteration by a reducing fluid caused a colour change to grey.

7.2.3.9 Sulphides

Coarse-grained pyrite (1 mm to 2 mm) occurs as disseminations in phyllic and silicic alteration and with other sulphides in semi-massive zones up to 50 cm wide, and in sulphide and quartz-calcite-barite veinlets. The other common sulphides are sphalerite, chalcopyrite and galena. The sphalerite is pale brown in colour indicating a low iron and high zinc content. It usually occurs with chalcopyrite in well formed crystals of 1 mm to 2 mm and these are partly replaced by black iron-rich sphalerite.

Pyrite also occurs in a fine-grained, framboidal habit in clasts in volcanic breccia in amounts varying from a few percent as disseminations to massive.

7.2.3.10 Oxidation and Enrichment

Supergene oxidation due to weathering is shallow with a depth of 10 m to 15 m. In zones of silicic alteration, the pyrite is leached giving residual vuggy silica with jarosite and hematite, for example at Romero. Supergene argillic alteration is developed from quartz-illite-pyrite, illite-chlorite-pyrite and propylitic alteration and gives white clay (kaolinite-smectite) with jarosite and hematite, and forms colour anomalies.

Rare copper oxide minerals, such as brochantite and blue copper carbonates, occur in outcrop. There is a thin zone of minor supergene chalcocite coating sulphides below the base of oxidation for 1 m to 2 m.

7.2.4 Geomorphology and Overburden

The Romero project is located in the valley of the south-flowing San Juan River. The relief within the project area is over 1,000 m with steep slopes. There are three geomorphological zones, as described in Section 5 above, ridges, valleys and canyons.

These geomorphological zones are interpreted to indicate a three-stage history of uplift and erosion:

1) Plateau Phase, of which the ridge tops with laterite are a remnant. The age of lateritization elsewhere in the Dominican Republic has been dated stratigraphically as Late Tertiary (post-Middle Oligocene);

2) Valley Phase, consisting of major uplift and river erosion to form broad valleys;

3) Canyon Phase, with the recent uplift and river erosion/down-cutting to form canyons which meander in the Canyon Phase.

The mineralization at the Romero project was exposed relatively recently during the valley and canyon Phases. For this reason sulphides are commonly exposed as there has been relatively little time for oxidation.

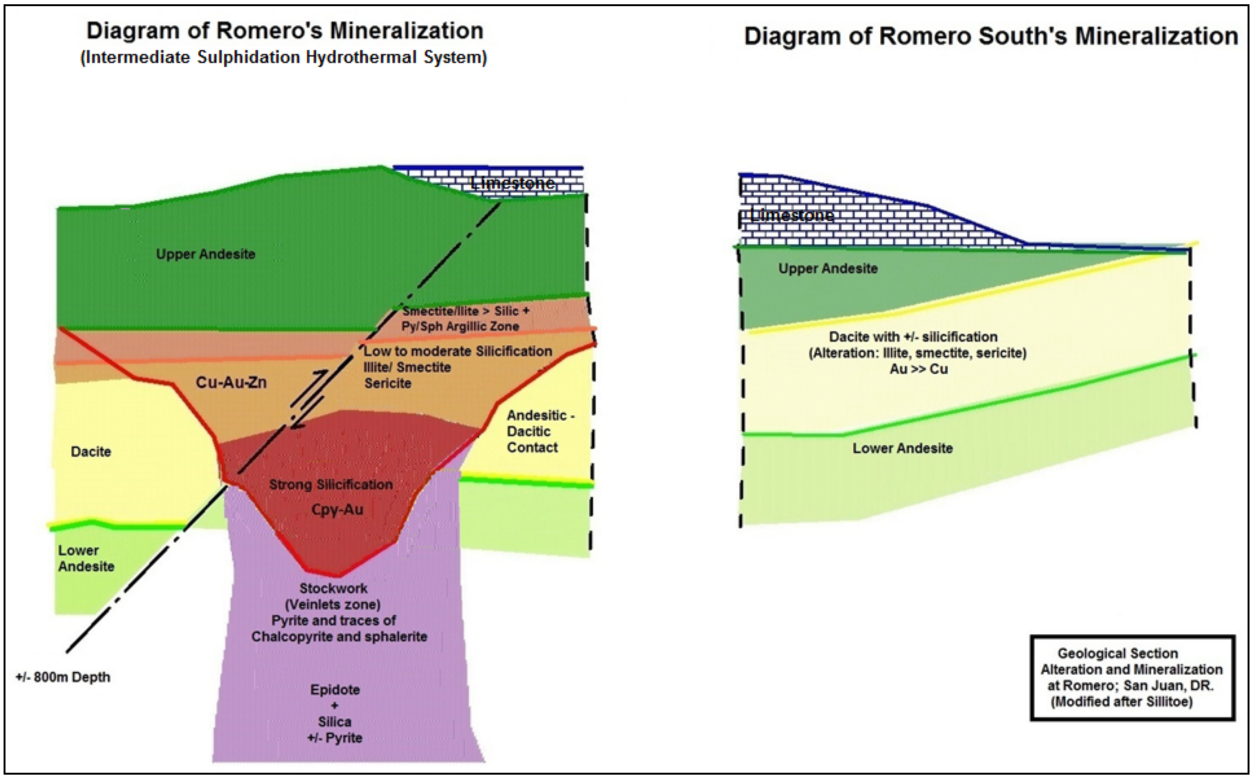
Unconsolidated Quaternary overburden deposits mapped are active river bed alluvium, river terraces, landslides and colluvium. Landslides are common especially in the canyon phase topography.

7.3 Gold and Base Metals Mineralization

Gold and associated base metal mineralization forms a stratiform body in dacite breccias. The stratiform style is shown in Figure 7.4. Alteration and mineralization can be traced for about 2,200 m from Romero south to Romero South. The altered unit is more than 200 m thick vertically at Romero.

Gold mineralization is related to quartz and sulphides. Coarse grained pyrite (1 mm to 2 mm) occurs as disseminations in phyllic and silicic alteration and with other sulphides in semi-massive zones up to 50 cm wide, and in sulphide and quartz-calcite-barite veinlets. The other common sulphides are sphalerite, chalcopyrite and galena. The sphalerite is pale brown in colour indicating a low iron and high zinc content. It usually occurs with chalcopyrite in well-formed crystals of 1 mm to 2 mm and these are partly replaced by black iron-rich sphalerite. Pyrite also occurs in a fine-grained, framboidal habit in clasts in volcanic breccia, in amounts varying from a few percent as disseminations to massive.

Figure 7.4: Cross Section through Romero and Romero South



Source: GoldQuest (2013)

8 Deposit Types

This section was taken from the 2014 Micon PEA, amended from Steedman and Gowans (2012) with more recent observations by R. H. Sillitoe (2013) and GoldQuest staff.

The features of the geological model for alteration and precious/base metals mineralization at Romero are as follows:

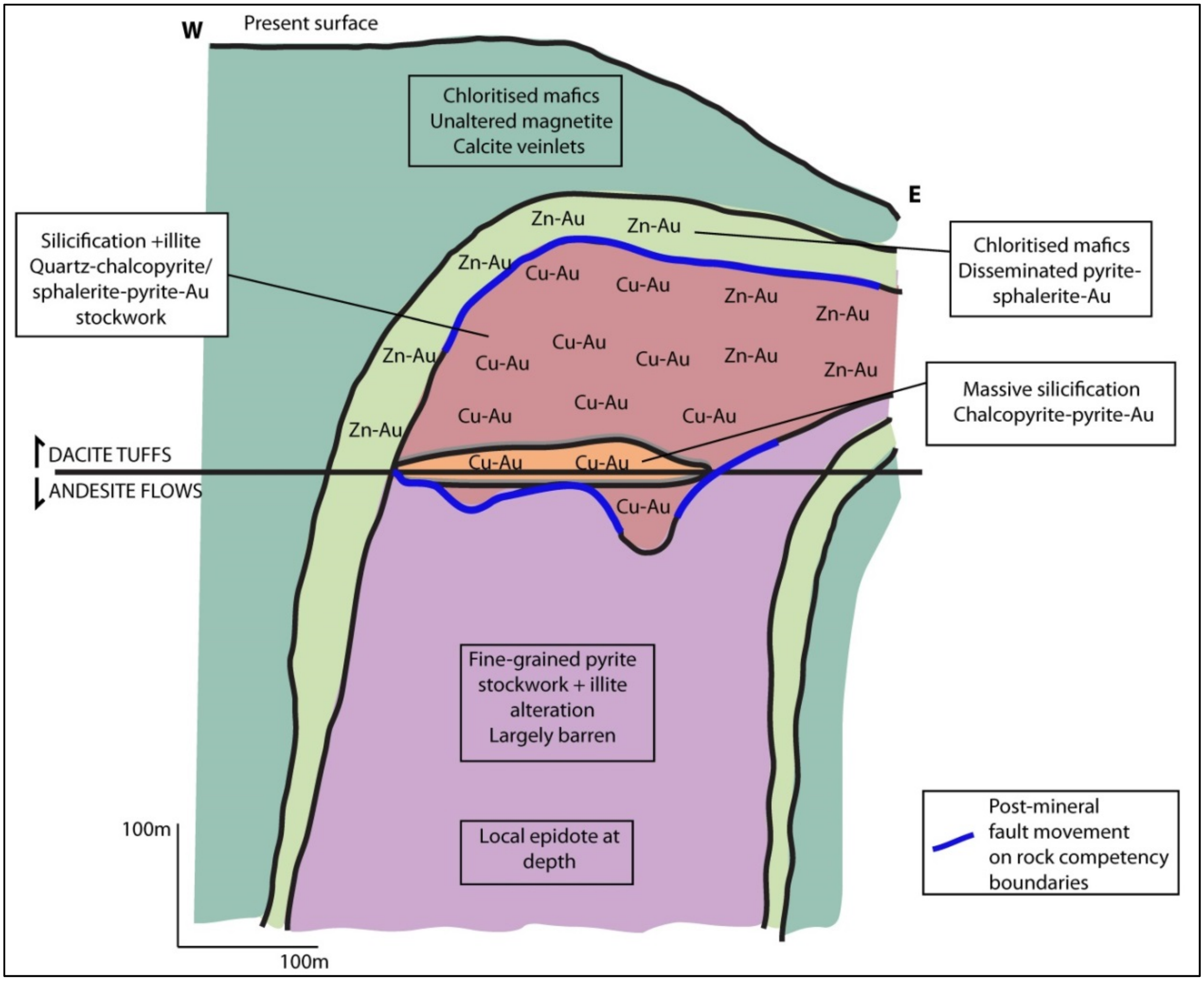
- Hosted by the Cretaceous-age Tireo Formation island arc sequence;
- The host rocks are subaqueous, felsic to intermediate volcanic and volcaniclastic rocks (rhyolite to dacite flows, possible domes, autobreccias, hyaloclastite sandstones to breccias) and non-volcanic sediments (limestones);
- Alteration and mineralization are epigenetic and of intermediate sulphidation epithermal style; and
- The gold-bearing chalcopyrite mineralization is hosted by silicified and illite-altered dacitic tuffs and underlain by a largely barren, vertically extensive pyritic stockwork (Figure 8.1) developed in andesitic rocks (Sillitoe, 2013);
- Upwards and laterally at Romero, the chalcopyrite gives way to sphalerite and a gold-zinc association predominates (Figure 8.1);
- Alteration and mineralization is generally stratabound within the dacitic volcaniclastic breccia (lithic lapilli tuff, with variable clast size from ash to block, also hyaloclastites). Bedding and lithological variations can be logged in the altered zones. May also be in massive lava units. The breccia clasts are dacite to rhyolite, hyaloclastic shards, and also mineralized clasts;
- The mineralized clasts in the dacite breccia are silicified with very fine grained pyrite, occasional quartz veinlets and no gold. The clasts were mineralized before being incorporated into the tuff;
- Alteration can be mapped for over 2.2 km north to south;
- The alteration is zoned vertically:
    - Propylitic alteration of the hanging wall (chlorite, epidote, quartz and silicification, pyrite and magnetite);
    - Quartz-illite-pyrite and quartz-pyrite in the mineralized zone. Quartz forms irregular veins in competent rock and matrix replacement in breccias. Alteration is stronger in the upper part of the zone and becomes weaker downwards and is pale green illite-chlorite-pyrite. The sulphides comprise disseminated to semi-massive pyrite with chalcopyrite, sphalerite and galena. The gold grade appears to correlate with silicification or quartz veining; and

  - Propylitic alteration in the footwall (chlorite-magnetite-epidote-quartz-pyrite-barite) with strong magnetite and barite.

- Gold is associated with silicification and quartz-sulphide veining;
- There are several stages of volumetrically minor hydrothermal breccias with sulphides (although most of the breccias are volcaniclastic);
- Veinlet breccias form in massive lava units;
- Barite is ubiquitous in breccias and veinlets, and forms pervasive fine-grained replacements;
- The alteration zonation shows a stratabound to stratiform geometry and indicates lateral fluid flow;
- There is a redox change in the fluid coincident with the change from quartz-illite-pyrite to propylitic alteration with magnetite. In some holes there is hematite-silica above and below illite. The hydrothermal fluid is interpreted to have been reducing with lateral flow in the main illite-quartz horizon, changing to oxidising with vertical flow into the hanging and footwall; and
- The favourable horizon has restricted outcrop and is masked by weakly altered rocks in the hanging wall and footwall.

Flow of the hydrothermal fluids is interpreted to have been lateral and related to the porosity and permeability of the host dacite breccias to form generally stratiform mineralized bodies with intermediate sulphidation epithermal characteristics.

Figure 8.1: Schematic Geological Section, Romero Deposit



Source: Sillitoe (2013).

There are several unusual or undetermined aspects to the deposit model which may have implications for future exploration.

9 Exploration

This section was taken from the 2014 Micon PEA, of which information was taken and amended from Hennessey et al. (2013).

9.1 Topography and Imagery

GoldQuest commissioned a detailed topographic map with 2 m contour intervals derived from Ikonos satellite imagery (1 m resolution) which provided a detailed base map for mapping, plotting drill holes and polygons, as well as a high resolution satellite image.

The company also carried out spectral interpretation for alteration mapping of an Aster satellite image (15 m resolution).

9.2 Geological Mapping

Geological mapping at Romero has been carried out for GoldQuest at 1:10,000 scale (Gonzalez, 2004) and at 1:2,000 scale (MacDonald, 2005; Redwood, 2006b, 2006c), with revision and additional mapping by Gold Fields (Dunkley and Gabor, 2008a, 2008b). A petrographic study of 15 samples was carried out by Tidy (2006).

During 2015 focused mapping of the Romero Trend to the north and more importantly to the south was carried out by the GoldQuest geology team. Mapping to date has followed the regional magnetic low trends, which coincide with the Romero Trend. Generally the magnetic lows in the vicinity of Romero and along trend have been coincident with areas of hydrothermal alteration.

9.3 Geochemistry

One of the main exploration techniques used in early exploration at Romero has been geochemistry. GoldQuest has taken 40 fine fraction stream sediment samples (minus 200 mesh), 1,090 soil samples and 1,176 rock samples, including channel samples.

Soil geochemical grids have been carried out over most of the areas of outcropping mineralization between Hondo Valle and La Higuera on 100 m by 100 m, and 50 m by 50 m grids and ridge and spur soil samples for reconnaissance. The area sampled on grids is about 2.0 km long north-south by 1.0 km across, and the total area sampled, including ridges and spurs, is about 4.0 km north-south by 3.0 km wide. A total of 1,090 soil samples have been taken.

Hand dug trenches were made to follow up on soil anomalies prior to drilling, and continuous channel samples were taken of the exposed bedrock.

9.4 Geophysics

9.4.1 Early Geophysics

GoldQuest obtained a regional airborne magnetic and radiometric survey flown on a 1-km line spacing for the SYSMIN program. Reprocessing was carried out by Gold Fields.

A Direct current induced polarization (DCIP) ground geophysical survey was completed by Quantec Geoscience Ltd, over the Las Tres Palmas project during the summer of 2011. A total of 44 east to west lines spaced at 200 and 100 m (depending on the priorities of the zones) with reading stations at 50 m over the lines which were surveyed, covering 77.75 line km over an area of approximately 15 km^2. The objective of the DCIP program was to define the chargeability (IP) and conductivity/resistivity responses of the underlying ground of the survey grid.

The survey delineated two anomalous (chargeability) corridors. The main corridor is coincident with the known mineralization at Romero South and Romero (Hondo Valle). It also coincides with a corridor of low resistivity, both of which had been delineated in a north to south direction for a distance in excess of 3.0 km across the central part of the grid. The second corridor, running parallel to the main corridor, is located at the eastern end of the grid and consists of two subsections, the northern section approximately 1.2 km long and the southern section of 0.8 km. In addition to the DCIP program GoldQuest completed a ground magnetic survey during the first quarter of 2012. The survey was completed using the company's magnetometers (GEM GSM-19 system) and field technicians. A total of 72.0 km of magnetometer survey was completed over the same grid used for the DCIP ground survey. Data were plotted and interpreted by external consultants and GoldQuest geologists. An integration of the ground geophysics (magnetic and DCIP), soil and rock geochemistry, alteration, lithology and structural mapping was used to define the sixth and seventh phases of drilling.

The results of the geophysical surveys are shown in Figures 9.1 to 9.3 of Steedman and Gowans (2012). They have been superseded by the maps from the 2012-2013 surveys. A total of 10 targets was identified for testing, based on chargeability, conductivity (resistivity), and magnetic responses, as well as taking into account the detailed and regional geology, alteration zones, surface geochemistry and the results of previous drill holes.

9.4.2 2012 - 2013 Ground IP Survey

In late 2012 and throughout the first half of 2013 GoldQuest contracted Insight Geophysics Inc. to conduct ground IP surveys over the Romero deposit and to expand the coverage to the north and west of the previous Quantec IP survey. The Insight IP survey consisted of 155 km of Gradient IP and 34 km of Insight sections, and produced chargeability and resistivity data looking to a depth of up to 500 m.

Two different grids were surveyed during the program. A north-south oriented grid at 200 m and 100-m spaced lines was conducted over the known mineralization at Romero to compare with the previous Quantec east-west surveys, and to potentially highlight any east-west trends in the mineralization, controlling structures, and/or an alteration package.

In addition to confirming the Romero trend, a component of north-northwest to south-southeast structures, inferred by resistivity lows, and similar potentially mineralized trends, inferred by chargeability highs, were observed to cross the main north-south Romero trend. These are interpreted to be potential secondary structural controls on the main north-south trend.

Insight sections have provided detailed vertical resolution and potentially resolved the contact between the lower andesite and the dacite lithological units, which is thought to be a nearly flat-lying control at Romero. Further, the altered and mineralized zones lying above this contact at Romero are visible as distinct chargeable anomalies, coincident with resistivity lows that indicate the location of the faults of the main north-south Romero trend.

In addition to this grid, an east-west survey using 200-m spaced lines was conducted over the Romero South deposit and to the north and west of the Romero deposit. This survey identified a new set of northwest-southeast to north-northwest to south-southeast-trending chargeability highs coincident with resistivity highs and lows, which has been named the Guama trend.

The Guama trend has several zones with slightly differently oriented target areas. The southern area strikes to the northwest-southeast and remains open at the limit of the survey. This area is 0.75 km wide by 2.5 km long and mostly occurs in the Loma Los Comios concession. It has not yet been drill tested. The central part of the Guama trend is north-northwest to south-southeast-trending and is very linear in geometry. It is 0.75 km wide and 2.3 km long and is, via initial drill testing, at this time believed to be related to the flat flying sediments (mudstones) which come closer to surface in the valley of the Guama creek, which cuts through the topography and is coincident with the anomaly. The northern area of the anomaly widens and generally has a circular orientation which is 1.6 km wide by 1.1 km long and open at the northern limit of the survey. It has been interpreted as a possible porphyry centre that could be related to the Romero trend, alteration and mineralized deposit. This area also falls in the Loma Los Comios concession and has not been drill tested to date.

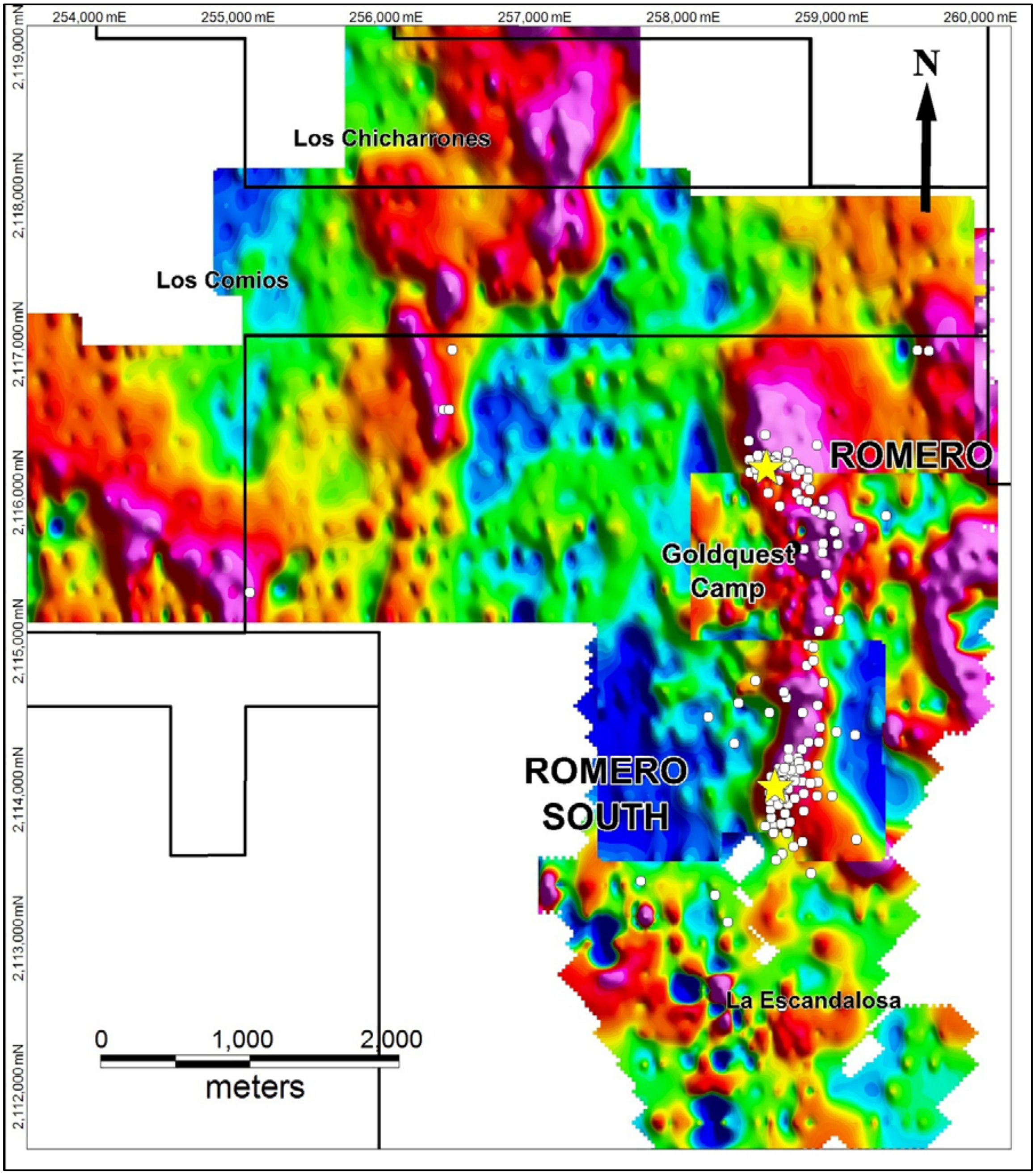
The chargeability map from the 2012-2013 surveys is shown in Figure 9.1, along with the drill hole locations for the Romero and Romero South drilling.

9.4.3 Airborne Z-Axis Tipper Electromagnetic (ZTEM) and Aeromagnetic Geophysics

During the first quarter of 2014 Geotech Limited (Geotech) was contracted to complete a 3,195 line-km helicopter-borne geophysical survey over the entire Goldquest concession package in the San Juan valley. The survey design utilized east-west oriented lines of a minimum length of 10 km with a spacing of 200 m, or 100 m over the core Romero Project area.

In a ZTEM survey, a single vertical-dipole air-core receiver is flown over the survey area in a grid pattern, similar to regional airborne EM surveys. Three orthogonal axis, air-core coils are placed close to the survey site to measure the horizontal EM reference fields. Data from the four coils are used to obtain the Tzx and Tzy Tipper (Vozoff, 1972) components at minimum six frequencies in the 30 to 720 Hz band. The ZTEM data provides useful information on geology using resistivity contrasts while magnetometer data provides additional information on geology using magnetic susceptibility contrasts.

Figure 9.1: 2012-2013 IP Chargeability Results



Source: GoldQuest (2013)

White dots are drill hole collars.

9.4.4 2014 Ground IP Survey

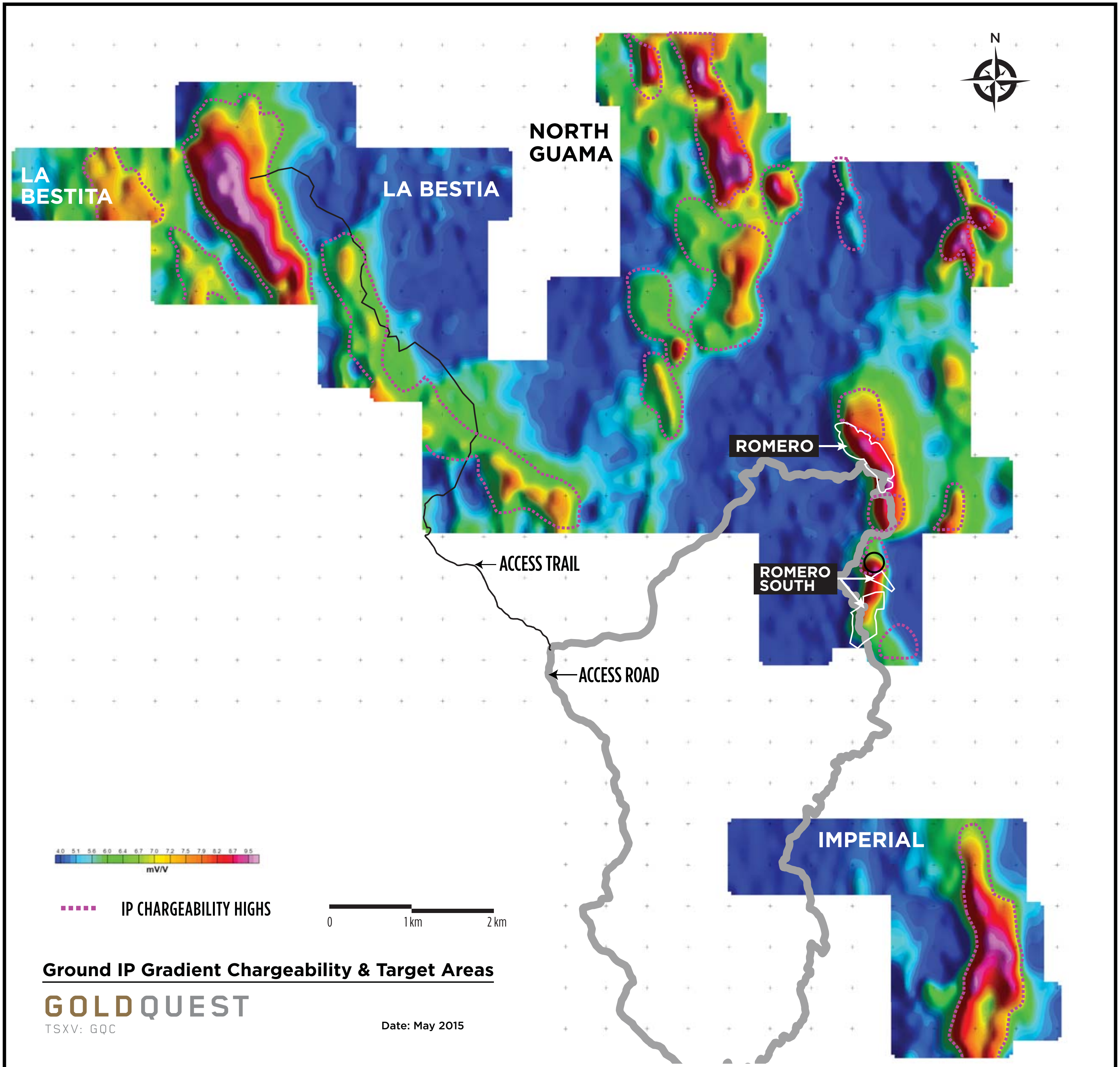
Continuing on from the 2013 Insight IP work, GoldQuest has completed 200 m spaced gradient array coverage to the north, south and to the west of the Romero and Guama trends. The 2014 Insight IP survey consists of 155 km of Gradient IP and 36 km of Insight sections from 37 sections. These have produced chargeability and resistivity data looking to a depth of up to 500 m. During the 2014 survey the La Bestia and Imperial targets were discovered. A summary map of the compiled IP results can be seen in Figure 9.2.

9.5 Deposit Model Confirmation

In January, 2013 Dr. Richard Sillitoe visited the project to assist in the determination of a deposit model and any mineralization vectors which could assist in the delineation or discovery of more mineralization in the Romero trend area. In the course of his work, Dr. Sillitoe examined drill core and field exposures of rocks. His findings have been incorporated into the geological interpretations in this report.

9.6 Summary of Exploration Results

Geological mapping, stream sediment and soil geochemistry and geophysics have confirmed a broad zone of gold and base metal mineralization over a strike length of about 2.2 km, with geophysical anomalies extending over 3.0 km. Several targets for further exploration were identified in the area by geophysics, and soil sampling and trenching programs have assisted in the planning and execution of subsequent drilling programs.



Ground IP Gradient Chargeability & Target Areas

10 Drilling

This section was taken from the 2014 Micon PEA, with information amended from Steedman and Gowans (2012).

10.1 Romero Trend Drilling

Eight programs of diamond drilling (Table 10.1) have been carried out in and around the Romero trend, on the Tireo property, by GoldQuest. As of the database freeze date for the present resource estimate this amounted to a total of 39,628.75 m in 150 holes. The average hole length was 264.2 m with holes in the Romero South area generally being shorter than those at Romero. In the preparation of Steedman and Gowans (2012) only drilling results from Phase 1, 2, 3 and 4 had been verified. Drilling in Phases 5 to 8 was completed after Micon's first site visit in July, 2011. Only drilling results from Phases 1 to 4 were employed in the 2012 mineral resource estimate.

Table 10.1: Drill Program Phases

Phase	Holes	Dates
1	LTP-01 to LTP-17	March - May, 2006
2	LTP-08 to LTP-33	November, 2006 - January, 2007
3	LTP-34 to LTP-42	April-May, 2010
4	LTP-43 to LTP-66	December, 2010 - March, 2011
5	LTP-67 to LTP-76	November - December, 2011
6	LTP-77 to LTP-91	February - April, 2012
7	LTP-92 to LTP-157*	June, 2012 - October, 2013
8	LTP-158 to LTP-164	May - October, 2014

* - Only results up to hole 150 were available for the mineral resource estimate.

Source: Micon 2014

Drilling in Phase 7 continued well into 2013 and was occurring during Micon's 2013 site visit. Its purpose was principally to define the extents of the Romero deposit and to provide enough infill drilling at both Romero and Romero South to model variograms allowing for the planning of the required amount of drilling to raise the mineral resource to the indicated category.

Drilling in Phase 8 was exploration focused and the holes were not drilled in the footprints of the mineral deposits and therefore had no impact on the mineral resources. All holes in the phase were drilled at geophysical targets south of Romero.

Table 10.2 shows a list of all drill holes on the Romero project trend, broken down by phase. Also indicated are those holes which intersected either the Romero or Romero South mineralized wireframes and were used in the mineral resource estimate presented in this report. Those holes not designated are generally along the mineralized Romero trend, between the two deposits.

Table 10.2: Romero Project Drill Holes

Hole-ID	Easting	Northing	Elevation	Length	Az	Dip	Zone Intercept
Phase 1							
LTP-01	258892	2115598	1089.78	148.44	270	-65	Romero
LTP-02	258890	2115598	1090.05	233.17	90	-70	Romero
LTP-03	258965	2115680	1065.04	149.35	270	-60	Romero
LTP-04	258987	2115595	1098.72	150.88	270	-75	Romero
LTP-05	258538	2114030	1076.82	19.79	270	-60	Romero South
LTP-06	258538.5	2114030	1076.96	99.2	310	-60	Romero South
LTP-07	258587	2113979	1109.6	109.73	310	-75	Romero South
LTP-08	258526	2113920	1111.79	80.72	270	-80	Romero South
LTP-09	258534	2113809	1104.81	79.24	304	-75	Romero South
LTP-10	258665	2113725	1124.67	97.62	304	-75	Romero South
LTP-11	258118	2114434	1080.21	41.75	160	-60	not designated
LTP-12	258321	2114527	1114.16	123.48	270	-65	not designated
LTP-13	258434	2114677	1121.8	67.5	270	-60	not designated
LTP-14	258929	2115143	1137.69	187.5	0	-90	not designated
LTP-15	257660	2113326	1190.65	126.7	0	-90	not designated
LTP-16	258246	2113051	1042.09	52.29	0	-90	not designated
LTP-17	258161	2113232	1055.57	45.72	0	-90	not designated
Phase 2							
LTP-18	258655	2114049	1120.61	268.3	0	-90	Romero South
LTP-19	258655	2113948	1142.84	121.92	0	-90	Romero South
LTP-20	258654	2113849	1129.88	102.11	0	-90	Romero South
LTP-21	258761	2113915	1150.79	106.68	0	-90	Romero South
LTP-22	258760	2113800	1146.66	115.82	0	-90	Romero South
LTP-23	258753	2113592	1126.36	105.16	0	-90	Romero South
LTP-24	258746	2113996	1163.89	129.54	0	-90	Romero South
LTP-25	258852	2113993	1179.35	143.26	0	-90	Romero South
LTP-26	258775	2114104	1115.1	307.24	0	-90	Romero South
LTP-27	258659	2114218	1120.73	170.69	0	-90	Romero South
LTP-28	258640	2114561	1111.69	89.92	0	-90	Romero South
LTP-29	258529	2114463	1082.9	85.34	0	-90	Romero South
LTP-30	258290	2114252	996.48	100.58	240	-60	not designated
LTP-31	258911	2115394	1103.62	150.88	0	-90	Romero
LTP-32	258759	2115564	1078.19	100.58	280	-70	Romero
LTP-33	259313	2115788	1186.96	251.46	0	-90	not designated
Phase 3							
LTP-34	258550	2113700	1125.51	82.93	0	-90	Romero South
LTP-35	258555	2113951	1093.29	89.95	0	-90	Romero South
LTP-36	258850	2113900	1155.05	134.16	0	-90	Romero South
LTP-37	258950	2113900	1167.37	170.74	0	-90	Romero South
LTP-38	259104	2114311	1275.36	323.2	180	-75	Romero South
LTP-39	258700	2114100	1104.31	180.2	0	-90	Romero South
LTP-40	258852.5	2113993	1179.48	192.09	0	-90	Romero South
LTP-41	258619	2114011	1107.56	112.81	300	-75	Romero South
LTP-42	258532	2113868	1108.23	74.7	0	-90	Romero South

Hole-ID	Easting	Northing	Elevation	Length	Az	Dip	Zone Intercept
Phase 4							
LTP-43	258539	2113755	1118.14	108.23	0	-90	Romero South
LTP-44	258555	2113650	1120.62	100.58	0	-90	Romero South
LTP-45	258498	2113696	1121.83	88.39	0	-90	Romero South
LTP-46	258608	2113714	1123.89	74.68	0	-90	Romero South
LTP-47	258717	2114156	1100.35	192.02	0	-90	Romero South
LTP-48	258700	2114050	1136.01	157.58	0	-90	Romero South
LTP-49	258700	2114000	1148.87	129.54	0	-90	Romero South
LTP-50	258805	2113986	1166.82	164.59	0	-90	Romero South
LTP-51	258646	2114089	1116.22	112.78	0	-90	Romero South
LTP-52	258590	2114084	1087.11	106.68	0	-90	Romero South
LTP-53	258697	2113885	1141.38	106.68	0	-90	Romero South
LTP-54	258632	2113783	1112.63	94.79	0	-90	Romero South
LTP-55	258644	2113652	1103.11	92.96	0	-90	Romero South
LTP-56	258590	2113842	1115.87	99.06	0	-90	Romero South
LTP-57	258668	2114010	1130.63	152.4	0	-90	Romero South
LTP-58	258615	2113511	1107.62	94.49	0	-90	Romero South
LTP-59	258810	2113381	1128.22	172.21	0	-90	not designated
LTP-60	258691	2113559	1111.53	94.49	0	-90	Romero South
LTP-61	258571	2113471	1102.63	143.26	0	-90	Romero South
LTP-62	258610	2113912	1135.91	121.92	0	-90	Romero South
LTP-63	258853	2114108	1150.08	419.1	0	-90	Romero South
LTP-64	258885	2115538	1104.17	178.31	0	-90	Romero
LTP-65	258944	2115788	1076.65	187.45	0	-90	Romero
LTP-66	258894	2115894	1071.62	172.21	0	-90	Romero
Phase 5							
LTP-67	258566	2113901	1110.63	85.34	0	-90	Romero South
LTP-68	258626	2113882	1133.47	108.2	0	-90	Romero South
LTP-69	258627	2113979	1128.13	124.97	0	-90	Romero South
LTP-70	258597	2113945	1121.09	105.16	0	-90	Romero South
LTP-71	258585	2114027	1098.48	73.15	0	-90	Romero South
LTP-72	258619	2114068	1102.79	114.34	0	-90	Romero South
LTP-73	258726	2114128	1098.66	153.92	0	-90	Romero South
LTP-74	258736	2114077	1105.85	124.97	0	-90	Romero South
LTP-75	258676	2114074	1130.16	124.97	0	-90	Romero South
LTP-76	258526	2113971	1088.8	54.86	0	-90	Romero South
Phase 6							
LTP-77	258746	2114213	1140.73	213.36	0	-90	Romero South
LTP-78	258792	2114261	1179.91	300.23	0	-90	Romero South
LTP-79	258870	2114363	1134.76	176.78	0	-90	Romero South
LTP-80-A	259114	2113607	1144.09	243.23	0	-90	not designated
LTP-81	258854	2114510	1135.33	216.41	0	-90	Romero South
LTP-82	258779	2114780	1175.57	202.69	0	-90	not designated
LTP-83	258659	2114151	1071.44	138.68	0	-90	Romero South
LTP-84	258862	2114262	1171.42	292.61	0	-90	Romero South
LTP-85	258862	2115009	1183.09	97.54	0	-90	not designated
LTP-86	258894	2114664	1159.04	211.84	0	-90	Romero South

Hole-ID	Easting	Northing	Elevation	Length	Az	Dip	Zone Intercept
LTP-87	258826	2114811	1200.82	109.73	0	-90	not designated
LTP-88	258787	2114918	1216.03	109.73	0	-90	not designated
LTP-89	258838	2115824	1123.72	213.36	0	-90	Romero
LTP-90	258503	2116119	1115.17	265.23	0	-90	Romero
Phase 7							
LTP-91	258711	2115942	1077.96	234.7	0	-90	Romero
LTP-92	258485	2116109	1108.82	398.98	0	-90	Romero
LTP-93	258527	2116121	1119.17	432.82	0	-90	Romero
LTP-94	258506	2116143	1124.91	406.91	0	-90	Romero
LTP-95	258503	2116089	1096.8	287.45	180	-80	Romero
LTP-96	258577	2116137	1131.35	381	0	-90	Romero
LTP-97	258505	2116192	1129.82	401.42	0	-90	Romero
LTP-98	258577	2116190	1132.59	432.82	0	-90	Romero
LTP-99	258458	2116137	1116.87	461.66	0	-90	Romero
LTP-100	258643	2116151	1115.97	505.05	0	-90	Romero
LTP-101	258395	2116166	1125.46	417.58	0	-90	Romero
LTP-102	258450	2116192	1122.56	403.86	0	-90	Romero
LTP-103	258644	2116113	1101.64	468.82	0	-90	Romero
LTP-104	258452	2116053	1084.67	381	0	-90	Romero
LTP-105	258587	2116026	1079.26	231.65	0	-60	Romero
LTP-106	258520	2115942	1118.45	704.08	0	-70	Romero
LTP-107	258708	2116060	1091.49	413.31	0	-90	Romero
LTP-108	258587	2116026	1079.26	449.58	0	-90	Romero
LTP-109	258734.6	2115880	1110.87	296.85	0	-90	Romero
LTP-110	258587	2116026	1079.26	327.66	180	-60	Romero
LTP-111	258771.2	2115994.6	1116.85	528.63	0	-90	Romero
LTP-112	258722	2116153	1117.5	522.73	0	-90	Romero
LTP-113	258520	2115942	1118.45	621.79	0	-90	Romero
LTP-114	258771.2	2115994.6	1116.85	509.03	270	-90	Romero
LTP-115	258733.5	2116097.5	1115.95	498.35	0	-90	Romero
LTP-116	258440	2116098	1100.49	414.53	0	-90	Romero
LTP-117	258800	2115963	1115.67	750.11	0	-90	Romero
LTP-118	258735	2116096	1116.69	419.3	260	-75	Romero
LTP-119	258399	2116080	1111.21	451.1	0	-90	Romero
LTP-120	258543	2116157	1131.93	762.05	0	-90	Romero
LTP-121	258735	2116096	1116.69	192.47	260	-75	Romero
LTP-122	258800	2115963	1115.67	469.39	220	-70	Romero
LTP-123	258618	2116128	1118.77	505.97	0	-90	Romero
LTP-124	258789	2116039	1124.61	510.54	260	-70	Romero
LTP-125	258625	2114600	1117.89	516.3	90	-60	Romero South
LTP-126	258789	2116039	1124.61	522.73	0	-90	Romero
LTP-127	258648	2116216	1135.02	650.19	0	-90	Romero
LTP-128	258752	2114462	1092.17	530.35	135	-82	Romero South
LTP-129	258789	2115880	1128.31	477.62	0	-90	Romero
LTP-130	258631	2114087	1109.26	503.22	0	-90	Romero South
LTP-131	258789	2115879	1128	535.22	250	-75	Romero
LTP-132	258789	2115879	1128	534.94	180	-65	Romero

Hole-ID	Easting	Northing	Elevation	Length	Az	Dip	Zone Intercept
LTP-133	258977	2114329	1210.84	522.73	0	-90	Romero South
LTP-134	259132	2115711	1082.9	644.64	0	-90	not designated
LTP-135	258997	2115087	1182.84	450.4	180	-65	not designated
LTP-136	258598	2115851	1091.43	614.17	360	-80	Romero
LTP-137	258499	2116330	1202.96	594.87	180	-75	Romero
LTP-138	258387	2116289	1136.88	557.78	0	-90	Romero
LTP-139	258565	2113972	1095.62	118.87	0	-90	Romero South
LTP-140	258584	2116146	1132.95	573.02	200	-80	Romero
LTP-141	258606	2113996	1118.21	150.88	0	-90	Romero South
LTP-142	258610	2113962	1127.99	111.25	0	-90	Romero South
LTP-143	258584	2116146	1132.95	388.62	200	-70	Romero
LTP-144A	258648	2116117	1100.91	451.1	200	-80	Romero
LTP-145	258648	2116117	1100.91	460.25	200	-70	Romero
LTP-146	258835	2115822	1124.86	350	190	-70	Romero
LTP-147	258782	2115879	1130.64	377.33	0	0	Romero
LTP-148	258880	2115798	1108.3	262.13	0	0	Romero
LTP-149	258880	2115798	1108.3	316.99	0	0	Romero
LTP-150	258790	2116079	1140	470.92	225	-60	Romero
LTP-151	258880	2115798	1119	364.24	180	-70	Romero
LTP-152	258880	2115798	1119	411.48	120	-70	Romero
LTP-153	258790	2116079	1140	371.86	0	-90	Romero
LTP-154	258880	2115798	1119	268.22	45	-70	Romero
LTP-155	258824	2114902	1249	548.64	95	-75	Romero
LTP-156	258850	2116261	1210	650.75	250	-70	Romero
LTP-157	258612	2112482	992	253.9	220	-50	Higuera
Phase 8							
LTP-158	258866	2115267	1134	409.96	0	-90	Romero Trend
LTP-159	259021	2113897	1196	591.31	0	-90	Romero Trend
LTP-160	258945	2115218	1159	312.42	0	-90	Romero Trend
LTP-161	259052	2115396	1170	316.99	0	-90	Romero Trend
LTP-162	257120	2117656	1479	323.09	0	-90	Romero Trend
LTP-163	257202	2118265	1502	288.04	0	-90	Romero Tremd
LTP-164	257351	2118873	1428	252.98	190	-70	Romero Trend

Easting and Northing are coordinates are in UTM NAD 27 Conus.

Azimuths are in degrees relative to grid north. They were corrected for magnetic declination of 10°19’ west.

Source: GoldQuest 2015

The drill contractor for all seven programs was Energold Drilling Corporation of Vancouver using man-portable, hydraulic Hydracore Gopher diamond drills, with NTW (56.0 mm diameter) and BTW (42.0 mm diameter) core (seeFigure 10.1). Supplies were brought to the rigs and core, sealed in wooden boxes, was transported out by mules rented from the local farmers.

Figure 10.1: Drill Rig at Romero

Source: Micon 2014

The Phase 1 program comprised 17 drill holes for 1,813.08 m in Hondo Valle, Los Tomates, Romero South and La Higuera (Hoyo Prieto) (holes LTP-01 to LTP-17). They were drilled between March 17, 2006 and May 6, 2006. The program is described in reports by MacDonald (2006) and Redwood (2006a). Magnetic susceptibility readings were taken from 10 holes from the Phase 1 program.

The Phase 2 program comprised 16 holes for a total of 2,349.48 m at Romero South and Hondo Valle (holes LTP-18 to LTP-33). The drilling was carried out between November 16, 2006 and January 29, 2007. The program is described in a report by Vega (2007).

The Phase 3 program was carried out at Romero South and comprised nine holes for 1,360.78 m (holes LTP-34 to LTP-42). It was carried out between April 15, 2010 and May 17, 2010. The program is described in a report by Gonzalez (2010).

The Phase 4 program comprised 24 holes for a total of 3,364.40 m including 21 holes in the Romero South area and three at Hondo Valle which were later added to the Romero interpretation (holes LTP-43 to LTP-66). The drilling was carried out between December 18, 2010 and March 22, 2011. The program is described in a report by Gonzalez (2011).

The Phase 5 program comprised 10 holes for a total of 1,069.88 m at Romero South (holes LTP-67 to LTP-76). The drilling was carried out between November 14, 2011 and December 6, 2011. The program is described in a report by Gonzalez (2011).

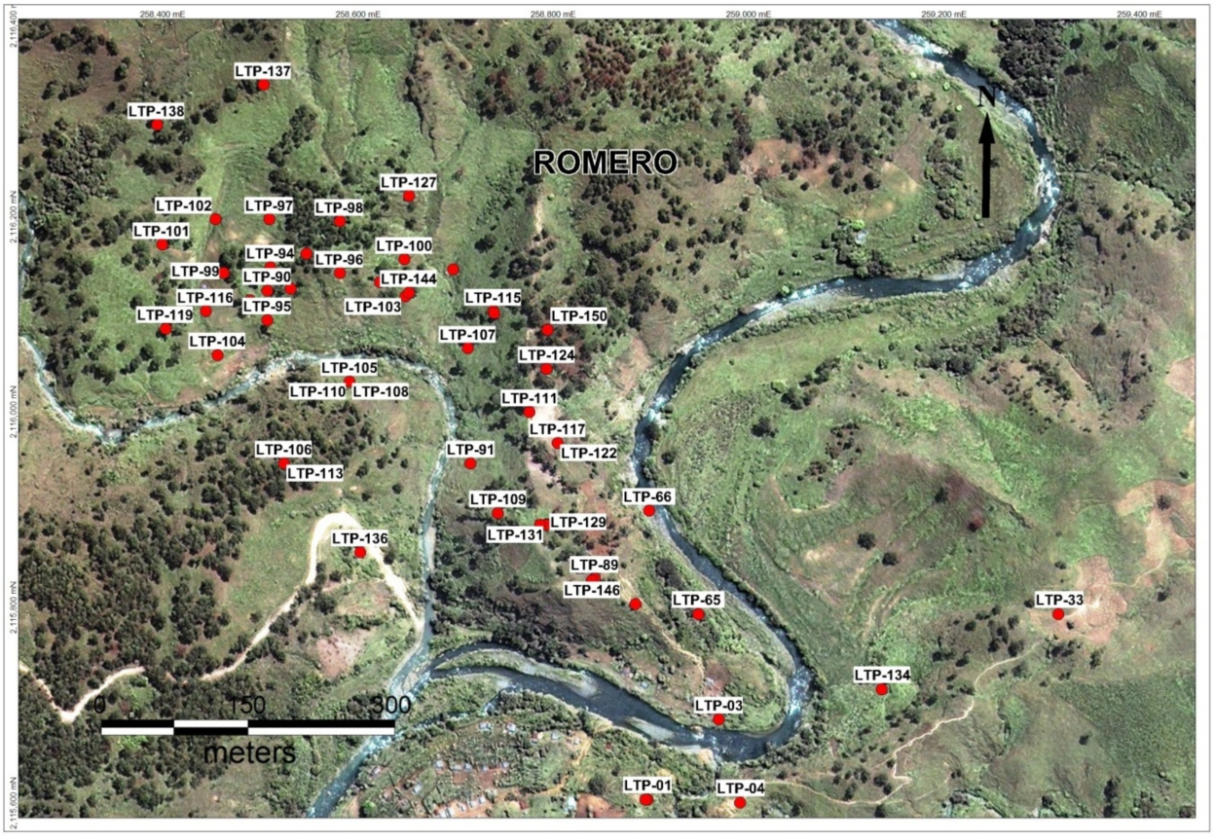
The Phase 6 and 7 programs consisted of 74 drill holes for 29,671.13 m at Romero/Hondo Valle, Los Tomates, and Romero South (holes LTP-77 to LTP-150). Their principal purpose was the delineation and definition of Romero and Romero South. The holes were drilled between February, 2012 and October, 2013 with intermittent brief breaks. The early portions of the program are described in reports by Gonzalez (2012).

The Phase 8 program comprised seven holes in the Romero Trend for a total of 2,494.70 m (holes LTP-158 to LTP-164). The drilling was carried out between May and October, 2014. All holes targeted new mineralization at geophysical targets outside of the Romero and Romero South deposits.

Down hole surveys were carried out from Phase 4 onwards. Drill hole deviations (if any) are expected to be minimal since most of the early drill holes are fairly shallow (i.e. averaging 106.65 m, 146.84 m, 151.20 m and 140.18 m for Phases 1 to 4 respectively) and only a few exceed 250 m.

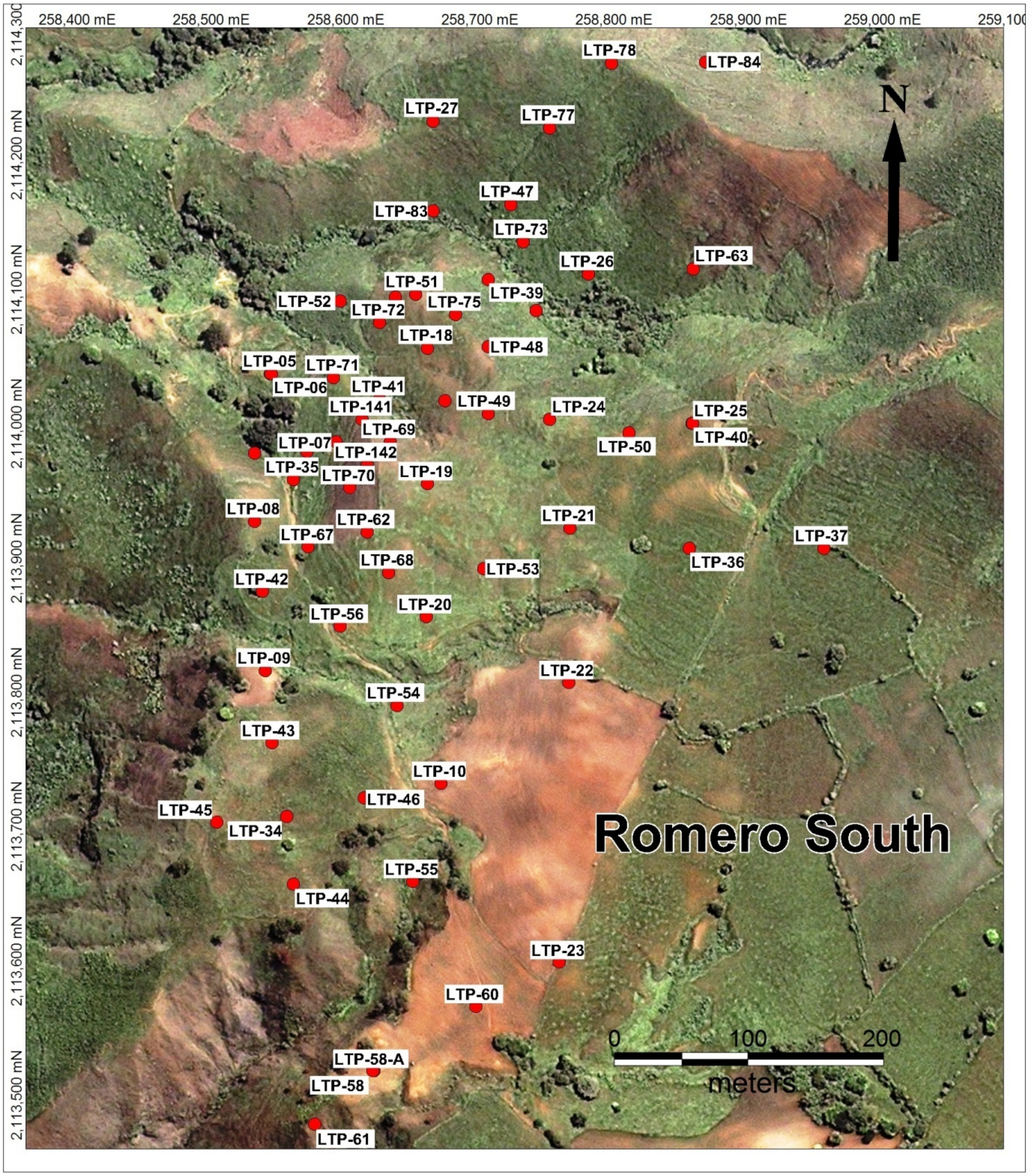
Plan views of the drill hole locations at Romero and Romero South are shown on satellite photos in Figure 10.2 and Figure 10.3, respectively.

Figure 10.2: Location of Drill Holes at Romero



Source: GoldQuest, 2013

Figure 10.3: Location of Drill Holes at Romero South



Source: GoldQuest, 2013

The geological drill logs record recovery, rock quality designation (RQD), structures, lithology, alteration and mineralization.

Drill platforms, mud sumps and access paths were re-contoured and re-vegetated after use.

Drill holes were capped and marked with plastic pipe set in cement.

Drill hole results, as disclosed in press releases by GoldQuest, are presented in Table 10.3 and Table 10.4 below. Table 10.3 shows those results available as of the 2012 mineral resource estimate (Steedman and Gowans, 2012). Table 10.4 shows those results disclosed afterward. Missing hole numbers were drilled on targets other than Romero and Romero South and are not reported here. GoldQuest did not routinely disclose copper assays until part way through the drill programs when the potential importance of those results became more apparent.

Table 10.3: Significant Gold Intersections from the Romero Project – Phase 1 to Early Phase 6

Hole No.	From (m)	To (m)	Interval (m)	Au (g/t)	Cu (%)	Location
LTP-01	0	20	20	0.98	*	Hondo Valle
LTP-02	0	42	42	1.68	*	Hondo Valle
including	0	20	20	2.65	*	
LTP-03	8	149.35	141.35	0.31	*	Hondo Valle
including	8	100	92	0.35	*	
LTP-05	0	14	14	0.5	*	Escandalosa Sur
LTP-06	0	20	20	0.26	*	Escandalosa Sur
LTP-07	26	86	60	2.07	*	Escandalosa Sur
including	38	76	38	3.15	*	
including	38	56	18	6.11	*	
LTP-08	38	64	26	0.84	*	Escandalosa Sur
including	38	50	12	1.74	*	
LTP-09	34	50	16	2.1	*	Escandalosa Sur
including	34	42	8	3.81	*	
LTP-10	60	84	22	0.31	*	Escandalosa Sur
LTP-14	8	58	50	0.28	*	Hondo Valle
LTP-18	60	108	48	0.29	*	Escandalosa Sur
LTP-19	78.46	110.56	32.1	0.37	*	Escandalosa Sur
LTP-20	65	87	22	0.27	*	Escandalosa Sur
LTP-21	78	104	26	0.24	*	Escandalosa Sur
LTP-22	74	112	38	0.17	*	Escandalosa Sur
LTP-23	62	70	8	0.18	*	Escandalosa Sur
LTP-24	102.46	129.54	27.08	0.33	*	Escandalosa Sur
LTP-26	124	153.9	29.9	0.2	*	Escandalosa Sur
LTP-27	115	127	12	0.11	*	Escandalosa Sur
including	161	170.69	9.69	0.15	*	
LTP-28	36	49.28	13.28	0.15	*	Los Tomates
LTP-30	96	100.58	4.58	0.13	*	Los Tomates
LTP-31	12	118	106	0.11	*	Hondo Valle
including	12	35.46	23.46	0.21	*	Hondo Valle
LTP-32	8	36.45	28.45	0.36	*	Hondo Valle
including	26	36.45	10.45	0.84	*	Hondo Valle
LTP-34	61.02	68.11	7.09	5.85	0.3	Escandalosa Sur
LTP-35	18	56	38	0.84	0.08	Escandalosa Sur
including	28	36	8	3.12	0.33	
LTP-36	No significant values					
LTP-37	No significant values					
LTP-38	282	318	36	0.12	0.02	Escandalosa Sur
LTP-39	66	92	26	11.39	0.28	Escandalosa Sur
including	68	86	18	16.33	0.29	
and	101.63	142	40.37	0.21	0.07	
LTP-40	178	192.09	14.09	0.18	0.02	Escandalosa Sur
LTP-41	25	78	53	3.02	0.09	Escandalosa Sur
including	36	52	16	9.39	0.18	

Hole No.	From (m)	To (m)	Interval (m)	Au (g/t)	Cu (%)	Location
LTP-42	35.23	58	22.77	1.33	0.1	Escandalosa Sur
including	38	48	10	2.74	0.2	
LPT-43	No significant values					
LPT-44	No significant values					
LTP-45	58.88	62.05	3.17	2.62	*	Escandalosa Sur
LTP-46	56.48	62	5.52	1.01	*	Escandalosa Sur
LTP-47	110	126	16	2.45	*	Escandalosa Sur
LTP-48	88.78	98	9.22	3.54	*	Escandalosa Sur
LTP-49	74	94	20	1.32	0.39	Escandalosa Sur
including	74	86	12	2.04	0.24	
LPT-50	No significant values					
LPT-51	No significant values					
LTP-52	46	58	12	0.32	*	Escandalosa Sur
LTP-53	84	92	8	0.46	*	Escandalosa Sur
LTP-54	57	63	6	0.4	*	Escandalosa Sur
LPT-55	No significant values					
LTP-56	42.37	69.06	26.69	0.37	nsv	Escandalosa Sur
including	55	61	6	0.97	nsv	
LTP-57	56.68	84	27.32	0.17	nsv	Escandalosa Sur
including	76	82	6	0.38	nsv	
LPT-58	No significant values					
LPT-59	No significant values					
LPT-60	No significant values					
LPT-61	No significant values					
LTP-62	63.5	100	36.5	2.74	*	Escandalosa Sur
including	63.5	76.63	13.13	6.6	*	
LTP-63	No significant values					Escandalosa
LTP-64	1.07	56	54.93	0.57	nsv	Hondo Valle
including	1.07	16	14.93	0.78	nsv	
LTP-65	50	79	29	2.18	0.25	Hondo Valle
including	58	75	17	3.45	0.42	
including	67.61	69.05	1.44	14.2	2.04	
LTP-66	111.82	133.97	22.15	0.66	0.12	Hondo Valle
LTP-67	34	42	8	1.95	*	Escandalosa Sur
	51.95	56	4.05	0.95	*	Escandalosa Sur
LTP-68	84	88.13	4.13	0.78	*	Escandalosa Sur
LTP-69	56	84	28	3.57	*	Escandalosa Sur
including	56	76	20	4.87	*	
and	96	100	4	0.98	*	
LTP-70	46	60	14	5.34	*	Escandalosa Sur
and	88	94	6	1.4	*	
LTP-71	20	40	20	4.04	*	Escandalosa Sur
LTP-72	64	68	4	1.51	*	Escandalosa Sur
and	96	100	4	2.18	*	
LTP-73	75.33	82	6.67	2.33	*	Escandalosa Sur
and	100	116	16	3.3	*	
LTP-74	70	88	18	1.01	*	Escandalosa Sur
and	98	110	12	0.83	*	

Hole No.	From (m)	To (m)	Interval (m)	Au (g/t)	Cu (%)	Location
LTP-75	85.78	102	16.22	5.5	*	Escandalosa Sur
including	88	99.68	11.68	7.51	*	
LTP-76	12	24	12	6.8	*	Escandalosa Sur
LTP-77	160	168	8	0.72	nsv	Escandalosa Sur
and	198	202	4	0.73	nsv	
LTP-79	52.27	68	15.73	0.91	nsv	Escandalosa Sur
including	60	68	8	1.28	nsv	
LTP-81	154	166	12	0.89	nsv	Los Tomates
and	194	198	4	0.55	nsv	
LTP-82	50	54	4	0.33	nsv	Los Tomates
LTP-83	34	56	22	5.99	0.23	Escandalosa Sur
including	38	52	14	9.07	0.24	
LTP-84	264	271.9	7.9	2.96	0.52	Escandalosa Sur
and	278	282	4	0.72	nsv	
LTP-85	26.6	36.61	10.01	0.53	nsv	Hondo Valle
LTP-86	136	138	2	0.34	nsv	Los Tomates
LTP-87	74	78	4	0.38	nsv	Los Tomates Norte
LTP-88	64	70	6	0.44	nsv	Los Tomates Norte
LTP-89	130	151.43	21.43	0.66	0.34	Hondo Valle
including	146	151.43	5.43	1.69	0.97	
and	177	205	28	0.67	0.13	
including	195	205	10	1.27	0.12	

* = no value reported, nsv = no significant values

Source: Micon 2014

Table 10.4: Significant Gold Intersections from the Romero Project – Late Phase 6 and Phase 7

Hole_ID	From (m)	To (m)	Interval (m)	Uncut Gold Grade (g/t)	Copper (%)	Gold Grade (cut to 50 g/t)
LTP-90	33	264	231	2.42	0.44	
including	33	91	58	1.36	0.04	
including	200	258	58	4.7	0.78	
including	103.74	264	160.26	2.9	0.62	
including	103.74	148	44.26	3.53	0.77	
including	180	203.97	23.97	1.14	0.78	
including	216	258	42	6.26	1.04	
including	216	228	12	16.95	2.14	
LTP-91	186	222	36	1.14	0.37	
including	191.95	206	14.05	2.36	0.72	
or	204	234.7	34.7	0.48	0.17	
LTP-92	28.2	82	53.8	0.63	0.02	0.63
and	120	144	24	7.5	0.86	6.88
and	212.5	372	159.5	4.45	0.95	4.14
including	212.5	288	75.5	9.01	1.06	8.35
including	243.93	288	44.07	15.03	1.43	13.9
including	320	346	26	0.54	2.04	0.54
LTP-93	44.58	100	55.42	1.27	0.03	1.27
and	119.97	378	258.03	4.47	1.27	3.44
including	126	324.47	198.47	5.69	1.54	4.34
LTP-94	68	95.21	27.21	0.67	0.05	0.67
and	131.23	366	234.77	7.88	1.43	4.71
including	139	349	210	8.77	1.56	5.21
including	142.5	246.12	103.62	13.17	1.55	7.74
including	142.5	178.85	36.35	28.16	1.9	14.88
LTP-95	24.41	42	17.59	1.79	0.03	1.79
and	54	91.75	37.75	0.6	0.01	0.6
and	184	285.9	101.9	0.73	0.15	0.73
LTP-96	122.49	311	188.51	3.14	1.07	2.83
including	169.12	203	33.88	14.21	1.38	12.48
and	346.84	381	34.16	0.45	0.59	0.45
LTP-97	185.48	222.59	37.11	0.57	0.28	0.57
and	230	278	48	1.41	0.21	1.41
and	312	391	79	2.33	0.29	2.33
LTP-98	184	294	110	0.57	0.24	0.57
including	220	270	50	1	0.32	1
and	361.05	432.81	71.76	0.53	0.16	0.53
LTP-99	124.1	164	39.9	0.62	0.07	0.62
and	254.34	335.45	81.11	0.51	1.31	0.51
and	367.86	400.81	32.95	0.45	0.03	0.45
LTP-100	184	210	26	1.13	0.3	1.13
and	240	256	16	0.8	0.16	0.8
and	353.32	476	122.68	2.64	0.33	2.5
including	398	442	44	6.35	0.53	5.97
LTP-101	268	289	21	1.89	0.07	1.89
and	388	400	12	0.17	0.01	0.17

Hole_ID	From (m)	To (m)	Interval (m)	Uncut Gold Grade (g/t)	Copper (%)	Gold Grade (cut to 50 g/t)
LTP-102	173.85	194	20.15	0.43	0.04	0.43
and	228	274	46	1.01	0.48	1.01
and	296	338	42	0.46	0.64	0.46
and	374	388	14	0.21	0.01	0.21
LTP-103	193.37	425	231.63	2.04	0.3	1.91
including	193.37	229	35.63	5.08	0.53	5.08
including	241	309	68	2.84	0.24	2.38
including	332.65	425	92.35	1.06	0.27	1.06
LTP-104	164	246	82	0.61	0.2	0.61
LTP-105	60	99	39	1.04	0.1	
and	119.47	231.65	112.18	0.87	0.43	
including	119.47	149	29.53	2.16	0.47	
LTP-106	195	361	166	0.67	0.16	
including	203	287	84	0.91	0.2	
LTP-107	145	246	101	1.6	0.74	
including	206	242	36	3.52	1.07	
LTP-108	64.79	109.46	44.67	1.49	0.03	
and	142	299	157	1.07	0.4	
including	165.5	202.69	37.19	3.31	1	
LTP-109	130	145.68	15.68	0.42	0.01	
LTP-110	97.97	109.73	11.76	0.55	0.01	
and	186.35	210.7	24.35	0.43	0.05	
LTP-111	163	243	80	0.93	0.85	
including	187	239	52	1.31	1.24	
including	191.75	227	35.25	1.58	1.65	
including	191.75	223	31.25	1.71	1.63	
LTP-112	188.75	204	15.25	0.27	0.03	
and	511	515	4	1.73	0.08	
LTP-113	No significant results					
LTP-114	237	301	64	0.93	0.16	
LTP-115	No significant results					
LTP-116	243	328	85	0.79	0.89	
LTP-117	173	239	66	0.47	0.16	
LTP-118	201	418.5	217.5	0.74	0.4	
including	273.22	322	48.78	2.06	0.71	
LTP-119	No significant results					
LTP-120	73	104.84	31.84	1.02	0.03	
and	131	165	34	0.32	0.22	
and	183	420	237	0.67	0.43	
including	335	392	57	2.16	0.85	
LTP-121	Hole stopped due to drilling problems					
LTP-125	63.08	68.58	5.5	0.36	-	0.36
and	354	369	15	0.36	-	0.36
and	407	413	6	0.35	-	0.35
LTP-126	176.45	209	32.55	0.17	-	0.17
and	221	249	28	0.17	-	0.17
LTP-127	410	458	48	0.17	0.04	0.17
	480.36	495	14.64	0.28	0.17	0.28

Hole_ID	From (m)	To (m)	Interval (m)	Uncut Gold Grade (g/t)	Copper (%)	Gold Grade (cut to 50 g/t)
LTP-128	92	134	42	0.57	-	0.57
and	245	261	16	0.28	-	0.28
and	346	382	36	0.61	-	0.61
LTP-129	210	216	6	1.68	0.66	1.68
and	234	265	31	0.45	0.13	0.45
LTP-130	79.35	89.46	10.11	2.72	0.09	2.72
and	124	140	16	0.76	0.35	0.76
LTP-131	212	240	28	0.42	0.06	0.42
LTP-132	136	266	130	1.22	0.24	1.22
including	185.03	202.04	17.01	6.21	0.9	6.21
LTP-133	281.43	318	36.57	0.38	0.12	0.38
LTP-134	No significant result					
LTP-135	442.8	449.58	6.78	4.62	0.01	4.62
LTP-136	526	538	12	0.63	0.07	0.63
LTP-137	250.87	310.22	59.35	0.53	0.06	0.53
and	380	502.72	122.72	0.92	0.24	0.92
including	400.83	466	65.17	1.3	0.31	1.3
LTP-138	129.85	164.69	34.84	0.53	0.05	0.53
and	210	243.47	33.47	0.62	0.03	0.62
LTP-139	21	42.13	21.13	4.58	0.24	4.57
LTP-140	127	396.35	269.35	2.35	0.56	2.12
including	246	278	32	9.95	1.58	9.95
LTP-141	33.55	62	28.45	10.11	0.31	7.03
and	74	88	14	0.35	0.14	0.35
LTP-142	41.92	100	58.08	4.03	0.21	2.74
including	46	76	30	7.69	0.37	5.19
LTP-143	118	333.76	215.76	2.54	0.6	2.54
including	150	184	34	10.94	1.87	10.94
LTP-144a	155	327	172	0.99	0.33	0.99
and	155	193	38	1.99	0.18	1.99
LTP-145	114	341	227	1.78	0.44	1.78
including	131	178	47	6.9	0.94	6.9
LTP-146	103.64	223	119.36	0.64	0.2	0.64
including	103.64	170	66.36	0.84	0.32	0.84
LTP-147	140	176	36	0.65	0.07	0.65
LTP-148	76.77	89	12.23	0.79	0.02	0.79
and	107	204.22	97.22	0.45	0.05	0.45
including	115.82	169	53.18	0.59	0.08	0.55
LTP-149	88.52	203	114.48	0.38	0.26	0.38
LTP-150	153.8	225.5	71.7	3.14	0.07	3.14
including	199.78	225.5	25.72	7.8	0.17	2.24
and	288.58	371	82.42	0.82	0.21	0.82

Source: Micon 2014

No results for the seven holes after LTP-150 were available at the time of estimation of the mineral resources used in this PEA. Five of the holes were targeted at generally distal areas around Romero. The sixth was drilled between Romero and Romero South and the seventh was at La Higuera, 1.5 km south of Romero South.

Recoveries of drill core were generally quite high, with the exception of local, isolated problem areas. GoldQuest began recording core recovery with hole LTP-74. From there to hole LTP-150 recoveries have averaged 94%.

It is Micon's opinion that there are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results received. Subject to appropriate analytical results (see Sections 11 and 12 below) the samples recovered are suitable for use in a mineral resource estimate.

Romero South is a relatively flat tabular deposit in which most drill holes intersected at roughly 90º representing approximately true intersections. To the northwest, the zone does roll over into a shallow northwest dip where true widths will be somewhat less than intersected widths.

Romero is a relatively more complex deposit shape in which mineralization has permeated a somewhat permeable host rock. The resulting mineralized shape is amoeba-like but has large contiguous areas of above cut-off mineralization and a relatively consistent dip and strike. Drill holes intersected it from various angles and dips as potential collar locations were limited by steep topography and restrictions about drilling close to creeks and rivers. The combination of the amoeboid shape and varying drill azimuths and dips means that there is no clear or consistent relationship between intersected widths and true widths. Section 14 provides figures which attempt to display the relationship.

10.2 Other Drilling

GoldQuest has also drilled 24 holes on the geophysical targets La Guama (LG-01 to LG-05), La Rosa (LR-01 and LR-02), La Bestia (LB-01 to LB-09) and Imperial (IMP-01 to IMP-08). La Guama is located about 1.5 km northwest of Romero, La Rosa is approximately 1 km northeast of Romero, La Bestia is approximately 8 km northwest of Romero and Imperial is approximately 2.5 km south of Romero South. All targets are chargeability highs from IP surveys; and varying amounts of sulphides, mainly pyrite, were encountered. These drill targets and their results do not affect the mineral resource estimate presented in this report and they will not be discussed further.

11 Sample Preparation, Analyses and Security

This section was taken from the 2014 Micon PEA with information amended from Steedman and Gowans (2012). In the preparation of that report only drilling results from Phases 1, 2, 3 and 4 were verified. Drilling in Phases 5, 6 and 7 was verified for the Micon PEA and the mineral resource used in this report.

11.1 Sampling Method and Approach

The initial indications of mineralization on the La Escandalosa concession were found by fine fraction stream sediment sampling and float sampling carried out as part of a regional stream sediment geochemistry exploration program.

The main exploration technique used for definition of drill targets was soil sampling. A total of 1,090 soil samples were taken in several programs between 2005 and 2010 and analyzed for gold and multi-elements. Soil samples were taken from the B horizon and were not sieved. The average sample weight was about 0.5 kg. Sampling was on grids of 50 m by 50 m, and 100 m by 100 m, and along ridges and spurs in reconnaissance areas. The area sampled on grids is about 2.0 km long north-south by 1.0 km across, and the total area sampled, including ridges and spurs, is about 4.0 km north-south by 3.0 km wide.

Rock sampling was carried out as grab samples of outcrop and float, and channel samples from hand-dug pits and trenches. A total of 1,176 rock samples was collected. Samples were 2 to 4 kg in weight and were analysed for gold and multi-elements. Surface rock samples are collected to check for the existence of mineralization, but not to quantify it, and were not used for resource estimation.

Diamond drilling was carried out using NTW (56.0 mm diameter) and BTW (42.0 mm diameter) core. Sample intervals in the core were selected by the geologist after geological logging. The sample intervals are generally 2.00 m. Priority was given to geological contacts so that some intervals may be shorter. In areas of low recovery the sample interval is between drill run markers. The median sample length is 2.00 m (n = 3519 samples captured in the Romero mineralized solid and 532 samples in the Romero South mineralized solid). The minimum sample length at Romero is 0.38 m and the maximum is 6.25 m. The minimum sample length at Romero South is 0.32 m and the maximum is 2.91 m. The core samples were cut lengthwise by diamond saw and one-half of the core was sampled, and the other half left in the core box for reference. Samples were collected in heavy duty clear plastic sample bags which were sealed with plastic cable-ties. A sample ticket was glued on the core box at the start of the sample interval. Another sample ticket was inserted in the bag and the number written on the outside of the bag with indelible marker pen.

The upper part of two holes were not sampled or analysed, although they were marked up with sample numbers; these were LTP-38 from 0 to 220 m due to no mineralization, and LTP-40 from 0 m to 142.36 m as it was a twin of hole LTP-25 designed to drill deeper to reach the target. In Phase 1 to 7, there were 14,474 analyses for core as well as 1,608 blanks, 265 pulp and 327 field duplicate samples, as well as 3,556 standards inserted.

11.2 Sample Security and Chain of Custody

Soil and rock samples were collected in heavy duty paper and plastic sample bags respectively, sealed with wire ties and plastic cable ties respectively. A detailed sample description form was filled in for each sample, and a tear-off sample ticket inserted in the bag.

Core samples were placed into wooden core boxes by the drillers. Core was collected from the drill rig by GoldQuest field assistants and taken to the core shack at Hondo Valle for logging and sampling.

The core was logged and marked for sampling by GoldQuest geologists. The core samples were cut lengthwise by diamond saw and one-half core was sampled. The other half was left in the core box for reference. All of the split core is stored at GoldQuest's core storage facility at Hondo Valle.

Stream sediment, soil, rock and core samples from the Phase 1 and 2 drill programs (holes LTP-01 to LTP-33) were shipped to ALS Chemex Ltd (ALS Chemex), Vancouver, Canada for preparation and analysis. This laboratory is independent of GoldQuest and complies with the requirements of international standards ISO 9001:2000 and ISO 17025:1999. The whole sample was shipped as there was no sample preparation facility in the Dominican Republic at that time.

The samples were bagged in nylon sacks and taken by GoldQuest vehicle to the GoldQuest office in Santo Domingo, where standard and blank samples were inserted and sample shipment forms prepared. The samples were then taken to Punta Cana by GoldQuest vehicle, about a four hour drive, and sent by air to Vancouver. It was found that the best air freight rates could be obtained from Punta Cana on direct holiday charter flights to Vancouver, with an average time of two to three days to reach the laboratory. Other courier and air freight routes from Santo Domingo were found by previous experience to be much more expensive, slower and prone to delays due to cargo being carried when space was available.

From September, 2007, all soil, rock and core samples from the Phase 3 and onward drill programs (hole LTP-34 and on) were prepared at Acme Analytical Laboratories Ltd.'s (Acme) new sample preparation facility in Maimon, Dominican Republic. Samples were delivered by GoldQuest vehicle. Acme is registered with ISO 9001:2000 and ISO 17025 accreditation.

11.3 Sample Preparation

Sample preparation for rock and core samples at ALS Chemex in Vancouver was to log the sample into the tracking system; record the weight; dry; crush the entire sample to >70% passing 2 mm; split off 1.5 kg; and pulverize the split to >85% passing 75 microns (method PREP-32). Coarse rejects and pulps are stored at the laboratory. Soil samples were prepared by sample login; record weight; dry, disaggregate and sieve sample to -80 mesh (method PREP-41). Some assay certificates indicate that for some soil sample orders a split of unspecified weight was pulverized to >85% passing 75 µm (method PUL-31).

Rock and drill core sample preparation by Acme in Maimon comprised logging the sample into the Acme tracking system with a bar code; dry in an electric oven; crush by Terminator jaw crusher to 80% passing -10 mesh (2 mm); and 300 g split by riffle splitter. The sample split was then shipped by courier, by Acme, to their laboratory in Santiago, Chile or Vancouver for pulverization to 95% passing -150 mesh (106 µm) (method R150). Soil samples were prepared by drying at 60°C; and sieving a 100 g split to -80 mesh. Coarse rejects for core, rock and soil samples were returned to GoldQuest and are stored at GoldQuest's core store in Bonao. Pulps are stored at Acme's laboratory in Chile.

11.4 Sample Analysis

There are a total of 1,176 rock sample analyses, 1,090 soil sample analyses and 14,611 drill core analyses, excluding QC samples.

ALS Chemex analysed samples in its Vancouver laboratory (VA assay certificate number prefixes) for gold by fire assay (30 g) with measurement by inductively coupled plasma atomic emission spectrometer (ICP-AES or ICP-ES) (method Au-ICP21, range 0.001 ppm to 10 ppm), with over-runs by fire assay (30 g) with atomic absorption spectrometry (AAS) finish (method Au-AA25). Multi-element analyses were done in a 53 element package (Ag, Al*, As, Au, B*, Ba*, Be*, Bi, Ca*, Cd, Ce*, Co, Cr*, Cs*, Cu, Fe, Ga*, Ge*, Hf*, Hg, In*, K*, La*, Li*, Mg*, Mn, Mo, Na*, Nb*, Ni, P, Pb, Pd, Pt, Rb*, Re*, S*, Sb, Sc*, Se, Sn*, Sr*, Ta*, Te*, Th*, Ti*, Tl*, U, V, W*, Y*, Zn, Zr*) by aqua regia digestion and a combination of inductively coupled plasma mass spectroscopy (ICP-MS) and ICP-AES (method ME-MS41). Major rock forming elements and more resistive minerals are only partly dissolved, and for elements marked (*), digestion is incomplete for most sample matrices. Over-runs for Ag, Cu, Pb and Zn were done by aqua regia digestion and AAS (method AA46).

Acme analysed core samples from holes LTP-34 to LTP-42 at its laboratory in Vancouver (DRG-series assay certificates) by fire assay by classical lead-collection on a 50 g sample with AAS analysis of the bead and a lower limit of detection of 5 ppb, and results were reported in ppb (method G6), or by fire assay fusion of a 50 g sample with detection by ICPES (method G601+G610). Over-runs above 10,000 ppb were re-analysed by fire assay on a 50 g sample with gravimetric analysis and reported in g/t (method G6Gr-50). Multi-elements were analysed in Acme's Vancouver laboratory in a 53 element ultra-trace level package including Au, Pt, Pd, Ag, Al*, As, B*, Ba*, Be*, Bi, Ca*, Cd, Ce*, Co, Cr*, Cs*, Cu, Fe, Ga*, Ge*, Hf*, Hg, In, K*, La*, Li*, Mg*, Mn, Mo, Na*, Nb*, Ni*, P*, Pb, Pd*, Pt*, Rb*, Re, S*, Sb, Sc*, Se, Sn*, Sr*, Ta*, Te, Th*, Ti*, Tl*, U*, V*, W*, Y*, Zn, Zr*) on a 15 g sample with aqua regia digestion (1:1:1) and ICP-MS analysis (method 1F05). Some elements (*) report partial concentrations due to refractory minerals. Over-limit analyses for Ag, Cu and Zn were re-analysed by four acid digestion on a 0.5 g split and ICP-ES analysis and reported in ppm for Ag and percent for Cu, Pb and Zn (method 7TD1).

Acme analysed core samples from holes LTP-43 to LTP-150 at its laboratory in Santiago by fire assay by classical lead-collection on a 30 g sample with AAS analysis of the bead and a lower limit of detection of 5 ppb. Results were reported in ppm (method G6). Over-runs above 10 ppm were re-analysed by fire assay on a 30 g sample with gravimetric analysis and reported in g/t (method G6Gr-30). Multi-element requests were analysed in Acme's Santiago laboratory in a 24 element ultra-trace level package including Au, Mo, Cu, Zn, Ag, Ni, Co, Mg, Fe, As, Sr, Cd, Sb, Bi, Ca, P, Cr, Mn, Al, Na, K, Hg, W, S) on a 15 g sample with aqua regia digestion (1:1:1) and ICP-ES analysis (method 7PD2). The gold fire assay was used for resource estimation rather than the ICP gold result.

GoldQuest reports that core assays performed since the freeze date for the mineral resource database continue to be completed by Acme using the procedures outlined above.

Acme analysed soil and rock samples initially for gold and multi-elements by the ultra-trace level package 1F, and later for gold by method G6 and multi-elements by method 7TX. These methods are described above.

Barium values are not representative due to the insolubility of barite in the aqua regia and multi-acid digestion used for the ICP analyses. In the sulphide zone Ba values are very low, despite abundant barite in places. In the oxide zone there are values up to 0.35% Ba, indicating some Ba in a more soluble mineral form, but still not representative of the total barium content. X-ray fluorescence (XRF) analyses are required to get accurate Ba analyses.

12 Data Verification

This section was taken from the 2014 Micon PEA. This section covers QA/QC data and results up to the freeze date for the mineral resource database used for the resource estimate used herein. Since that time QA/QC procedures have remained the same.

12.1 Assay Laboratory Data Verification

Both ALS Chemex and Acme laboratories maintain in-house quality assurance/quality control (QA/QC) programs involving the insertion of blank, duplicate and certified reference standards into the sample stream.

12.2 GoldQuest Data Verification

GoldQuest initially carried out QA/QC for the drill programs by the insertion of three certified standard reference materials (CSRM), three blanks and two core duplicates per 100 samples, giving 7% QC samples. From Phase 4 drilling on, GoldQuest QA/QC, included the insertion of five CSRM, two blanks, two field duplicates and two preparation duplicates per every 100 samples, giving 11% QC samples.

The results of the QC samples were checked upon receipt of the analytical results from the laboratory. If the QC sample results fell beyond the acceptable limits, described in Sections 12.2.1 to 12.2.4, the laboratory was notified and requested to investigate the problem, and, if necessary, to re-analyse all or a portion of the batch. Once the sample order passed QC it was approved and entered into the company database.

Similar QA/QC procedures were carried out by GoldQuest for stream sediment, soil and rock samples. The results are not described in this report as these data were not used for the mineral resource estimation.

12.2.1 Certified Standard Reference Materials

CSRM number OxD27 was used for the Phase 1 drill program, SF12 was used for the Phase 2 drill program, and CDN-GS-P5B and CDN-GS-P8 were used for the Phase 3 drill program and, CDN-ME-2, CDN-ME-6, CDN-ME-7 and CDN-ME-11 were used for the Phase 4 program. Three CSRM were inserted per 100 samples. The results were evaluated using performance gates. The results are accepted if they are within plus or minus two standard deviations (SD) of the recommended value. A single value lying between plus or minus 2 SD and 3 SD is also acceptable, but two consecutive values between plus or minus 2 SD and 3 SD are rejected, as are any values greater or less than 3 SD.

OxD27 and SF12 were produced by Rocklabs Ltd., New Zealand. OxD27 has a certified value of 0.416 ± 0.025 (1 SD) g/t Au. SF12 has a certified value of 0.819 ± 0.028 (1 SD) g/t Au.

CSRMs CDN-GS-P5B and CDN-GS-P8, CDN-ME-2, CDN-ME-6, CDN-ME-7, CDN-ME-11, CDN-CM-18, CDN-CM-24, CDN-FCM-6, CDN-CM-12A, CDN-CM-13A, CDN-ME-16, CDN-ME-1205 and CDN-ME1206 were produced by CDN Resource Laboratories Ltd., British Columbia, Canada. The recommended values and the “Between Lab” standard deviations (SD) are shown in Table 12.1.

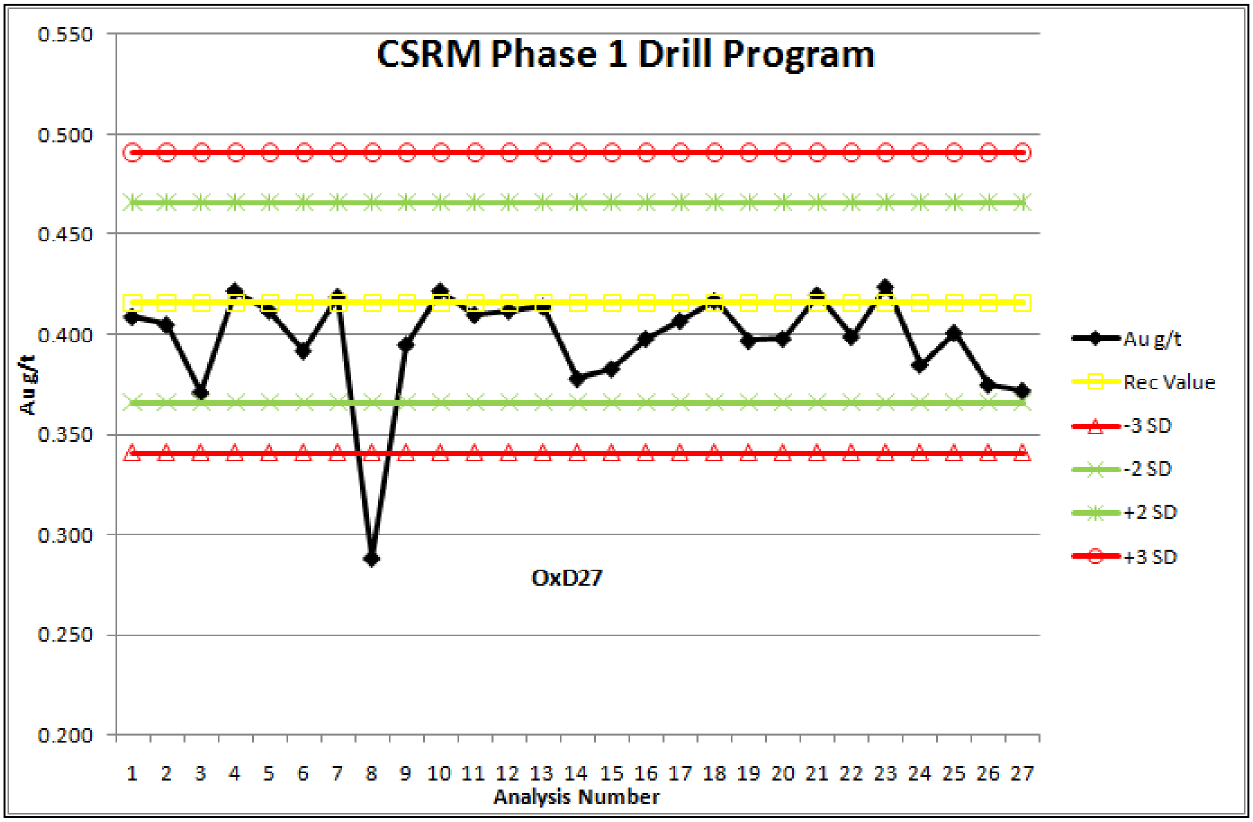
Table 12.1: Standard Reference Material Utilized by GoldQuest

Standard	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	SD	Remarks
OXD27	0.416± 0.05					2	Used in Phase 1
SF12	0.819± 0.056					2	Used in Phase 2
CND-GS-P5B	0.44 ± 0.04					1	Used in Phase 3
CND-GS-P8	0.819 ± 0.028					1	Used in Phase 3
CDN-ME-2	2.10 ± 0.11	14.0 ± 1.3	0.480 ± 0.018		1.35 ± 0.10	2	Used in Phase 4, 5, 6
CDN-ME-6	0.270 ± 0.028	101 ± 7.1	0.613 ± 0.034	1.02 ± 0.08	0.517 ± 0.040	2	Used in Phase 4, 5 , 6, 7
CDN-ME-7	0.219 ± 0.024	150.7 ± 8.7	0.227± 0.016	4.95± 0.30	4.84 ± 0.17	2	Used in Phase 4, 5 , 6, 7
CDN-ME-11	1.38 ± 0.10	79.3 ± 6.0	2.44 ± 0.11	0.86 ± 0.10	0.96 ± 0.06	2	Used in Phase 4, 5 , 6, 7
CDN-CM-18	5.28 ± 0.35		2.42 ± 0.22			2	Used in Phase 7
CDN-CM-24	0.521 ± 0.056	4.1 ± 0.4	0.365 ± 0.02			2	Used in Phase 7
CDN-FCM-6	2.15 ± 0.16	156.8 ± 7.9	1.251 ± 0.064	1.52 ± 0.06	9.27 ± 0.44	2	Used in Phase 7
CDN-GS-12A	12.31 ± 0.54					2	Used in Phase 7
CDN-GS-13A	13.20 ± 0.72					2	Used in Phase 7
CDN-ME-16	1.48 ± 0.14	30.8 ± 2.2	0.671 ± 0.036	0.879 ± 0.040	0.807 ± 0.040	2	Used in Phase 7
CDN-ME-1205	2.20 ± 0.28	25.6 ± 2.4	0.218 ± 0.012	0.13 ± 0.004	0.369 ± 0.03	2	Used in Phase 7
CDN-ME-1206	2.61 ± 0.20	274 ± 14	0.79 ± 0.038	0.801 ± 0.044	2.38 ± 0.15	2	Used in Phase 7

Source: Micon 2014

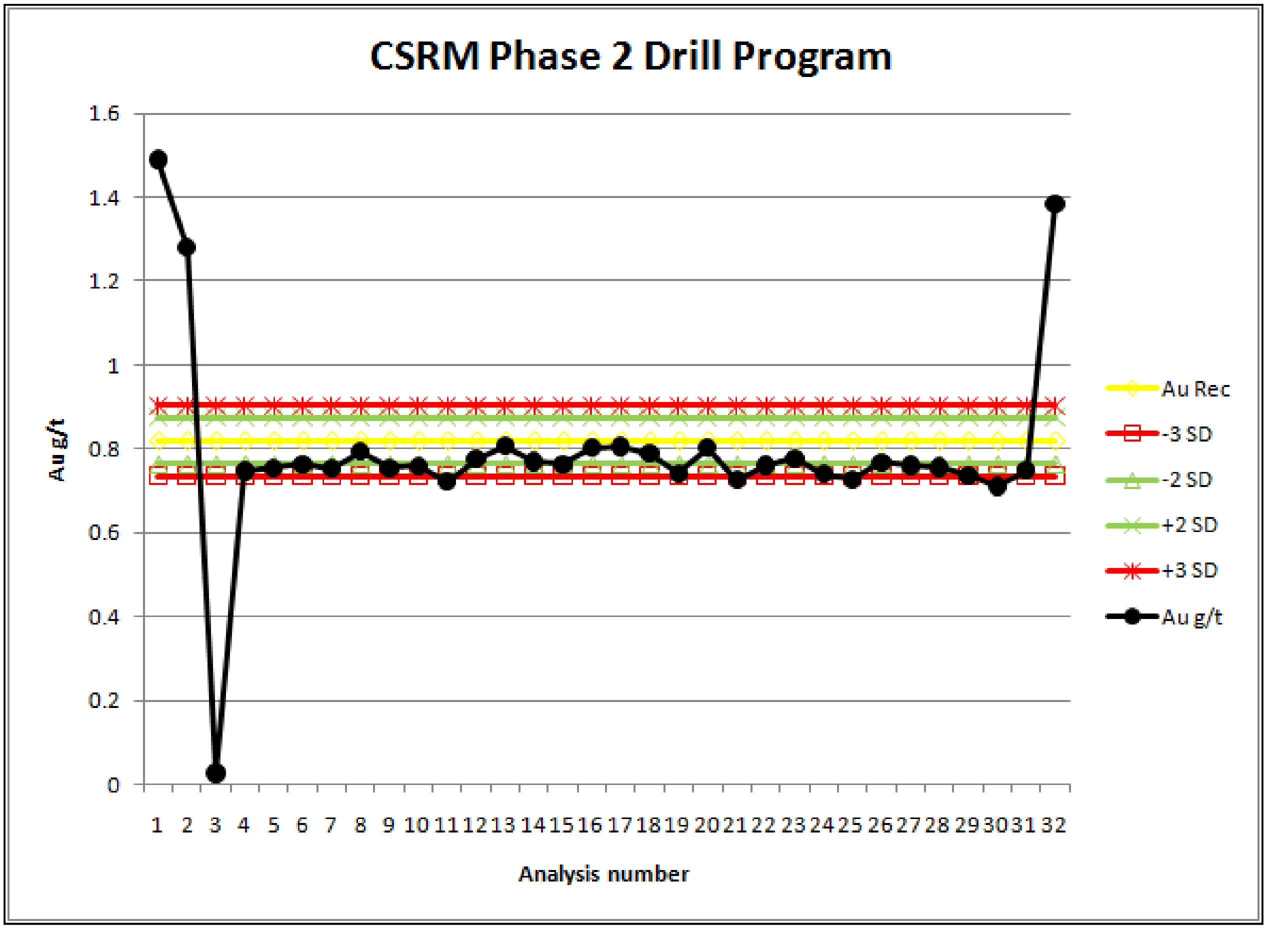
Gold results for the CSRMs for Phase 1 to 3 are shown in Figure 12.1 to Figure 12.3, respectively. There is one exception in the Phase 1 drill program, and four exceptions from the Phase 2 drill program where Au is ± 3 SD.

Figure 12.1: CSRM Plot for Phase 1 Drill Program



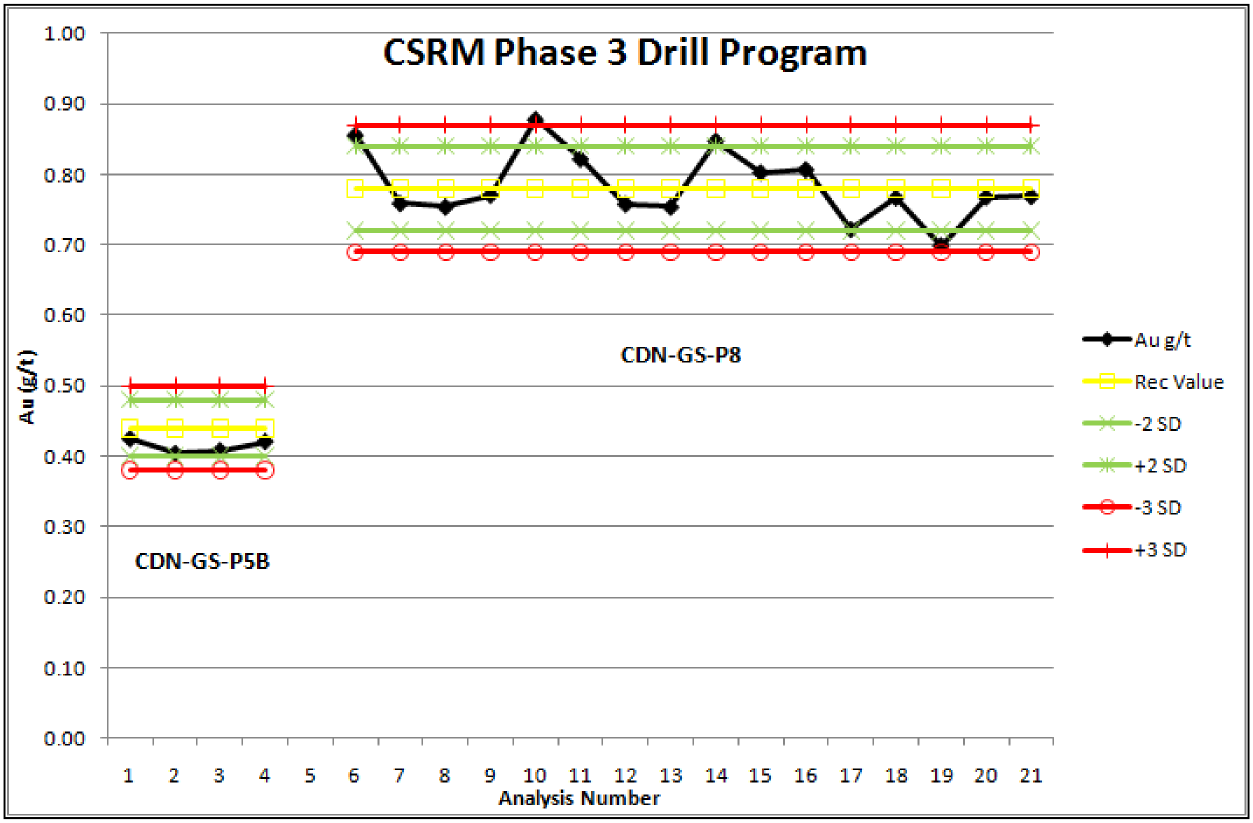
Source: Micon 2014

Figure 12.2: CSRM Plot for Phase 2 Drill Program



Source: Micon 2014

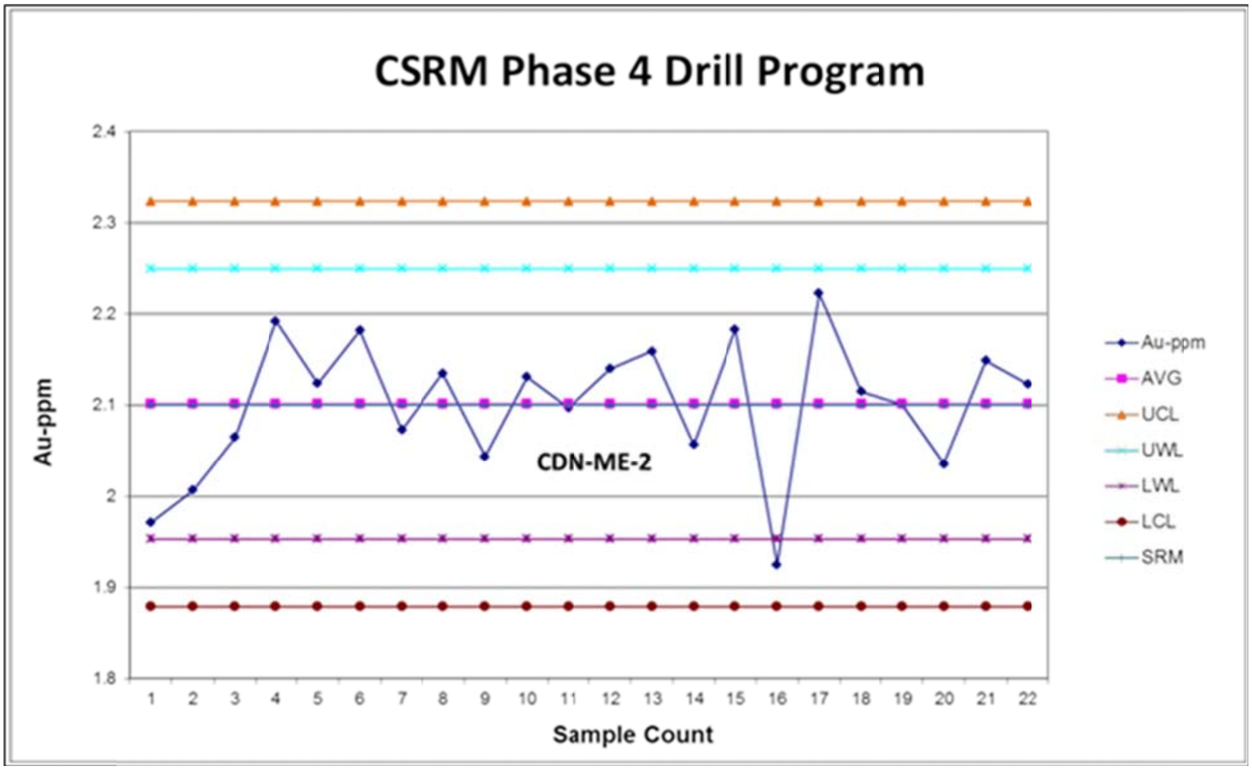
Figure 12.3: CSRM Plot for Phase 3 Drill Program



Source: Micon 2014

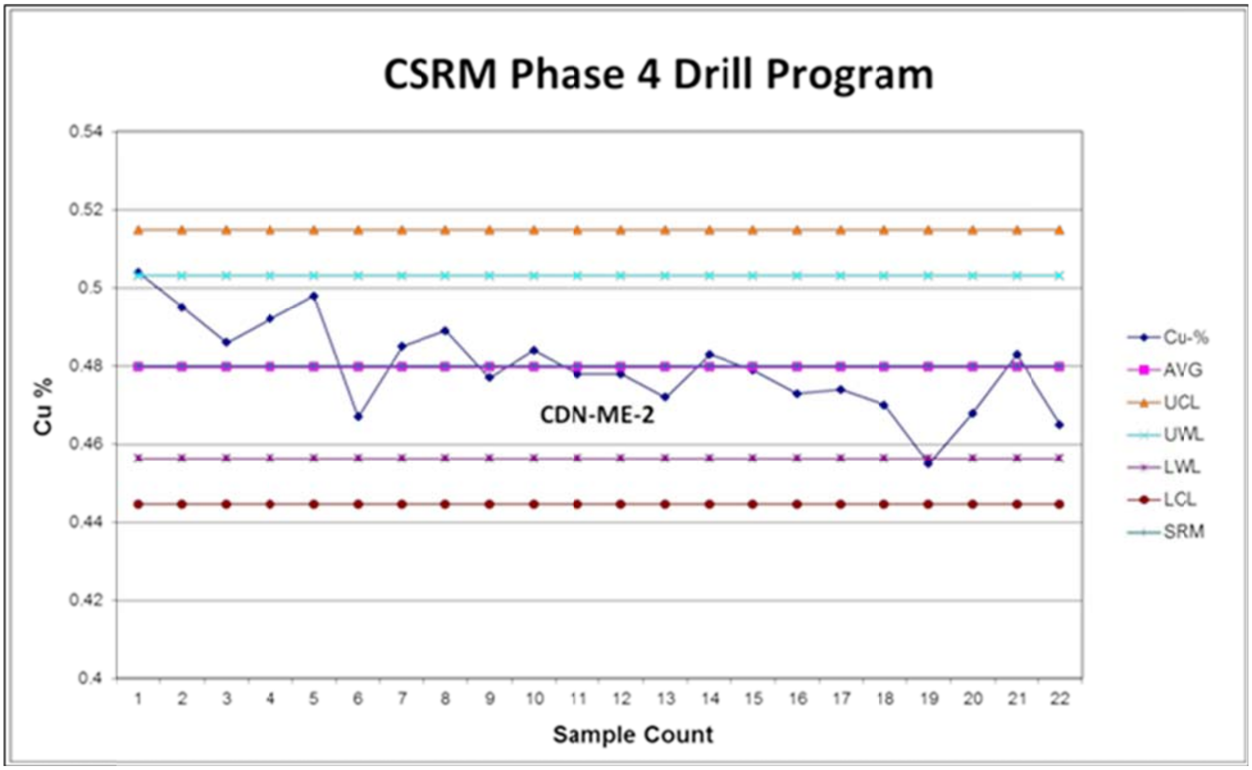
In Phase 4 drilling, GoldQuest introduced four multi-metal reference standards to monitor the laboratory's analytical performance on both gold and base metals. The more widely used of these is CDN-ME-2 for which the results are shown in Figure 12.4 and Figure 12.5. These results demonstrate the laboratory's proficiency.

Figure 12.4: CSRM Plot for Phase 4 Drill Program - Gold



Source: Micon 2014

Figure 12.5: CSRM Plot for Phase 4 Drill Program - Copper

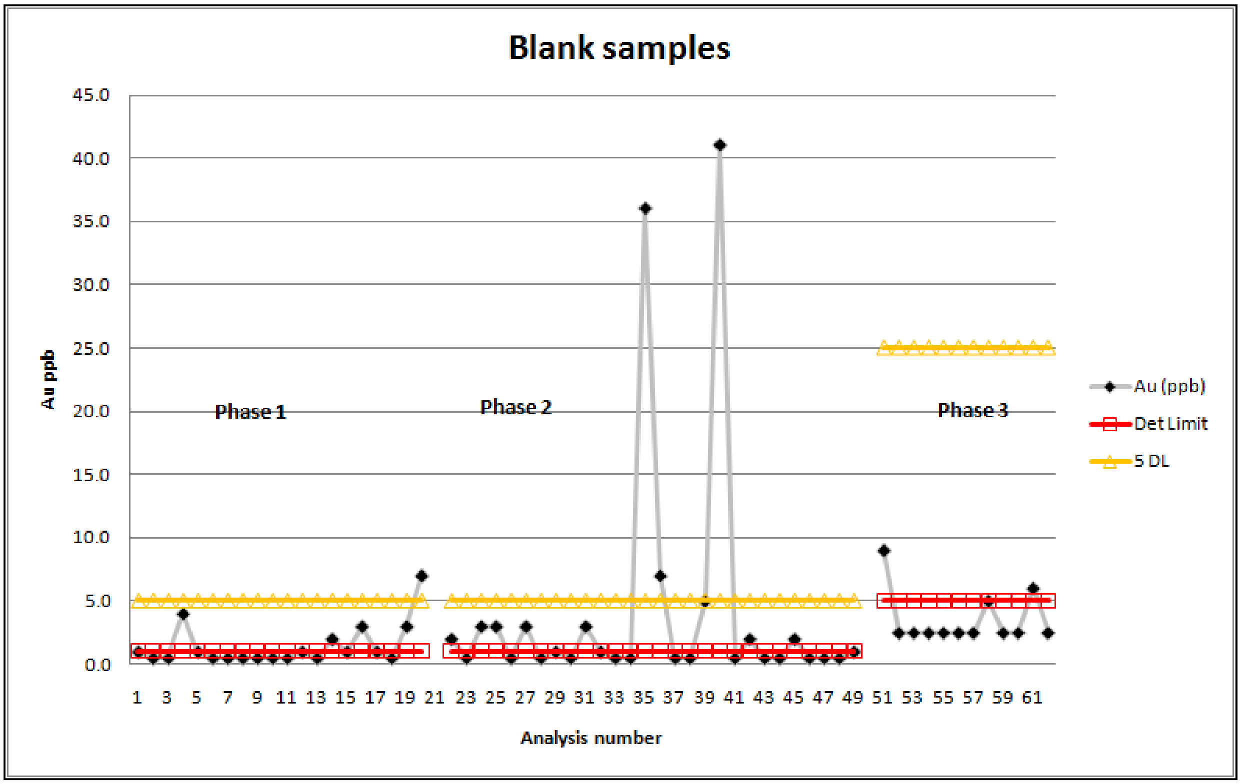


Source: Micon 2014

12.2.2 Blank Assays

Three blank samples were inserted per 100 samples. The blank used was silica sand. The plot of blank analyses for gold is shown in Figure 12.6. The blank results are generally within acceptable limits, defined as 5 times the detection limit, with three exceptions in the Phase 2 drill program. Since these were in intervals with no significant values, GoldQuest decided not to reanalyse the intervals at the time.

Figure 12.6: Plot of Blank Samples for Phase 1 to Phase 3 of the Drill Program



Source: Micon 2014

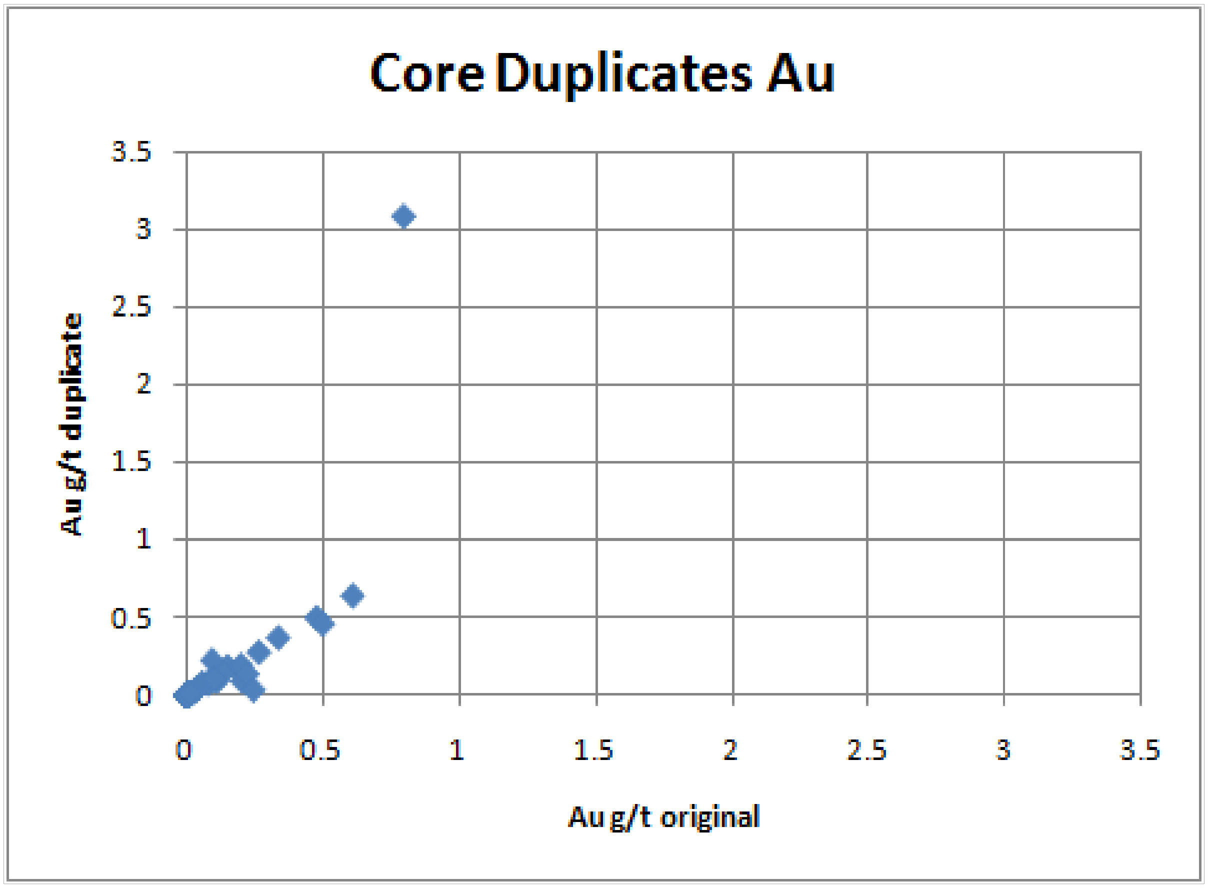
Values below detection replaced by half the detection limit to avoid negative numbers.

12.2.3 Core Duplicates

Two core duplicates were taken for every 100 samples. The core duplicate is a quarter core sample taken by cutting the reference half core sample in two with a diamond saw. A plot of all the core duplicates is shown in Figure 12.7 and shows one outlier sample which may be the result of geological variability, or a laboratory error. In Figure 12.8, the outlier sample has been removed and shows good repeatability of all the other samples.

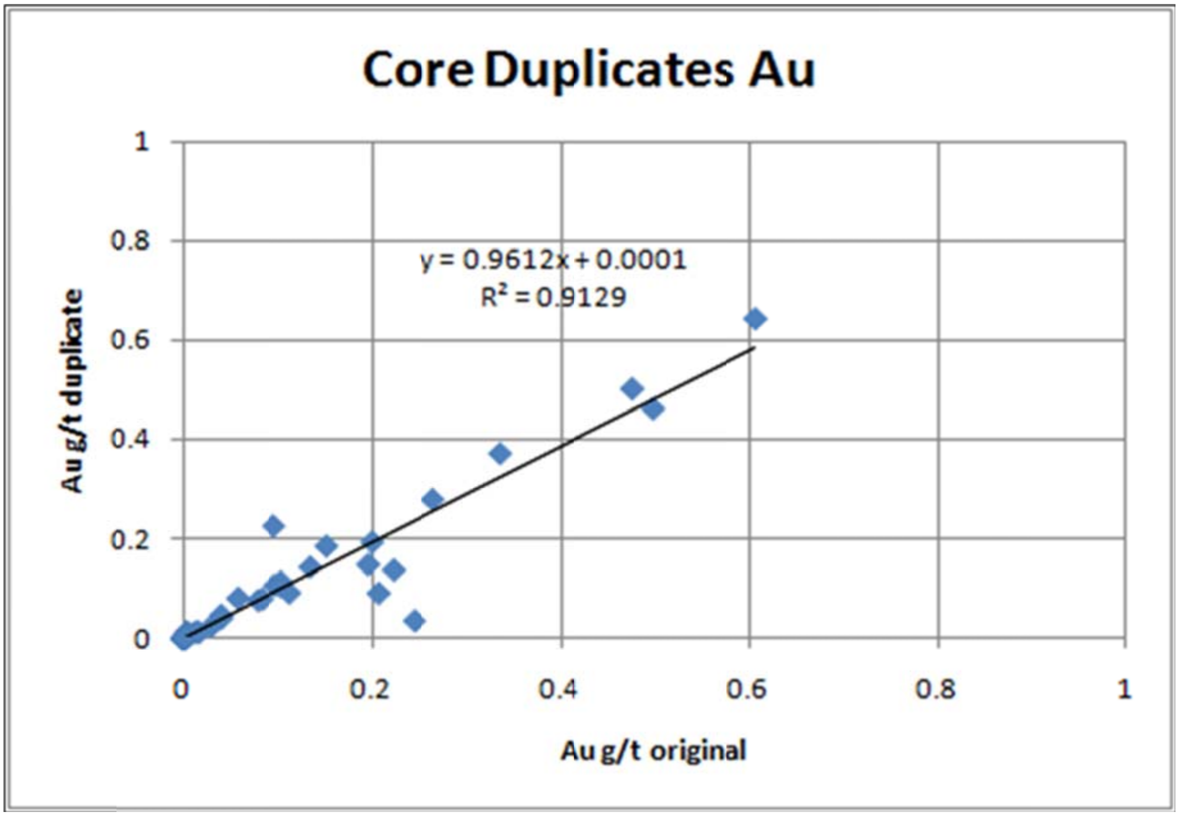
Although there appears to be good repeatability, in 2012 Micon did not recommend continued use of core duplicates due to the inherent geological variability.

Figure 12.7: Plot of Core Duplicate Analyses for Au, Phases 1 to 3 of the Drill Program



Source: Micon 2014

Figure 12.8: Plot of Core Duplicate Analyses for Au, Phases 1 to 3 of the Drill Program



(with one outlier removed)

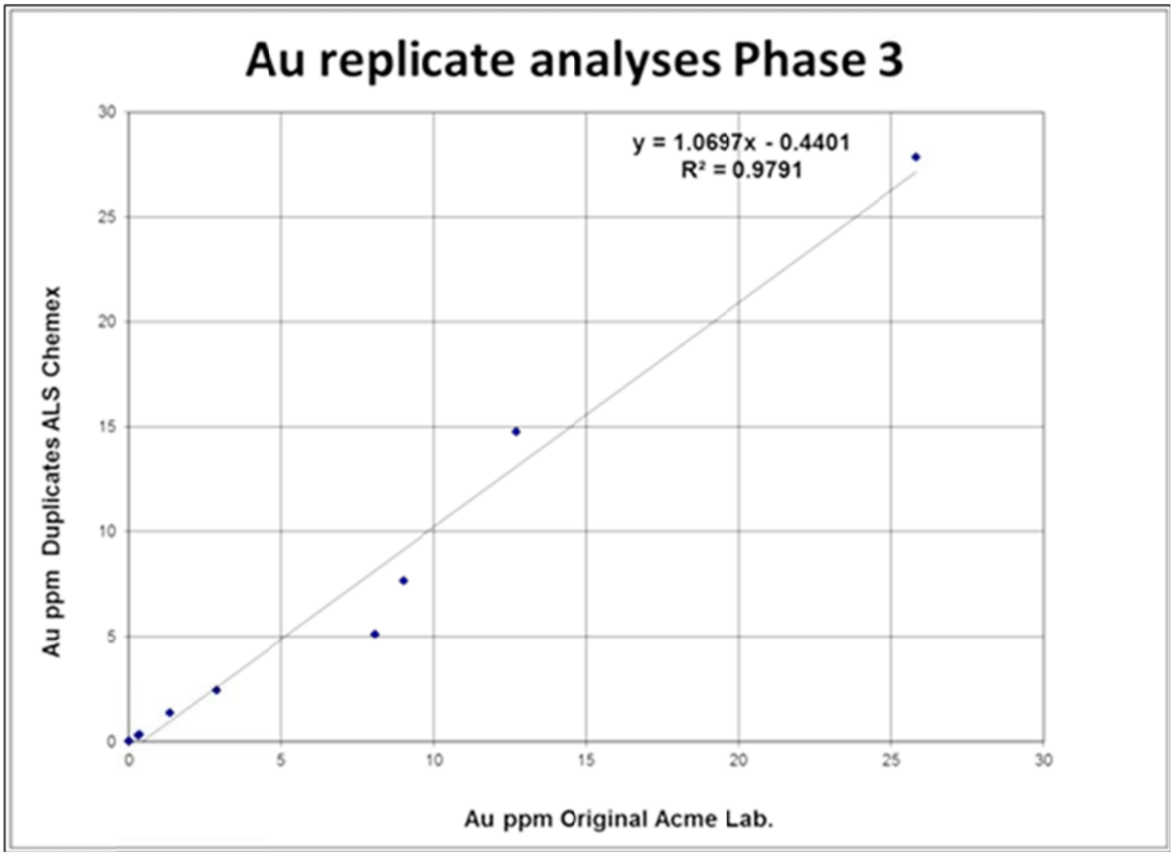
Source: Micon 2014

12.2.4 External Laboratory Repeats

Replicate analyses of the same sample pulp were made at a third party, certified laboratory on 55 sample pulps from Phase 3 of the drill program. The 55 sample pulps were selected above a cut-off of 0.2 g/t Au, out of 501 analyses (excluding QC samples), representing 11% of the total. These were sent, with 2 CSRMs and 2 blanks for QC, to ALS Chemex in Vancouver for analysis for Au by Au-AA23 (FA30g-AAS) and multi-elements by ME-ICP41. A cut-off grade was used to select replicate samples rather than selection at random since the latter would have resulted in the majority of the check samples being below detection or of very low grade, due to the stratiform nature of the mineralization.

The gold results are plotted in Figure 12.9 and show a very good correlation between the two laboratories.

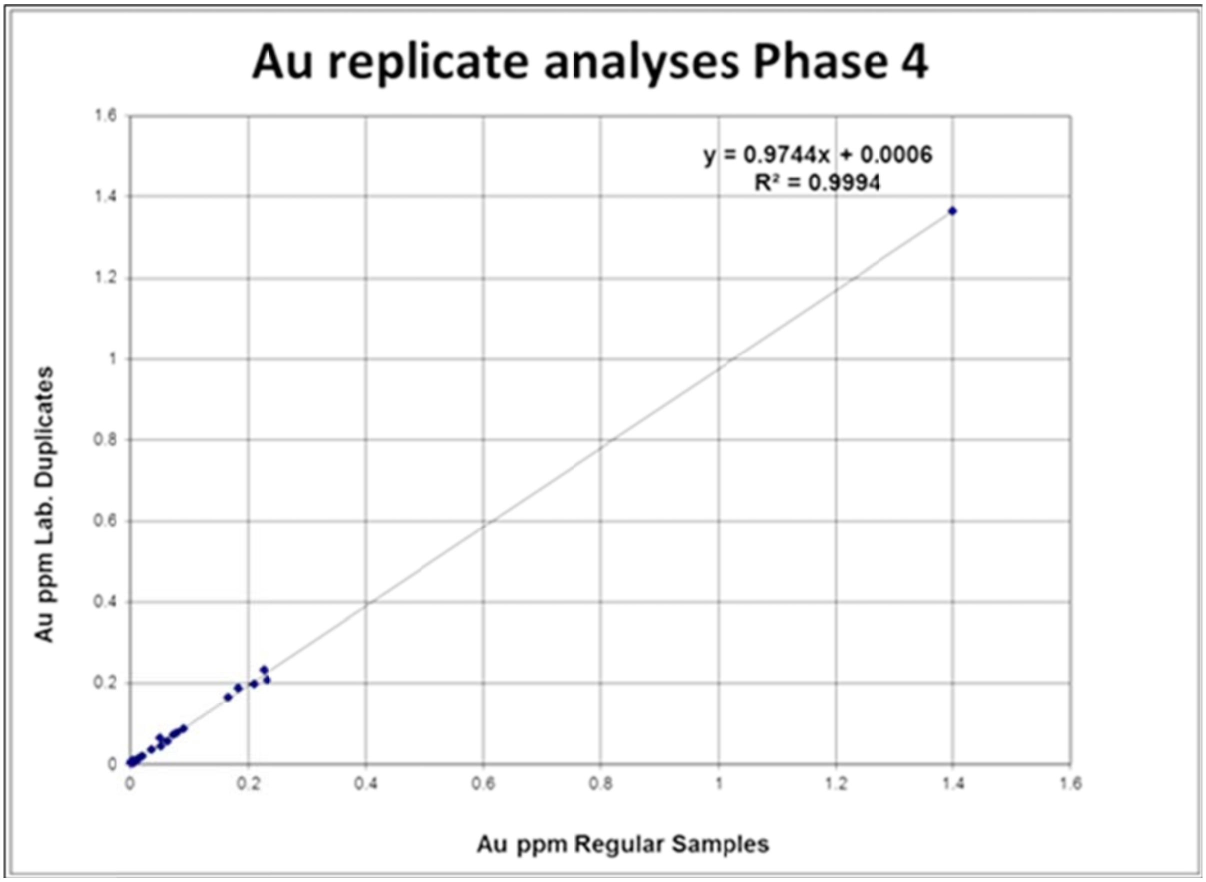
Figure 12.9: Plot of Replicate Analyses for Phase 3 of the Drill Program



Source: Micon 2014

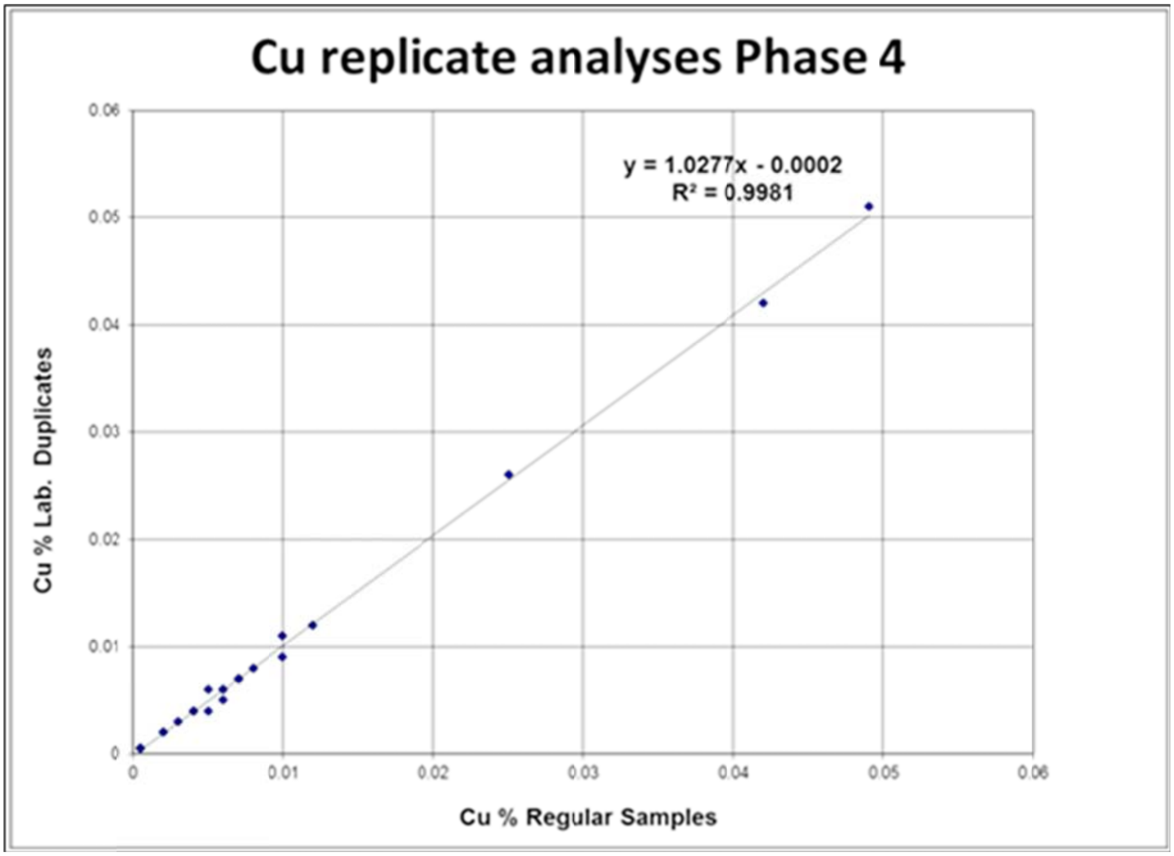
In Phase 4 drilling, replicate analyses were conducted for both gold and base metals. The correlation for all elements (i.e. Au, Ag, Cu, Pb and Zn) is good. Only one sample replicate (i.e. sample number 16978) appeared as an outlier and this is most likely due to a sample switch. The scatter plots for Au and Cu are shown in Figure 12.10 and Figure 12.11, respectively.

Figure 12.10: Plot of Replicate Analyses for Phase 3 of the Drill Program



Source: Micon 2014

Figure 12.11: Plot of Replicate Analyses for Phase 3 of the Drill Program



Source: Micon 2014

Later QA/QC plots for phases 5, 6 and 7 generally produced similar results. There are several dozen of them and it is beyond the scope of this report to reproduce them all. The ones presented are considered representative of the type of QA/QC program conducted. Field duplicate control charts occasionally produced points which fall well off the 45º agreement line at higher grades. However, this is to be expected occasionally when sampling the other half of the core in a high grade sample.

12.3 Micon Data Verification

12.3.1 2011 Validation

During its 2011 site visit and in preparation for the 2012 report (Steedman and Gowans, 2012) Micon completed data validation. Only drilling results from Phase 1, 2, 3 and 4 were verified. Drilling in Phases 5, 6 and 7 was completed after Micon's first visit to site in July, 2011. Micon verified the data used by:

- Visiting the property and confirming the geology in July, 2011;
- Confirming drill core intervals including mineralized intersections;
- Checking the location of the Phase 1 to 4 drill holes in the field; and
- Reviewing Phase 1 to 4 QA/QC analysis.

For the 2012 resource estimate Micon used Excel files exported from the Access database and supplied by GoldQuest. All of these were checked against digital PDF assay certificates supplied by the analytical labs. There was no problem with verification of assay certificates with original analyses by ALS Chemex and Acme.

At the time Micon considered the sample preparation, security and analytical procedures to be adequate to ensure the integrity and credibility of the analytical results used for mineral resource estimation. The use of control samples (i.e. standards, blanks and duplicates) was rigorous and this, coupled with the monitoring of the laboratory's performance on a real time basis, ensured that corrective measures (if need be) are taken at the relevant time and gave confidence in the validity of the assay data used in the resource estimate. However, the use of silica sand as "blanks" does not monitor contamination between samples during the crushing stage; accordingly, Micon recommended that blank material which requires crushing and pulverizing is employed so that contamination can be monitored during this process as well.

On the whole, there was a steady improvement noted in the QA/QC protocols from Phases 1 to 3 and on to Phase 4 when GoldQuest adopted multi-metal standards to cope with the mineralization types encountered. Micon considered that the analytical work completed to-date was monitored closely enough to ensure representative assays.

Micon concluded that:

- Exploration drilling, drill hole surveys, sampling, sample preparation, assaying, and density measurements had been carried out in accordance with best current industry standard practices and are suitable to support resource estimates.
- Exploration and drilling programs were well planned and executed and supply sufficient information for resource estimates and resource classification.
- Sampling and assaying includes quality assurance procedures.
- Exploration databases were professionally constructed and are sufficiently error-free to support resource estimates.

12.3.2 2013 Validation

The presence of copper mineralization at Romero and Romero South is obvious from a review of a representative selection of drill core from the two deposits. As expected from a deposit showing frequent multi-percent copper assays, chalcopyrite is easily visible in core.

During its site visit Micon collected two duplicate quarter core samples and a composite grab sample from a rock outcrop in the Escandalosa Creek which exposes the edge of the Romero South deposit. The results are presented in Table 12.2 below.

Table 12.2: Micon Check Sampling Results

Sample No.	Original Assay		Re-assay		Comment
	Au (g/t)	Cu (%)	Au (g/t)	Cu (%)	
664	-	-	0.71	0.2	Outcrop in creek at Romero South
665	22	3.54	26	3.05	¼ core duplicate
666	10.5	6.37	14.3	6.74	¼ core duplicate

Source: Micon 2014

The assay results show remarkably close agreement for quarter-core field duplicate samples and confirm the presence of high grade copper and gold mineralization.

12.3.3 Database Verification

The geological database is the foundation of a resource estimate. Therefore, Micon performed a thorough review of the data to ensure the reliability of the estimate. The review of the data was performed in Micon's Toronto offices. Some errors were detected and corrected including:

Correction of the drill hole collar surveys; some updated collar locations were adjusted using the topographic surface grid provided by GoldQuest.

Detailed review of down hole surveys, assay data, density measurements. Correction of silver assay results which were suspiciously high and determined to be a unit error (silver assays in ppb instead of ppm). Given this, Micon decided to cross check the entire assay table against results independently downloaded from the laboratory for all available assay certificates. 84% of the assay results were checked. See Table 12.3 for a summary of results.

Table 12.3: Romero Project Assays Table Cross Check Validation Results Summary

Description	Count of Au Checks*
Chemex	
No results	12
OK	1,499
OK-Detection Limit	244
Not found	2,263
Acme	
OK	8,281
OK-Detection Limit	1,294
OK-Over Limit	118
Switch	208
Not found	0
Grand Total	**13,919**

* - Copper, silver and zinc assay entries were also checked.

Source: Micon 2014

12.4 Micon Comments

Micon considers the sample preparation, security and analytical procedures employed to be adequate to ensure the validity of assays. The QA/QC protocols employed by GoldQuest are sufficiently rigorous to ensure that sample data are appropriate for use in a mineral resource estimate.

13 Mineral Processing and Metallurgical Testing

The 2015 test program completed at the ALS metallurgical laboratories (ALS) in Kamloops, BC was managed by JDS. The program was designed primarily to develop a flowsheet that would produce one copper concentrate with a focus on improved gold recovery and to provide additional support to the metallurgical design criteria developed in the earlier stages of testwork.

13.1 Summary of Metallurgical Testing

GoldQuest conducted grinding and flotation tests on drill core and bulk samples generated by the Romero underground exploration program, between 2011 and 2014. A series of test programs investigated the feasibility of producing a copper concentrate and pyrite concentrate for recovery of gold. In 2011, a composite sample from Romero South was sent to Resource Development Inc, (RDI) to look at gravity separation and cyanide leach tests. A second sample sent to RDI was subjected to grinding, abrasion, cyanide leach and flotation tests. From 2013 through June 2014, ALS completed two test programs on six metallurgical composite samples. Samples 1 to 3 of test program KM3650 were composited based on variable head grades to the mill. The samples represented high gold and copper grades (HAu/HCu), high gold and low copper grades (HAu/LCu), and low gold and high copper grades (LAu/HCu). The second ALS test program, KM4076, involved three new composites representing Romero Indicated Resources, Romero Inferred Resources and Romero South Resources. In 2015, the most recent test program, KM4601, was completed. Test program KM4601 used the six samples from the previous ALS test programs and was focused on improving the recovery of gold and silver to a single copper concentrate.

13.2 Historical Testwork

Metallurgical test programs were completed in 2011, 2013 and 2014 on metallurgical composites selected by GoldQuest. The following list of historical metallurgical test reports were reviewed for this study:

- Resource Development Inc., “Scoping Metallurgical Study for Las Escandalosa and Las Animas Oxide Ores, Dominican Republic”, September 8, 2011. (RDI, 2011);
- ALS Metallurgy Kamloops, KM3650 “Metallurgical Flowsheet Development Testing on Three Composite Samples from the Romero Deposit”, June 6, 2013 (ALS, 2013); and
- ALS Metallurgy Kamloops, KM4076 “Metallurgical Flowsheet Development Testing on Three Composite Samples from the Romero Deposit”, June 16, 2014 (ALS, 2014).

In 2011, a composite sample “RDI Composite No. 1”, was constructed from Romero South assay reject samples (RDI, 2011) for gravity separation and cyanide leach tests. A second composite sample was subjected to grinding, abrasion, cyanide leach, and flotation tests.

Two metallurgical test programs were completed at ALS Metallurgical from 2013 to 2014. Three metallurgical composite samples were constructed for test program KM3650; Sample 1 (High Au/High Cu), Sample 2 (High Au/Low Cu), and Sample 3 (Low Au/High Cu). Three different metallurgical composites were constructed for test program KM4076; Romero Indicated, Romero Inferred, and Romero South. The test programs included the evaluation of the chemical and mineralogical characteristics of the composites, comminution work, flotation tests, and gold gravity and cyanidation leach recovery.

The comminution results from the ALS Metallurgical test programs are summarized in Table 13.1 and were used for the development of the updated flowsheet:

Table 13.1: Historical Test Comminution Results used for the Development of the New Flowsheet

Program	Sample	Bwi	P_{80}	Close Screen Size	Ai	Smc
		(Kwh/Tonne)	(µm)	(µm)		(A X B)
RDI	Second Program Sample	12.8		150	0.2078	
KM3650	Sample 1	13.9	70	106	0.183	36.9
	Sample 2	15.9	78	106	0.125	35.5
	Sample 3	14.1	80	106	0.275	35.7
KM4076	Romero Indicated	15	79	106		
	Romero Inferred	16	80	106		
	Romero South	14.4	80	106		

Source: JDS 2015

13.3 Recent Test Work

This technical report is based predominantly on the results of the ALS Metallurgy program KM4601,

ALS Metallurgy Kamloops, KM4601 “Metallurgical Evaluation of Samples from the Romero Deposit”, April 8, 2015 (ALS, 2015).

The objective of the 2015 metallurgical test program KM4601 was to continue the development of the Romero flowsheet by improving the recovery of gold and silver to produce one copper concentrate. Kinetic and batch rougher and cleaner tests were used to optimize reagent dosage, primary and regrind sizing and pH control. Confirmatory gravity and cleaner tests were conducted on all available samples with the optimized conditions.

13.3.1 Composite Characteristics

The six previously used composite samples at ALS were used for the 2015 test program. A summary of the composite head assays is displayed in Table 13.2.

Table 13.2: Chemical Composition of the Composites

	Assay Results							
Composite	**Cu (%)**	**Zn (%)**	**Fe (%)**	**S (%)**	**Au (g/t)**	**Ag (g/t)**	**CuOx (%)**	**CuCN (%)**
Sample 1	1.96	0.24	7.8	8.59	6.74	6	0.032	0.08
Sample 2	0.17	0.59	6.7	7.01	3.53	10	0.003	0.015
Sample 3	2.65	0.14	9.2	10.3	0.52	4	0.027	0.072
Romero Inferred	0.44	0.86	5.4	5.44	1.47	3	0.005	0.027
Romero Indicated	0.78	0.12	6.6	6.22	3.01	3	0.013	0.024
Romero South	0.31	0.18	4.1	4.39	3.5	2	0.004	0.013

Source: ALS 2015

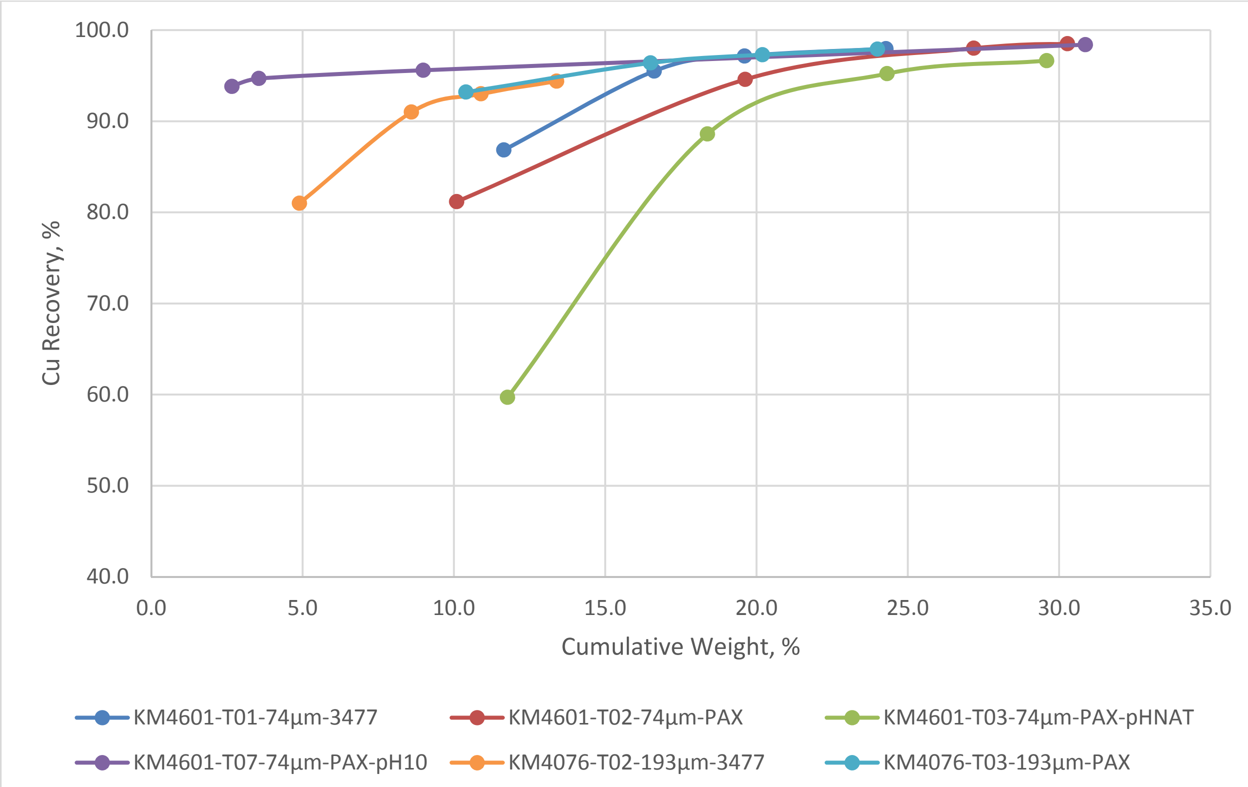
13.3.2 Rougher Flotation Tests

Rougher optimization tests were conducted using the Romero Indicated composite. The following conditions were targeted during the optimization:

- Primary grind K_{80} of 75 µm;
- A coarser primary grind of 190 µm was targeted in the previous test program, KM4076;
- Flotation time and mass pull;
- Copper sulphide collectors PAX and 3477; and
- Lime addition for pH control.

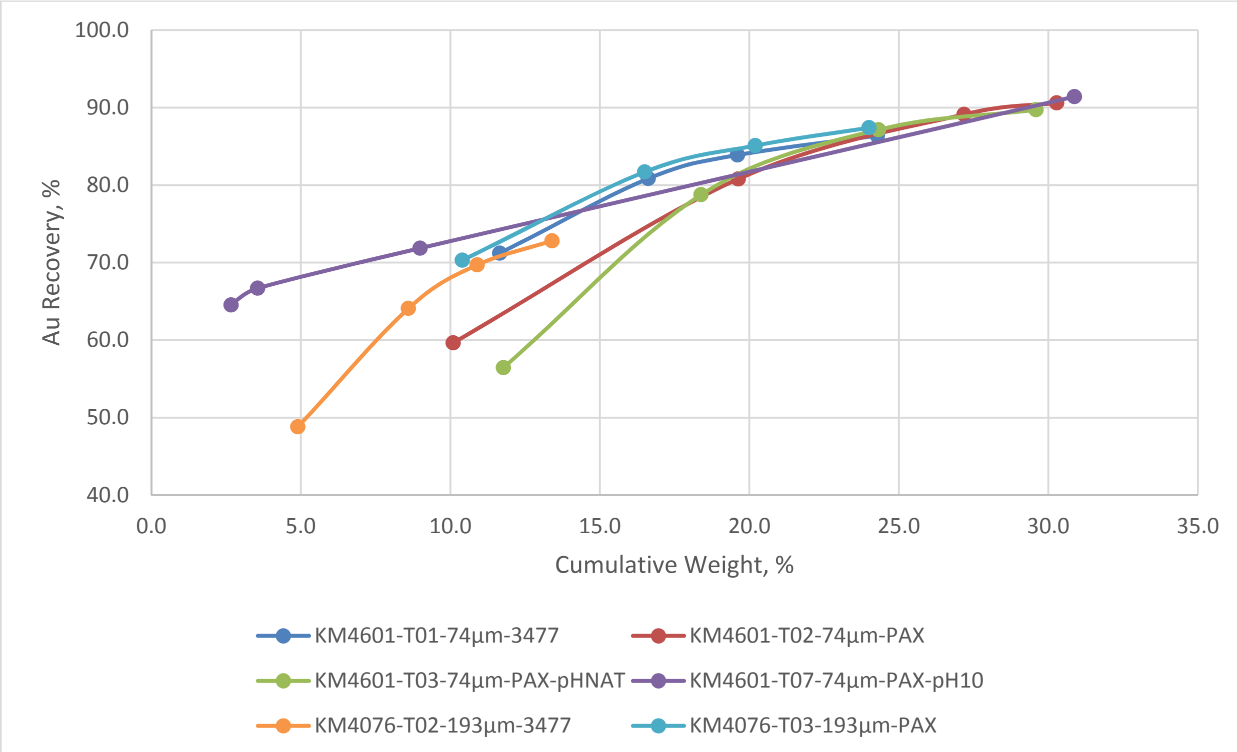
The rougher optimization tests identified that approximately 98% of the copper and 88% of the gold can be recovered with an aggressive mass pull of 30%. The required mass pull was directly correlated to the slow kinetics associated with the gold bearing particles. A primary grind of 74 µm using PAX at a pH of 10 were chosen as the optimized conditions. Figure 13.1 and Figure 13.2 show a comparison of the optimization tests of KM4601 along with relevant historical results.

Figure 13.1: Rougher Optimization Copper Recoveries versus Mass Pull



Source: ALS KM4076 and 4601 Test Programs

Figure 13.2: Rougher Optimization Gold Recoveries versus Mass Pull



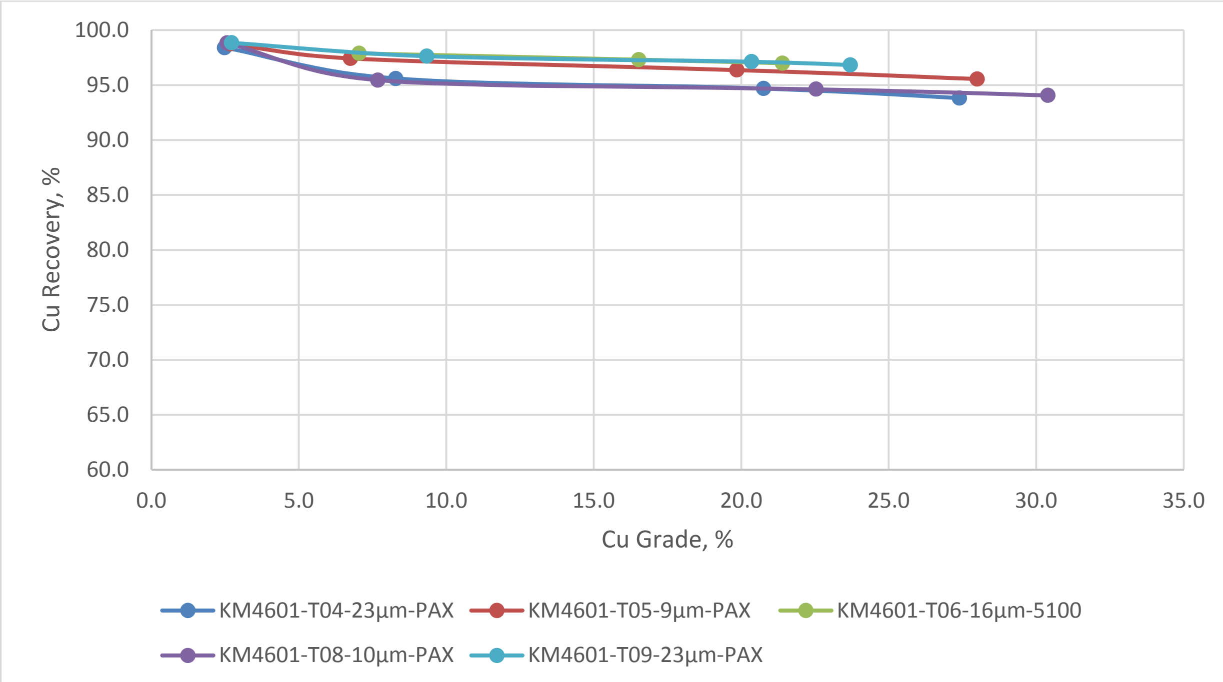
Source: ALS KM4076 and 4601 Test Programs

13.3.3 Cleaner Flotation Results

Batch Cleaner flotation tests were carried out on the Romero Indicated composite to investigate the effect of regrind discharge size and collector type. Lime was used to maintain a pH of 11.5 in the cleaners with the addition of collectors PAX and 5100.

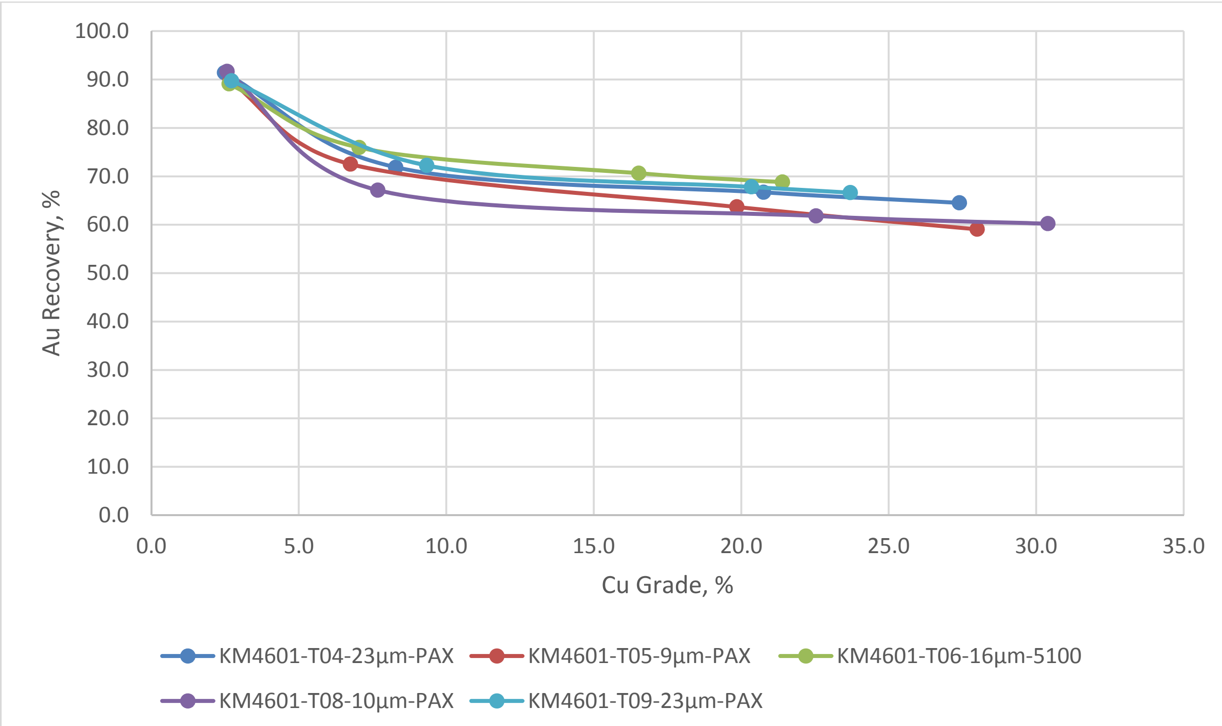
The results shown in Figure 13.3 indicated that an average of 95% of the copper could be recovered to a saleable concentrate of 25% copper. An average of 63% of the gold reported to the final con as shown in Figure 13.4.

Figure 13.3: Batch Cleaner tests - Copper Recoveries



Source: ALS KM4076 and 4601 Test Programs

Figure 13.4: Batch Cleaner Tests - Gold Recoveries



Source: ALS KM4076 and 4601 Test Programs

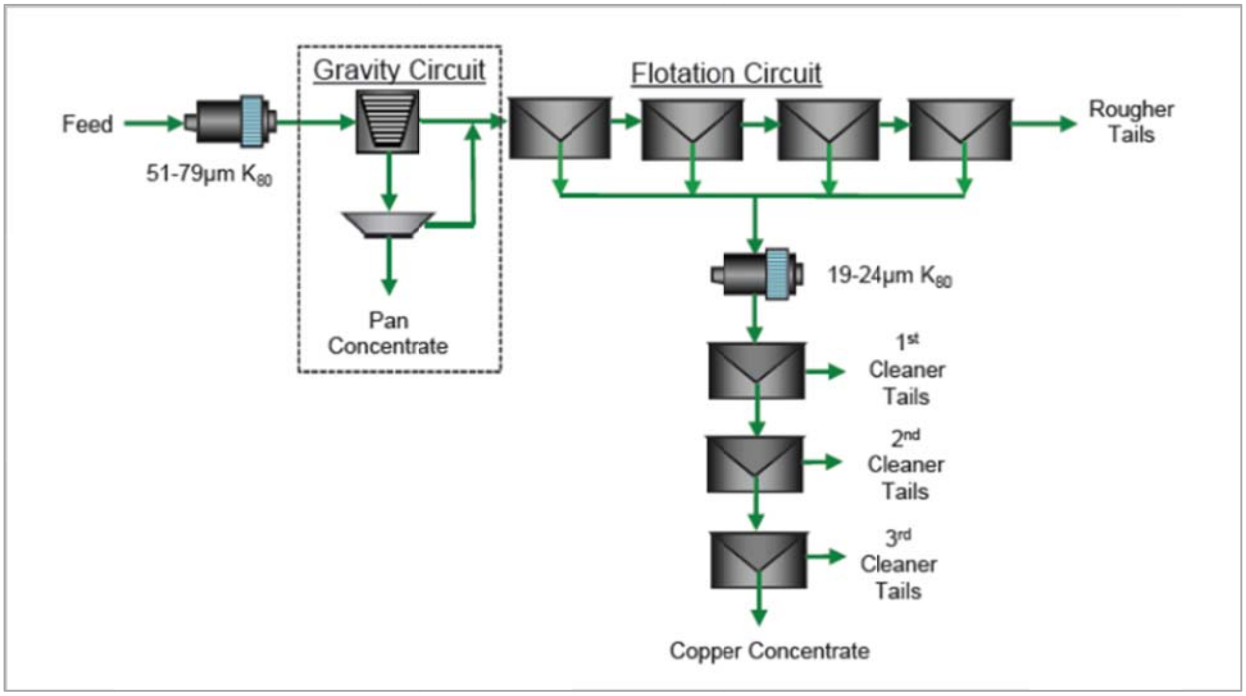
13.3.4 Gravity and Flotation Results

A gravity circuit was incorporated into the flowsheet to improve gold recoveries and investigate the combined recovery of gravity and flotation concentration. All six composites were subjected to the flowsheet using the rougher and cleaner optimized conditions:

- Primary Grind K_{80} of 75 µm;
- Regrind P_{80} of 23 µm;
- Copper sulphide collector PAX ; and
- Lime addition to maintain pH of 10.0 in the roughers and pH of 11.5 in the cleaners.

The batch cleaner flotation flowsheet used for all six samples is shown below in Figure 13.5.

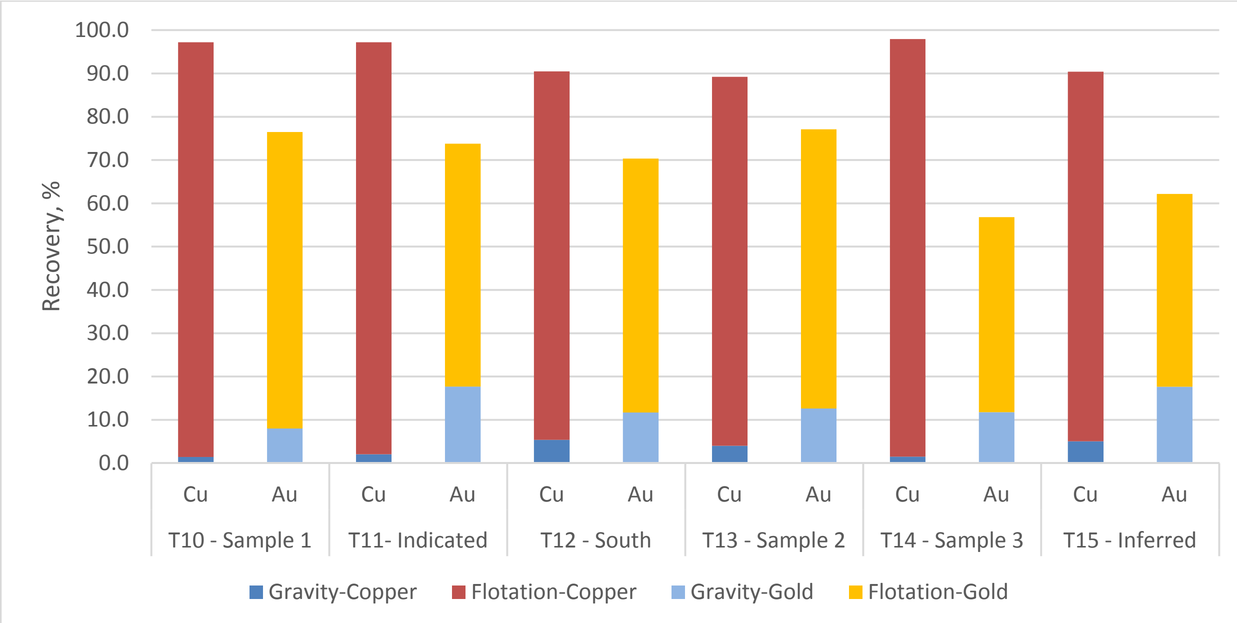
Figure 13.5: Batch Cleaner Flowsheet, taken from ALS Report KM4061



Source: ALS 2015

The addition of the gravity circuit resulted in up to 17.7% of gold feed reporting to the gravity concentration. Copper recovery was unaffected by the introduction of the gravity circuit. Three composites were unable to attain a saleable copper concentrate greater than 20% Copper due to dilution by pyrite and zinc. The recovery results are presented in Figure 13.6.

Figure 13.6: Combined Gravity and Flotation Recovery Results



Source: ALS 2015

13.3.5 Filtering Results

Filtering tests were conducted on a concentrate and tail composite as a scoping test to assess the amenability of the samples for vacuum filtration. The results are presented in Table 13.3. Inadequate preparation of the samples involving proper thickening deemed these tests unreliable and do not provide representation of in-plant filtering. Additional test work has been recommended for the next stage of engineering in addition to tailings analysis for paste backfill and dry stack.

Table 13.3: Filter Leaf Test Results

Parameter	Units	Test 11 - Final Tail	Test 9, 10, 11, 14 Combined Concentrate
pH	-	10	11.5
Solids S.G.	-	2.65	4.16
Particle Size K80	mm	77	22
Filter Area	cm^2	63.6	63.6
Filter Media	-	Whatman #1	Whatman #1
Filtration Rate	ml/sec	11	6.9
Estimated Sample Weight	g	150	150
Pulp Density	%	30	30
Pick up Time	sec	2880	1060
Dry Time	sec	N/A	90

Source: ALS 2015

13.4 Process Design

The process design criteria and proposed flowsheet were based on test 11 of ALS program KM4601 (KM4601-11GCl), results from previous test programs on Romero samples and industry standards, and vendor recommendations where test work was not available. The flowsheet includes crushing, grinding, gravity, flotation, dewatering and filtration unit operations.

13.4.1 Comminution Circuit

Based on the mineralized material hardness seen from grinding comminution test work, it was assumed a jaw crusher would reduce the underground material from 80% passing 600 mm to 150 mm in one stage. The grinding circuit will include a SAG mill, pebble crusher and ball mill. The SAG mill and ball mill were sized using a combination of the SMC and the Bond ball mill work index results from previous test programs, in conjunction with the JKSimMet grinding simulation software, Bond equation and efficiency factors. A SAG efficiency factor of 1.5 was used with a SAG power to ball mill power ratio of 40:60. The power requirements were calculated using average Life of Mine (LOM) daily tonnage with an assumed plant availability of 90% and a final target particle size of P_{80} 75 μm.

Table 13.4: Process Design Critera

Mill Process Design Parameters	Unit	Value	Mill Operating Parameters	Unit	Value
Operating Parameters			SAG Mill		
Daily Dry Tonnage	t/d	2,500	Number of SAG Mills	-	1
Availability	%	90	Mill Outside Diameter	ft	18 (5.5 m)
Hourly (Instantaneous) Throughput	t/h	115.7	Mill Length-EGL	ft	9 (2.7 m)
Ore Specific Gravity	-	2.94	Percent of Critical Speed (VS)	%	72
Ball Mill Work Index	kWh/t	15	Mill Speed	rpm	13
Abrasion Index	-	0.07	Percent Volume Total Charge	%	25
Feed Size,K80	µm	150,000	Percent Volume Steel Charge	%	9
Final Grind Size, P80	µm	75	Tons of Steel Charge	t	28
SAG Mill			Ore Specific Gravity	-	2.94
Final Grind Size	µm	900	Slurry Pulp Density	% sol	72
SAG Efficiency Factor	-	1.5	Slurry Specific Gravity	-	1.91
Transmission Loss Factor	-	1.05	Charge Specific Gravity	-	3.73
Power Required	kWh/t	7.42	Charge Density	lb/ft3	233
Unit Power Consumption	kW	859	Mill Power Draw	kW	952
Power Requirement	HP	1,152	Mill Power Draw	hp	1,277
Installed Power	HP	1,275	Mill Operating Parameters	Units	
% Power Utilized	%	90	Ball Mill		
Ball Milling			Number of Mills	-	1
Discharge Size P80	µm	75	Mill Diameter	ft	13 (4.0 m)
EF1 - Dry/Wet Grind	-	1	Mill Length	ft	21 (6.5 m)
EF2 - Open/Closed Circuit Grinding Factor	-	1	Mill Diameter Inside Liners	ft	13
EF3 - Diameter Efficiency Factor	-	0.915	Mill Length Inside Liners	ft	20
EF4 - Oversized Feed Factor	-	1	Volume Inside Mill	ft³	2,448
EF5 - Fine Grinding Factor	-	1	Percent Volume Loading of Balls	%	35
EF6 - N/A - Rod Mill Only	-	1	Ball Loading, ton(ne)s	s.t.	124
EF7 - Low Ratio of Reduction Factor	-	1	Percent of Mill Critical Speed	%	76
EF8 - N/A - Rod Mill Only	-	1	Mill Speed	rpm	16.47
Transmission Loss Factor	-	1.05	Bulk Density of Ball Charge	lb/ft³	290
Power Requirement	kWh/t	12.09	Makeup Ball Size	in	3
Power Required	kW	1,400	Ball Size Factor	-	0.56
Power Required	hp	1,876	Kilowatts per ton Balls	kW/t	10.96
Installed Power	hp	2,000	Mill Power Draw	kW	1,362
Power Utilized	%	94	Mill Power Draw	hp	1,826

Source: JDS 2015

The diameter, length and motor size for the mills were confirmed by vendors. Additional comminution test work is scheduled for the next stage of engineering to confirm the mineralization hardness.

13.4.2 Gold Recovery

The results from KM4601 cleaner tests 09Cl, without gravity, and 11GCl, with gravity, on the Romero Indicated sample at a target copper concentrate of 20% recovered approximately 68 and 74% gold, respectively. With the 4 to 5% additional gold recovered to the final concentrate, the economics indicate a gravity circuit should be included until further test work is completed.

13.4.3 Flotation

The flotation circuit design criteria was based on ALS KM4601-GCl11 flowsheet, reagent dosages, mass-pull and flotation times. The test parameters and results for KM4601-GCl11 are shown in Table 13.5 and Table 13.6, respectively. The flotation circuit feed size used for design was P80 = 75 microns with a target regrind particle size of P80 = 23 microns. The flowsheet included rougher flotation, followed by regrind of the rougher concentrate and three stages of cleaning. In the next stage of engineering it is recommended that additional testwork to optimize the flowsheet and gold recovery be continued.

13.4.3.1 KM4601-11 Romero Indicated Composite Test Procedures and Results

Table 13.5: KM4601-GCl11 Test Parameters

Stage	Reagents Added g/tonne			Time (minutes)			pH	Redox
	Lime	PAX	MIBC	Grind	Cond.	Float		
Natural							7.0	32
COPPER CIRCUIT:								
Rougher 1	350	5	15		1	2	10.0	82
Rougher 2	√	4	15		1	2	10.0	43
Rougher 3	√	3	15		1	2	10.0	52
Rougher 4	√	2	15		1	4	10.0	58
Regrind	650			25			11.0	20
Cleaner 1	200	20	23		1	10	11.5	-20
Cleaner 2	√	6	15		1	8	11.5	-35
Cleaner 3	√	4			1	6	11.5	-19

Source: ALS Test Results KM4601

Table 13.6: KM4601-GCl11 Cumulative Results

Cumulative	Cum. Weight		Assay - percent or g/t						Distribution - percent					
Product	%	grams	Cu	Zn	Fe	S	Ag	Au	Cu	Zn	Fe	S	Ag	Au
Product 1	1.0	20.9	1.57	0.27	43.8	50.5	23	53.5	2.1	2.0	7.3	9.0	6.0	17.7
Product 1 to 2	4.1	81.6	18.9	2.52	33.1	39.7	49	57.1	97.2	72.0	21.5	27.7	50.1	73.8
Product 1 to 3	5.0	98.8	15.7	2.10	30.8	36.5	43	48.1	97.5	72.4	24.3	30.8	52.9	75.3
Product 1 to 4	11.9	237.6	6.55	0.89	20.7	23.4	21	21.3	97.9	73.9	39.3	47.4	61.6	80.2
Product 1 to 5	33.9	675.8	2.32	0.33	15.4	16.6	10	8.74	98.8	77.0	83.0	96.1	83.5	93.5
Product 6	66.1	1318.6	0.02	0.05	1.6	0.34	1	0.31	1.2	23.0	17.0	3.9	16.5	6.5
Feed	100.0	1994.4	0.80	0.14	6.3	5.87	4	3.17	100	100	100	100	100	100

Source: ALS Test Results KM4601

13.4.4 Regrind

Flotation tests were completed at a range of particle sizes. A P80 = 23 microns was chosen as the target particle size to achieve liberation of the copper and gold minerals. Eliason Tests were conducted to provide an estimate of the energy required to regrind the rougher concentrate. The results indicated a specific energy requirement of 14.6 kWh/t was required and was the basis for sizing the regrind mill.

13.4.5 Dewatering and Filtering

The thickener sizing and reagent requirements were based on vendor recommendations for similar concentrates and tailings of a similar grind size.

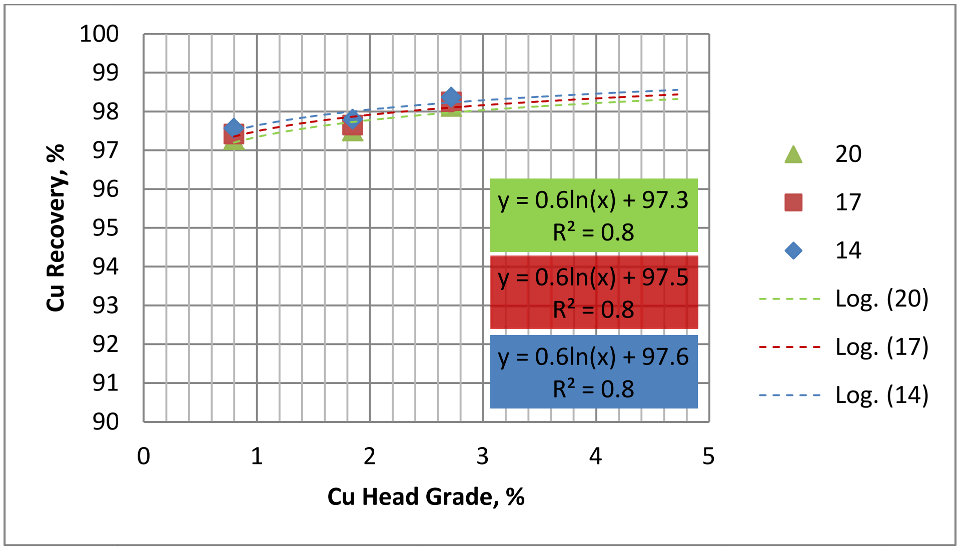
Preliminary filtration test work completed by ALS was sent to vendors for their recommended sizing, based on the performance of their equipment. The concentrate will be filtered in a pressure filter and the tailings by two disc filters. The target moisture content for the copper concentrate is 8% and 12.5% for the tailings.

13.5 Metallurgical Predictions

An analysis of the open circuit cleaner tests performed during ALS program KM4601 was undertaken to predict the copper, gold and silver recoveries. Three cleaner tests, M4601-10GCl,11GCl,14GCl, samples Romero Indicated, Sample 1 and Sample 3,were used as the basis to model recoveries versus head grades. Results from the tests were plotted at 14, 17 and 20% concentrate grades to develop a correlation between recovery and head grade.

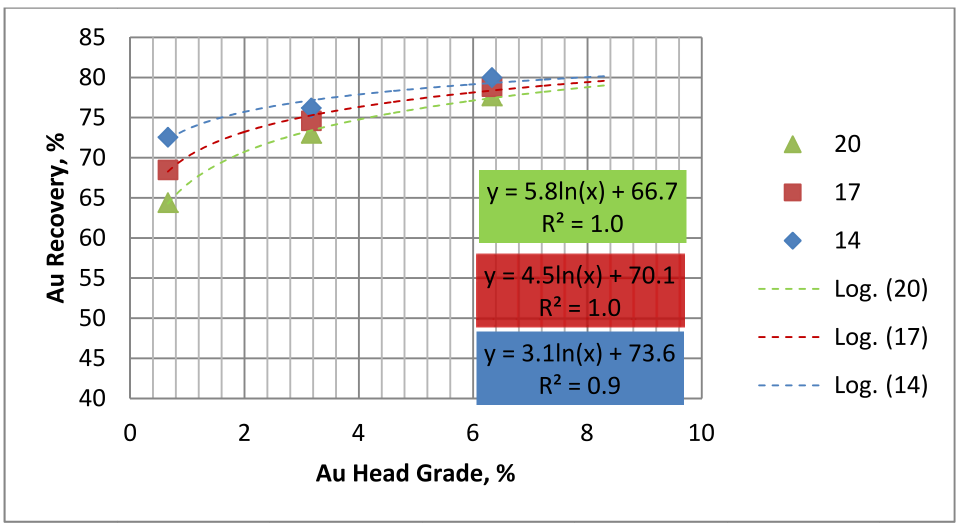
Copper, gold and silver head grades were plotted against their respective recoveries to develop the models outlined in Figure 13.7, Figure 13.8, Figure 13.9 and depict the resulting correlations.

Figure 13.7: Copper Recovery with respect to Copper Head Grade



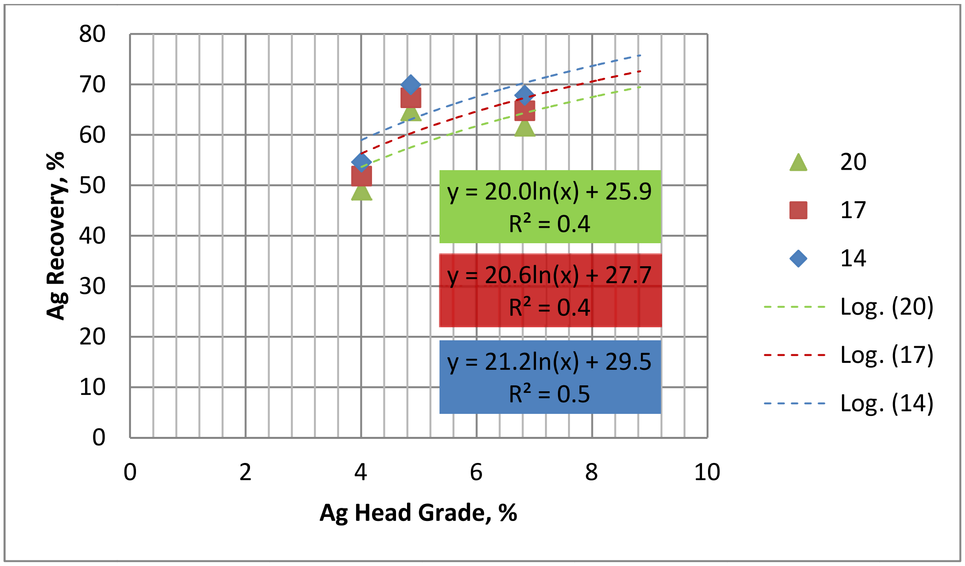
Source: ALS Test Program KM4601

Figure 13.8: Gold Recovery with respect to Gold Head Grade



Source: ALS Test Program KM4601

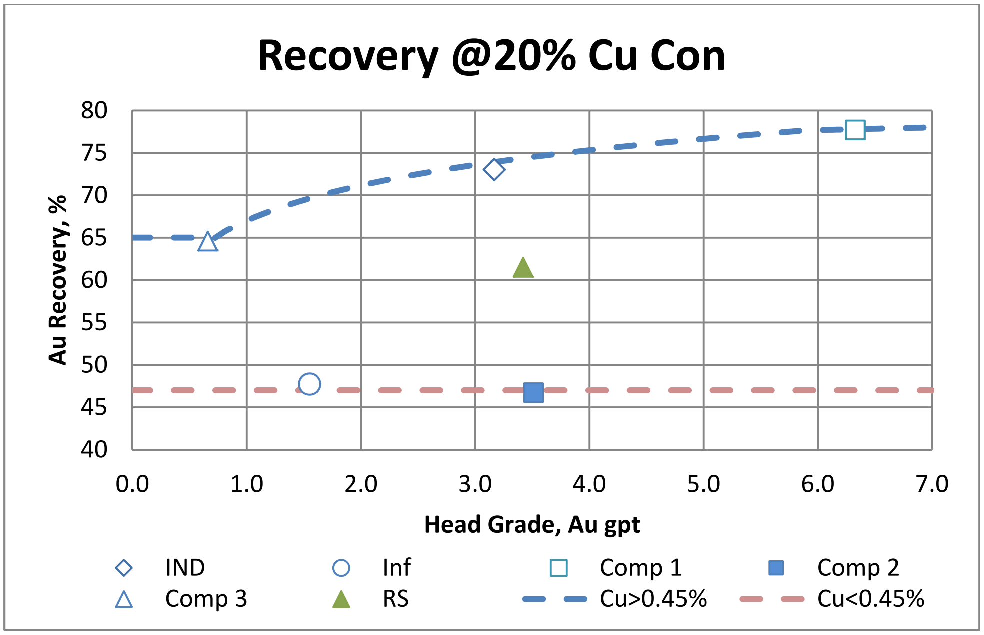
Figure 13.9: Silver Recovery with respect to Silver Head Grade



Source: ALS Test Program KM4601

The graph below shows the relationship between gold recovery and head grade at a copper concentrate of 20.

Figure 13.10: Recovery at 20% Copper Concentrate



Source: ALS Test Program KM4601, JDS 2015

The results from the test program KM4601 were used to predict the recoveries of copper, gold and silver at a copper concentrate grade of 20% for the LOM average head grades. A lower copper grade was chosen to maximize gold recovery to the final concentrate. Table 13.7 displays the predicted LOM metallurgical forecasts.

Table 13.7: Predicted LOM Metallurgical Recoveries of the Romero Deposit

Product	Wt%	Cu (%)	Ag (g/t)	Au (g/t)	Cu Rec (%)	Ag Rec (%)	Au Rec (%)
Copper Concentrate	3.92	20	54	76.9	96.8	49.8	75
Tailings	96.1	0.03	2.1	1	3.2	50.2	25
Feed	100	0.81	4.25	4.02	100	100	100

Source: JDS 2015

13.6 Product Quality Predictions

Copper concentrates produced from Romero and Romero South Indicated composites of test program KM4601 were submitted to ALS Minerals Vancouver for a multi-element ICP scan. The concentrates tested contained no deleterious elements and will not encounter smelter penalties. The results are presented in Figure 13.13 below.

Figure 13.11: Multi-element ICP Scan Results of Copper Concentrates

Element	Symbol	Unit	Romero Indicated Composite Test 9	Romero Indicated Composite Test 11	Romero South Composite Test 12
Copper	Cu	%	23.7	24.9	9.2
Gold	Au	g/t	56.4	58.4	75.8
Silver	Ag	g/t	59.9	59.9	32.1
Iron	Fe	%	28.2	27.8	36.6
Antimony	Sb	g/t	29.3	23.4	33
Arsenic	As	g/t	430	306	605
Bismuth	Bi	g/t	24.2	23.2	5.6
Cadmium	Cd	g/t	109	134	311
Calcium	Ca	%	0.09	0.07	0.08
Cobalt	Co	g/t	20	17	96
Lead	Pb	g/t	461	410	1,720
Magnesium	Mg	%	0.15	0.14	0.17
Manganese	Mn	g/t	50	40	130
Molybdenum	Mo	g/t	143	128	161
Phosphorus	Pb	g/t	<100	<100	<100
Selenium	Se	g/t	40	50	70
Sulphur	S	%	37.1	36	44.4
Zinc	Zn	%	2.74	3.3	4.78

Notes:

a) Full minor elements determinations can be found appended to the ALS report in Appendix IV

B) Copper, Sulphur and gold analysis performed by ALS Metallurgy Kamloops

C) Sulphure analysis was completing using LECO

Source: Source: ALS Test Program KM4601

14 Mineral Resource Estimate

14.1 Introduction

The Romero project contains two distinct zones of mineralization, Romero, and Romero South in a 2.2 km-long area of anomalous gold and base metals (see Figure 14.1). Mineral resources for the latter zone, previously known as La Escandalosa, were estimated by Micon in 2011 and published in August, 2012 (Steedman and Gowans, 2012). The mineral resource estimate presented in this report supersedes that estimate and was originally published in the 2014 Micon PEA.

Figure 14.1: Relative Location of the Romero Project Mineralized Zones

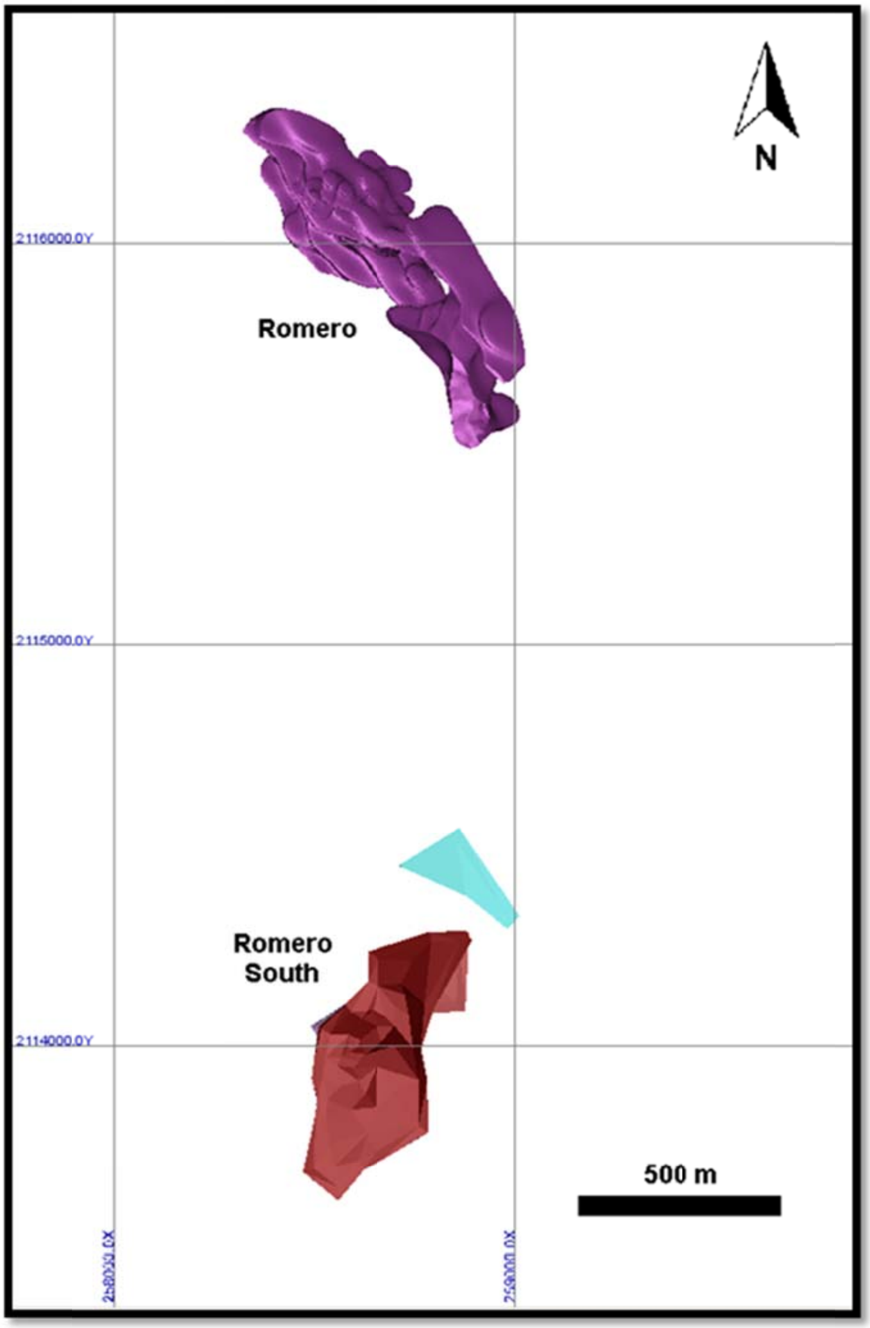


Figure supplied by GoldQuest (2013).

14.2 Mineral Resource Estimation Procedures

The mineral resource estimates for the Romero project deposits presented in this report are in accordance with NI 43-101 and follow the CIM Definition Standards – For Mineral Resources and Mineral Reserves as adopted by CIM Council on November 27, 2010 which state as follows:

"Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

"A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

"The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase "reasonable prospects for economic extraction" implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports."

Based on the CIM definitions the mineral resource estimate was carried out as described below.

14.2.1 Supporting Data

The Romero project database provided to Micon comprises 150 drill holes with a total of 39,629 m of drill core and containing 14,474 samples. Assays for gold, silver, copper and zinc were available for these holes. This database was the starting point from which the two mineralized envelopes, Romero and Romero South, were modelled.

From the entire database Micon used the data contained within the interpreted mineralization wireframes to estimate resources. The number of holes and samples used in the estimate were 113 drill holes and 4,199 samples, totalling 8,228 m of mineralized intercepts.

14.2.2 Topography

The project topography comes from a digital terrain model (DTM) constructed by GoldQuest based on purchased IKONOS satellite data. Some surveyed collar elevations were corrected using this topographic surface.

14.2.3 Geological Framework

The Romero project contains gold, silver, copper and zinc mineralization as described in Sections 7 through 10 of this report. This interpretation, along with input and guidance from GoldQuest staff was used to model the mineralization wireframes.

14.2.4 Local Rock Density

Bulk density measurements of core samples were taken by local technicians and geologists employed by GoldQuest using the weight-in-air, weight-in-water comparison method.

A total of 877 measurements were delivered to Micon from which average densities were calculated for the Romero and Romero South deposits, as well as for the surrounding waste rock. A few suspicious, extremely low values, less than 2.36, were not used. The overall average density value of the Romero project is 2.77 g/cm^3. Table 14.1 below summarizes the statistics of the calculations.

Table 14.1: Romero Project Average Density within the Envelopes

Deposit	Measurements (ea)	Min. (t/m^3)	Max. (t/m^3)	Avg. Value (t/m^3)
Romero South	113	2.36	4.22	2.71
Waste Rock	98	2.36	4.22	2.71
Mineralized Rock	15	2.44	3.23	2.72
Romero	714	2.4	4.72	2.78
Waste Rock	517	2.4	4.21	2.72
Mineralized Rock	197	2.4	4.72	2.94
Grand Total/Average	**827**	**2.36**	**4.72**	**2.77**

Source: Micon 2014

14.2.5 Population Statistics

Basic statistics were determined for the entire database. For the selected intervals in the mineralized envelopes, the results are as follows:

Table 14.2: Romero Basic Population Statistics

Variable	Romero				Romero South			
	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
Number of samples	9,383	9,383	9,383	9,383	4,184	4,184	4,184	4,184
Minimum value*	0.000	0.00	0.000	0.000	0.00025	0.002	0.0004	0.000
Maximum value	288.6	186.00	21.941	20.020	68.500	98.000	2.714	3.870
Mean	0.690	2.419	0.181	0.164	0.346	0.902	0.031	0.041
Median	0.100	1.000	0.013	0.020	0.014	0.262	0.007	0.010
Variance	27.191	21.211	0.402	0.293	4.286	6.062	0.010	0.025
Standard Deviation	5.215	4.606	0.634	0.541	2.070	2.462	0.101	0.158
Coefficient of variation	7.554	1.904	3.493	3.31	5.975	2.730	3.231	3.829

* - Zero value means missing assays assumed to be zero
Source: Micon 2014

14.2.6 Three-Dimensional Modelling

GoldQuest provided Micon with a preliminary 3D wireframe representing the interpreted mineralized envelope of the Romero deposit. The Romero South envelope, which had previously been interpreted by Micon, was reviewed and updated accordingly to account for the additional drilling completed since 2011.

Given that Romero project is a multi-element mineral resource, the Romero and Romero South envelopes prepared by Micon were defined using the in-situ contained metal value from the gold, silver, copper and zinc assays. The metal prices assumed for this calculation were; Au = US$1,400/oz, Ag = US$22.50/oz, Cu = US$3.18/lb and Zn = US$0.95/lb. These metal prices were derived from a long term consensus metal price forecasting service (Consensus Economics Inc.) which surveys 26 banks and economic monitoring units for short and medium term metal price predictions.

The metal value was calculated using the following formula:

- Metal Value = (Au g/t x Au price) + (Ag g/t x Ag price) + (Cu % x Cu price) + (Zn % x Zn price)

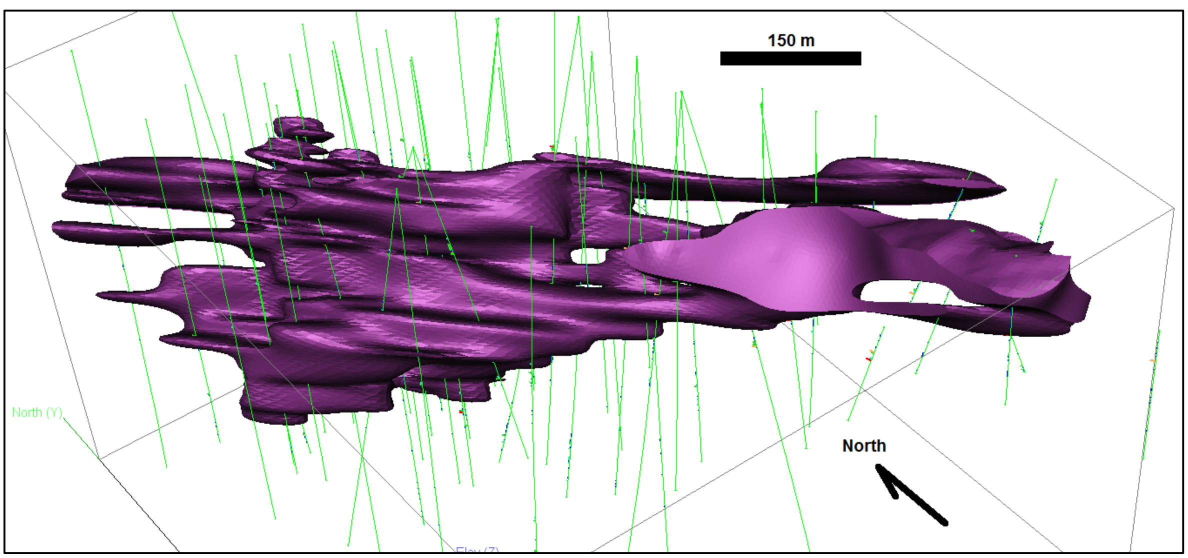
Gold and Silver units are in ppm and copper and zinc prices are in weight percent. Applying unit adjusting factors to prices, we have:*

- Metal Value in-situ = (Au g/t x US$45.01) + (Ag g/t x US$0.72) + (Cu % x US$70) + (Zn % x US$21)

The Romero deposit is complex with locally high gold and copper grades, along with zinc and silver grades which are not necessarily coincident. The interpretation of the mineralization and its envelope construction was performed by an implicit modelling method using Leapfrog Geo software. A contained metal value cut-off of US$20 was used along with other constraining parameters, such as interpreted dip and strike anisotropy, interactively until the desired envelope shape was achieved.

The Romero South deposit is simple set of stacked, flat-lying lenses. The mineralized envelope was updated using a US$15 cut-off metal value and the wireframe was constructed by conventional manual triangulation methods. Figure 14.2 and Figure 14.3 show 3D isometric views of the final interpreted mineralization lenses and intersecting drill holes.

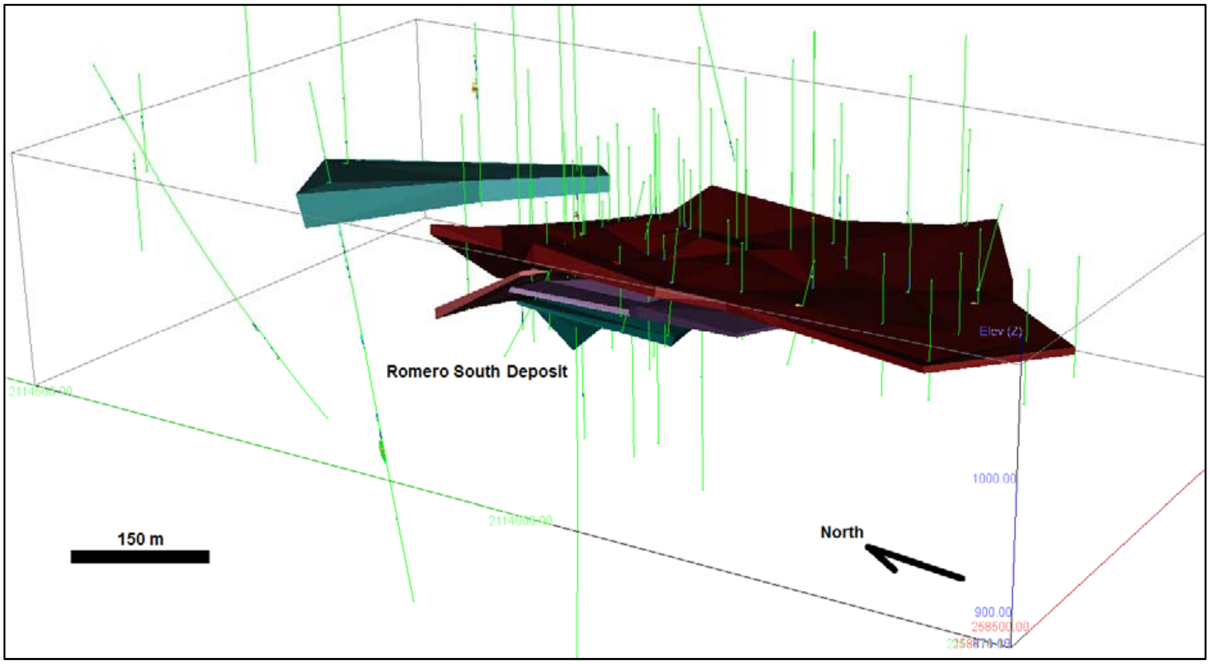
Figure 14.2: Romero Deposit Resulting Wireframe



(Looking down dip to the north-east)

Source: Micon 2014

Figure 14.3: Romero South Deposit Resulting Wireframes



(Looking down dip to the north-east)
Source: Micon 2014

Romero South shows three stacked lenses and a fourth lens to the north. The centre lens of the three stacked lenses was discontinuous and had to be separated into a zone 2 north and zone 2 south making for five separate zones. Zone 2 south and north were combined for variography as one is the along strike extension of the other.

14.2.7 Data Processing

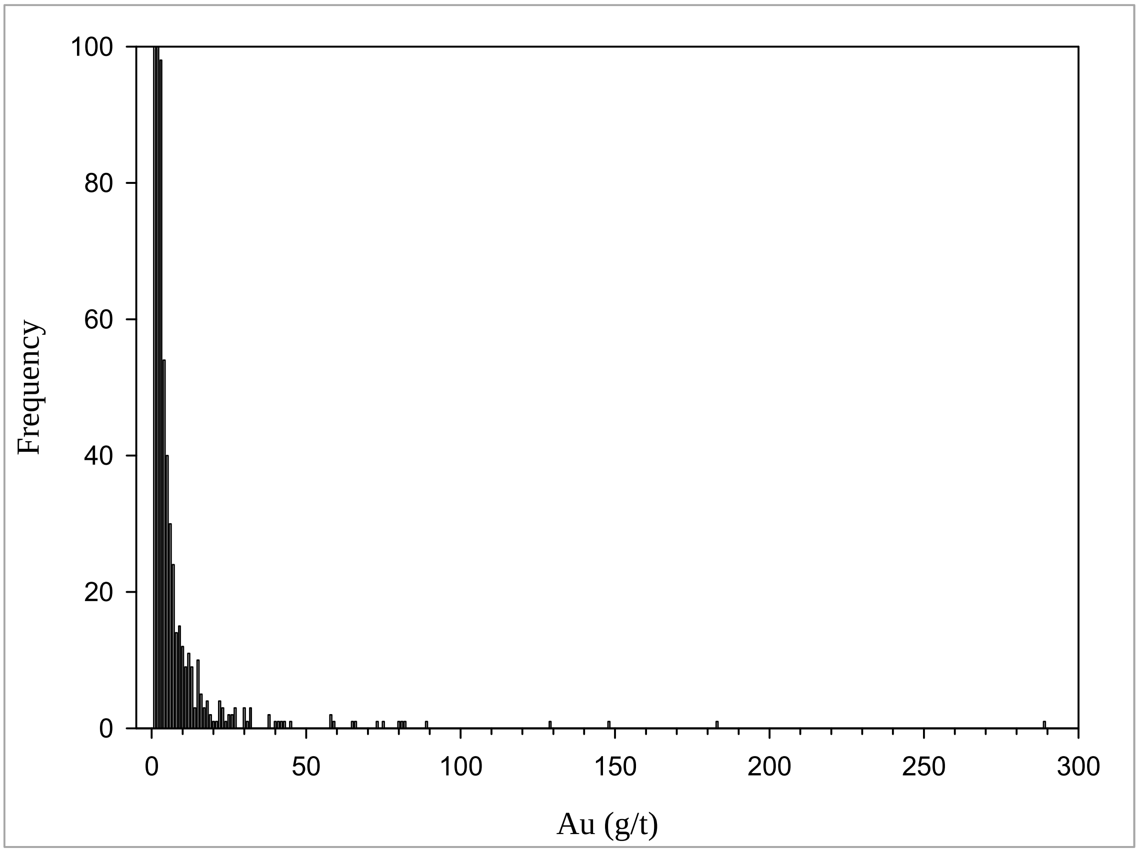
In order to complete the resource estimate the following procedures and analyses were performed.

14.2.7.1 High Grade Restriction

Gold, silver, copper and zinc data within the mineralized envelopes were examined for outlier values using histograms and probability plots. These are useful tools for the identification of the limits of log-normally distributed populations and the identification of any outlier values. These plots were reviewed and decisions made on capping values for the elements in question in order to prevent nugget effect from creating inappropriately high amounts of metal in the block model.

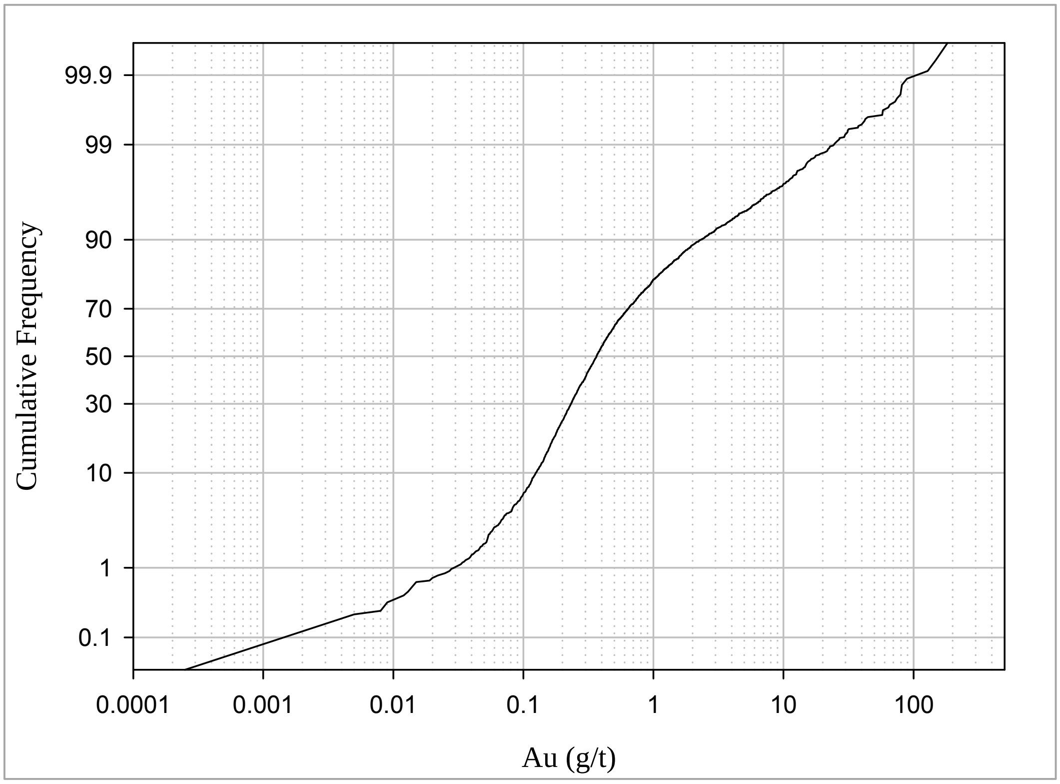
An example histogram and probability plot are shown in Figure 14.4 and Figure 14.5. Log normal populations plot as straight lines on probability plots. The upper point at which the straight line breaks down is often accepted as the capping value.

Figure 14.4:Romero Deposit Gold Histogram



Source: Micon 2014

Figure 14.5: Romero Deposit Gold Probability Plot



Source: Micon 2014

The grade capping values used in the Romero project mineral resource estimates are set out in Table 14.3 below.

Table 14.3: Romero Project Grade Capping

Element	Romero		Romero South	
	Cap Grade	Samples Capped	Cap Grade	Samples Capped
Au (g/t)	72.2	10	20.5	7
Ag (g/t)	60	8	15	16
Cu (%)	6.37	9	1.25	5
Zn (%)	6.91	7	1.65	9

Source: Micon 2014

14.2.7.2 Compositing

After grade capping, the selected intercepts were composited to 2 m equal length intervals with a minimum acceptable length of 1 m for those last composites of the intercept. Composites shorter than this were deleted so as not to introduce short sample bias. The composite length decision was made based on the average original sampling length. Table 14.4 shows the basic population statistics for the composited data.

Table 14.4: Romero Project Population Statistics for 2-m Composites

Variable	Romero							
	Au (g/t)	**Au CAP (g/t)**	**Ag (g/t)**	**Ag CAP (g/t)**	**Cu (%)**	**Cu CAP (%)**	**Zn (%)**	**Zn CAP (%)**
Number of samples	3,454	3,454	3,454	3,454	3,454	3,454	3,454	3,454
Minimum value	0.00025	0.00025	0.01	0.01	0.001	0.001	0.001	0.001
Maximum value	218.200	72.200	97.00	60.00	13.969	6.37	16.259	6.91
Mean	1.607	1.496	3.485	3.441	0.432	0.42	0.314	0.303
Median	0.381	0.381	2	2	0.138	0.138	0.1	0.1
Geometric Mean	0.473	0.472	2.265	2.263	0.122	0.122	0.093	0.093
Variance	43.85	23.836	28.833	23.431	0.691	0.512	0.529	0.333
Standard Deviation	6.622	4.882	5.37	4.841	0.831	0.715	0.727	0.577
Coefficient of variation	4.120	3.262	1.541	1.407	1.923	1.702	2.317	1.907
Variable	**Romero South**							
	Au (g/t)	**Au CAP (g/t)**	**Ag (g/t)**	**Ag CAP (g/t)**	**Cu (%)**	**Cu CAP (%)**	**Zn (%)**	**Zn CAP (%)**
Number of samples	591	591	591	591	591	591	591	591
Minimum value	0.005	0.005	0.002	0.002	0.001	0.001	0.000	0.000
Maximum value	68.500	20.500	86.170	15.000	1.398	1.25	3.547	1.650
Mean	2.19	2.006	2.233	1.882	0.156	0.155	0.170	0.161
Median	0.473	0.473	1.190	1.190	0.090	0.090	0.040	0.040
Geometric Mean	0.643	0.639	0.396	0.39	0.074	0.074	NC	NC
Variance	25.103	13.499	27.522	6.605	0.036	0.035	0.118	0.078
Standard Deviation	5.010	3.674	5.246	2.570	0.189	0.186	0.343	0.28
Coefficient of variation	2.288	1.832	2.350	1.366	1.210	1.196	2.018	1.740

Source: Micon 2014

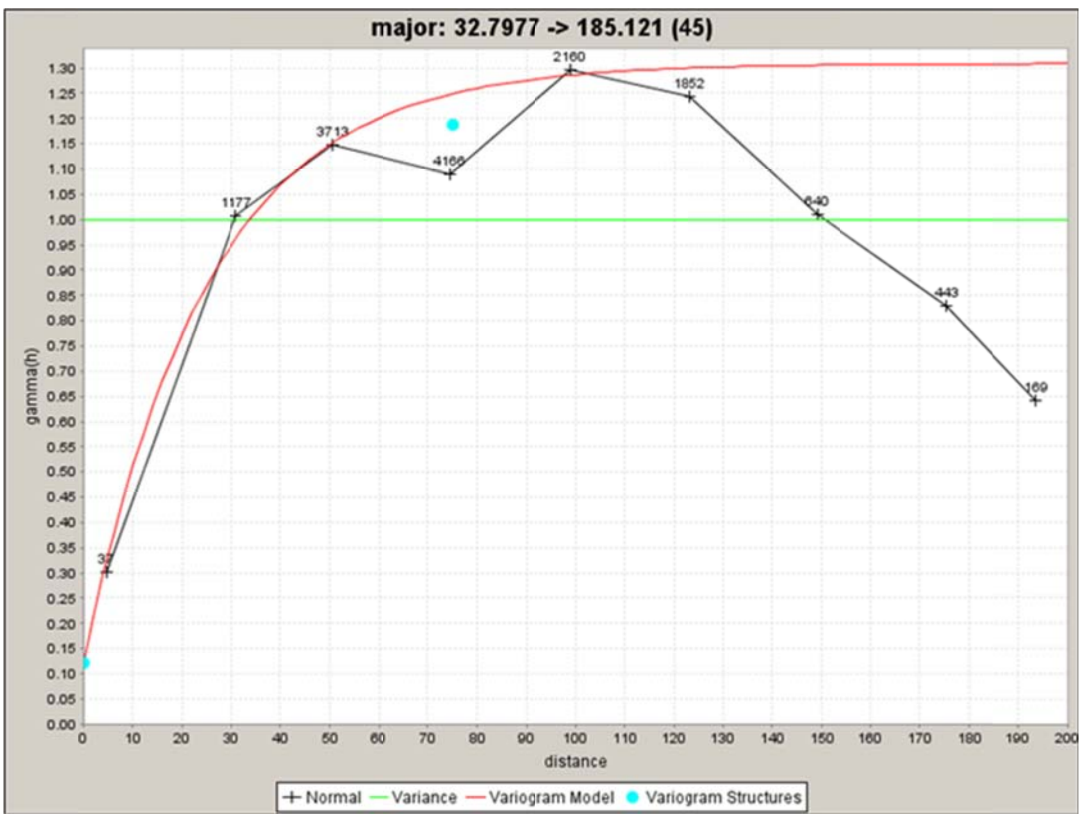
14.2.8 Variography

Variography is the analysis of the spatial continuity of grade. Micon performed various iterations with 3D variograms in order to obtain the necessary parameters for grade interpolation.

First down-the-hole variograms were developed for each zone to determine the nugget effect (y coordinate intercept of the variogram, or zero range variability) to be used in the modelling of the 3D variograms. As representative examples, Figure 14.6 and Figure 14.7 show the resulting major axis variograms for gold in both zones.

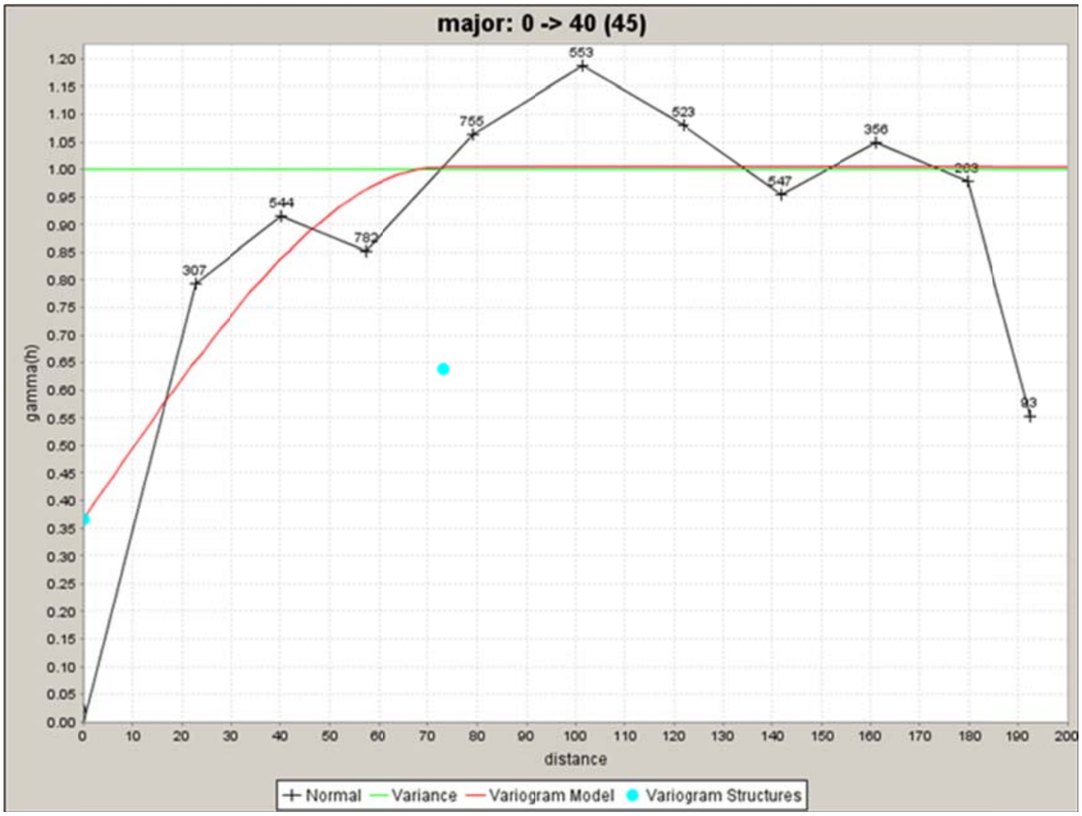
Variography should be performed on data from regular, coherent mineralized shapes with geological support. In that regard Romero South presented four different mineralized layers (see Section 14.2.6) and five zones where variograms were tested. Variograms could be modelled only for zones 1 (upper) and zone 2 north and 2 south combined. The variograms parameters from these were used in zone 3 and zone 4. Except for zone 3 and 4 at Romero South, Micon ran variograms for all elements in all zones.

Figure 14.6: Romero - Major Axis Variogram for Gold



Source: Micon 2014

Figure 14.7: Romero South - Major Axis Variogram for Gold



Source: Micon 2014

14.2.9 Continuity and Trends

The Romero and Romero South zones present good grade continuity; however, these two zones have clearly different orientations and dip. Romero has a strike of 325º and a 45º northeast dip while Romero South has a 20º strike of its long axis with almost no dip, and a partial plunge in the northern portion of the deposit of about -20º northeast.

The mineralization trends are well defined in both Romero and Romero South, but Romero presents a thicker zone of mineralization.

14.3 Mineral Resource Estimation

14.3.1 Block Model

Two block models were constructed; the first one contains the Romero deposit, and the second block model Romero South. A summary of both block models' definitions and data is listed in Table 14.5 below.

Table 14.5: Romero Project Block Model Information Summary

Description	Romero	Romero South
Dimension X (m)	1,200	1,300
Dimension Y (m)	600	1,500
Dimension Z (m)	560	600
Origin X (Easting)	258,100	258,000
Origin Y (Northing)	2,116,275	2,113,300
Origin Z (Upper Elev.)	1,120	1,410
Rotation (º)	305	0
Block Size X (m)	10	10
Block Size Y (m)	4	10
Block Size Z (m)	4	2

Source: Micon 2014

14.3.2 Search Strategy and Interpolation

Grade interpolation parameters were derived from the results of the variographic analysis. These parameters were used in the ordinary kriging (OK) grade interpolation to fill the blocks in the model. The search parameters used are set out in Table 14.6.

Table 14.6: Romero Project Ordinary Kriging Interpolation Parameters

			Variogram Parameters								Search Parameters		
Element	Rock* Code(s)	Pass	Az (°)	Plunge (°)	Dip (°)	Nugget	Sill	Range Major Axis (m)	Range Semi Major Axis (m)	Range Vertical Axis (m)	Min. Samples	Max. Samples	Max Samples per Hole
Au	ROM6	1	185	-32	-30	0.117	1.187	75	55	50	6	12	2
	ROM6	2	185	-32	-30	0.117	1.187	150	110	100	4	8	2
	ROM6	3	185	-32	-30	0.117	1.187	150	110	110	2	8	2
Ag	ROM6	1	62	-4	-45	0.052	0.886	75	60	50	6	12	2
	ROM6	2	62	-4	-45	0.052	0.886	150	120	100	4	8	2
	ROM6	3	62	-4	-45	0.052	0.886	150	120	100	2	8	2
Cu	ROM6	1	190	-35	-24	0.111	1.299	75	50	50	6	12	2
	ROM6	2	190	-35	-24	0.111	1.299	150	100	100	4	8	2
	ROM6	3	190	-35	-24	0.111	1.299	150	100	100	2	8	2
Zn	ROM6	1	195	-38	15	0.100	0.999	80	50	50	6	12	2
	ROM6	2	195	-38	15	0.100	0.999	160	100	100	4	8	2
	ROM6	3	195	-38	15	0.100	0.999	160	100	100	2	8	2
Au	ROMS1-5**	1	40,140	0, -26	0	0.366	0.638	70, 80	50, 60	50, 60	6	12	2
	ROMS1-5**	2	40,140	0, -26	0	0.366	0.638	140, 160	100, 120	100, 120	4	8	2
	ROMS1-5**	3	40,140	0, -26	0	0.366	0.638	140, 160	100, 120	100, 120	2	8	2
Ag	ROMS1-5**	1	40,140	0, -26	0	0.177	0.821	70, 80	50, 60	50, 60	6	12	2
	ROMS1-5**	2	40,140	0, -26	0	0.177	0.821	140, 160	100, 120	100, 120	4	8	2
	ROMS1-5**	3	40,140	0, -26	0	0.177	0.821	140, 160	100, 120	100, 120	2	8	2
Cu	ROMS1-5**	1	40,140	0, -26	0	0.133	0.876	70, 80	50, 60	50, 60	6	12	2
	ROMS1-5**	2	40,140	0, -26	0	0.133	0.876	140, 160	100, 120	100, 120	4	8	2
	ROMS1-5**	3	40,140	0, -26	0	0.133	0.876	140, 160	100, 120	100, 120	2	8	2
Zn	ROMS1-5**	1	40,140	0, -26	0	0.174	0.828	70, 80	50, 60	50, 60	6	12	2
	ROMS1-5**	2	40,140	0, -26	0	0.174	0.828	140, 160	100, 120	100, 120	4	8	2
	ROMS1-5**	3	40,140	0, -26	0	0.174	0.828	140, 160	100, 120	100, 120	2	8	2

* - Rock codes Romero (ROM6), Romero South (ROMS1, ROMS2, ROMS3, ROMS4 and ROMS5).

** - Romero South has multiple horizontal zones as described above. There were only minor differences in many of the parameters for the different elements in ROMS1-5. For simplification it was determined that there was no need to present them separately. More than one azimuth or range has been presented in each row.

Source: Micon 2014

14.3.3 Prospects for Economic Extraction

The mineral resource has been constrained using economic assumptions which considered underground mining scenarios. The economic assumptions used are listed in Table 14.7 below.

Table 14.7:Romero Mineral Resource Estimate Economic Assumptions

Description	Underground Romero	Underground Romero South
Mining Method	Sublevel Open Stoping	Room and Pillar
Au price US$/Oz	1,400.00	1,400.00
Ag price US$/Oz	22.50	22.50
Cu price US$/lb	3.18	3.18
Zn price US$/lb	0.95	0.95
Au recovery %	76.6	76.6
Ag recovery %	85.0	85.0
Cu recovery %	90.0	90.0
Zn recovery %	90.0	90.0
Price Weighted Avg. Recovery %	76.7	76.7
Mining Cost US$/t	30.00	24.00
Mill Cost US$/t	12.50	12.50
G&A Cost US$/t	2.50	2.50
Overall Cost US$/t	45.00	39.00

Source: Micon 2014

The Romero project mineral resources were evaluated and reported from the calculated contained metal value for each block (including gold, copper, silver and zinc values, Section 14.2.6) using the cost, commodity price and recovery parameters in Table 14.7 above. A dollar NSR value of payable metal was determined for the two cut-offs used. For the purposes of reporting the mineral resources, Micon selected an NSR cut-off of US$60 (overall cost/price weighted recovery) as an estimate of what might be a reasonable marginal cost of extraction at Romero and US$50 as the marginal cost of extraction at Romero South.

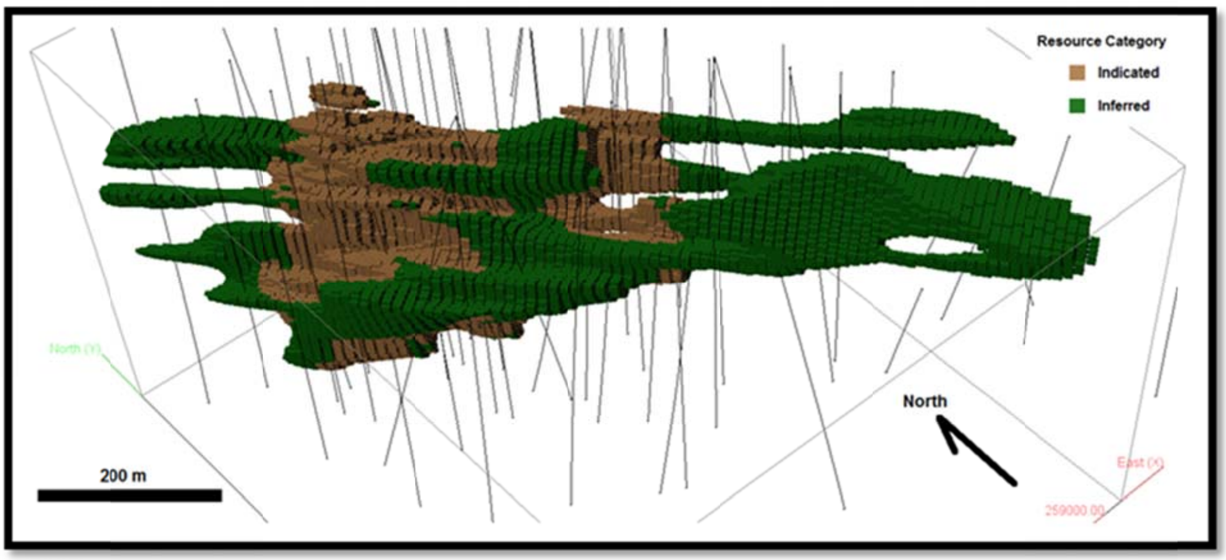
14.3.4 Mineral Resource Categorization

The mineral resource estimates for Romero and Romero South have been categorized into the indicated and inferred categories (see Figure 14.8 and Figure 14.9). No measured resources have been determined at this time. The criteria for classification is as follows:

- Indicated resources are those blocks that are within the range outlined in interpolation pass 1 of Table 14.6 and which have been interpolated using three or more drill holes;
- Inferred resources are all those remaining blocks that do not meet the criteria of the indicated category (pass 2 and 3 of Table 14.6);

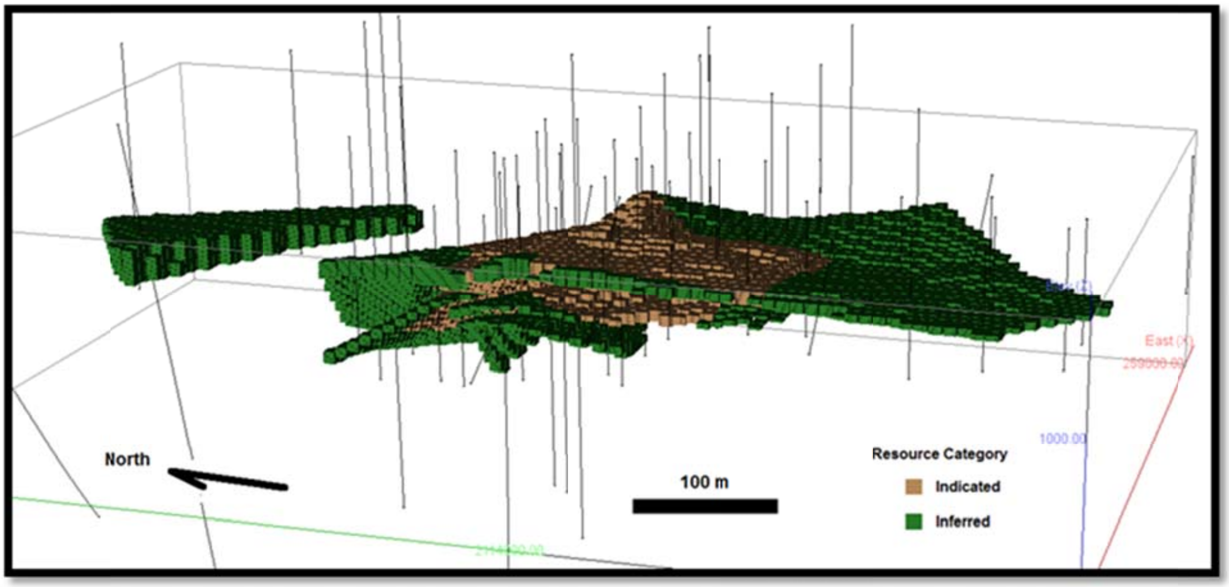
These rules were combined with a visual check of the model to make sure the indicated resource has a regular, continuous shape and is not broken up creating the "spotted dog effect" (scattered isolated islands of indicated resource). Some indicated blocks were downgraded in this checking process.

Figure 14.8: Romero Block Model Isometric View - Resource Category



Source: Micon 2014

Figure 14.9: Romero South Block Model Isometric View - Resource Category



Source: Micon 2014

14.4 Mineral Resources

The mineral resources determined for the Romero project are set out in Table 14.8.

Table 14.8: Romero Project Mineral Resources

Category	Zone	Tonnes (x 1,000)	Au (g/t)	Cu (%)	Zn (%)	Ag (g/t)	AuEq (g/t)	Au Ounces (x 1,000)	AuEq Ounces (x 1,000)
Indicated	Romero	17,310	2.55	0.68	0.30	4.0	3.81	1,419	2,123
	Romero South	2,110	3.33	0.23	0.17	1.5	3.80	226	258
Total Indicated Resources		19,420	2.63	0.63	0.29	3.7	3.81	1,645	2,381
Inferred	Romero	8,520	1.59	0.39	0.46	4.0	2.47	437	678
	Romero South	1,500	1.92	0.19	0.18	2.3	2.33	92	112
Total Inferred Resources		10,020	1.64	0.36	0.42	3.8	2.45	529	790

Note: AuEq g/t = (Au g/t)+(Ag g/t)/62.222)+(Cu %)/0.642)+(Zn %)/2.1491)

Source: Micon 2014

The present report and mineral resource estimates are based on exploration results and interpretation current as of October 10, 2013. The effective date of the mineral resource estimates is October 29, 2013.

It is Micon's opinion that there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues which exist that would adversely affect the mineral resource estimates for Romero and Romero South presented above. The mineral resources presented herein are not mineral reserves as they have not been subject to adequate economic studies to demonstrate their economic viability. They represent in-situ tonnes and grades and have not been adjusted for mining losses or dilution.

A portion of the mineral resource estimate has been designated as inferred as there has been insufficient exploration to define it as an indicated or measured mineral resource. It is uncertain if further exploration will result in upgrading to an indicated or measured mineral resource category.

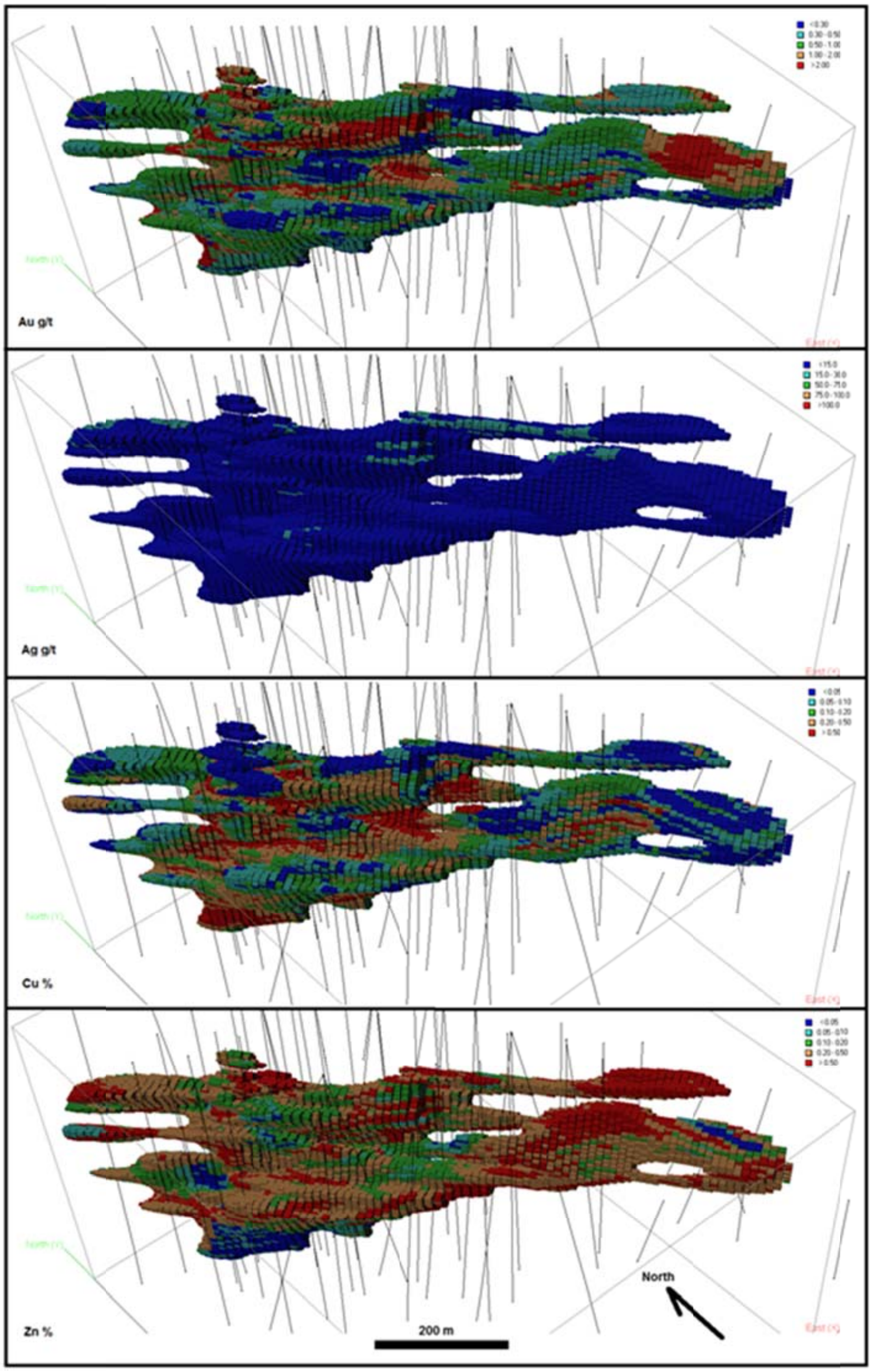
14.4.1 Responsibility for Estimation

The mineral resource estimates for the Romero and Romero South deposits have been prepared and categorized for reporting purposes by B. T. Hennessey, P.Geo. and A. J. San Martin, MAusIMM(CP), of Micon, following the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum. Both Mr. Hennessey and Mr. San Martin are Qualified Persons as defined by NI 43-101 on the basis of training and experience in the exploration, mining and estimation of mineral resources of gold deposits. Both Messrs. Hennessey and San Martin are independent of GoldQuest.

14.4.2 Block Model Isometric Views

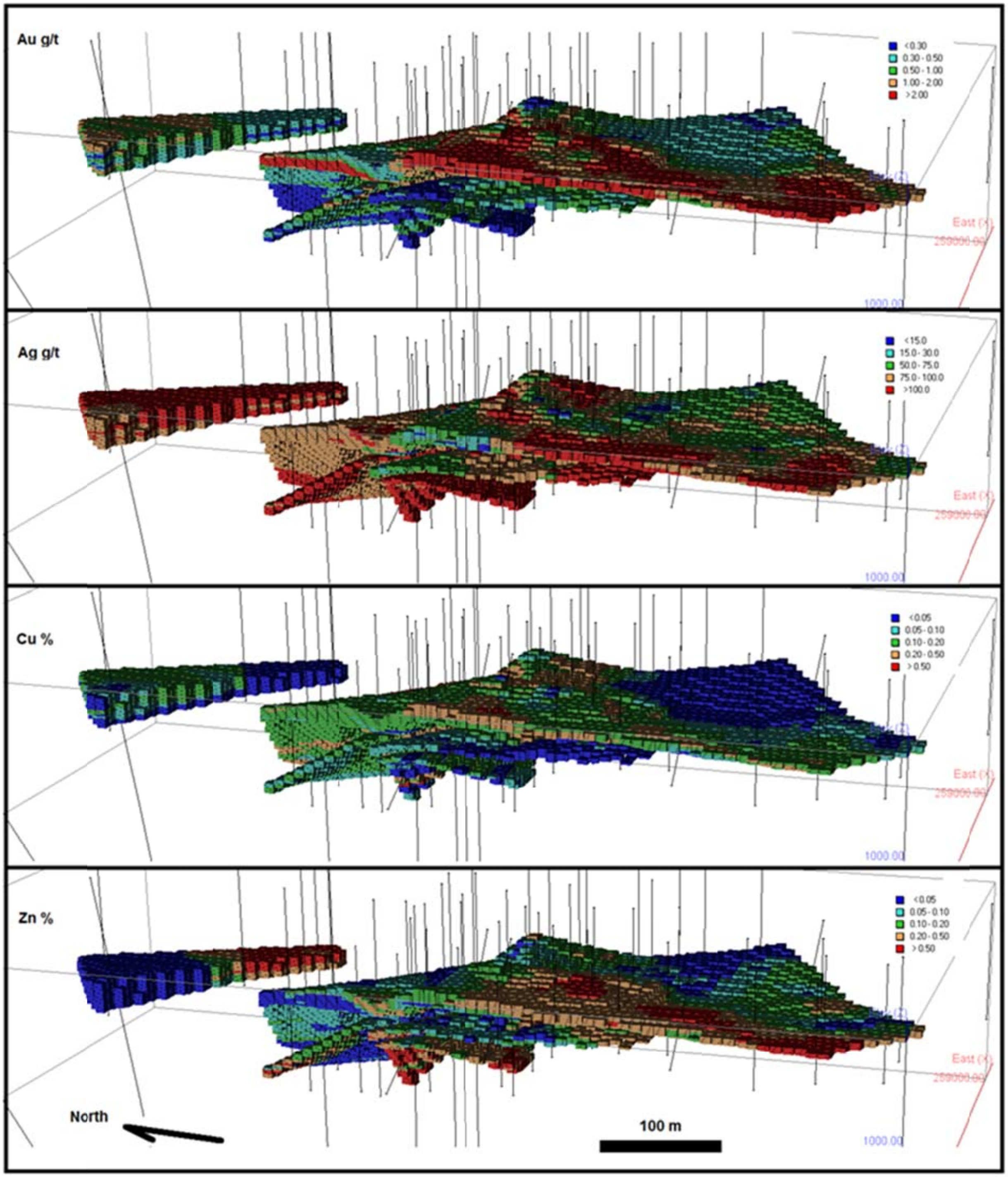
Figure 14.10 and Figure 14.11 graphically show the grade of the mineral resources tabulated above as 3D isometric views of the block model.

Figure 14.10: Romero Block Model Isometric View - Grade Distribution



Source: Micon 2014

Figure 14.11: Romero South Block Model Isometric View - Grade Distribution



Source: Micon 2014

14.5 Sensitivity to Cut-off

Micon has prepared tables of the mineral resource sensitivity to changes in the dollar NSR cut-off. That data can be seen in Table 14.9 to Table 14.10 below.

Table 14.9: Romero Indicated Resources Sensitivity to NSR Cut-off

Category	Cut-off (US$)	Cum. Tonnage	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au-Eq (g/t)	Au Ounces	Au-Eq Ounces
Indicated	>150	6,230,000	5.21	4.6	0.94	0.36	6.92	1,043,000	1,386,000
Indicated	140	6,810,000	4.92	4.6	0.93	0.35	6.60	1,077,000	1,446,000
Indicated	130	7,470,000	4.64	4.5	0.91	0.35	6.29	1,114,000	1,510,000
Indicated	120	8,200,000	4.36	4.5	0.89	0.34	5.97	1,149,000	1,575,000
Indicated	110	9,090,000	4.06	4.4	0.87	0.34	5.64	1,187,000	1,648,000
Indicated	100	10,100,000	3.77	4.4	0.84	0.33	5.31	1,226,000	1,723,000
Indicated	90	11,390,000	3.47	4.3	0.81	0.33	4.95	1,269,000	1,811,000
Indicated	80	13,000,000	3.15	4.2	0.77	0.32	4.57	1,317,000	1,909,000
Indicated	70	14,950,000	2.84	4.1	0.73	0.31	4.19	1,367,000	2,013,000
Indicated	**60**	**17,310,000**	**2.55**	**4.0**	**0.68**	**0.30**	**3.81**	**1,419,000**	**2,123,000**
Indicated	50	20,080,000	2.28	3.9	0.63	0.30	3.46	1,471,000	2,231,000
Indicated	40	23,400,000	2.02	3.8	0.57	0.29	3.11	1,522,000	2,338,000
Indicated	30	27,490,000	1.78	3.7	0.51	0.28	2.76	1,573,000	2,440,000

(reported cut-off in bold)

Source: Micon 2014

Table 14.10: Romero Inferred Resources Sensitivity to NSR Cut-off

Category	Cut-off (US$)	Cum. Tonnage	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au-Eq (g/t)	Au Ounces	Au-Eq Ounces
Inferred	>150	1,460,000	3.84	5.1	0.58	0.48	5.04	180,000	237,000
Inferred	140	1,690,000	3.61	5.0	0.57	0.48	4.79	196,000	261,000
Inferred	130	1,990,000	3.36	4.9	0.55	0.48	4.52	215,000	289,000
Inferred	120	2,370,000	3.10	4.7	0.54	0.48	4.24	236,000	323,000
Inferred	110	2,830,000	2.86	4.6	0.52	0.48	3.97	260,000	361,000
Inferred	100	3,410,000	2.62	4.5	0.50	0.47	3.69	287,000	405,000
Inferred	90	4,080,000	2.39	4.4	0.48	0.47	3.43	314,000	450,000
Inferred	80	5,020,000	2.14	4.3	0.46	0.47	3.14	346,000	507,000
Inferred	70	6,340,000	1.88	4.2	0.43	0.47	2.83	383,000	577,000
Inferred	**60**	**8,520,000**	**1.59**	**4.0**	**0.39**	**0.46**	**2.47**	**437,000**	**678,000**
Inferred	50	11,850,000	1.33	3.9	0.34	0.45	2.12	506,000	808,000
Inferred	40	17,340,000	1.07	3.8	0.28	0.43	1.76	596,000	983,000
Inferred	30	24,420,000	0.87	3.6	0.23	0.41	1.48	685,000	1,160,000

(reported cut-off in bold)

Source: Micon 2014

Table 14.11: Romero South Indicated Resources Sensitivity to NSR Cut-off

Category	Cut-off (US$)	Cum. Tonnage	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au-Eq (g/t)	Au Ounces	Au-Eq Ounces
Indicated	>150	950,000	5.34	1.5	0.29	0.19	5.90	163,000	180,000
Indicated	140	1,040,000	5.11	1.5	0.28	0.19	5.66	171,000	189,000
Indicated	130	1,120,000	4.93	1.5	0.28	0.19	5.47	177,000	197,000
Indicated	120	1,210,000	4.73	1.5	0.27	0.19	5.27	184,000	205,000
Indicated	110	1,310,000	4.54	1.5	0.27	0.19	5.06	191,000	213,000
Indicated	100	1,420,000	4.34	1.5	0.26	0.19	4.86	198,000	222,000
Indicated	90	1,540,000	4.13	1.5	0.26	0.19	4.64	205,000	230,000
Indicated	80	1,660,000	3.94	1.5	0.25	0.18	4.45	210,000	237,000
Indicated	70	1,800,000	3.74	1.5	0.25	0.18	4.23	216,000	245,000
Indicated	60	1,940,000	3.55	1.5	0.24	0.17	4.03	221,000	251,000
Indicated	**50**	**2,110,000**	**3.33**	**1.5**	**0.23**	**0.17**	**3.80**	**226,000**	**258,000**
Indicated	40	2,300,000	3.12	1.4	0.22	0.17	3.57	231,000	264,000
Indicated	30	2,550,000	2.87	1.5	0.21	0.16	3.30	235,000	270,000

(reported cut-off in bold)

Source: Micon 2014

Table 14.12: Romero South Inferred Resources Sensitivity to NSR Cut-off

Category	Cut-off (US$)	Cum. Tonnage	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Au-Eq (g/t)	Au Ounces	Au-Eq Ounces
Inferred	>150	240,000	5.10	2.1	0.22	0.25	5.59	39,000	43,000
Inferred	140	280,000	4.74	2.2	0.22	0.27	5.25	43,000	47,000
Inferred	130	320,000	4.47	2.2	0.23	0.28	4.99	46,000	51,000
Inferred	120	360,000	4.24	2.2	0.23	0.29	4.77	49,000	55,000
Inferred	110	400,000	4.05	2.2	0.23	0.29	4.57	52,000	59,000
Inferred	100	460,000	3.76	2.2	0.22	0.28	4.27	56,000	63,000
Inferred	90	520,000	3.53	2.2	0.22	0.27	4.03	59,000	67,000
Inferred	80	610,000	3.23	2.1	0.22	0.27	3.73	63,000	73,000
Inferred	70	760,000	2.84	2.2	0.21	0.25	3.32	69,000	81,000
Inferred	60	1,060,000	2.34	2.2	0.20	0.21	2.78	80,000	95,000
Inferred	**50**	**1,500,000**	**1.92**	**2.3**	**0.19**	**0.18**	**2.33**	**92,000**	**112,000**
Inferred	40	2,190,000	1.53	2.4	0.17	0.18	1.91	107,000	134,000
Inferred	30	3,090,000	1.21	2.5	0.15	0.18	1.58	120,000	157,000

(reported cut-offin bold)

Source: Micon 2014

14.6 Block Model Checks and Validation

A block model is a three dimensional representation of the estimated tonnage and grade in a given mineralized envelope. As such, it should be validated in order to give the best level of confidence possible. Micon has carried out four methods of validation to accomplish this goal.

14.6.1 Statistical Comparison

The average grade of the informing composites within the mineralized envelope was compared to the average grade of the all the resulting blocks. Table 14.13 below shows the results for all four elements of the mineral resource.

Table 14.13: Romero Project 2-m Composites vs. Blocks

Deposit	Grade	Block Model Average	2m Composite Average
Romero	Au g/t	1.140	1.496
	Ag g/t	3.300	3.441
	Cu %	0.327	0.420
	Zn %	0.318	0.303
Romero South	Au g/t	1.467	2.006
	Ag g/t	2.000	1.882
	Cu %	0.147	0.155
	Zn %	0.149	0.161

Source: Micon 2014

As expected the block model grades have been smoothed and are generally somewhat lower than the grade of the informing samples.

14.6.2 Comparison to Other Interpolation Methods

As a comparison to OK, Micon also interpolated grades using the inverse distance squared (ID^2) method for Romero and Romero South. As can be seen in Table 14.14 and Table 14.15, the comparisons are very close.

Table 14.14: Comparison of OK and ID^2 Grades for Gold and Copper

Category	Zone	Tonnes (x 1,000)	Au (g/t)		Cu (%)	
			OK	ID^2	OK	ID^2
Indicated	Romero	17,310	2.55	2.55	0.68	0.68
	Romero South	2,110	3.33	3.35	0.23	0.23
Inferred	Romero	8,520	1.59	1.60	0.39	0.39
	Romero South	1,500	1.92	1.92	0.19	0.18

Source: Micon 2014

Table 14.15: Comparison of OK and ID^2 Grades for Zinc and Silver

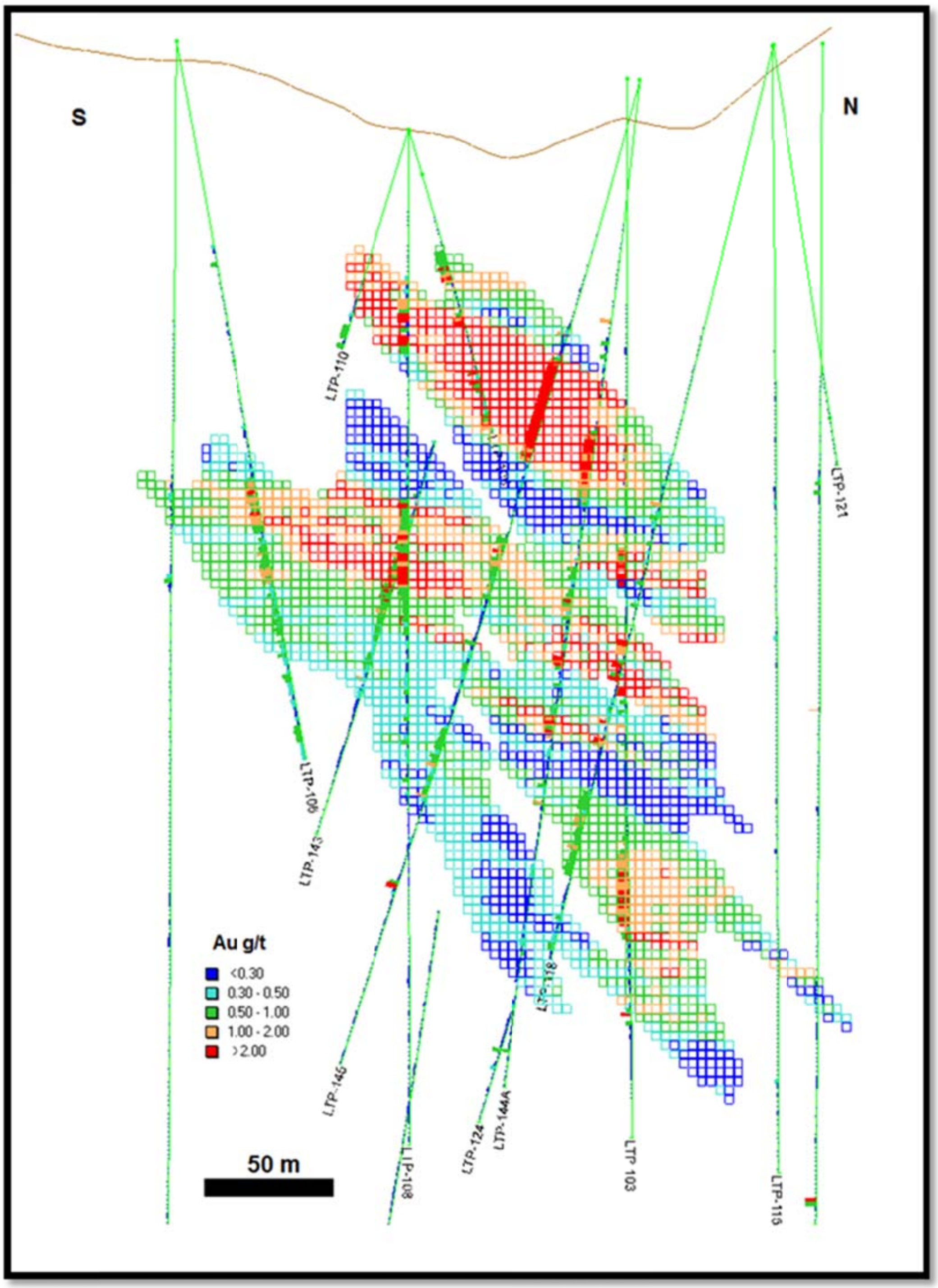
Category	Zone	Tonnes (x 1,000)	Zn (%)		Ag (g/t)	
			OK	ID^2	OK	ID^2
Indicated	Romero	17,310	0.30	0.31	4.0	4.1
	Romero South	2,110	0.17	0.17	1.5	1.5
Inferred	Romero	8,520	0.46	0.46	4.0	4.0
	Romero South	1,500	0.18	0.18	2.3	2.2

Source: Micon 2014

14.6.3 Visual Inspection

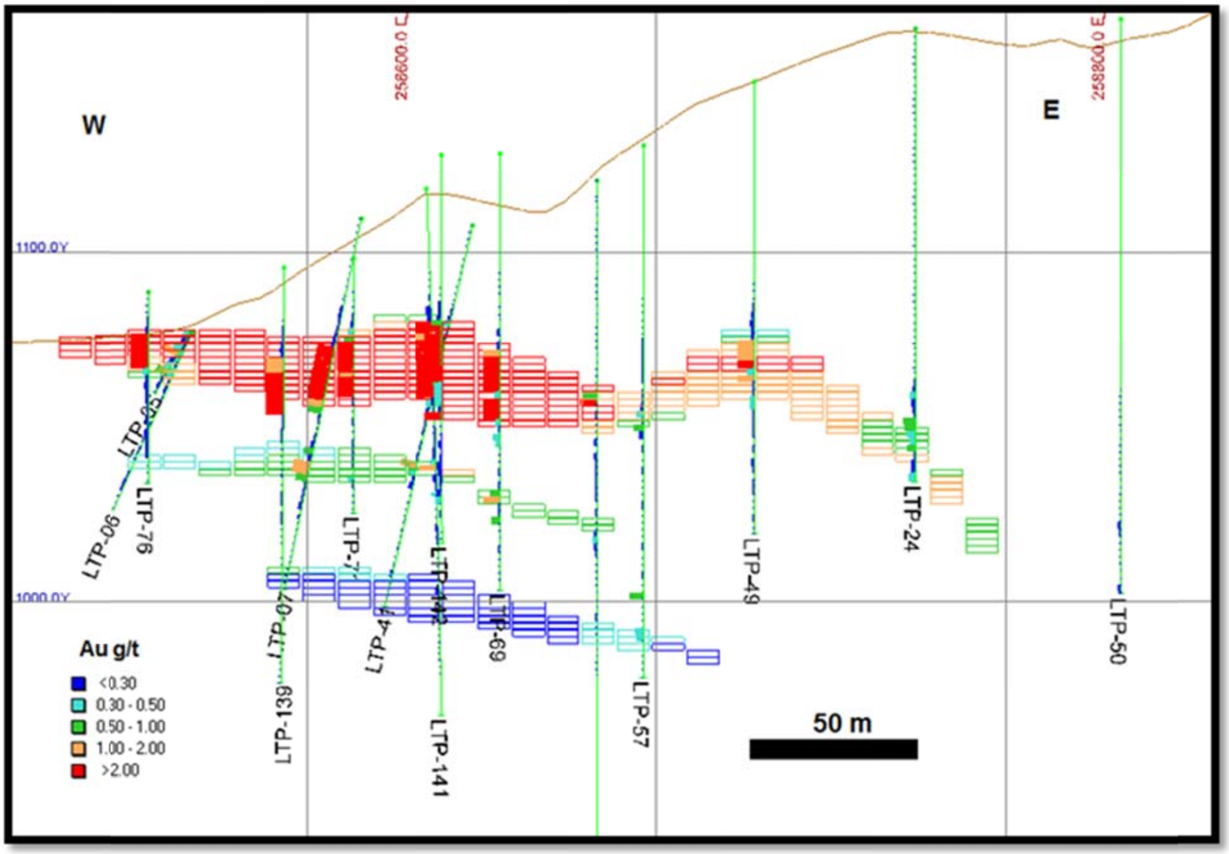
The block models and drill holes were reviewed on section to ensure that the grade distribution in the blocks honoured the neighbouring drill hole data. Figure 14.12 and Figure 14.13 show typical results.

Figure 14.12: Romero Typical Vertical Section



Source: Micon 2014

Figure 14.13: Romero South Typical Vertical Section



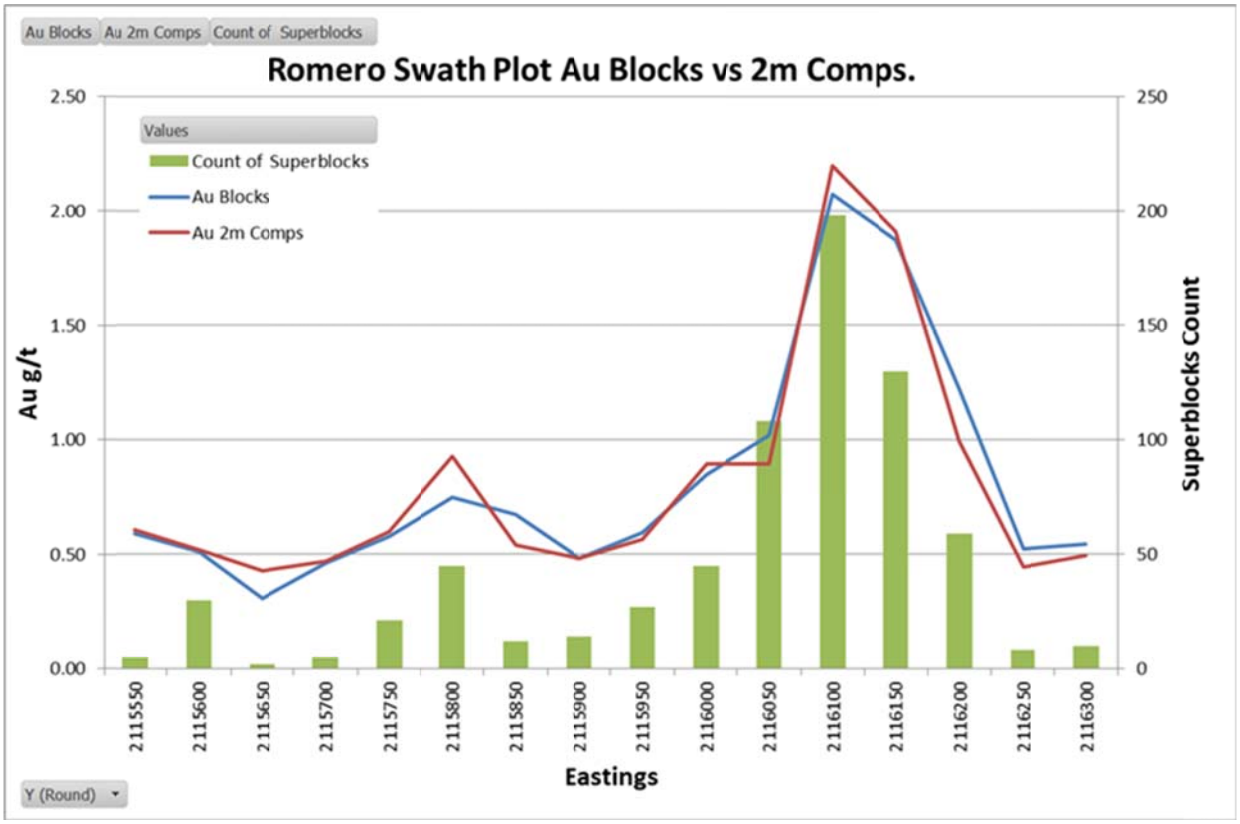
Source: Micon 2014

14.6.4 Trend Analysis

Trend analysis is an exercise involving the super blocking (averaging of groups of data) of grade data and comparing the resulting block model values to the source informing composites. The results are plotted in a swath plot following the strike of the deposit. Broad grade trends in the block model should respect the grade trends in the informing data.

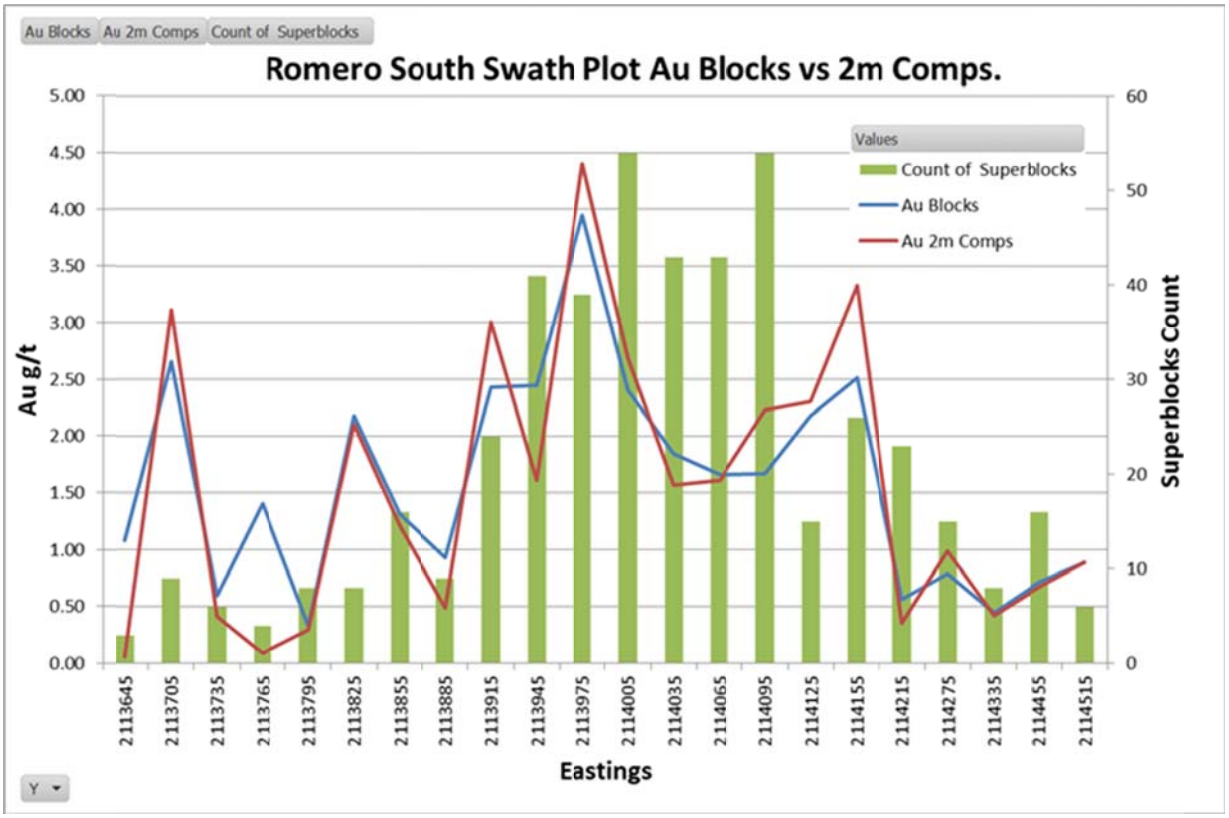
The gold swath plots for Romero and Romero South are shown in Figure 14.14 and Figure 14.15. Reasonable agreement with minor smoothing of extremes can be seen.

Figure 14.14: Romero Trend Analysis Chart for Gold



Source: Micon 2014

Figure 14.15: Romero South Trend Analysis Chart for Gold



Source: Micon 2014

15 Mineral Reserve Estimates

Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the GoldQuest project.

16 Mining Methods

16.1 Introduction

The potentially mineable resources at the Romero deposit will be extracted using a combination of longhole stoping (LH) and drift and fill (DF) underground mining methods with paste backfill to reach a target production rate of 2,500 tonnes per day (tpd or t/d) over a mine life of ten years. LH stoping will account for about 30% of total production and the remaining 70% will come from DF. The Romero deposit will be accessed from surface via a spiral decline and all mineralized material and waste rock will be trucked out of the mine via this decline. Three ventilation raises will be required in addition to the spiral decline to circulate the required amount of air through the Romero underground workings.

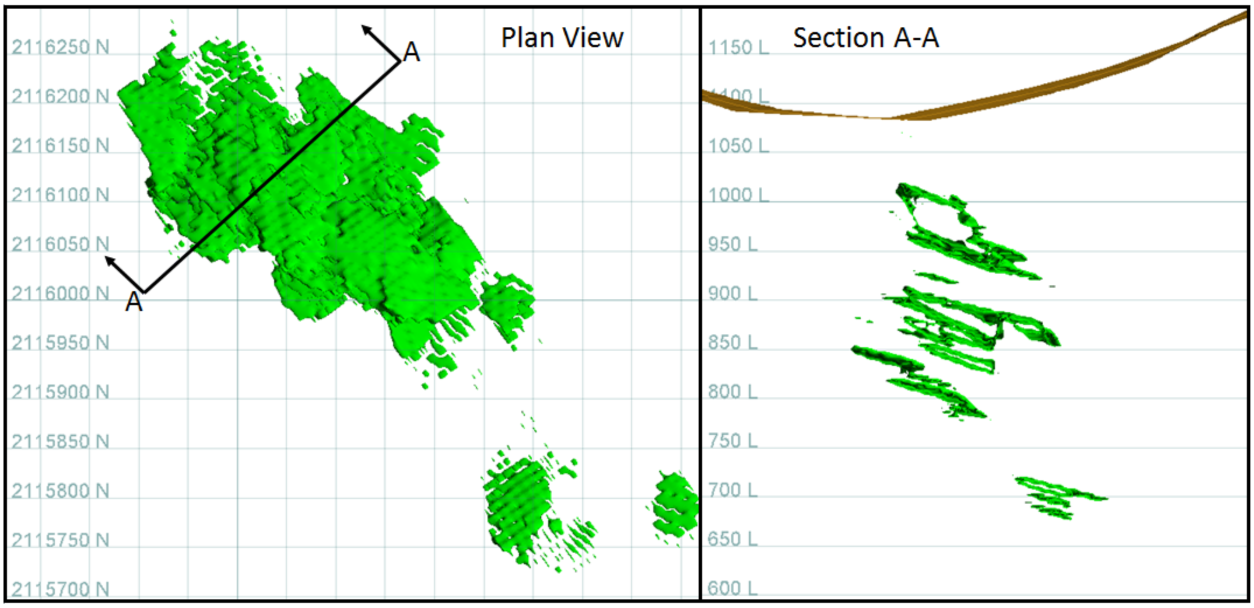
Indicated and Inferred mineral resources were included in the mine design and schedule optimization process. Inferred resources are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the Inferred resources would be upgraded to a higher resource category. The PEA LOM plan contains approximately 14% Inferred resources. No mineral resources have been classified in the Measured category.

16.2 Deposit Characteristics

High grade mineralization at the Romero deposit takes the shape of vertically stacked sub-parallel irregular lenses which generally dip to the northeast at an average angle of 20°. Each lens ranges in thickness from 10 m to 40 m thick in the middle and generally tapers to zero width at the edges, but the continuity of the lenses in all directions is inconsistent. The spacing between lenses is also inconsistent but ranges from zero to 50 m. Generally, lower grade mineralization surrounds the higher grade lenses.

The strike length of the main portion of the potentially economic resource is 430 m with two smaller pods of high grade mineralization approximately 200 m along strike to the southeast of the main portion. The deepest mining level is 420 m below surface (680 mL) and the highest mining level is 85 m below surface (1015 mL), meaning the total vertical extent of the potentially economic resource is 335 m. Perpendicular to strike, the potentially mineable resource is about 170 m wide.

Figure 16.1: Grade Shell at $70 NSR



Source: JDS 2015

16.3 Geotechnical Analysis and Recommendations

Geotechnical information for the project is limited. According to Micon, rock quality designation (RQD) values had been assigned for much of the exploration drill core; however, no geotechnical drilling or testing has been carried out. According to Micon's examination of exploration drill core, the ground conditions in zones of high grade mineralization are good, whereas the overlying material in the hangingwall is considerably less competent.

More geotechnical data will be needed at the pre-feasibility stage for the mine design. Geotechnical drilling and testing should be conducted to collect the necessary data. Oriented drilling and logging of the core should target the planned mining areas to establish geotechnical design criteria for the hangingwall, the mineralized zone and the footwall. Major geological fault structures should also be tested.

16.4 Hydrogeology Analysis and Recommendations

No hydrological data is currently available. According to the exploration drill logs, no major groundwater inflows were observed. Hydrological data should be collected as part of the geotechnical drill program, and a groundwater model should be established based on this information.

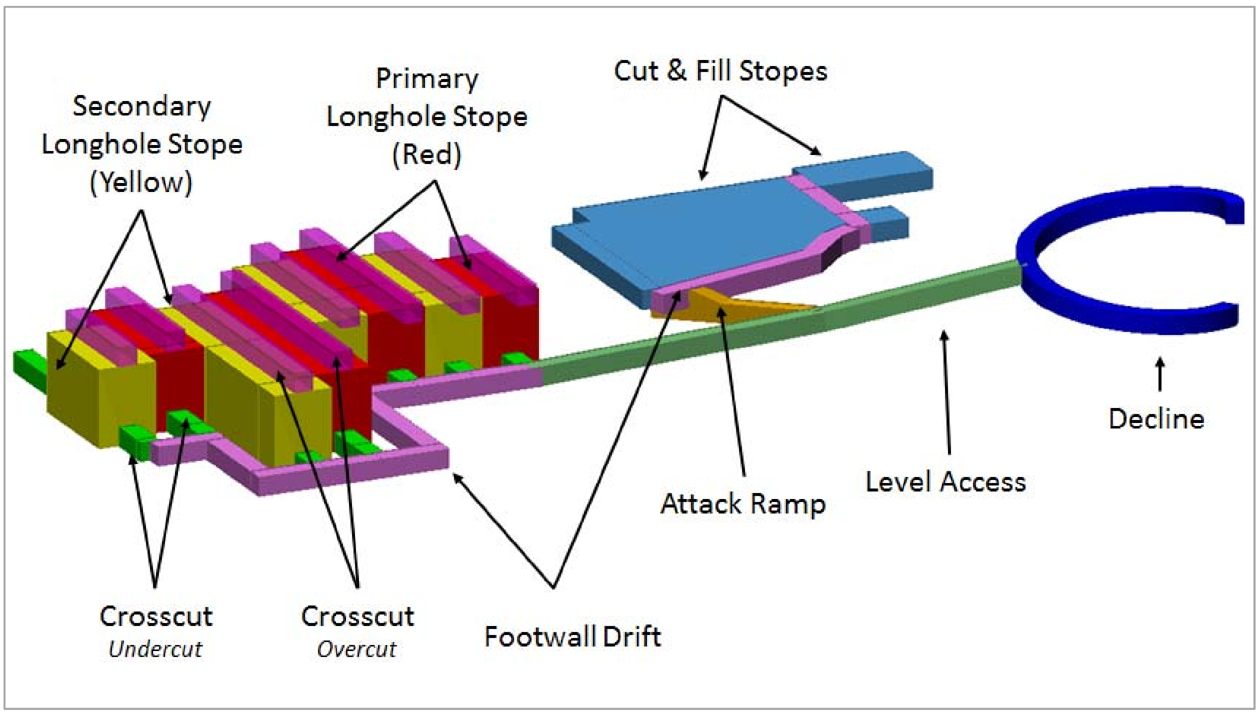
Low groundwater inflow rates of 350 m^3/day were assumed in this PEA.

16.5 Mining Methods

JDS selected LH stoping as the mining method for the areas which are thickest and most continuous and DF for the narrower and more irregular areas. Both methods will use cemented paste backfill. Areas amenable to LH stoping were selected first and DF was selected for the remainder. LH stoping accounts for about 30% of the total mined resources in the mine plan while cut and fill accounts for the remaining 70%. Mining of the Romero deposit has been designed and scheduled such that multiple mining blocks will be in production simultaneously, thereby allowing the mine to reach target production before the spiral decline has reached its ultimate depth. Generally, though, each individual mining block will be mined in an overhand (bottom-up) manner.

The total average mining dilution calculated for the overall deposit was 9.5%.

Figure 16.2: Perspective View of a Typical Mining Level (980 Level)



Source: JDS 2015

16.5.1 Drift and Fill Mining

The DF method is typically used for mineralization that is irregular in shape, has a shallow dip, and has relatively high value mineable material. It is a preferred mining method in poor or uncertain ground conditions, since the span of the openings can be kept small and backfill is used for support of the hangingwall.

DF is a development-intensive mining method, in which a footwall drive is established across the entire length of the mining zone and production drifts are mined perpendicular to the footwall drive in a primary-secondary sequence. After the primary drift is mined, it is backfilled with cemented fill, so that the secondary drift may be mined alongside. The walls of the secondary drift expose the cemented backfill of the primary drifts.

The minimum drift dimensions for DF mining at Romero were assumed to be 5 m width by 5 m high. Mining dilution from overbreak was assumed to be 0.25 m on each wall plus 0.25 m in the floor for a total average mining dilution of 7.8% dilution by mass in DF stopes. The grade carried in the dilution material for DF mining was calculated to be 0.62 g/t Au, 1.34 g/t Ag, and 0.19% Cu for both Indicated and Inferred material.

16.5.2 Transverse Longhole Stoping

LH stoping is a semi-selective and productive underground mining method, and well suited for steeply dipping or massive deposits with good continuity. It is typically one of the most productive and lower-cost mining methods applied across many different styles of mineralization. Transverse LH stoping will be the mining method used at Romero for the thickest, most continuous portions of the potentially mineable resource. This method can achieve high production rates as multiple stopes can be in operation at once on a level.

The LH stopes at Romero were designed to be 15 m wide, 20 m tall, and no longer than 30 m (average length 25 m). The extraction sequence will generally be overhand (bottom-up) and will follow a primary-secondary pattern. Each stope is accessed by a 5 m by 5 m crosscut above and below. In cases where the stoping block accessed by a set of crosscuts is longer than 30 m and has been divided into multiple stopes, the stopes will be extracted and filled with paste backfill in a retreating manner.

The total mining dilution for longhole stopes was calculated to be 10.8% by mass for primary and secondary stopes combined. The underlying assumptions were 0.50 m overbreak per stope for all primary stope walls, 2.5 m overbreak per stope for all secondary LH stope walls, and 0.25 m overbreak per stope for floors. Crosscuts were diluted with 0.5 m total wall overbreak and 0.25 m floor overbreak. The average grade contained in the LH stope dilution was calculated to be 1.53 g/t Au, 2.12 g/t Ag, and 0.39% Cu for both Indicated and Inferred material.

16.6 Mine Design

16.6.1 Optimization

Mine planning for the Romero project was conducted by JDS using Maptek Vulcan 3D and Minemax iGantt software.

Mine design was carried out based on an estimated net smelter return (NSR) value for each block in the resource model. The NSR value was calculated based on estimated selling prices for gold, copper, and silver, estimated metallurgical recoveries, selling costs, and mining dilution (see Table 16.1).

Table 16.1: NSR Calculation Inputs

Parameter	Unit	Input
Prices and Royalty		
Copper price	US$/lb Cu	2.70
Gold price	US$/oz Au	1,200
Silver price	US$/oz Ag	17.00
NSR Royalty	% NSR	1.25
Mining Parameters		
External Mining Dilution - Cut & Fill	%	7.3
External Mining Dilution - Longhole	%	16.1
Mining Recovery	%	95
Recovery to Concentrates		
Copper Concentrate		
Copper	Avg. %	96.5
Gold	Avg. %	75.2
Silver	Avg. %	53.9
Concentrate Grades and Moisture Content		
Copper concentrate grade	%	20.0
Copper concentrate moisture content	%	8.0
Percent Payable Metal		
Copper Concentrate		
Copper	%	96.5
Gold	%	90.0
Silver	%	95.0
Minimum Unit Deductions		
Copper Concentrate		
Copper	%	1.00
Gold	g/t	0
Silver	g/t	0
Smelting & Refining Costs		
Copper concentrate Treatment Cost	US$/DMT	90.00
Copper refining charge	US$/lb Cu	0.10
Gold refining charge	US$/oz. Au	6.00
Silver refining charge	US$/oz. Ag	0.96
Transport cost	US$/WMT	100.00
Deleterious elements charge	US$/DMT	-
Insurance Costs		
Insurance	US$/$1K value	0.495

Source: JDS 2015

NSR cutoff grade was calculated based on estimated mining, processing, and general and administrative (G&A) operating costs. Vulcan software was used to create NSR grade shells which served as the basis for mine design. iGantt was used to optimize the mine production schedule by maximizing the NPV, subject to constraints including maximum lateral development rates, maximum production rates, maximum backfill rates, minimum backfill cure times, and extraction sequence.

The cutoff grades used to generate grade shells were NSR values of $50/tonne for LH stoping areas and $70/tonne for DF mining areas. These grade shells were then used as the basis for stope design. First, a $50/tonne grade shell was generated using Vulcan and LH stopes designed in the thickest, most continuous zones. Next, a $70/tonne grade shell was generated using Vulcan, the LH stopes subtracted, and the remainder used as the design basis for DF stopes.

16.6.1.1 Access

The Romero deposit will be accessed via a spiral decline. A decline was selected over a shaft to provide early access to the mineralized zones and to reduce initial capital. The decline will be used to haul mineralized material and waste and as general access. The decline will also be used as an exhaust airway.

The decline will descend to a final depth of approximately 415 m below surface (685 masl) and will break through on surface directly above the Romero deposit and close to the proposed mill location to minimize mine to mill haulage.

The size of the decline was selected according to required clearances for the chosen mobile equipment and required ventilation during development and production. It was determined that a 4.5 m wide by 5.0 m high profile would be suitable for a 30 t haul truck. The decline will be driven at a -15% gradient. Level access crosscuts and attack ramps are planned to be developed off the decline at a 4.5 m by 5.0 m profile.

Level access crosscuts are designed to be located every 25 m vertically along the spiral decline to provide access to the potentially mineable resource. Attack ramps would provide the access from the access drifts directly to each mining level and would have a maximum gradient of +/- 15 %. Once a given level has been completely mined and backfilled, the back of the attack ramp access is planned to be slashed down and a ramp would be constructed with the slashed rock to access the next cut above.

16.7 Mine Services

16.7.1 Mine Ventilation

The ventilation system for the Romero operation has been designed to dilute and remove dust, diesel emissions and blast fumes. The ventilation network was modelled using Ventsim software.

Three ventilation raises are planned to be developed for air circulation. Two raises will be developed near to and in parallel with the decline by drop raising on a 4 m by 4 m profile. Of these raises, one will be used for fresh air and the other for exhaust air. Lateral ventilation drifts at 4 m by 4 m profile will be required to follow the decline and connect the ventilation circuits to the decline and level access. A third raise, 250 m long, will be driven by a contract raisebore crew about 260 m southeast of the spiral decline to ventilate the southern areas of the mine.

Airflow requirements were based on expected diesel emissions of the underground mining fleet required at peak mine production. The power rating of each piece of equipment was determined, and the utilization factors representing the equipment in use at any time, were applied to estimate the amount of air required. Industry standard recommends 0.06 m^3/s for each kW of diesel powered equipment operating at the work site. Using a 10% safety factor this gave a final airflow requirement of 205 m^3/s.

The mine ventilation circuit designed consists of a "push" system with one intake raise and two exhaust raises in different areas of the mine; additionally, air will exhaust out the mine ramp. The return air raises (RAR) will be required to keep air velocity on the ramp at or below 6 m/s. The fresh air raise (FAR) will also act as a means of secondary egress.

Auxiliary fans will be used to ventilate the advancing development and active production levels. Fresh air will be sourced from the FAR and distributed using the auxiliary fans through ventilation ducting to the active mine areas.

16.7.2 Water Supply

Service water will be required mainly for drilling, dust suppression and washing of development faces. Water will be supplied from a 55,000 liter service water tank located close to the portal and will be gravity fed to the underground work areas via 100 mm diameter pipelines. Pressure reduction valves will be installed along the decline as required. The service water tank will be refilled with underground mine water or externally sourced water.

16.7.3 Dewatering

To control groundwater inflows a network of 43kW submersible pumps will be installed at specific dewatering sumps with staged pumping to the surface. Some of this water will be re-used as mine service water.

16.7.4 Electrical Distribution

Electrical power will be supplied from the site substation to the west of the process plant distributed to the portal site via feeder cable. If the main power line goes down a diesel powered standby generator will be used to provide emergency power for mine dewatering.

High voltage 4.16kV cable will enter the mine via the decline and will be distributed to electrical substations located close to the main mining blocks. From the substations step down transformers will provide 600V to the various electrical equipment and junction boxes.

Estimated annual power consumption from the underground mine is summarized in

Table 16.2, major electrical power consumption in the mine arises from the following:

- Main and auxiliary ventilation;
- Mine dewatering pumps;
- Underground mobile equipment;
- Portable air compressors; and
- Refuge stations.

Table 16.2: Projected Annual Electrical Power Consumption

Years	Total MWh	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10
MWh/year	88,744	805	8,662	11,049	8,353	7,395	8,605	9,460	10,474	11,126	8,487	4,327

Source: JDS 2015

16.8 Unit Operations

16.8.1 Drilling

Development headings are planned to be driven with electro-hydraulic two-boom jumbos. Blastholes with 45 mm diameter would be drilled to a depth of 4.2 m. The advance per round is assumed to be 4.0 m. It is envisioned that one jumbo can drill between two to three rounds per shift.

Production drilling for the longhole stopes would be performed by longhole drills. Blastholes with 64 mm diameter would be drilled in a fan pattern from the overcut to the undercut.

16.8.2 Blasting

Development rounds would be charged by an ANFO loader. Lifter holes would be loaded with packaged emulsion. Blasting is planned to be initiated by non-electric (NONEL) detonators.

For longhole production blasting, ANFO would be used together with NONEL detonators and Pentex boosters.

16.8.3 Ground Support

After mucking and scaling is complete, ground support would be installed by a mechanized scissor bolter. Typical ground support in access development is planned to consist of 2.4 m long Swellex bolts in the back and 1.8 m Swellex bolts in the walls at a 1.2 m by 1.5 m pattern with 6-gauge welded wire mesh installed to the floor. In intersections, 4.6 m Super Swellex bolts would be used for deep ground support.

It was assumed that 5% of the development will be in poor ground conditions, which would require shotcreting. A shotcrete machine would be used to apply shotcrete at 50 mm thickness.

16.8.4 Mucking

Blasted material from development headings would be mucked with a 4.6 m^3 LHD directly to a haul truck or to a remuck bay. Broken material from longhole stopes would be mucked by remote control LHD.

16.8.5 Hauling/Hoisting

30 t haul trucks would drive on the decline to surface, where they would dump the material on mineralized material or waste stockpiles in close proximity to the portal.

Haulage profiles for all production levels were generated to calculate equipment hours for the fleet.

16.8.6 Backfill

The selected mining methods require the placement of backfill for full extraction of the mineralized zones. Primary stopes require cemented backfill to provide stability to exposed backfill walls when mining the secondary stopes. Secondary stopes and attack ramps can be backfilled with lower or no cement content.

Paste backfill would be used to minimize the storage requirements for process plant tailings on surface. The paste would be mixed at a paste plant and pumped through pipelines underground to the stopes. A cement content of 4% was assumed for cemented paste fill of primary stopes. Further test work will be required to determine the optimum cement content, curing time and achievable backfill strength.

Underground development waste may be used for uncemented backfill in attack ramps and secondary stopes to minimize waste haulage to surface.

16.9 Mine Equipment

The mobile equipment fleet to support the mining operation is summarized in Table 16.3.

Table 16.3: Mobile Equipment Fleet (average number of units)

Underground Equipment	Year -1	Year 1	Year 5
Jumbo 2 Boom	1	3	2
Longhole Drill	1	1	1
Diamond Drill	0	1	0
Bolter	1	4	3
30 t Haul Truck	1	5	4
4.6 cu. m LHD	1	3	3
Scissor Lift	1	2	2
ANFO Loader	1	2	1
Boom Truck	1	1	1
Fuel/Lube Truck	1	1	1
Toyota	4	4	4
Personnel Carrier	1	1	1
Shotcrete Machine	1	1	1
Forklift/Telehandler	1	1	1
Portable Welder	1	1	1
Grader	1	1	1
Front End Loader (Surface)	1	1	1

Source: JDS 2015

16.10 Mine Personnel

The underground mine would operate on two 11-hour shifts (day shift / night shift), 365 days per year with four crews on rotation. Two crews would be on site at any time with the other crews off site on break. Both hourly mining and maintenance personnel and salaried supervisors and technical staff would work on the same 2 x 2 rotation.

Hourly personnel were estimated based on development and production rates, operation productivities and maintenance requirements.

Underground mining personnel requirements are summarized in Table 16.4.

Table 16.4: Underground Mine Operations Personnel

Position	Avg. Quantity	Hourly/Salary
Mining Operations		
Mine Supervisor/Shift Boss	4	Salary
Coverage Miner	2	Hourly
Production Drill Operator	2	Hourly
Jumbo Operator	6	Hourly
Ground Support	18	Hourly
Development Service	11	Hourly
Blaster	3	Hourly
LHD Operator	10	Hourly
Truck Driver	16	Hourly
Backfill/Construction	6	Hourly
Nipper	3	Hourly
Backfill Plant Operator	4	Hourly
Maintenance		
Heavy Duty Mechanic	14	Hourly
Electrician	6	Hourly
Technical Services		
Chief Mine Engineer	1	Salary
Senior Mine Engineer	2	Salary
Vent/Project Engineer	2	Salary
Geotechnical Engineer	1	Salary
Surveyor/Mine Technician	4	Salary
Chief Geologist	1	Salary
Production Geologist	2	Salary
Diamond Driller	4	Salary
Diamond Drill Geologist	1	Salary
Total Underground	**123**	

Source: JDS 2015

16.11 Mine Production Schedule

Mine scheduling for the Romero project was conducted by JDS using Minemax iGantt software. The scheduler seeks to optimize the NPV of a 2,500 tonne per day operation subject to constraints of development rates, production rates, and backfill rates.

Underground production was considered to have started as soon as first mineralization is mined. Mining blocks with higher grade (NSR $/t) mineralization were targeted in the early stages of the mine life to optimize project economics.

Annual mine production statistics are provided in Table 16.5.

Table 16.5: Annual Mineralized Material, Waste and Backfill Schedule

Year	TOTAL	Year										
		-1	1	2	3	4	5	6	7	8	9	10
Mineralized Material Mined												
Tonnes (kt)	7,737		614	913	913	913	913	913	913	913	540	196
Avg Au Grade (g/t)	4.02		4.21	3.75	5.33	5.21	4.63	3.97	3.20	2.97	3.06	1.80
Avg Ag Grade (g/t)	4.25		3.12	3.36	4.81	4.66	4.45	4.89	3.73	4.10	4.98	4.56
Avg Cu Grade (%t)	0.81		0.70	0.87	0.82	0.91	0.86	0.73	0.76	0.74	0.90	0.87
Ounces Au (koz)	1,000		83	110	156	153	136	116	94	87	53	11
Ounces Ag (koz)	1,056		62	99	141	137	130	143	109	120	86	29
Tonnes Cu (kt)	63		4	8	8	8	8	7	7	7	5	2
Waste Mined												
Tonnes (kt)	903	40	413	98	15	10	32	38	84	56	83	35
Backfill Placed												
Tonnes (kt)	4,158		146	511	521	511	307	350	548	679	375	210

Source: JDS 2015

The development schedule was based on estimated cycle times for jumbo development.

All waste development during pre-production is shown as capital development.

During the production phase, the decline, ventilation drifts and raises are considered sustaining capital development, but crosscuts and drifting on the levels were included in operating costs.

Annual development meters are summarized below in Table 16.4.

Table 16.4: Annual Development Schedule

Year	TOTAL	Year										
		-1	1	2	3	4	5	6	7	8	9	10
Capital Lateral	10,941	591	5,359	1,048	107	42	293	456	1,045	640	981	379
Capital Vertical	1,186	104	721	248							113	
Operating Lateral	2,394		753	323	112	95	184	133	247	199	212	137

Source: JDS 2015

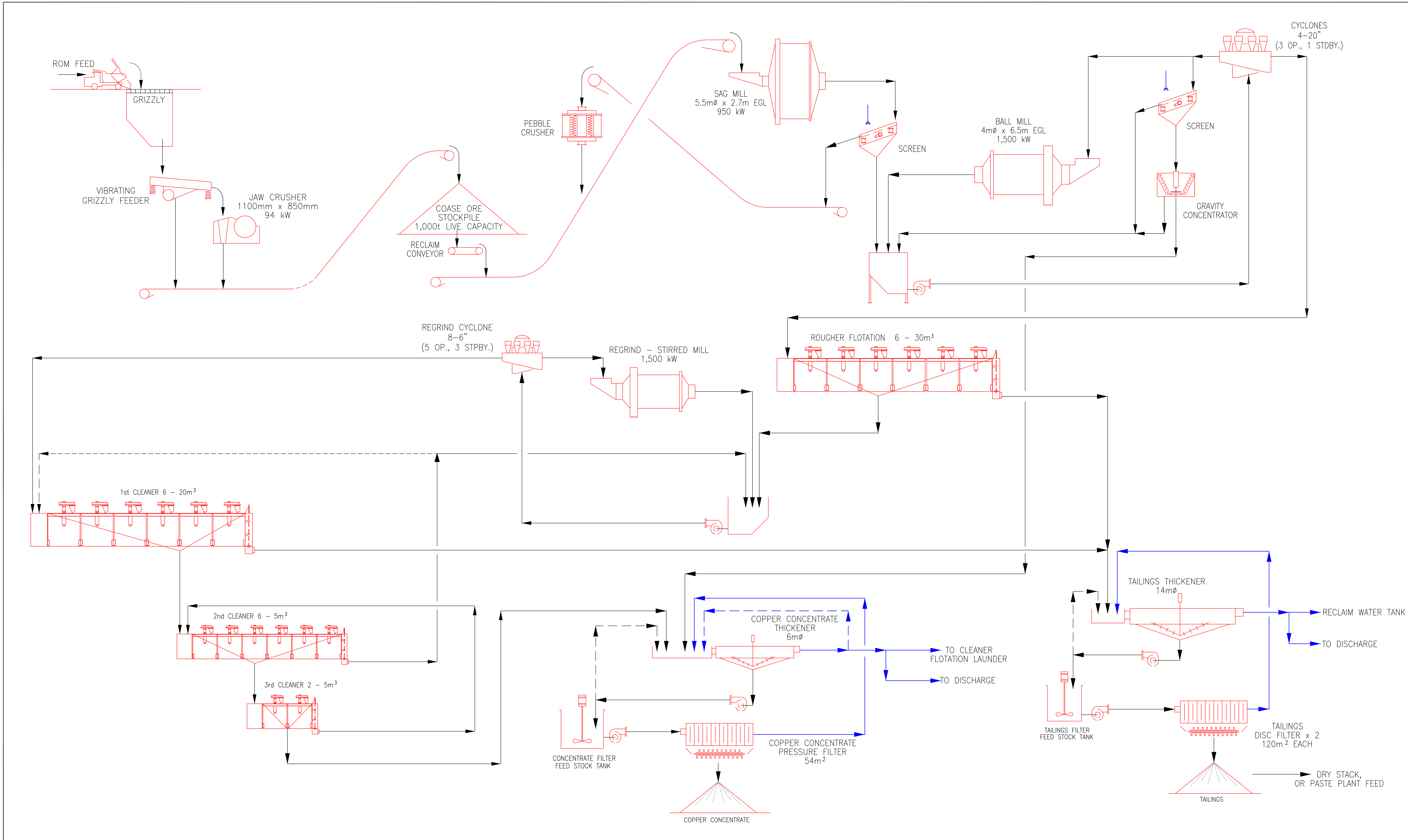
17 Process Description/Recovery Methods

The process design criteria and flowsheets have been developed based on the metallurgical testwork results from historical and current testwork programs as described in Section 13 using industrial design factors as noted. The testwork has shown that Romero mineralization can be treated using conventional mineral processing techniques for the recovery of copper concentrate.

17.1 Introduction

Figure 17.1 presents a conceptual flowsheet of the processing plant for the Romero Project.

A simplified description of the mineralization processing at the mine site is summarized in this section with details following in the descriptions of each unit operation.



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	
REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	
REFERENCE DRAWING	REF. DWG No.						

SCALE:	None	DATE:
DESIGNED:	J.D.S.	Jan 23 15
DRAWN:	B. Wong	Jan 23 15
CHECKED:		
APPROVED:		

GOLDQUEST

JDS Energy & Mining Inc.

TITLE
GOLDQUEST MINING CORP.
ROMERO PROJECT SIMPLIFIED PROCESS FLOWSHEET

PROJECT No. | DRAWING No. 100-PF-001 | REV No.

Mineralized material or plant feed will be dumped by truck into a jaw crusher's feed hopper and vibrating feeder. The jaw crusher will reduce the run-off-mine material, ROM, to a product size, $P_{80,}$ of 150 mm. The crushed feed will be conveyed from the crushing plant located near the portal to the coarse feed stockpile. The material will be reclaimed by one of two apron feeders under the stockpile and conveyed to the SAG mill in the process plant.

The SAG mill discharge will feed the SAG vibrating screen. The screen oversize material will circultate back to the SAG mill feed chute by a series of conveyors and a pebble crusher. Screen undersize material will combine with the ball mill discharge and gravity circuit tailings before being pumped to the grinding cyclones. The coarser fraction in the cyclone underflow feeds the ball mill for further grinding and approximately 25% of the underflow will feed the gravity circuit. The cyclone overflow will gravitate to the rougher flotation cells at a P_{80} target of 75 µm.

The flotation circuits have been sized based on test work recently completed at ALS Metallurgical in Kamloops, B.C (ALS). The laboratory retention time required for effective rougher and cleaner flotation has been scaled up by 2.5 and 4 times respectively. PH values of 10 and 11.5 will be maintained in the copper rougher flotation circuit and cleaner flotation circuits, respectively.

Copper rougher flotation concentrate will be pumped to the copper regrind circuit. The slurry will be pumped to a cluster of cyclones and cyclone underflow will undergo further grinding in the copper stirred regrind mill to a P_{80} particle size of 23 µm. The cyclone overflow will flow by gravity for further processing in the cleaner flotation circuit. There will be three stages of copper flotation cleaning.

The first cleaner concentrate will report to the second cleaner cells and the tailings will be pumped to the final tailings thickener. Second cleaner concentrate will advance to the third cleaner flotation cells and the tailings will be either sent back to regrind or fed back to the head of the first cleaner circuit. Third cleaner tailings will be sent back to the second cleaner flotation circuit. Third cleaner concentrate and gravity concentrate will form the final copper concentrate.

Tailings will be dewatered and filtered for use as dry stack tailings or pumped to the paste backfill plant.

Dewatered and filtered copper concentrate will be loaded into trucks for transport to the port facility and shipped to refineries for further processing.

Reagents will be shipped to site in the form of standard drums, totes and bulk bags.

17.2 Design Criteria

The design criteria and mass balance for the Romero Project have been developed using results from testwork, industry standards, Vendor recommendations and the average feed head grades and tonnage from the mine plan.. The results are summarized in Table 17.1.

Table 17.1: Design Criteria

Description	Units	Nominal/Design
Operating Data		
Daily feed throughput	t/d	2,500
Annual plant throughput	t/a	912,500
Ore Characteristics		
Ore Solids Density	SG	2.94
JK Drop-Weight Parameters - A		68.3
b		0.5
t_a		0.3
Bond ball mill work index, Wi	kWh/t	15.3
Bond abrasion index, Ai	g	0.2
Head Grade (Average LOM)	%Cu	0.81
	%Au	4.02
	%Ag	4.25
Production Rates		
Overall Crusher Availability	%	65
Overall Plant Availability	%	90
Final Copper Concentrate		
Concentrate mass pull	%	3.92
Concentrate production, daily	dry tpd	98 nominal, 118 Design
Concentrate grade	% Cu	20
Recovery	% Cu	96.8
	% Au	75.0
	% Ag	49.8
Tailings		
Methodology		Dry Stack and/or Paste Backfill

Source: JDS 2015

Further metallurgical test work will be completed in the next stage of engineering to provide confirmatory and/or additional information as discussed in Section 13.

The metallurgical plant is designed to process 2,500 dry tonnes with a plant availability of 90%. Annual throughput is targeted at 912,500 dry tonnes.

17.3 Process Plant Description

17.3.1 Primary Crushing, Feed Storage and Reclaim

The crushing circuit consists of a stationary grizzly, rock breaker, truck dump pocket, vibrating feeder, jaw crusher, and belt feeder. A vibrating feeder with VFD drive will draw feed out of the dump pocket and provide a constant feed of material to the jaw crusher. Crushed material will discharge coarse feed onto a 924 mm (36”) overland conveyor to the 1,000 t live coarse feed stockpile.

Two apron feeders with variable speed drives will reclaim feed from the coarse feed stockpile and feed the SAG mill feed conveyor. Each apron feeder will be capable of sending full tonnage to the mill. A weightometer on the SAG mill feed conveyor will control the speed of the apron feeders. Crushing design criteria is shown in Table 17.2.

Table 17.2: Crushing Design Citeria

Description	Units	Nominal and Design
Crushing and Coarse Feed Stockpile		
Maximum lump size (stationary grizzly opening)	mm	600
Crusher Type	-	Jaw
Crusher Power Installed	kW	94
Estimated Product, P80	mm	150
Coarse Feed Storage (Live)	-	Stockpile, 1,000 tonnes

Source: JDS 2015

17.3.2 Grinding

Reclaimed material will feed a 950 kW, 2.7 m diameter by 5.5 m long SAG mill driven by a variable speed motor, which will enable the SAG mill to vary a power draw for circuit optimization under varying feed material conditions. SAG mill discharge will feed a 1.2 m x 3.7 m vibrating screen with a deck aperture of 12.5 mm. The screen undersize will be pumped to the cyclone feed pumpbox; the screen oversize will recycle to the SAG mill feed conveyor via pebble recycle conveyors.

Grinding media will be added to the SAG mill feed conveyor through an automated ball feeder system that consists of a ball bin, a feeder system, and a counter.

Table 17.3: Grinding Design Criteria

Description	Units	Nominal and Design
Grinding		
Primary Grinding		
Primary Grinding Mill Type		SAG
Number of Mills		1
Mill Length, EGL	m	2.7
Mill Diameter	m	5.5
Mill Power Installed	kW	950
Estimated Product Size, P80	microns	900
Screening		
SAG Discharge Screen		Vibrating
Size	m	1.2 x 3.7
Aperture Size	mm	12.5
Secondary Grinding		
Secondary Grinding Mill Type		Ball Mill
Number of Mills		1
Mill Diameter	m	4.0
	m	6.5
Mill Power Installed	kW	1,500
Target Product Size, P80	microns	75

Source: JDS 2015

SAG discharge screen undersize material will be combined in the cyclone feed pumpbox with the ball mill discharge and gravity circuit tailings. The slurry will be pumped to a cyclone cluster for size classification. The underflow from the cyclopac will be fed to the ball mill and gravity circuit while the overflow will be sent to copper rougher flotation. The target particle size P_{80} of the cyclopac overflow is 75 µm. The gravity concentrate will report directly to the copper concentrate thickener.

Process water will be added directly to the SAG mill feed chute to maintain a target slurry density in the SAG mill. Process water addition to the cyclone feed pump box will be controlled to maintain pump box level and/or cyclone feed density.

17.3.3 Copper Processing

This section describes the copper processing circuit; this circuit includes flotation, regrinding, and concentrate dewatering and handling.

17.3.3.1 Rougher Flotation

Table 17.4: Flotation Circuit

Description	Units	Nominal	Design
Roughers			
Number of cells	-	6	
Cell Volume	m^3	30	
Concentrate Mass Pull	%	33.9	

Source: JDS 2015

Slurry from the cyclone overflow will gravitate to the rougher flotation circuit which consists of one bank of six 30 m^3 cells. The cells will use a combination of reagents (PAX, MIBC and Lime), agitation, and air to recover the copper sulphides, gold and silver for further processing.

Rougher concentrate froth will be collected in a common launder which feeds a standpipe. Slurry collected in the standpipe will be pumped to the regrind circuit for further mineral liberation. The copper content of the rougher feed, rougher concentrate, cleaner concentrate, and copper rougher tailings will be determined by inline samples that report to the on-stream analyzer for metallurgical analysis. Copper rougher tailings will report to the tailings thickener.

17.3.3.2 Regrind Circuit

Rougher concentrate will be pumped to the regrind cyclone feed pump box. Cyclone underflow will feed the stirred mill and the cyclone overflow flow will gravitate to the first cleaner circuit.

Table 17.5: Regrind Circuit Summary

Description	Units	Nominal and Design
Regrind		
Mill type		Stirred
Number of Mills		1
Power Installed	kW	1500
Target Product Size, P80	microns	23

Source: JDS 2015

17.3.3.3 Cleaner Flotation

The cleaner circuit will comprise of six 20 m^3 first cleaners, six 5 m^3 second cleaners and two 5 m^3 third cleaners. Slurry from the regrind cyclone overflow and stirred mill discharge will feed the first cleaner cells. The first cleaner concentrate will be collected in a common launder that flows by gravity to the first cleaner concentrate standpipe. The concentrate is pumped to the second cleaners. Concentrate from the second cleaner reports to the third cleaner cells. The third cleaner concentrate will be pumped to the concentrate thickener. Each staged cleaner flotation tailings will be pumped back to the previous stage of flotation, with the exception of the first cleaner tailing, which will be directed to the tailings thickener.

Table 17.6: Flotation Circuit Summary

Description	Units	Nominal and Design
1st Cleaners		
Number of cells		6
Cell Volume	m^3	20
Concentrate Mass Pull	%	11.9
2nd Cleaners		
Number of cells		6
Cell Volume	m^3	5
Concentrate Mass Pull	%	5
3rd Cleaners		
Number of cells		2
Cell Size	m^3	5
Concentrate Mass Pull	%	3.92

Source: JDS 2015

17.3.3.4 Concentrate Dewatering and Storage

The concentrate dewatering circuits will remove water from the concentrate slurry to permit shipping of the concentrate as damp filter cake. Testwork to confirm the equipment sizing will be completed in the next stage of engineering.

The thickening operation concentrates suspended solids by gravity settling. Flocculant will be added as a dilute solution to the thickener to agglomerate fine solid particles which assist the settling of the fine particles. Settled solids will be raked to the center discharge cone where the thickened slurry is withdrawn using one of two centrifugal pumps for transfer to the concentrate stock tank. The concentrate stock tank will provide eight hours of surge capacity between the six meter diameter concentrate thickener and concentrate pressure filter. The concentrate stock tank will be agitated to prevent sanding out of solids. A centrifugal slurry pump feeds thickened slurry from the concentrate stock tank to the concentrate filter. The thickener overflow solution is pumped to the cleaner circuit for process dilution water and as launder spray water.

A horizontal or vertical pressure filter is used for final concentrate dewatering. The pressure filter is a series of cloth covered plates on a rack. Concentrate is pumped into the chambers between the plates through channels and the plates are squeezed together using a hydraulic piston. The filter then undergoes a blow operation to push out any remaining free water; the piston releases and the plates separate allowing concentrate cake to freely fall down through bomb-bay doors to the floor below. The filter then undergoes a wash cycle to remove any remaining solids attached to the filter cloth.

Filtrate recovered from the squeezing process flows by gravity to the concentrate filtrate tank and is then pumped to the concentrate thickener.

Copper concentrate will be transferred by front-end loader to trucks for transport to the port of Haina near Santo Domingo. The concentrate will be transferred to sea containers at the port before being shipped to markets in Europe and Asia.

Table 17.7: Dewatering Design Summary

Description	Units	Nominal and Design
Dewatering and Filtration		
Thickener Type		High Rate
Thickener Underflow Density	%	60
Thickener Diameter	m	6*
Filter Type		Pressure, Horizontal or Vertical
Final Concentrate Moisture Content	%	8
Storage Method		Bulk

*sizing is preliminary based on Vendor recommendations.

Source: JDS 2015

17.3.4 Reagents Handling

Reagents consumed within the flotation circuits will be prepared and distributed by the reagent handling circuits. This facility will include mixing and storage for PAX, MIBC, Flocculant and lime. All reagent areas will be bermed with sump pumps to transfer spills to the final tailings pump box, with the exception of the Flocculant area, which will circulate any spills back to the storage tank. The reagents will be mixed, stored and then delivered through a supply loop with dosage controlled by flow meters and manual control valves. The storage tanks have been sized for a minimum of one day. The reagents will be delivered in powder form, with the exception of MIBC and antiscalant which will be delivered as solution.

The following table presents the estimated annual consumption for each reagent.

Table 17.8: Estimated Annual Reagent Consumption

Reagents	Use	Annual Consumption (tonnes)	Delivered Form
PAX	Sulphide Collector	100	850 kg box/bag
Lime	pH Modifier	1095	900 kg sack
MIBC	Frother	89	1 t tote
Flocculant	Fine Particle Agglomeration	37	25 kg bag
Antiscalant	Scale Inhibitor	9	220 kg drum

Source: JDS 2015

17.3.4.1 Collector; PAX (Potassium Amyl Xanthate)

PAX is is proposed to be used as a flotation reagent in the copper circuit. It promotes the flotation of selected sulphide particles contained within the mineralized feed. It will be delivered to the plant in the form of 850 kg bags of dry solid product. The bags will be lifted using the flotation aisle crane onto a hopper. The solids will discharge into an agitated mixing tank, which will blend the solids with fresh water to a solution of 10% by weight of the dissolved product. From the mixing tank, the solution will be discharged by gravity to a storage tank.

At the PAX storage tank outlet, a pump will transfer the solution to a supply loop. The supply loop will deliver PAX solution as required directly into the copper rougher flotation and cleaner circuits.

17.3.4.2 Frother – MIBC

The frother, MIBC, will be used as a flotation froth stabilizer. Frothers strengthen bubbles in flotation cells, enabling them to support the load of the activated mineral particles. The ready to use reagent will be transported to site in 1-tonne totes and metered directly to the flotation circuit.

17.3.4.3 Flocculant

Flocculant will be received in 25 kg bags and will be prepared by a vendor supplied mixing system. Bags of solid product will be loaded into a hopper from which the particles will be slowly fed into the system via an educator to generate a concentration of 0.25% into a mix tank. From the mix tank the flocculant will be transferred by gravity to a storage tank for delivery to the copper concentrate and final tailings thickeners.

17.3.4.4 Lime

Lime will be delivered in 900 kg sacks and mixed to a concentration of 20% solids for delivery to the flotation circuit for pH control..

17.3.4.5 Antiscalant

Antiscalant will be shipped to the plant in 220 kg drums. The antiscalant will be added at a rate of 10 g per tonne.

17.3.5 Tailings

Final tailings will be collected in a 14 m diameter thickener. Flocculant will be added to assist the settling of the fine particles. Settled solids will be withdrawn using one of two centrifugal pumps for transfer to the disc filters for deposition as dry stack tailings or paste backfill. The thickener overflow solution is pumped to the plant as make-up or spray water in the grinding and rougher flotation circuits. Excess water will be treated in the water treatment plant for discharge.

Table 17.9: Tailings Thickener Design Summary

Description	Units	Nominal and Design
Dewatering and Filtration		
Thickener Type		High Rate
Thickener Underflow Density	%	60
Thickener Diameter	m	14*
Filter Type		2 Disc Filters
Final Concentrate Moisture Content	%	12.5
Storage Method		bulk

*sizing is preliminary based on Vendor recommendations.

Source: JDS 2015

17.3.6 Plant and Instrumentation Air

17.3.6.1 Plant Air Compressors

The primary consumers of compressed air are: the primary crushing plant, and the filters. Minor users of compressed air are: dust collection/suppression, samplers, on-stream analyzer, SAG mill gear lubrication system, ball mill gear lubrication system and air hose stations located throughout the plant.

There are two compressed air systems. The systems will be located at the crushing plant and at the process plant. The plant and instrument air receivers will be located in the compressor room and the remaining receivers will be at their respective points of application. The air system will be set up such that if a power failure occurs, the instrument air loop will not flow back into any other loop.

17.3.6.2 Flotation Air

Two 150 kW blowers with a capacity of 60 - 140 Am^3/min at 40 kPag will provide air to the flotation circuits.

17.3.7 Assay Laboratory

The Assay Laboratory will consist of a sample preparation/metallurgical module and a wet laboratory module. The two containers will be housed in a heated pre-engineered building separated by a sample lay down area with roll up door access for truck drop off. The Laboratory will be performing test work for the underground mine workings, the mill, and the environmental group.

18 Project Infrastructure and Services

18.1 General

The project envisions construction of the following key infrastructure items:

- 13.3 km of new site access road;
- 4.5 km of upgraded access road;
- 3.5 km of new Dry Stacks Tailings Facility (DSTF) access road;
- Fresh/Fire water storage tank;
- Process plant building;
- Truck shop;
- Warehouse and maintenance building;
- Process building;
- Primary crusher building;
- Explosive storage;
- Dry stack tailings storage facility;
- Emergency backup power generator;
- Communications systems;
- Sewage treatment plant;
- Fresh water pumps;
- Process water tank;
- On-site substation; and
- Upgraded Sabaneta substation.

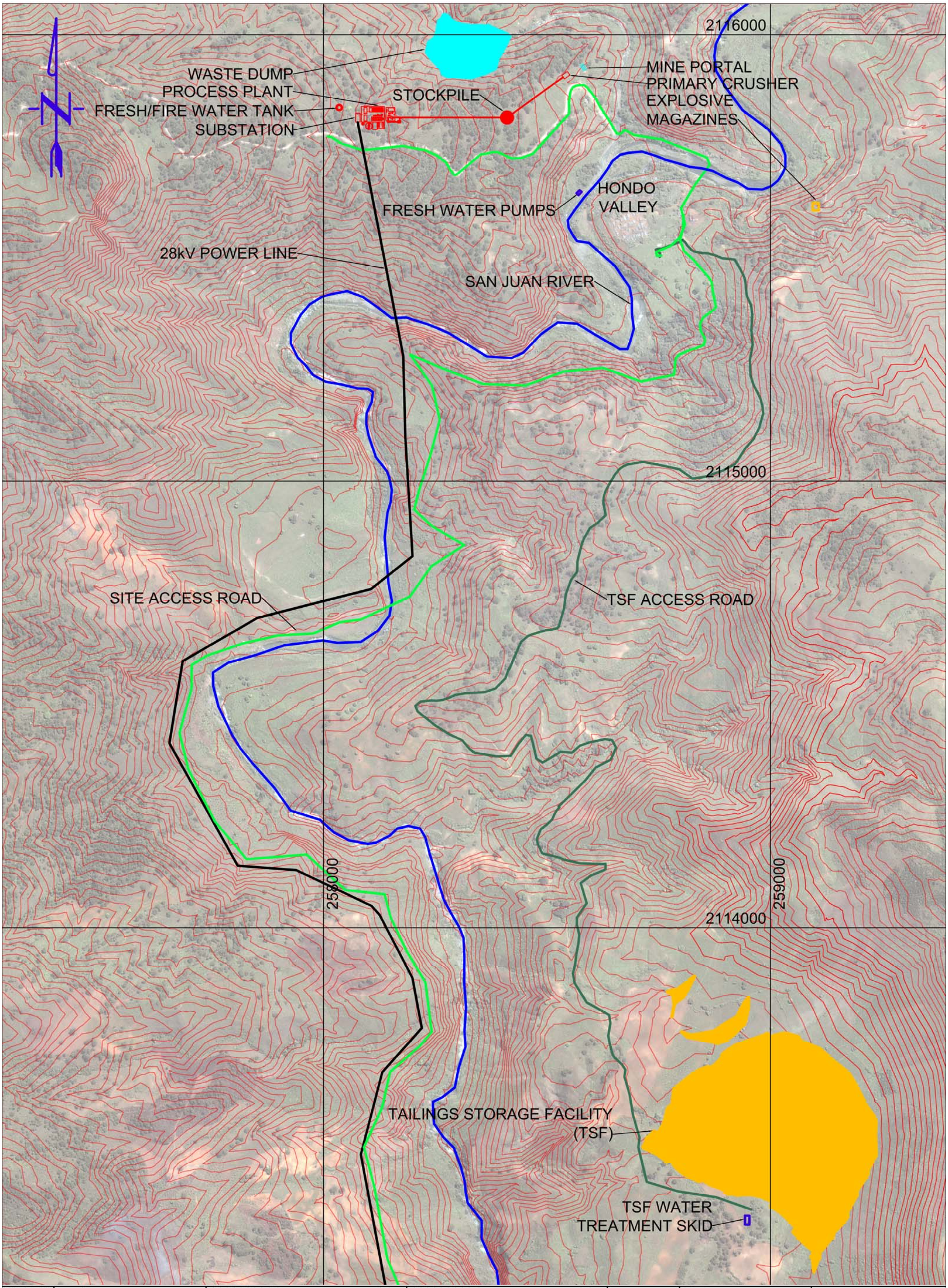
18.1.1 General Site Arrangement

The overall site arrangement is shown in Figure 18.1.

The site has been configured for optimum construction access and operational efficiency. Primary buildings have been located to allow easy access from the site access road and utilize existing topography to minimize bulk earthworks volumes. The primary crusher has been located as close as safely possible to the portal and at an elevation that facilitates mill feed conveying. Existing roads are upgraded and reused wherever possible. The new site access road follows the most cost effective geographic path.

18.2 Site Access Road

The existing 4.5 km section of site access road that starts at Sabaneta dam and runs parallel to the Sabaneta reservoir will receive significant upgrades to accommodate the increased traffic. Traveling north past the Sabaneta reservoir 13.3 km of new site access road will be constructed along the most cost effect and safe route towards Hondo Valle. The road will be widened to 6 m with new gravel, grading and compaction. It will be suitable for transportation of concentrate trucks, fuel trucks, mobilization of construction equipment and ongoing operational requirements.



JDS Energy & Mining Inc.

GOLDQUEST

TSXV: GQC

GOLDQUEST ROMERO PROJECT
SITE LAYOUT

SITE-001

100-10-001

A

18.3 TSF Access Road

Currently there is a narrow access road to the proposed tailings storage facility. This 3.5 km road will require significant upgrades and route changes to accommodate increased haul truck traffic.

18.4 Power Supply

The anticipated peak operational electrical demand will be 7 MW. Electrical power will be supplied from the nearby Sabaneta Dam. A 15km 28kV overhead power line and onsite substation will need to be constructed. The existing substation at Sabaneta Dam will also require upgrades to complete the tie-in.

There are several potential run of river (ROR) sites near the mine. Further data must be collected in future phases to determine the reliability of such a station to supply 7 MW of consistent power. However for the purposes of this PEA site power is assumed to be provided by Sabaneta Dam.

18.5 Construction Power

Standalone diesel generators will supply 1 MW of power during site construction. These will be rented to reduce project capital costs.

18.6 Camp

Existing facilities in Hondo Valle and surrounding villages will be used to house both the construction crews and operations employees. No temporary camp is required.

18.7 Process Plant

The process plant will be a 75 m x 35 m pre-engineered building with a covered roof and no wall cladding. The preliminary layout will utilize a narrow footprint in order to minimize cut/fill volumes in a challenging geographical area. It contains milling, flotation, regrind, concentrate thickening, filter presses, concentrate storage/loadout, reagent storage and electrical rooms.

18.8 Primary Crusher Building

The primary crusher will not require a covered building. Only the crusher operating control room will isolated from the environment. The remaining crusher structure will be constructed on a concrete pad.

18.9 Truck/Maintenance Facility

A 15 m x 15 m sprung building containing two bays will be used to service mining mobile equipment and process plant mechanical equipment. This will be attached to the process plant to minimize the overall site footprint and increase operational efficiency. Tire changing and large vehicle assembly will take place outdoors and utilize rough terrain mobile equipment.

18.10 Warehouse, Mine Dry and Administration Building

The warehouse will be located within close proximity to the truck shop and process plant. It will be constructed using seacans to reduce cost and expedite construction. It will have a footprint of 160 $m^{2.}$

The mine dry and administration buildings will be constructed using cost effective sprung buildings on concrete pads. These facilities will be located in Hondo Valle which is located approximately 650 m away from the process plant

18.11 Communications / IT

The process facilities and offices will include a wired and wireless computer network and satellite phone system.

A hand-held radio system will be used for voice-communication between personnel in the field.

18.12 First Aid / Emergency Services

A qualified nurse or first-aid attendant will be provided on-site. The first aid room will be located beside the administration building. The ambulance and fire truck will be parked at the ready outside the process plant.

Buildings will be equipped with smoke, carbon monoxide and heat detectors, overhead sprinklers, hydrants / hoses and appropriate chemical fire extinguishers.

18.13 Explosives Storage and Magazines

Explosives will be stored at a secured and monitored site located approximately 800 m from the main plant and populated, high traffic areas. All infrastructure items includes powder magazine and detonator magazine.

18.14 Bulk Fuel Storage and Delivery

Diesel fuel will be stored in a 75,000L dual wall fuel tank located near the truck shop. The tank will have an internal submersible pump capable of delivering 40 GPM to all site vehicles. Diesel will be delivered to mobile equipment by the fuel and lube truck. A small spill containment pad will be installed around the fueling station.

18.15 Fresh Water Supply

Fresh water will be pumped from the adjacent San Juan River system. The pump station will be located as close as possible to the process plant and access road. This water will be used to feed the process plant, potable water skid, firewater system and reagent preparation.

18.15.1 Fresh/Firewater Tank and System

The fresh/firewater tank will be dual purpose 12 m high by 12 m diameter tank serving as freshwater and firewater storage. Tank internal risers on all non-firewater suction lines will ensure a minimum volume of 470,000 L. This capacity will allow for approximately two hours of firefighting capability.

The buried firewater network will be pressurized by two pumps (one electric, one diesel stand-by). This network will be connected to all buildings requiring fire protection.

18.16 Potable Water and Sewage Treatment

Potable water and sewage treatment systems will be included with all the other facilities in Hondo Valle. These will be permanent fixtures for the duration of the mine life.

18.16.1 Water Treatment

Any surplus water generated by the mining operation will be treated and tested prior to being discharged back into the environment. A permanent water treatment skid will be installed at the DSTF.

18.17 Freight

Freight will be delivered to site on the access road and offloaded at the warehouse or other designated area.

18.18 Tailings Storage Facility

SRK Consulting (Canada) Inc. (SRK) completed PEA level design and costs for the Tailings Storage Facility (TSF) and their entire work is summarized in the their *"Romero Project: Preliminary Economic Assessment Tailings Storage Facility Design"* memo dated May 19, 2015. This section is a summary of the memorandum.

18.18.1 Design Criteria and Assumptions

Design criteria and assumptions adopted for completion of the alternatives assessment and preliminary TSF design are summarized as follows:

- Tailings production over the 10 year life of the mine is approximately 7.4 Mt, at a mill throughput of 2,500 t/d;
- Approximately 42% (3.1 Mt) of the tailings will be used as underground paste backfill, with the remaining 58% (4.3 Mt) being disposed of on surface;
- Tailing specific gravity of 2.82 was calculated based on mineralized material SG of 2.94 and mass pull of 4% (JDS 2015);

- For sizing of alternative tailings storage facilities considering alternate tailings deposition methods, conventional slurry tailings dry density was assumed to be 1.28 t/m^3, and the dry density of dewatered dry stack tailings was assumed to be 1.70 t/m^3. This equates to a required storage volume of 3.4 M m^3 of slurry tailings or 2.5 M m^3 for dry stack tailings;
- Slurry tailings was assumed to have a solids density of 40%, and dry stack tailings moisture content was assumed to be about 12% by weight;
- The climate in the project area is warm to hot subtropical. Rainfall was estimated to exceed 1,000 mm per annum, with rainy season extending from April to November (Micon 2014). Details regarding the magnitude and daily distribution of the rainfall events are not available, and for the purpose of the design it was assumed that high intensity short duration rainfalls dominate the wet season;
- As far as practical all contact and non-contact water must be separately handled. Non-contact water must be diverted and contact water must be collected and treated (if necessary) prior to discharge;
- The project is located in a seismically active area, with relatively frequent severe earthquakes like the Haiti 2010 earthquake (Peak Ground Acceleration (PGA) of 0.5 g). Based on regional seismicity data (Frankl et al. 2011) the design criteria for seismic design was determined to be an earthquake with a PGA of 4.9 m/s2 (equivalent to 0.5 g), commensurate with a 2% probability of exceedance in 50 years;
- For sizing of alternate tailings storage facilities, containment structures shall be designed with 2.5H:1V upstream, and 3H:1V downstream slopes. Dry stack tailings will be constructed with an overall slope of 3H:1V;
- There has been no foundation characterization at any of the alternate tailings storage facility sites. Borehole log information suggest that the surficial soils consist of a thin layer of organic soils overlying residual soils, overlying competent bedrock. The overburden thickness ranges between 10 and 60 m in the Romero deposit area and generally less than 10 m in the Romero South area (Niemi 2015);
- There is no information available about the tailings geochemistry. For the purpose of this assessment it has been assumed that the tailings would be geochemically benign;
- There was no consideration of lease or property boundaries in the site selection; and
- It is not known whether the Dominican Republic has specific tailings design guidelines or standards. For the purposes of this preliminary assessment it has been assumed that the design will be carried out in according with industry best practice, which includes but is not limited to the Canadian Dam Association Dam Safety Guidelines (CDA 2013).

The level of engineering presented in this technical memorandum is conceptual. This is deemed suitable to provide capital and operating costs to a level of accuracy of ±30% for inclusion into the revised PEA.

18.18.2 Alternatives Assessment

18.18.2.1 Alternative Deposition Methods

Alternate deposition methods considered for the project included:

- Conventional low solids content (typically less than 30% by volume) slurry;
- Thickened tailings slurry (solids content typically between 30% and 65%), but still pumpable with centrifugal pumps;
- Paste tailings (solids content typically in excess of 65%) requiring positive displacement pumps; and
- Filtered (i.e. dry stack tailings) tailings.

The tailings material is expected to be liquefiable in an undrained state. As a result, any tailings deposition strategy that results in containment of wet solids will require an extremely robust tailings containment structure. Therefore, filtered tailings definitely poses significant advantages. However, the initial capital costs, and subsequent high operating cost of the filtered tailings is high.

The terrain at the Romero site does have significant topographical relief and as a result, there might be valleys that would offer opportunity to provide cost effective containment structures for depositing of conventional slurry tailings. Therefore, to evaluate the possible, spectrum of tailings storage options, both conventional low solids slurry and filtered tailings disposal was evaluated.

Slurry tailings could be deposited either sub-aqueously or subaerially. Given the high seismically active area, and assuming that the tailings are geochemically benign, it would be more reasonable to assume subaerial deposition.

18.18.2.2 Alternative Deposition Sites

<u>Conventional Slurry Tailings</u>

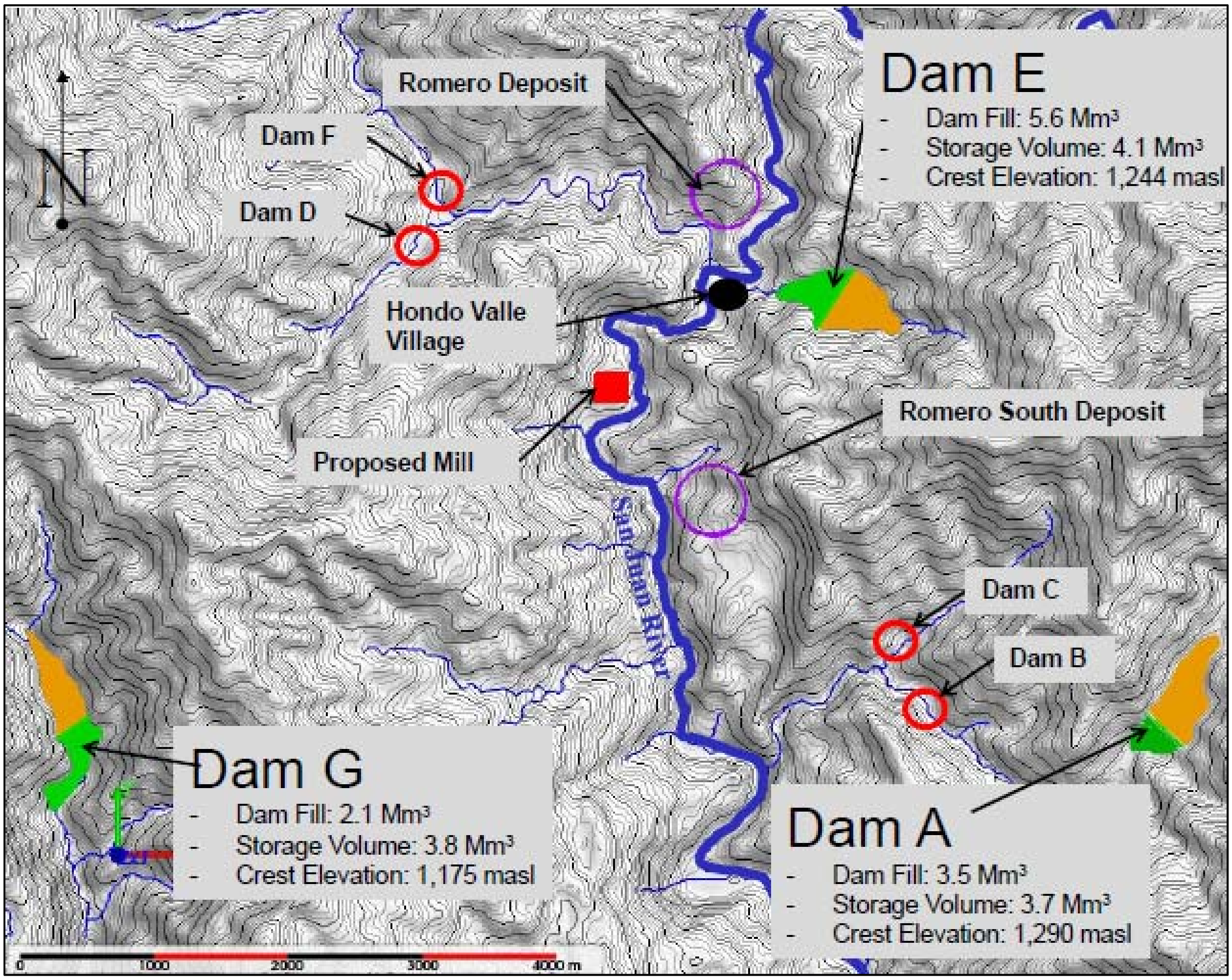
The search for a suitable tailings site was limited to an area within a 20 km radius from the proposed mill location as illustrated on Figure 18.2. Considering the significant topographical relief associated with the site this distance was considered to be the maximum practical distance before pumping tailings may become cost prohibitive.

Site selection was done using topographical and cadastral mapping supplied by JDS supplemented by regional Aster GDEM worldwide elevation data, and imported into GIS software Global Mapper (Global Mapper 2014). Screening volumetric take-offs were calculated assuming dams with an upstream slope of 2.5H:1V, and a downstream slope of 3H:1V and a 10 m wide crest. The dams models and volume take-offs were obtained using the Muck 3D software package (Muck 3D 2015).

A total of seven sites were evaluated as shown in Figure 18.2, with a brief description of each site provided in Table 18.1.

The closest site, Dam E, has the lowest storage efficiency, with 5.6 Mm^3 of fill required to create a storage volume of 4.1 Mm^3 while the most efficient location, Dam G, is the furthest away in a valley west of the project area, requiring a tailings pipeline of over 7 km long with an elevation gain of over 500 m. Dam A is located at moderate distance from the proposed mill site, with favorably small catchment, but the dam fill volume is nearly equal to the created storage volume.

Figure 18.2: Slurry Tailings Storage Alternatives



Source: SRK 2015

Table 18.1: Summary of Tailings Dam Locations Evaluated

Site	Location	Description	Status
Dam A	Upper reach of San Juan River east bank tributary 2.5 km south of Romero South deposit; about 4.4 km from confluence;	Dam fill volume almost equal to created storage volume; Relatively small catchment area.	Shortlisted
Dam B	Mid-reach of San Juan River east bank tributary 2.5 km south of Romero South deposit;1.8 km from confluence;	Large catchment area; difficult terrain for diversion channels.	Not considered further
Dam C	Side-branch of San Juan River east bank tributary 2.5 km south of Romero South deposit;1.8 km from confluence;	Very steep valley gradient; Cannot provide sufficient storage volume.	Not considered further
Dam D	Side-branch of west bank tributary of San Juan River 4.5 km north of Romero South deposit, about 3.4 km from confluence.	Large catchment area; requires excessive non-contact water diversion structures.	Not considered further
Dam E	Mid-reach of San Juan River east bank tributary 4.5 km north of Romero South deposit; 0.7 km from confluence.	Close to proposed mill and small catchment area; Dam fill volume exceeds the created storage volume.	Shortlisted
Dam F	Mid-reach of west bank tributary of San Juan River 4.5 km north of Romero South deposit; about 3.4 km from confluence.	Large catchment area; Steep valley gradient, with insufficient storage volume created by a dam.	Not considered further
Dam G	Not in the San Juan River valley. One valley over to the west; 4.7 km (straight line) south-west from proposed mill site.	Relatively small catchment; Smallest dam fill required to create required storage.	Shortlisted

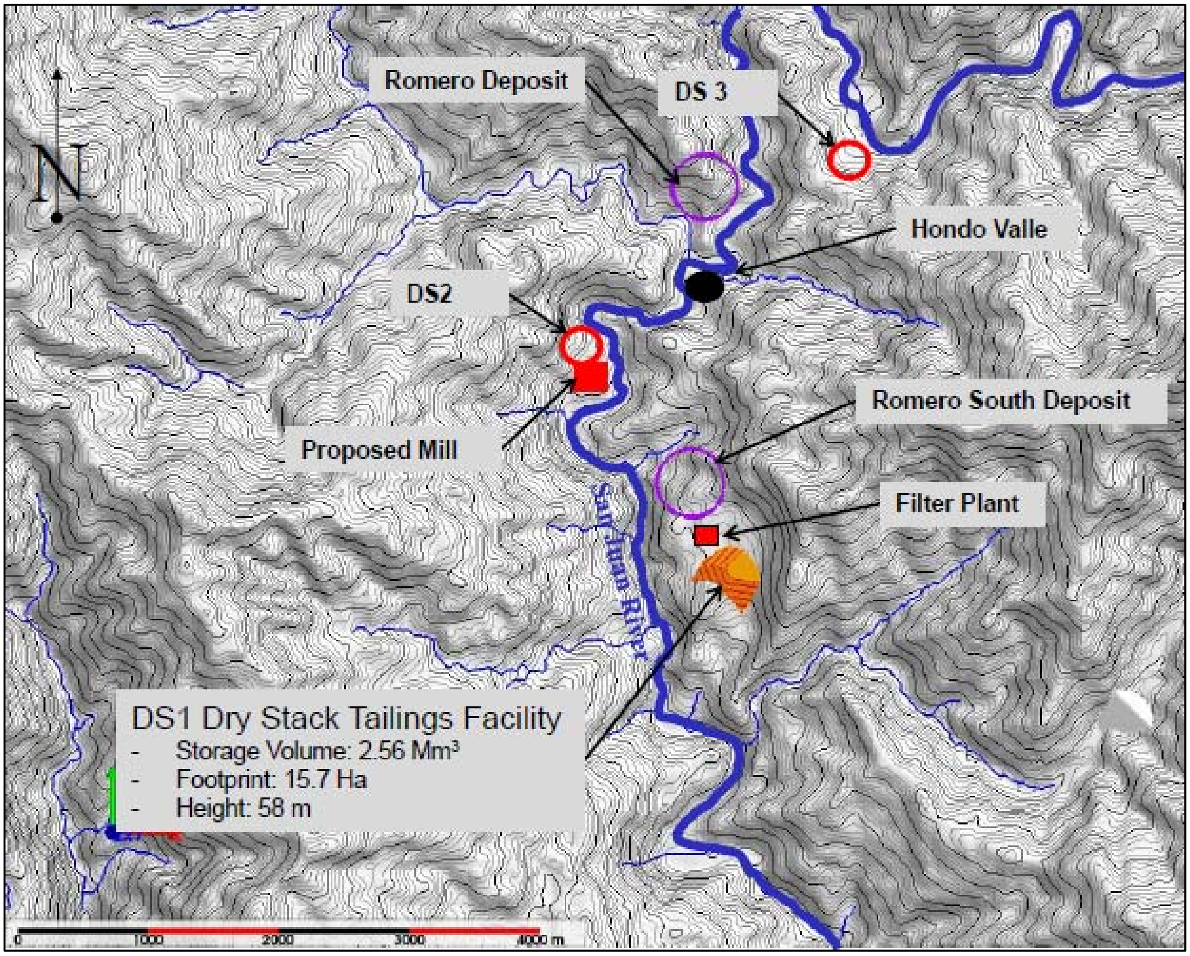
Source: SRK 2015

Dry Stack Tailings

Three sites were considered for dry stack tailings as illustrated in Figure 18.3. Two of these, sites DS2 and DS3 did not provide sufficient storage capacity dues to the steep terrain, and the requirement to maintain an overall outer slope of 3H:1V.

The only area, relatively close to the mill and sufficiently large to accommodate the volume of tailings is the large terrace adjacent to the Romero South deposit, above the east bank of the San Juan River, shown in Figure 18.3. Three dry stack configurations were evaluated at this location, ranging between a large footprint with low overall height, to a small footprint with subsequent large height.

Figure 18.3: Dry Stack Tailings Storage Alternatives



Source: SRK 2015

18.18.3 Selected Alternative

The conventional slurry tailings option has the advantage of being the most common, well defined and understood tailings deposition method. The simple tailings deposition strategy also makes for a low operational cost; however, the high pumping heads (in excess of 200 m static head difference) that has to be overcome does challenge this fact.

The steep topography results in poor storage efficiency, with the containment structure occupying close to the same, if not more material that the required tailings storage volume. This also limits the ability to effectively stage containment dam construction over the life of the mine, with a large proportion of the fill volume being required as the starter dam. This means that the initial capital cost would be high, with little opportunity to defer capital over the life of mine.

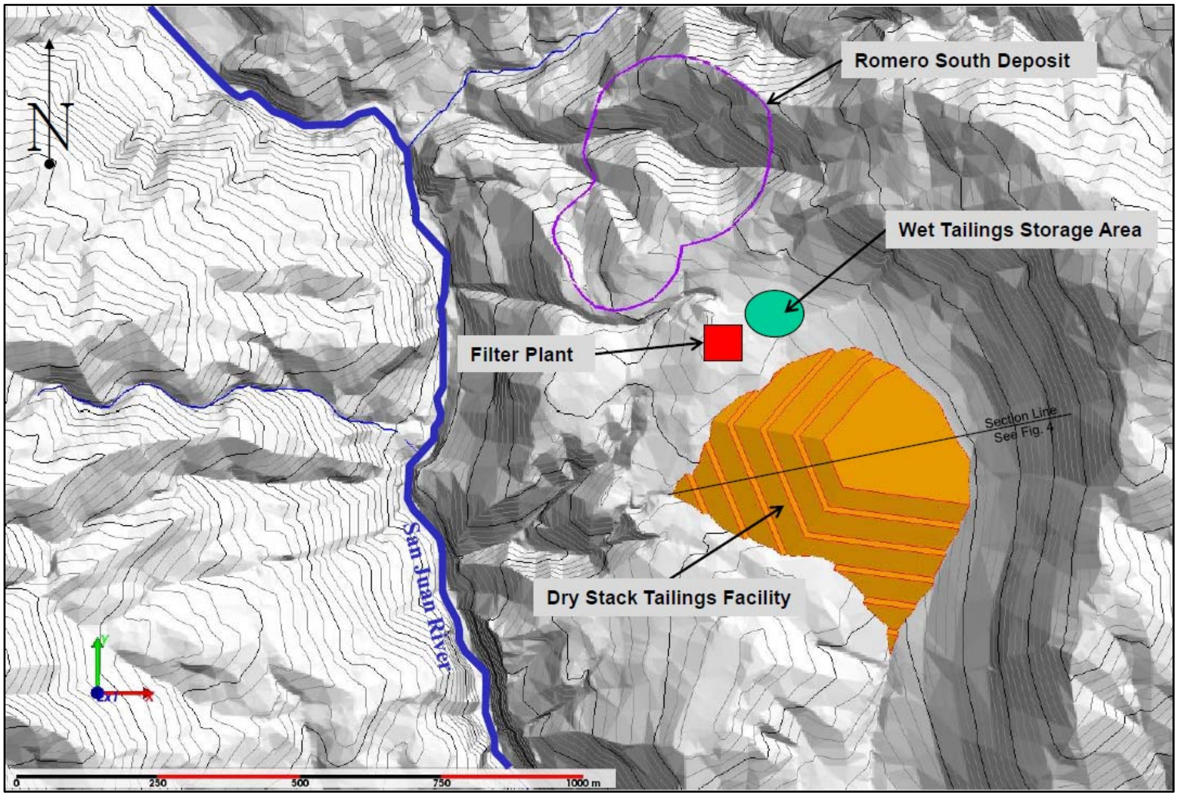
The steep gradient also makes for challenging and cost intensive water diversion structures to ensure diversion of upstream non-contact water. Finally, the presence of a very large tailings containment dam, in a seismically active, and environmentally sensitive area is not desirable, and may be challenging to obtain regulatory approvals for.

Dewatered, dry stack tailings, if properly compacted and provided with sufficient underdrain facilities significantly reduces the risk of tailings liquefaction, and therefore effectively mitigates against the high seismicity of the region. The reduced footprint offered by the higher density tailings, and the fact that a large containment dam is not necessary eliminates substantial upfront capital costs as well as facilitates water management, both related to contact and non-contact water.

The high capital cost associated with constructing the filter plant is a disadvantage; however, compared against the cost of the very large containment dams required for conventional slurry tailings, this is not considered that significant. The operating cost of dry stacking will be higher than for conventional slurry tailings, especially considering the high degree of quality control that will be required to ensure that proper moisture control and compaction can be achieved, especially during the wet season.

Considering all of the above, the conclusion was reached that dewatered (i.e. dry stack) tailings deposition would be the most suitable tailings deposition method for the Romero Project. The final selected location for this facility is presented in Figure 18.4.

Figure 18.4: Dry Stack Tailings Conceptual Design – Plan View



Source: SRK 2015

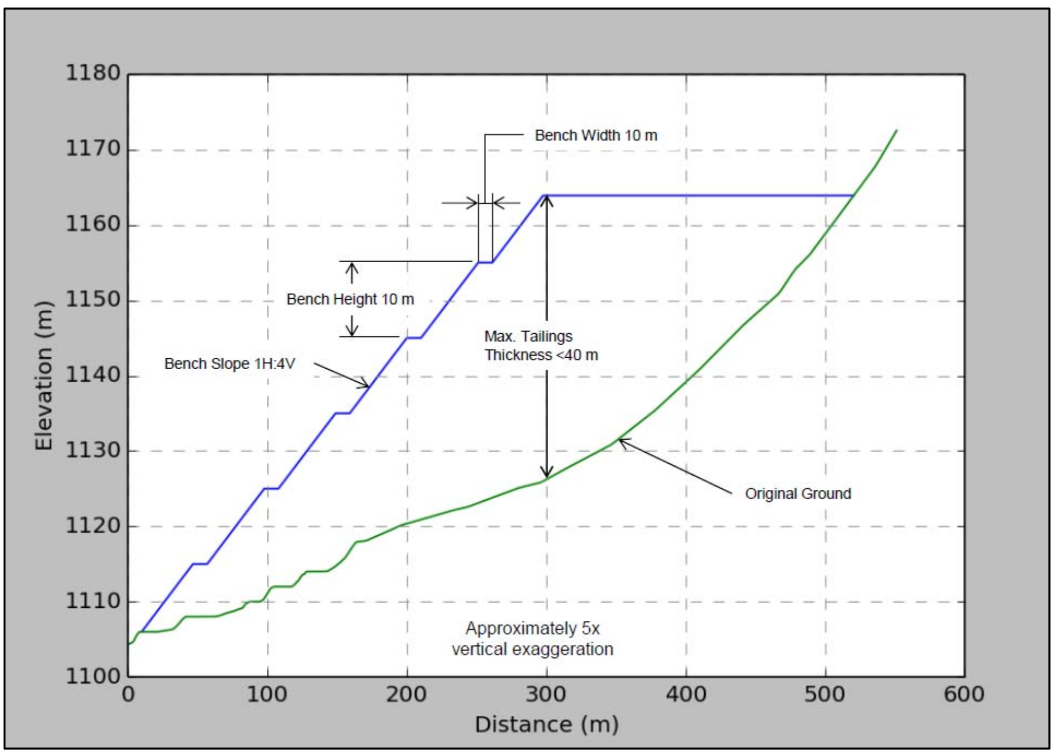
18.18.4 Dry Stack Design and Operation

Geometry

Using assumed foundation conditions and associated material properties, a preliminary pseudo static stability assessment was carried out to confirm the requirements for ensuring minimum factors of safety for the dry stack tailings facility. The analysis confirmed that the overall outside slope of the tailings facility should be lowered to 4.5H:1V (i.e. 12º). Therefore, for the design presented in this assessment the dry stack will be constructed in 10 m high benches with 10 m setbacks, using inter-bench slopes of 4H:1V, to yield an overall slope of 4.5H:1V as illustrated in Figure 18.5.

Although the maximum elevation difference between the toe and the crest of the facility is approximately 58 m, the actual tailings thickness does not exceed 40 m due to the rising topography. The total footprint occupied by the facility is about 15.7 ha.

Figure 18.5: Dry Stack Tailings Conceptual Design – Section View



Source: SRK 2015

Underdrain

A comprehensive underdrain system is required to facilitate rapid drainage of the placed dry stack material. This underdrain is critical towards ensuring that the liquefaction risk of the facility remain as small as possible.

The dry stack footprint will be cleared and grubbed of any vegetation. About 0.3 m of organic topsoil will subsequently be stripped and stockpiled adjacent to the dry stack for later use. A series of finger drains, 0.6 m thick, will be constructed prior to tailings deposition using 50 mm minus drainage gravel. This primary drainage fill will be covered with a 0.3 m thick pea gravel filter zone. The finger drains will occupy about one third of the total dry stack footprint once complete. Staged construction will occur, advancing upslope as construction of the dry stack progresses.

Waste Rock Shell

Surface erosion from the exposed side slopes of the dry stack facility will be a major concern. To best manage this, the entire face of the facility will be clad in a shell of geochemically suitable waste rock (or quarry rock if required). The waste rock shell will be progressively raised as the dry stack facility is constructed over the life of mine. The waste rock shell will have a horizontal thickness of 2 m, which translates to a normal thickness of about 0.5 m along the inter-bench side slopes. The benches will receive a cladding of at least 0.5 m of waste rock.

Dry Stack Placement

The filter plant will be constructed immediately adjacent to the dry stack facility. The filter plant will produce a dewatered tailings product which will be stockpiled adjacent to the plant. A loader will be used to place this material into 35 ton haul trucks for transport to the dry stack facility. It is anticipated that two trucks will be required to keep up with the daily production demand.

The filtered tailings will be end-dumped and spread into horizontal lifts not exceeding 0.3 m thick using a CAT D6 Dozer (or grader). A 10 T vibrating drum sheepsfoot roller will then be used to compact the placed material to a Standard Proctor density of at least 95%, at ± 2% of optimum moisture content. At all times the surface will be graded to ensure any runoff is shed from the facility so as to limit any possibility of ponding water on the dry stack facility.

Diligent quality control and quality assurance testing of the compaction and moisture content of the placed tailings is of paramount importance. Moisture conditioning of the stack must be carried out to ensure that the required placement moisture is as specified. During dry periods additional wetting up of the material may be required using a water truck. During wet periods, the material may have to be spread out and allowed to dry.

During the wet season, there may be periods when the rainfall is sufficiently intense that tailings placement is not practical to achieve the intended densities. If there is not sufficient room on the dry stack to temporarily stockpile this material for later spreading and drying, the material can be stored in the Wet Tailings Storage Area adjacent to the dry stack facility. This facility has a storage capacity of about 56,000 tonnes, which is equivalent to 22 days of tailings production. The facility occupies a footprint of about 2 hectares and contains a waste rock (or quarry rock) containment berm that has upstream and downstream side slopes of 1.5H:1V, a 2 m wide crest width and is about 10 m high at its maximum.

Water Management

Design Flood

The project life is 10 years. The site is subject to short duration high intensity storms, and therefore for the purpose of this assessment the design flood has been set as the 24 hour duration, 100-year recurrence storm. Site specific intensity duration frequency (IDF) curves are not available but nearby data from for a site in Haiti, 110 km southwest of the Romero site (Heimhuber 2013) suggest the design flood equates to a 5.94 mm/hr of precipitation event.

Non-Contact Water

Upstream non-contact water will be managed by constructing diversion channels. The diversion channels will bend along opposite sides of the dry stack facility from a central high point behind the highest part of the facility. The two catchments associated with these two channels, west and south are about 26.5 and 19.3 ha in size respectively. This diverted non-contact water is ultimately directed towards existing natural creeks that drain towards the San Juan River downstream of the dry stack facility as illustrated in Figure 18.6.

The diversion channels were sized based on the larger of the two catchments, yielding a design flow of 0.45 m^3/s associated with the design flood event. The diversion channels will be excavated into natural ground with a minimum depth of 1.2 meters with side slopes of 2H:1V and base width of 1 m. No seepage containment is planned for the diversion channels, but they will be riprapped to protect against scour. For preliminary planning the riprap was assumed to be 250 mm nominal sized rock placed 0.5 m thick, resulting in a requirement for 3.2 m^3 of riprap per lineal meter of channel.

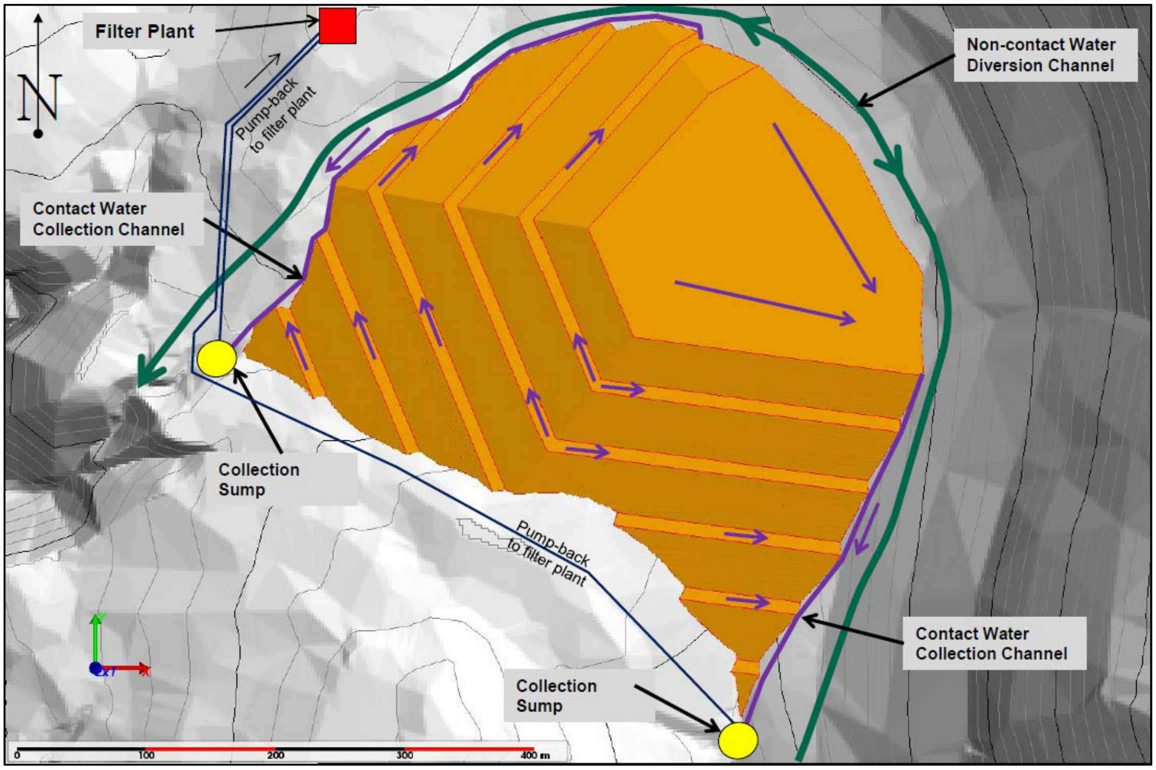
Contact Water

Contact water from the surface of the dry stack facility will be collected through swales constructed on each of the benches. The water will then be directed via collection channels along the west and east perimeter of the facility towards two collection sumps as illustrated in Figure 18.6. Water collected in these sumps will be discharged to the environment if it meets discharge criteria, or alternately be returned to the filter plant for recycle back to the processing plant.

The catchment area of the contact water swales is limited to the surface area of a bench, which is a maximum of about 3 ha, resulting in a swale flow requirement of about 2.3 m^3/s. The collection channels for the contact water are assumed to collect about half of the total volume associated with the surface area of the dump, resulting in a flow requirement of about 6.0 m^3/s.

Water emerging from the finger drains will also be directed towards the two collection sumps.

Figure 18.6: Dry Stack Tailings Water Management



Source: SRK 2015

18.18.5 Dry Stack Closure

A key assumption was that the tailings would be geochemically benign and therefore no permanent infiltration or oxygen reducing cover is required. The primary closure objective would therefore be to create a stable landform. The biggest long term risk associated with creating a stable landform is to ensure erosion protection. The waste rock shell along the outer slopes of the dry stack facility has been designed to achieve this during the operational stage, and given the climatic regime, it is anticipated that natural re-vegetation of these slopes would occur over time, which would further serve to provide erosion protection, as well as allow the facility to better blend into the landscape.

At closure no further re-sloping would be carried out, but drainage swales will be upgraded as required. The upstream non-contact water diversions will be breached. The top surface of the dry stack will be covered with a layer of 0.3 m of organic soil to facilitate rapid re-vegetation.

19 Market Studies And Contracts

19.1 Market Studies

At this time, no market studies have been completed. No contractual arrangements for concentrate trucking, port fees, shipping, smelting or refining exist at this time. There are no contracts in place for the sale of copper concentrate. It is assumed that the concentrate produced at the Romero mine would be marketed to international smelters in Asia and Europe. No deleterious elements have been identified or considered at this time.

The smelter terms used in the economic analysis are based on recent marketing terms from similar projects and are demonstrated in Table 19.1.

Table 19.1: NSR Parameters used in the Economic Analysis

NSR Parameters	Unit	Cu Concentrate
Smelter Payables		
Cu Payable	%	96.5
Au Payable	%	97.5
Ag Payable	%	90
Minimum Deduction in Conc	%	1
Au Minimum Deduction	g/t	0.6
Ag Minimum Deduction	g/t	20
TC/RCs		
Treatment Charge	$/dmt conc	85
Cu Refining Charge	US $/lb	0.085
Au Refining Charge	US $/oz	6
Ag Refining Charge	US $/oz	0.5
Transport Costs		
Moisture Content	%	8
Transport to Port	US$/wmt conc	$100.00
Total	**US$/wmt conc**	**$100.00**
	US$/dmt conc	**$108.70**

Source: JDS 2015

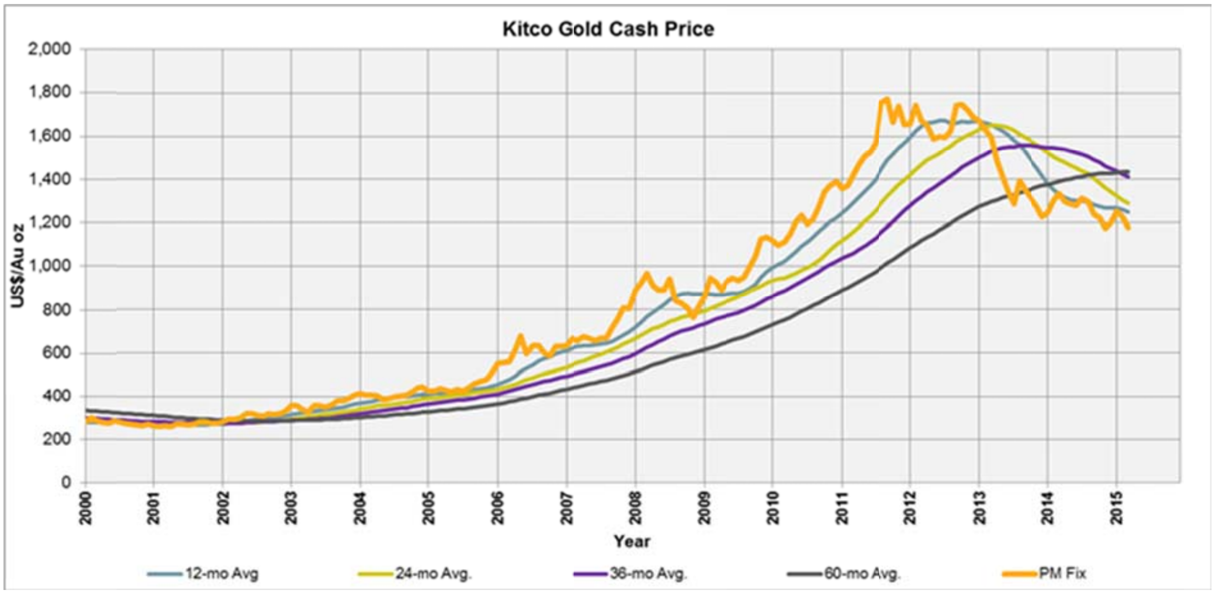
19.2 Royalties

The economic analysis has considered a 1.25% NSR royalty on all revenues. LOM royalties amount to $15M.

19.3 Metal Prices

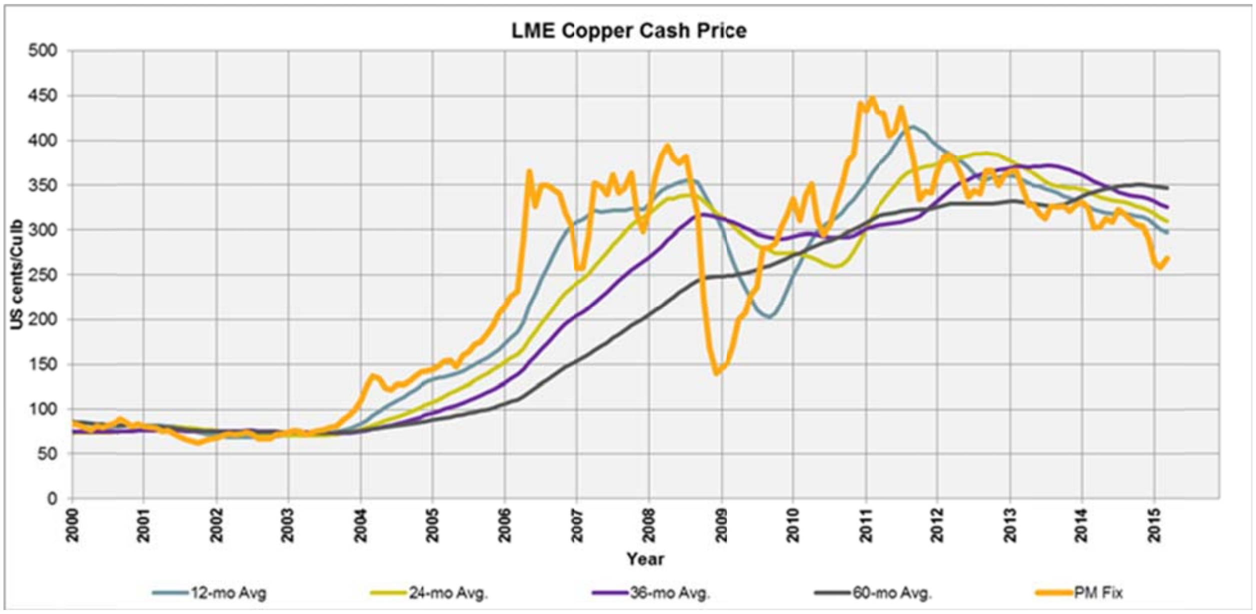
The base and precious metal markets benefit from terminal markets around the world (London, New York, Tokyo, Hong Kong) and fluctuate on an almost continuous basis. Historical metal price for copper are shown in Figure 19.1 through Figure 19.2 and demonstrate the change in metal price from 1998 through to 2015.

Figure 19.1: Historical Gold Price



Source: JDS 2015

Figure 19.2: Historical Copper Price



Source: JDS 2015

Base Case pricing used in the economic analysis is in line with recent publications and spot metal pricing as at April 2015. The metal prices used in the economic analysis are presented in Table 19.2.

Table 19.2: Metal Prices & F/X Rate used in the Economic Analysis

Metal Price & F/X Rate	Unit	Value
Cu Price	US$/lb	2.90
Au Price	US$/oz	1,225
Ag Price	US$/oz	17.00
F/X Rate	US$:C$	0.8

Source: JDS 2015

20 Environmental Studies, Permitting and Social or Community Impact

The following section is taken from the 2014 Micon PEA and includes direct citations from baseline and assessment reports completed by AMEC in 2013 and 2014, identified by indented, italicized text.

20.1 Environmental Studies and Issues

A rapid biodiversity assessment was undertaken to establish the wider biological sensitivity of the area. From a project strategic point of view the following biodiversity aspects need special consideration during future planning:

The Romero Project is located in the globally significant Cordillera Central Corridor Key Biodiversity Area. For future project development, further investigations will be required to determine if there are any vulnerable or irreplaceable biodiversity communities or habitats present.

The Project area is located in close proximity to two National Parks:

- The José del Carmen Ramírez National Park in which the Critical Endangered Eleutherodactylus schmidti (amphibian) is found; and
- The Armando Bermudez National Park, which has been demarcated for the protection of its large-scale ecological processes and species.

Encroachment into these areas has to be avoided, while the potential presence of the Eleutherodactylus schmidti has to be investigated should future activities be planned.

Although some level of anthropogenic disturbance is associated with the project area, the IUCN Red Data List indicates that there are two Critically Endangered Species that can occur in the project vicinity:

- Hispaniolan Crestless Toad (Peltophryne fluviatica); and
- Ridgway's Hawk (Buteo ridgwayi).

The initial definition of the water quality sampling network was set up on the San Juan River watershed, installation of a weather station and training of GoldQuest‟s personnel has been undertaken. Two sampling campaigns were undertaken in 2013 (AMEC 2014).

The Romero Project is also located on the San Juan River, upstream of the Sabaneta Reservoir that provides irrigation to downstream agricultural lands. At least three small villages use the San Juan River. Water quality is slightly basic and overall low in most metals.

20.2 Environmental Management

More detailed environmental management plans will be developed as the project planning progresses; however, it is assumed that the project will be developed to international best practice standards and conform with the IFC Social and Environmental Performance Standards for environmental protection and management.

20.2.1 Tailings

Tailings will be backfilled to the underground mines and to a dry stack tailings management facility. The various tailings streams will need to be characterized during the metallurgical program at the next stage of the project.

It is likely that some of the tailings streams will be potential acid generating and metal leaching (PAG/ML). Any PAG/ML tailings should preferentially be backfilled underground so that the material will flood after closure and have less opportunity to oxidize or release contaminants to the receiving environment.

20.2.2 Waste Rock

The quantity of waste rock from underground development will need to be disposed of initially on surface. As mining progresses, some waste rock may be used to partially backfill secondary stopes and inactive areas of the mine. The expected waste rock types will need to be sampled and characterized during the next stage of the project. If any of the rock types are identified as PAG/ML, kinetic tests will need to be conducted to estimate the time to onset of acid generation and to develop a management strategy.

20.2.3 Water Management

It is assumed that seepage and runoff water will be controlled, collected and monitored on a regular basis for all facilities during all phases of the project. An assessment will be needed to determine the effects on flows from hydro-electric power development on the upper San Juan River and the Sabaneta Reservoir.

A key feature of the current process design is that cyanide is not proposed for mineral processing. This is preferable for project permitting in the Dominican Republic.

Water and waste management planning will need to protect the San Juan River watershed flows and water quality for the surrounding villages and the Sabaneta Reservoir users

20.3 Permitting Requirements

20.3.1 Constitution of the Dominican Republic

The current Constitution was adopted and entered into force on January 26, 2010. It is the general framework to ensure the functioning of the state. The Constitution emphasises the protection of property and the importance of such in Article 51. However, Article 17 states that "mining and hydrocarbon deposits and, in general, all non-renewable resources, may only be explored or exploited by private parties, under sustainable environmental criterion, in accordance with concessions, agreements, licenses, permits or quotas, under the conditions determined by law".

A company undertaking mining operations in the Dominican Republic must take into account that the Dominican State is a necessary participant in any mining operation, and that the minerals are owned by the State, although the entity awarded with a concession has the right to profit from the extracted minerals (AMEC 2013c).

20.3.2 Mining Law

Law No. 146 enacted on June 4, 1971 and regulation No. 207-98 dated June 3, 1998, are the general mining laws by which mining in the Dominican Republic is governed. It is these Laws that codifies the State is the owner of all mineral deposits, of any nature, on Dominican soil, and that the exploitation or mining of such deposits are undertaken by means of concessions or agreements granted exclusively by the Government.

All concessions granted within national territory are exclusively governed by the laws and courts of the Dominican Republic. When foreigners are the concessionaires, such concessionaires are deemed to have validly waived of any right to diplomatic protection in relation to the concession.

Law No. 146 created the General Mining Directorate, as the administrative body charged with implementing the Law and regulating mining activities in the Dominican Republic, this has in the meantime been amended, with the mining administrative powers now vested in the Ministry of Energy and Mining.

20.3.2.1 Presidential Decrees

Decree No. 613-00, dated August 25, 2000, regarding the creation of the National Council for Mining Development.

Decree No. 839-00 dated September 26, 2000, regarding the declaration of mining as an activity of the highest priority of the Dominican State, thereby instructing the Corporate Mining Authority to enter into certain agreements regarding the development of certain mining sectors of the country.

Decree No. 947-01 dated September 19, 2001, regarding the creation of Industrial mining parks for whom the tax incentives of the Dominican Industrial Free Zone Law No. 8-90 are extended to.

Law No. 123-71, dated 10 May 1971, along with its regulation of enforcement, also regulate certain mining activities, namely the extraction of sand, gravel, chippings, rocks and similar materials (AMEC 2013c).

20.3.3 Environmental Law

The General Law with respect to the environment is No.64-00, dated August 18, 2000, governing all environmental related issues in the Dominican Republic. This Law creates five Vice-Ministries for Environmental Resources:

- Water Management Issues;
- Biodiversity;
- Protected Areas;
- Forest Resources; and
- Marine Resources.

This Law creates the governmental authority the Ministry of Environment and Natural Resources to oversee and regulate this Law. The Law sets out the general rules regarding conservation, protection, improvement and restoration of the environment and natural resources.

Article 38 of 64-00 establishes the process of environmental evaluation, in order to prevent, control and mitigate the impacts over the environment and natural resources caused by works, projects and other activities. This process includes the development of the following instruments:

- Environmental impact Statement;
- Strategic environmental evaluation;
- Environmental impact study;
- Environmental report;
- Environmental license;
- Environmental permit;
- Environmental audit; and
- Public consultation.

The Ministry of Environment and Natural Resources requires projects conduct environmental impact studies in order to obtain an environmental license. The activities triggering studies in the mining sector include the following: development, exploitation and processing of metallic and non-metallic mining; exploration and mining prospection; extractive metallurgy; artisan mining; mining parks; and aggregate processing plants; among others (AMEC 2013c).

20.3.4 Conventions, Treaties and Protocols

The Dominican Republic is a member of the following international bodies: ACP, AOSIS, BCIE, CARICOM (observer), CD, CELAC, FAO, G-77, IADB, IAEA, IBRD, ICAO, ICC (national committees), ICRM, IDA, IFAD, IFC, IFRCS, IHO, ILO, IMF, IMO, Interpol, IOC, IOM, IPU, ISO (correspondent), ITSO, ITU, ITUC (NGOs), LAIA (observer), MIGA, NAM, OAS, OPANAL, OPCW, PCA, Petrocaribe, SICA (associated member), UN, UNCTAD, UNESCO, UNIDO, Union Latina, UNWTO, UPU, WCO, WFTU (NGOs), WHO, WIPO, WMO, WTO.

Specifically related to environmental and social protection, the Dominican Republic is signatory to the following conventions, treaties and accords:

Basel Convention on the control of Trans-boundary Movements of Hazardous Wastes and their Disposal (accession July 10, 2000).

- American Convention on Human Rights (accessionJanuary 21, 1978);
- Convention on Biological Diversity (signed May 23, 2001);
- Convention on International Trade in Endangered Species of Wild Fauna and Flora (signed March 17, 1987);
- International Covenant on Economic, Social and Cultural Rights (accession January 4,1978);
- Indigenous and Tribal Populations Convention, 1957 (accession: June 23, 1958);
- Kyoto Protocol (accession: February 12, 2002);
- Ramsar Convention (accession:September 15, 2002); and
- Stockholm Convention on Persistent Organic Pollutants (accession: May 4, 2007).

20.3.5 Potential Permitting Risks

Permitting of a new mine carries some risk due to the the proximity of the project to a national park and the San Juan and La Guama Rivers. As project plans progress, it will be important to not encroach on the park, to complete thorough and scientifically defensible baseline environmental studies and to conduct an effective engagement and consultation program from the community to the national level.

20.4 Social and Community Aspects, Stakeholder Consultation

In terms of social setting, the Romero Project is located in a remote area of the Dominican Republic. Population densities are low, with only a few villages located within the exploration area and the closest large city approximately 40 km south of the project.

Three settlements have been identified as falling within the Project area:

- Hondo Valle;
- La Hilguera; and
- La Cienaga Vieja (AMEC 2013c).

These villages may be directly or indirectly impacted (physically, economically, positively and potentially negatively) by future project activities. GoldQuest will continue consultation with these communities throughout the on-going exploration activities and future Project planning process.

The following social aspects need specific consideration in both the planning and continuous engagement process:

- Primary agricultural practices are important downstream of the Project area. Irrigation practices seem to take place on a large scale.
- AMEC understands that the opinion and role that the Church plays in society is very important. Strategic planning and engagement with the Church is essential.
- The secondary investigation indicates that the oldest male in a family holds the decision making role. This is important to take into consideration in the Stakeholder Engagement Plan (SEP).
- The financial status quo might lead to an influx of employment seekers into this remote area where an extra burden might be placed on services; a forecast increase in the population due to mining justifies taking due consideration that the local healthcare services are not overburdened.
- The public schooling system, especially in rural, more inaccessible areas is fairly poor. This presents an opportunity to GoldQuest in terms of social investment, but taking due care that such support provides for a sustainable community infrastructure, i.e., able to be maintained after mining investment ceases.
- Various villages are located along the road to the project area. The sphere of influence of the project may also include these villages. The potentially affected communities will therefore have to be clearly defined to ensure that a pro-active stakeholder engagement process can be implemented.
- Depending on the location of the facilities and the mining method chosen, some resettlement might be necessary and therefore the development of a Resettlement Policy Framework

(RPF) is recommended, as well as gaining an understanding of when the census of directly affected people can be locked (AMEC 2014).

- The project proposed in this PEA is not expected to require any resettlement. Some land acquisitions will likely be necessary for the proposed tailings facility, mill site, and ancillary facilities.

20.5 Social Management

GoldQuest has an environmental policy in which they commit to:

- Communicate openly and transparently about their activities;
- Provide information to their shareholders about the environmental aspects of their business;
- Meet and, where appropriate, exceed applicable legal and other requirements of their operating licenses;
- Ensure that all GoldQuest's employees and subcontractors are familiar with their policies and act accordingly; and
- Strive to continuously improve their work practices in order to reduce the impact of their activities on the environment.

20.6 Reclamation and Closure Requirements

Initial closure bonding is estimated at $US400,000, with final closure costs estimated at $19M. This cost estimate will need to be re-calculated at the next stage of project planning.

21 Capital Cost Estimate

The capital cost (CAPEX) of the project has been estimated based on the scope defined in previous sections of this report. The following parties have contributed to the preparation of the CAPEX estimates in the specific areas:

JDS:

- Process Plant;
- Plant infrastructure and services, including roads, water management, ancillary buildings, and fuel storage;
- EPCM and Indirect costs relating to the process plant, infrastructure, and tailings facility;
- Owner's Costs;
- Contingency;
- Select Mining equipment; and
- Mining.

SRK

- Dry stack tailings facility and haulage cost estimation.

21.1 Capital Cost Summary

The capital cost estimate was prepared using first principles, applying project experience and avoiding the use of general industry factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study. Given that assumptions have been made due to a lack of available engineering information, the accuracy of the estimate and/or ultimate construction costs arising from the engineering work cannot be guaranteed. The target accuracy of the estimate is ±30%.

Costs are expressed in US$ with no escalation unless stated otherwise. Foreign exchange rates of C$1.00:US$0.80 are used where applicable.

The estimate is based on the assumption that contractors would mobilize only once to carry out their work and are not already mobilized on site performing other work.

Total life of mine capital costs are estimated to be $235.3M, Pre-production capital costs amount to $143.1M. Capital costs during production years total $92.3M. Contingency for the project totals $26.5M. The costs are summarized below in Table 21.2.

Sustaining and closure capital cost estimates amount to $92.3M and were assumed to occur from 2019 to 2029 with a majority of these costs for tailings earthworks. Mine equipment that is included in the sustaining and closure capital costs account for the ancillary, spares and other miscellaneous mine equipment that are assumed to not be leased.

Closure costs amount to $19M and were assumed to occur in years 10 and 11 immediately after plant closure.

Table 21.1: Capital Cost Summary

Description	Pre-Production (US$M)	Sustaining/ Closure (US$M)	Total (US$M)
Mining	14.9	61.5	76.4
Site Development	9.7	0	9.7
Crushing & Handling	7.1	0	7.1
Process Plant	35.6	0	35.6
On-Site Infrastructure	26.1	0	26.1
Tailings & Waste Rock Management	2.6	6.6	9.2
Project Indirects	9.9	0	9.9
Engineering & EPCM	12.7	0	12.7
Owner's Costs	3.1	0	3.1
Closure	0	19	19
Subtotal	**121.7**	**87.1**	**208.8**
Contingency (20%)	21.4	5.1	26.5
Total Capital Costs	**143.1**	**92.3**	**235.3**

Source: JDS 2015

21.2 Basis of Estimate

All CAPEX costs have been expressed in Q2 2015 US dollars, there are no allowances for escalation included in the estimate. The estimated costs include mine stripping, mine development, site preparation, process plant equipment and facilities, ancillary facilities, roadworks, powerplant and fuel storage, and utilities. The estimate has been considered to have an overall accuracy of +-30% and assumes the project would be developed on an EPCM basis.

The following parameters and qualifications should be considered when reviewing the project CAPEX:

- No allowances have been made for exchange rate fluctuations over the life of mine;
- Force majeure issues;
- Future scope changes;
- Project interest, insurances and financing cost;
- Land acquisition, compensation cost, and sunk costs; and
- Operational insurances such as business interruption insurance and machinery breakdown.

Data for the CAPEX has been obtained from numerous sources, including:

- PEA Level engineering design;
- Budgetary quotations have been obtained for major equipment and infrastructure items;
- QP experience;
- Labour rates obtained from local Dominican Republic contractors; and
- Data from recently completed projects of a similar size, method, and location.

The following assumptions were used in the CAPEX estimates:

- Suitably qualified and experienced construction labour would be available at the time of execution of the project;
- Qualified construction personnel are available in the local community to assist the project;
- No geotechnical and drainage issues are present, therefore, no allowance for special ground preparation was made;
- Borrow sources for construction are available from within the mine limits;
- A power and water supply capable of supplying the required demand of the processing plant is assumed to be available;
- Access road maintenance; and
- No extremes in weather would be experienced during the construction phase and as such, no allowances are included for construction-labour stand-down costs.

21.3 Mine Capital Cost Estimate

The Romero deposit will be accessed from surface via a spiral decline located directly above the deposit, with a portal on surface near the proposed mill site. All mineralized plant feed and waste will be trucked out of the mine to surface via the spiral decline. The mine plan for Romero has been designed and scheduled such that multiple areas of the mineral resource material will be mined simultaneously and independently, thereby allowing the mine to reach target production before the spiral decline has reached its ultimate depth.

The mining work required for the project is shown below in Table 21.2.

Table 21.2: Mining Capital Cost Estimate

Description	Pre-Production (US$M)	Sustaining/ Closure (US$M)	Total (US$M)
UG Development	2.6	27.2	29.8
UG Mobile Equipment	5.6	31.4	37
UG Stationary Equipment	6.2	3	9.2
First Fills	0.4	0	0.4
Total Mining CAPEX	**14.9**	**61.5**	**76.4**

Source: JDS 2015

21.4 Site Development

Site development costs carried in the CAPEX include clearing and grubbing, mass earthworks, access roads, and internal site roads. Cost estimates are based on historical data and relative experience.

21.4.1 Site and Access Roads

Construction will commence with three major road building projects. The first will be an upgrade to the existing 4.5 km stretch of road starting at the Sabaneta Dam power generating station and ending north of the dam reservoir. This section will be upgraded to handle increased traffic loads.

The second is a new site access road starting from the north end of the dam reservoir and finishing at Hondo Valle. This is a 13.3 km stretch of completely new road. The route has been designed to minimize grade for operations traffic and cut/fill volumes during construction.

The third road is the dry stack tailings facility access road. This is a 3.5 km road that will utilize the existing rough terrain route as much as possible with some completely new built sections to minimize grade. The road will be widened to accommodate CAT 730 rock trucks to the DSTF.

The Site Development work required to support plant operations is shown below in Table 21.3.

Table 21.3: Site Development Capital Cost Estimate

Description	Pre-Production (US$M)	Sustaining/ Closure (US$M)	Total (US$M)
Mass Earthworks	2.4	0	2.4
Site & Access Roads	7.3	0	7.3
Total Site Development Costs	9.7	0	9.7

Source: JDS 2015

21.5 Processing Cost Estimate

The process plant design incorporates primary jaw crushing and a vibrating feeder, coarse feed overland feed and reclaim conveying, SAG and ball milling, rougher flotation, vertical regrind milling, and three stages of cleaner flotation, concentrate dewatering, and tailings will be dewatered and filtered for dry stack tailings or the paste backfill plant.

The estimate has been prepared based on new budget quotes for major mechanical equipment and high level estimates for bulk take-offs on detailed earthworks, concrete, structural and internal steel, and major pipelines.

Factors have been applied to cover in-plant electrical distribution, instrumentation, piping, and allowances for minor mechanical equipment and platework. Estimates for reagent systems, utility supply (air/water), PLC control, and fire protection have been based on database pricing.

The Crushing and Process Plant works required to support plant operations is shown below in Table 21.4.

Table 21.4: Crushing & Process Plant Capital Cost Estimate

Description	Pre-Production (US$M)	Sustaining/ Closure (US$M)	Total (US$M)
Crushing & Handling			
Primary Crushing	4.3	0	4.3
Reclaim System	2.8	0	2.8
Process Plant			
Grinding Area	11.9	0	11.9
Flotation Area & Regrind	13.3	0	13.3
Concentrate Thickening & Loadout / Filtering	3.5	0	3.5
Finals Tailings	3.9	0	3.9
Reagents	1.3	0	1.3
Process Plant Utilities	1.6	0	1.6
Total Crushing & Process Plant Costs	42.6	0	42.7

Source: JDS 2015

21.5.1.1 Earthworks and Civil Works

Earthwork MTO’s were based on AutoCAD models and by using limited topographical survey information, and thus require further review when detailed topographical data becomes available. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Latin America.

21.5.1.2 Concrete

Concrete MTO’s were based on preliminary layouts and/or included as estimated allowances based on similar plants. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Latin America.

21.5.1.3 Mechanical Equipment

The following major process equipment was sized based on the design criteria and vendor recommendations if required budget quotes were obtained, as detailed below in Table 211.6.

Table 21.5: Summary of Quoted Equipment

Description	Quote Vendor	Estimate (US$M)
Primary Jaw Crusher	FLS	0.3
Rockbreaker	Metso	0.1
SAG Mill	FLS	3.1
Ball Mill	FLS	2.3
Vertical Regrind Mill / Screen / Circuit	FLS	2.8
Cyclone Feed Pumps	Weir	0.3
Liner Handlers	RME	0.6
Reagent Packages	Various	0.8
Pumps (Various)	Weir	0.7
Samplers	Outotec	0.5
Thickeners	Outotec	0.5
Cranes	CRS	0.3
Feeders	Metso/IEM	0.4
Disc Filters/Pressure Filter	Metso/Outotec	2.6
Cyclones	Krebbs/Outotec	0.3
Gravity Concentrator	FLS Knelson	0.2
Flotation Cells	Outotec	2.9
Total Quoted Equipment		**18.2**

Source: JDS 2015

The following equipment has been sized based on the process design criteria and costs were determined based on database pricing:

- Magnets;
- Grizzly;
- Dust Collectors;
- Primary Vibrating Screen;
- Pebble Crusher;
- Conveyors;
- Reclaim Apron Feeders;
- Flotation Air Blowers;
- Flotation O/H Crane;
- PSA/OSA Analyzers;
- Concentrate Thickeners; and
- Compressors.

21.5.1.4 Structural Steelwork

Structural steelwork MTO's were based on preliminary layouts and/or included as estimated allowances based on similar plants. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Latin America.

The unit rate includes supply, shop detailing, fabrication, surface preparation and final painting in the shop, transport to site, site erection and paint touch-up as required.

21.5.1.5 Platework

The following mechanical bulks have been sized based on the process design criteria:

- Cleaner and rougher flotation conditioning tanks;
- Fresh/fire water tanks;
- Process water tank; and
- Chutes and bins.

The remaining mechanical bulks, such as pumps, launders, vessels and receivers, were factored as a percentage of overall mechanical costs for each area. Unit rates carried in the CAPEX were based on benchmarked data for similar projects in Latin America.

The unit rate includes supply, shop detailing, fabrication, surface preparation and final painting in the shop, transport to site, site erection, and paint touch-up.

21.5.1.6 Piping, Electrical and Instrumentation

Piping, Electrical and Instrumentation costs were factored from mechanical equipment pricing for the crushing and process plant areas based on actual historical factors for similar plants in Latin America.

21.5.1.7 Installation

JDS has reviewed and applied a blended labour rate of $25/hr based on local Dominican Republic contractors. The unit man-hours has been adjusted by a productivity factor of 1.8 which was confirmed by local contractors. Labour rates are based on a 50-hour work week, which is typical for remote projects located in Latin America.

The labour rate includes the following items:

- Base rate per hour;
- Sick time;
- Holiday pay;
- Insurance;
- Health and welfare;
- Small tools and consumables;
- Safety gear and clothing;
- Site supervision;
- Mobilization and demobilization;
- Transportation – turnaround;
- Site and head office overhead; and
- Contractor Mark-Up & Profit.

The estimate has been based on the majority of the work being carried out under fixed price or re-measurable unit price contracts under a normal development schedule. No allowance has been included for contracts on a cost plus or fast-track accelerated schedule basis.

The erection of tankage, structural, mechanical, piping, electrical, instrumentation, and civil works would be performed by experienced contractors, using a mix of local and out-of-town labour to achieve the required quality and meet the project schedule.

21.6 Infrastructure Capital Cost Estimate

The on-site Infrastructure required to support the plant operations is shown below in Table 21.7.

Table 21.6: On-Site Infrastructure Capital Cost Estimate

Description	Pre-Production (US$M)	Sustaining/ Closure (US$M)	Total (US$M)
Electrical Supply & Distribution	12.8	0	12.8
Water Supply & Distribution	1.2	0	1.2
Assay Laboratory	2	0	2
Construction/Permanent Camps – n/a	0	0	0
Sewage / Effluent Treatment Plants	3.2	0	3.2
Admin Offices / Ancillary Buildings	1.6	0	1.6
Bulk Diesel Storage & Distribution	0.2	0	0.2
Plant Mobile Fleet	5.1	0	5.1
Total On-Site Infrastructure Costs	26.1	0	26.1

Source: JDS 2015

21.6.1.1 Water Storage and Treatment

During construction the following tanks will be erected:

- 1 – 12.0 m diameter by 12.0 m tall fresh/firewater carbon steel tank;
- 1 – 3.0 m diameter by 5.0 m tall process water carbon steel tank;
- 1 - 2.4 m diameter by 3.0 m tall gland water carbon steel tank;
- A potable water skid will be used to produce quality water for human consumption at site; and
- All site runoff and DSTF runoff water will be collected and treated prior to being discharged into the environment.

21.6.1.2 Site Power Supply/Fuel Storage

The project requires 7 MW of dedicated and uninterrupted power delivery. This will be accomplished by tying into the existing hydroelectric Sabaneta Dam. The major infrastructure components include the following:

- 1 - 28Kv Powerline stepped down to 4160V complete with cable & equipment;
- 1 - 28Kv to 4160V 5Mva substation;
- 1 - 4160V to 480V 2.5Mva transformers; and
- Significant upgrades to the existing substation at the Sabaneta dam.

Further optimization is possible with further study work on several run of river sites near the project site.

21.6.1.3 Fuel Storage and Distribution

Fuel storage and distribution requirements have been estimated and will be accomplished with the following equipment:

- 1 – 75,000L Fuel tank complete with piping and electrical components.

21.6.1.4 Ancillary Buildings/Camps

The following ancillary buildings are included in the CAPEX estimate:

- Administration building complete with a mine dry, and first aid;
- Assay Laboratory;
- Plant maintenance warehouse; and
- Plant truck shop complete with minor equipment.

The costs of ancillary and support buildings have been estimated based on historical unit rates per area for similar projects. In addition to the building structures, the cost includes the supply of the buildings electrics, fittings, and furnishings. Construction and permanent camps are not required as there are existing facilities on-site and local villages in the area to provide manpower to support the project. Earthworks required for the project have been carried in the overall site development. The total cost for the ancillary building has been estimated at $3.6M.

The cost to supply power and water services to the buildings and camps form part of the water and electrical supply and distribution costs. In addition, reagent storage facilities are included in the process plant cost estimate.

Mobile Equipment

Mobile fleet required to support plant operations is shown below in Table 21.7.

Table 21.7: Plant Support Mobile Equipment CAPEX Estimate

Description	Equipment Count	Total Capital Costs (US$M)
Hydraulic Excavator - Cat 349DL	1	0.4
Skid Steer - Cat 326D	1	0.1
Wheel Loader (910K)	1	0.3
Truck - Dump (10 m3)	1	0.3
F/E Loader (966K) (Dry Stack)	1	0.5
Rock Truck (Dry Stack)	2	1
CAT CS56 - Packer (Dry Stack)	1	0.3
CAT D6 Dozer (Dry Stack)	1	0.3
Motor Grader (Caterpillar 140H)	1	0.2
5 T Flat Deck Truck (HIAB)	1	0
Roll Off Truck	1	0.5
5 T Fork Lift Zoom-Boom - Terex GTH-5519	1	0
65T Rough Terrain Crane	1	0.7
Light Vehicles (Ford F350)	4	0.2
Ambulance	1	0.1
Pipe Fusing Machine (Able to Fuse 28" DR17)	1	0
Diesel Pit Dewatering Pumps	4	0.1
Portable Diesel Light Plants	4	0.1
Total Mobile Equipment Capital Costs	28	5.1

Source: JDS 2015

21.7 Dry Stack Tailings Facility Capital Cost

The Dry Stack Tailings Facility required to support the plant operations is shown below in Table 21.8. The cost estimate, which is deemed accurate to ±40% and suitable for inclusion in the PEA economic model was developed using a combination of first principle costs develop by SRK, and site specific unit rates provided by JDS. This cost estimate excludes the cost of the filter plant, as well as the equipment required to place the tailings. Those costs are included elsewhere in the economic model and Capex estimate.

Table 21.8: Tailings Management Facility Capital Costs

Description	Pre-Production (US$M)	Sustaining/ Closure (US$M)	Total (US$M)
Tailings Management Facility	2.6	6.6	9.2
Total Tailings Management Facility Costs	2.6	6.6	9.2

Source: SRK and JDS 2015

21.8 Indirect Cost Estimate

21.8.1.1 Summary

Indirect costs total an estimated $9.9M, equal to 12.1% of the total direct costs. The various cost centres that comprise the indirect costs are described in the following sections.

21.8.1.2 Heavy Construction Equipment

Heavy Construction Equipment costs have been calculated to be $0.8M, which equates to 1.0% of the direct costs less mining and mobile equipment. Costs are intended to cover an 80T crane and miscellaneous heavy equipment for the duration of the project to support the construction.

21.8.1.3 Field Indirect Costs

Field indirect costs have been calculated to be $3.0M, which equates to 4.0% of the direct costs less mining and mobile equipment. Costs are intended to cover the following:

- Temporary Construction Facilities: work areas and bays, roads, walks and parking areas, temporary buildings, temporary utilities for power and sewage, other minor temporary construction.
- Construction Services: general and final clean-up, material handling and warehousing, craft training and testing, onsite services (soils exploration and soil testing, all labour and material costs, concrete testing and security), operation and maintenance of temporary facilities, surveying, pre-operational testing and start-up.

21.8.1.4 Freight and Logistics

Freight and logistics have been calculated to be $2.9M, which equates to 7.0% of the equipment and material costs less mining equipment. Costs include ocean freight and inland freight, this figure is based on factored historical data for similar projects in Latin America.

21.8.1.5 Vendor Representatives

Vendor representatives have been calculated to be $0.6M, which equates to 2.0% of the equipment and material costs less mining and mobile equipment. This figure is based on factored historical data for similar projects in Latin America.

21.8.1.6 Start-Up and Commissioning / Capital Spares

Start-Up & commissioning/capital spares have been calculated to be $2.0M, which equates to a combined 7.0% of the equipment and material costs less mining and mobile equipment. This figure is based on factored historical data for similar projects in Latin America.

21.8.1.7 First Fills

First fills have been calculated to be $0.6M, which equates to 2.0% of the equipment and material costs less mining and mobile equipment. This figure is based on factored historical data for similar projects.

21.9 EPCM

EPCM services have been calculated to be $12.7M or 14.0% of the direct and indirect costs, which includes detailed engineering, procurement, project management and home office services as well as construction management. This was calculated on direct and indirect costs excluding mine equipment and mine development.

21.10 Owners Cost Estimate

For the purpose of the PEA estimate, $3.0M or 3.1% of the total direct, indirect, and EPCM costs were selected to cover the Owner's Costs. Owner's costs include Insurance, Owner's team costs, pre-production, and Project development. This figure is based on factored historical data for similar projects in Latin America.

21.11 Project Sustaining Capital and Closure Cost Estimate

Ongoing capital requirements for the mine production period totals $92.3M over the life of mine. This cost covers the construction of the tailings management facility and closure costs to sustain the ongoing operation of the project. The Sustaining and Capital and Closure costs required for the project are shown below in Table 21.9.

In addition, sustaining capital is required for mining and the dry stack tailings facility throughout the life of mine.

Table 21.9: Sustaining / Closure Cost Summary

Description	Sustaining / Closure (US$M)
Year 1	28.1
Year 2	4.4
Year 3	4.2
Year 4	2.3
Year 5	3.8
Year 6	6.1
Year 7	5.9
Year 8	7.0
Year 9	5.0
Year 10	13.4
Year 11	12.1
Total Sustaining Costs	**92.3**

Source: JDS 2015

21.12 Contingency

For the purpose of the PEA estimate, $26.5M or 20.0% of the total direct, indirect, EPCM, and Owner's costs.

The contingency reflects the potential growth in CAPEX within the same scope of work. The contingency includes variations in quantities, differences between estimated and actual equipment and material prices, labour costs and site-specific conditions. It also accounts for variation resulting from uncertainties that are clarified during detail engineering, when designs and specifications of the basic engineering scope are finalized.

Contingency is an amount of money allowed in an estimate for cost which, based on past experience, are likely to be encountered, but are difficult or impossible to identify at the time the estimate is prepared. It is an amount expected to be expended during the course of the project. Contingency does not include scope changes, force majeure, labour disruptions or lack of labour availability.

21.13 Duties and Taxes

Local taxes on contractor-supplied materials and installation labour are not included in the estimate.

21.14 Escalation

No escalation costs have been included in the project, all costs and prices are expressed in Q2 2015 US dollars.

22 Operating Cost Estimate

22.1 Operating Cost Summary

Operating costs in this section of the report include mining, processing, tailings, and administration up to the production of concentrate from the site. Mine operating costs incurred during the construction phase (pre-production Years -2 and -1) are capitalized and form part of the capital cost estimate. Concentrate transportation, treatment and refining charges, and royalties are discussed in Section 22.

The operating cost estimate is broken into four major sections:

- Mining;
- Processing;
- Tailings; and
- General & Administrative (G&A).

The operating cost estimate is based on a combination of experience, reference projects, budgetary quotes, first principle calculations and factors as appropriate with a preliminary economic assessment study. All consumable costs include a 18% value add tax (VAT). Power has been assumed at $0.15/kWh.

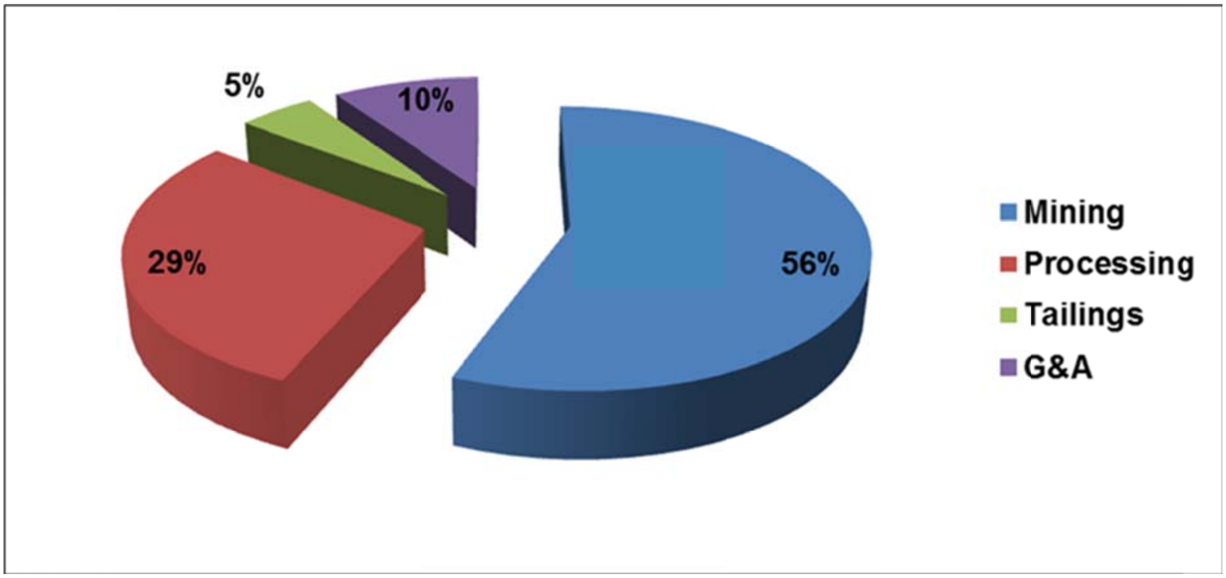
The total operating unit cost is $52.78/t processed. Life of Mine operating costs and total unit costs are summarized in Table 22.1. Figure 22.1 shows the distribution of unit operating costs.

Table 22.1: Breakdown of Estimated Operating Costs

Operating Costs	$/t milled	LOM M$
Mining	29.61	299.0
Processing	15.53	120.2
Tailings	2.64	20.5
G&A	5.00	38.7
Total	**52.78**	408.3

Source: JDS 2015

Figure 22.1: Unit Operating Cost Distribution



Source: JDS 2015

22.2 Mine Operating Cost Estimate

Mine operating costs were built using a combination of first principals engineering and scaling from other similar level studies JDS has prepared for Canadian underground operations. Mine operating costs are summarized below in Table 21.2. Costs are subdivided into operating categories.

Table 22.2: Mining Operating Cost by Category

Category	$/tonne processed
Labour	8.97
Fuel	1.74
Equipment (maintenance, oil/lube, tires) & Power	6.39
Consumables (drilling, ground support, electrical, ventilation, pipe, cement and explosives)	12.51
Grand Total by Activity	**29.61**

Source: JDS 2015

Capitalized development summarized in Table 21.2 equates to $3.85/tonne processed.

Mining Labour is summarized in Table 22.3 and includes the following burdens to account for in-country benefits, training, production bonus and potential ex-patriot benefits & costs:

- Staff: Years -1 to 3: 50%, after year 3: 40%; and
- Houry: Years -1 to 3: 75%, after year 3: 50%.

22.3 Processing Operating Cost Estimate

Process operating costs were developed using labour rates as provided from GoldQuest and sufficient personnel to operate the process plant, factored maintenance cost, budget quotes for consumables and a factored power requirement. . Process operating costs are summarized below in Table 22.3. Costs are subdivided into operating categories.

Table 22.3: Processing Operating Cost by Category

Category	$/tonne processed
Labour	3.37
Equipment Maintenance & Consumables (Reagents, Media, Liners and other Wear Parts)	5.33
Power	6.83
Grand Total by Activity	**15.53**

Source: JDS 2015

Process Labour includes 1.35 burden for salaried employees and 1.3 burden for hourly employees to account for in-country benefits, training, production bonus and potential ex-patriot benefits & costs.

Equipment maintenance was calculated by applying a factor of 3.5 to major process equipment cost. Costs for media were determined using engineering calculations based on mill power draw. Reagents requirements from recent testwork and budget quotes from vendors were used to calculate the cost of reagents. Mill liners and wear parts for major equipment were based on vendor recommended requirements and quotes.

Power costs were determined using 80% of installed power for major equipment.assuming $0.15/kWh.

22.4 General and Administration Operating Cost Estimate

The average G&A operating cost for the supporting facilities and administration for a typical year are estimated to be $4.6M per year (or $5/t processed). These costs are assumed to consist of both fixed costs, independent of plant throughput or mining rate, and partially variable, changing in direct proportion to the plant throughput rate. The G&A costs are summarized in Table 22.4.

Table 22.4: G&A Operating Cost by Category

Category	$/tonne processed	Comments/Description
Labour	1.51	All labour except contractors
Services	3.49	Support Equipment, Legal, IT, Transportation for employees, Freight, etc.
Total	**5.00**	

Source: JDS 2015

22.4.1.1 G&A Services

G&A Sevices amount to $3.2M/yr and are summarized in Table 22.5.

Table 22.5: G&A Services

G&A Services	$M/Yr	$/t processed
Equipment	0.7	0.79
Health Safety, Medicals & First Aid	0.1	0.11
Environmental	0.1	0.08
Office Admin Supplies	0.1	0.08
Legal & Insurance	0.3	0.33
Recruitment/training /safety programs	0.1	0.11
Consultants General	0.2	0.22
Community Involvment	0.1	0.11
IT & Communications	0.2	0.17
Water Treatment	0.3	0.27
Freight	0.8	0.82
Transportation Employees	0.4	0.39
Total G&A Services	**3.2**	**3.48**

Source: JDS 2015

23 Economic Analysis

23.1 Summary

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal prices, head grades, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment of skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 of this report (presented in 2015 dollars). The economic analysis has been run with no inflation (constant dollar basis).

It must be noted that this PEA is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the results of the preliminary economic assessment will be realized.

23.2 Basis of Analysis

One metal price scenario was utilized to prepare the economic analysis. However, a sensitivity analysis on the metal prices was completed and is outlined in Section 23.6.

All costs, metal prices and economic results are reported in US dollars (US$ or $) unless stated otherwise. LOM plan tonnage and grade estimates are demonstrated in Table 23.1.

.

Table 23.1: Life of Mine Plan Summary

Summary of Results	Unit	Value
Indicated Material	M tonnes	6.6
Inferred Material	M tonnes	1.1
Total Mill Feed	M tonnes	7.7
Indicated		
Cu	%	0.83
Au	g/t	4.1
Ag	g/t	4.1
Inferred		
Cu	%	0.67
Au	g/t	3.6
Ag	g/t	5.2

Source: JDS 2015

23.3 Assumptions

The following economic assumptions were used in the economic analysis:

- Discount rate of 6% (sensitivities using other discount rates have been calculated) - refer to Section 23.6;
- Closure cost of $19M was considered;
- Nominal 2015 US dollars;
- Revenues, costs and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Working capital was calculated as 2-months of operating costs (mining, processing, tailings management, G&A) in Year 1 (assumed to be required in Year -1). The working capital is recuperated during the last year of production (Year 10);
- Results are presented on a 100% equity basis; and
- No management fees or financing costs have been considered.

The economic analysis excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.).

Table 23.2 outlines the metal price assumption used in the economic analysis. The reader is cautioned that the metal prices used in this study are only estimates based on recent historical performance and there no guarantee that they will be realized if the project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 23.2: Metal Prices used in the Economic Analysis

Metal Price & F/X Rate	Unit	Value
Cu Price	US$/lb	2.9
Au Price	US$/oz	1,225
Ag Price	US$/oz	17
F/X Rate	US$:C$	0.8

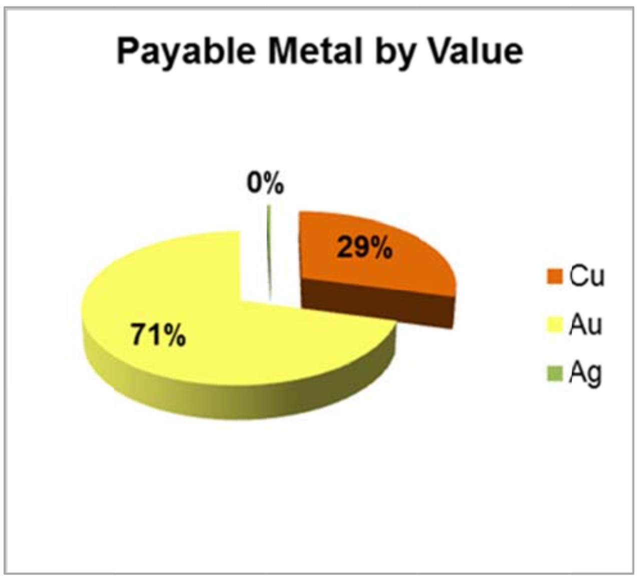
Source: JDS 2015

23.4 Revenues

Mine revenue is derived from the sale of copper concentrate into the international marketplace. No contractual arragements for refining exist at this time. Details regarding the terms used for the economic analysis can be found in the Market Studies Section 19 of this report. Figure 23.1 demonstrates the revenues by metal.

Total smelter revenues amount to (net of royalties) $1,174M over the 10 year mine life.

Figure 23.1: Payable Metal by Value



Source: JDS 2015

23.5 Taxes

The project has been evaluated on an after-tax basis to provide a more indicative value of the potential project economics. High-level tax assumptions were considered in order to calculate approximate annual taxes payable. The assumptions used were based on the known tax regime in the jurisdiction. Total taxes for the project amount to $187M.

The following assumptions were used in the preparation of the tax calculations for the Romero project and used in the economic model:

- Tax calculations are based on 100% ownership of the Romero project;
- All taxes are paid in the year incurred;
- Withholding taxes on repatriation to Canadian parent company are not considered;
- All sales are recognized in year of production;
- Cash requirements to fund the project are provided by equity;
- A 15% declining balance depreciation method was considered on all capital expenditures beginning in Y-2;
- Net asset tax of 0.5%;
- Corporate income tax of 27%;
- Maximum of 20% loss carryforward per year;
- Export withholding tax of 5%; and
- Local community tax of 5%.

23.6 Results

The reader is cautioned that this PEA is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves and, as such, there is no certainty that the PEA economics will be realized. The PEA uses 14% inferred mineralized material.

The project is economically viable with an after-tax internal rate of return (IRR) of 34% and a net present value using a 6% discount rate (NPV6%) of $219M using the Base Case metal prices. Table 21.5 summarizes the economic results of the project.

The break-even gold price for the project (using the Base Case metal prices) is approximately $628/oz, based on LOM presented herein and a copper price of US$2.90/lb.

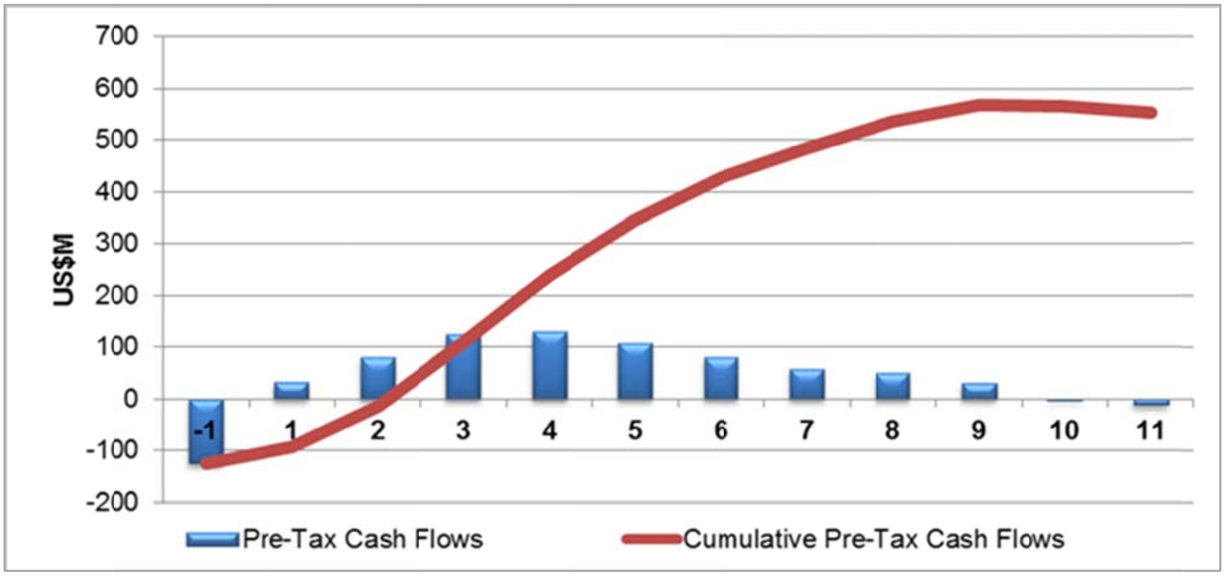
Table 23.3 demonstrates the economic results. Figure 23.2 demonstrates the projected cash flows for the project.

Table 23.3: Summary of Economic Results

Results	Unit	Value
Gross Revenues	US$M LOM	1,174
Total Operating Cost	US$/t milled	52.78
	US$M LOM	408
Net Operating Income	US$M	765
Pre-Production Capital (Incl. Contingency)	US$M	143
Sustaining Capital (Incl. Contingency	US$M	92
Total Capital (Incl. Contingency)	US$M	235
LOM Pre-Tax Free Cash Flow	US$M	530
Average Annual Pre-Tax Free Cash Flow	US$M/yr	58
Pre-Tax $NPV_{6\%}$	US$M	355
Pre-Tax IRR	%	46
Pre-Tax Payback	Years	2.3
NPV to Pre-Production CAPEX	times	2.5
Taxes	US$M	187
LOM After-Tax Free Cash Flow	US$M	343
Average Annual After-Tax Free Cash Flow	US$M/yr	37
After-Tax $NPV_{6\%}$	US$M	219
After-Tax IRR	%	34
After-Tax Payabck	Years	2.7
Break-Even Au Price‡	US$/Au oz	628
Cash Cost*	US$/Au oz	813
Cash Cost Net of By-Products**	US$/Au oz	572

(‡) Based on constant Cu price of US$2.90/lb
(*) Cash Cost = (Treatment Charges + Refining Charges + Royalties + Operating Costs + Sustaining & Closure Capital Costs)/Payable Au oz
(**) Cash Cost Net of By Products = ((Treatment Charges + Refining Charges + Operating Costs + Sustaining & Closure Capital Costs) – (Payable Cu lbs * 2.90/lb) – (Payable Ag oz * $17/oz)) / Payable Au oz
Source: JDS 2015

Figure 23.2: Annual After-Tax Cash Flows



Source: JDS 2015

23.7 Sensitivities

A sensitivity analysis was performed on the Base Case metal pricing scenario to determine which factors most affect the project economics. The analysis revealed that the project is most sensitive to metal prices, followed by head grade and operating costs. The project showed less sensitive to changes in capital costs.

Table 23.4 along with Figure 23.3 outline the results of the sensitivity test performed on the after-tax $NPV_{6\%}$ for the Base Case evaluated.

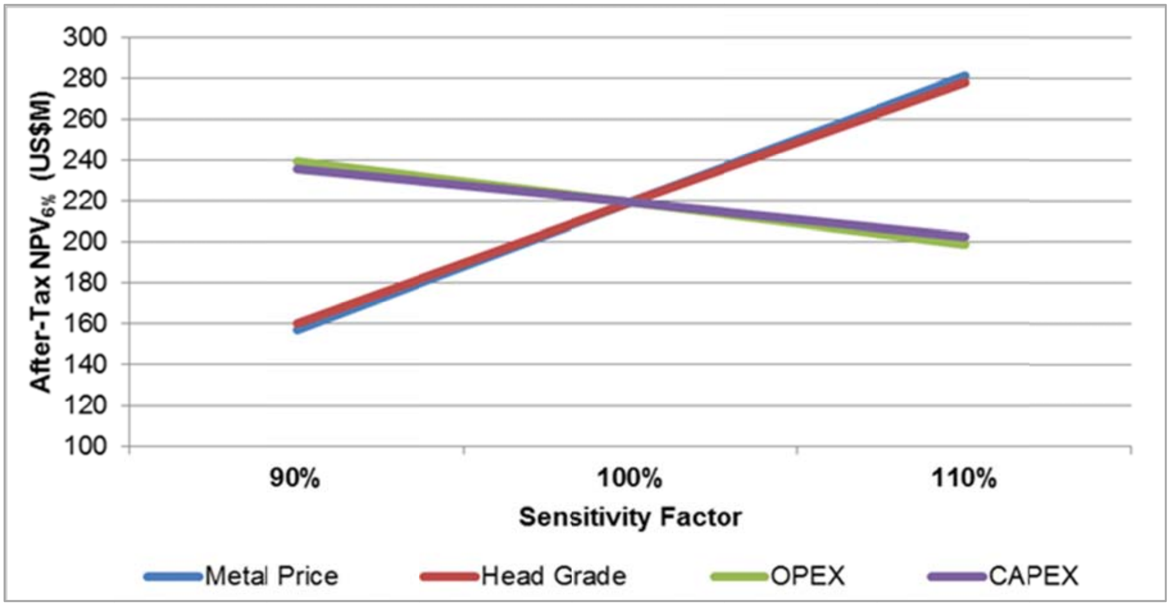
The project was also tested under various discount rates. The results of this sensitivity test are demonstrated in Table 23.5.

Table 23.4: After-Tax Sensitivity Test Results

After-Tax $NPV_{6\%}$ (US$M)			
Variable	90%	100%	110%
Metal Price (Combined)	156.7	219.1	281.6
Cu Price	201.1	219.1	237.2
Au Price	175	219.1	263.3
Head Grade	160.3	219.1	278
OPEX	239.5	219.1	198.8
CAPEX	235.7	219.1	202.6

Source: JDS 2015

Figure 23.3: After-Tax Sensitivity Test Results



Source: JDS 2015

Table 23.5: Discount Rate Sensitivity Test Results

Discount Rate	Pre-Tax NPV (US$M)	After-Tax NPV (US$M)
0%	530	343
5%	379	236
6%	355	219
8%	311	188
10%	272	161

Source: JDS 2015

	Unit	LOM	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Metal Prices																
Cu	US$/lb	**2.90**	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90
Au	US$/oz	**1,225**	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225	1,225
Ag	US$/oz	**17.00**	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Production Schedule																
Underground *(Romero)*																
Indicated Material																
Indicated Mineralized Mined	**M tonnes**	**6.6**				**0.5**	**0.9**	**0.9**	**0.9**	**0.8**	**0.6**	**0.7**	**0.8**	**0.5**	**0.1**	**0.0**
Cu	%	**0.83%**	0.00%	0.00%	0.00%	0.70%	0.89%	0.81%	0.88%	0.86%	0.79%	0.80%	0.77%	0.93%	0.96%	0.00%
Au	g/t	**4.08**	0.00	0.00	0.00	4.50	3.59	5.23	5.12	4.68	4.03	3.27	2.90	3.21	1.79	0.00
Ag	g/t	**4.09**	0.00	0.00	0.00	3.04	3.38	4.70	4.49	4.02	3.96	3.52	4.27	5.03	4.88	0.00
Inferred Material																
Inferrred Mineralized Material	**M tonnes**	**1.1**				**0.2**	**0.1**	**0.0**	**0.0**	**0.1**	**0.3**	**0.3**	**0.2**	**0.1**	**0.0**	**0.0**
Cu	%	**0.67%**	0.00%	0.00%	0.00%	0.70%	0.59%	1.09%	3.55%	0.89%	0.60%	0.65%	0.56%	0.61%	0.59%	0.00%
Au	g/t	**3.65**	0.00	0.00	0.00	3.32	6.08	8.16	14.44	4.03	3.83	3.02	3.32	1.86	1.83	0.00
Ag	g/t	**5.17**	0.00	0.00	0.00	3.36	3.03	7.67	22.72	9.79	7.07	4.24	3.33	4.57	3.52	0.00
Total Underground Mineralized Material	**M tonnes**	**7.7**	**0.0**	**0.0**	**0.0**	**0.6**	**0.9**	**0.9**	**0.9**	**0.9**	**0.9**	**0.9**	**0.9**	**0.5**	**0.2**	**0.0**
Total Underground Mining Rate	tpd	**2,314**	0	0	0	1,682	2,500	2,500	2,500	2,500	2,500	2,500	2,500	1,479	536	0
Total Mine Production																
Total Mineralized Material	M tonnes	**8**	0.0	0.0	0.0	0.6	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.5	0.2	0.0
Mine Production Rate	tpd	**2,314**	0	0	0	1,682	2,500	2,500	2,500	2,500	2,500	2,500	2,500	1,479	536	0
Milling Schedule																
Total Mineralized Material	tonnes	**7.7**	0.0	0.0	0.0	0.6	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.5	0.2	0.0
Plant Throughput	tpd	**2,314**	0	0	0	1,682	2,500	2,500	2,500	2,500	2,500	2,500	2,500	1,479	536	0
Head Grades																
Cu	%	**0.81%**	0.00%	0.00%	0.00%	0.70%	0.87%	0.82%	0.91%	0.86%	0.73%	0.76%	0.74%	0.90%	0.87%	0.00%
Au	g/t	**4.02**	0.00	0.00	0.00	4.21	3.75	5.33	5.21	4.63	3.97	3.20	2.97	3.06	1.80	0.00
Ag	g/t	**4.25**	0.00	0.00	0.00	3.12	3.36	4.81	4.66	4.45	4.89	3.73	4.10	4.98	4.56	0.00
Au Equiv	g/t	**5.39**	0.00	0.00	0.00	5.38	5.21	6.73	6.74	6.10	5.22	4.47	4.23	4.58	3.28	0.00
Recovery to Cu Concentrate																
Overall Recoveries																
Overall Recovery	Cu %	**96.8%**	0.0%	0.0%	0.0%	96.8%	96.8%	96.8%	96.8%	96.8%	96.8%	96.8%	96.8%	96.8%	96.8%	0.0%
	Au %	**75.0%**	0.0%	0.0%	0.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	0.0%
	Ag %	**49.8%**	0.0%	0.0%	0.0%	49.8%	49.8%	49.8%	49.8%	49.8%	49.8%	49.8%	49.8%	49.8%	49.8%	0.0%
Metal in Concentrate	Cu M lbs	**134**	0.0	0.0	0.0	9.1	16.9	16.0	17.7	16.8	14.2	14.7	14.3	10.3	3.6	0.0
	Au k oz	**750**	0.0	0.0	0.0	62.3	82.6	117.3	114.6	101.9	87.3	70.3	65.4	39.8	8.5	0.0
	Ag k oz	**526**	0	0	0	31	49	70	68	65	71	54	60	43	14	0
Cu Concentrate Grade	Cu %	**20%**	20.0%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%
	Au g/t	**76.9**	0.0	0.0	0.0	93.7	66.8	100.4	89.0	83.1	84.3	65.5	62.6	52.9	32.0	0.0
	Ag g/t	**53.9**	0.0	0.0	0.0	46.1	39.7	60.1	52.8	52.9	68.9	50.7	57.3	57.2	53.8	0.0
Cu Concentrate Produced	k dmt	**303**	0	0	0	21	38	36	40	38	32	33	33	23	8	0
	k wmt	**330**	0	0	0	22	42	40	44	41	35	36	35	25	9	0
Pull Factor		**25**	0	0	0	30	24	25	23	24	28	27	28	23	24	0
Payable Cu in Cu Concentrate	M lbs	**127**	0	0	0	9	16	15	17	16	13	14	14	10	3	0
	M US$	**369**	0	0	0	25	47	44	49	46	39	41	39	28	10	0
Au Payable in Cu Conc	k oz	**726**	0	0	0	60	80	114	111	99	84	68	63	38	8	0
	M US$	**889**	0.0	0.0	0.0	73.9	97.7	139.3	135.9	120.9	103.5	83.2	77.4	47.0	10.0	0.0
Ag Payable in Cu Conc	k oz	**298**	0	0	0	16	22	42	38	36	46	30	35	25	8	0
	M US$	**5.1**	0.0	0.0	0.0	0.3	0.4	0.7	0.6	0.6	0.8	0.5	0.6	0.4	0.1	0.0
Cu TC	M US$	**25.8**	0.0	0.0	0.0	1.8	3.3	3.1	3.4	3.2	2.7	2.8	2.8	2.0	0.7	0.0
Cu RC	M US$	**10.8**	0.0	0.0	0.0	0.7	1.4	1.3	1.4	1.4	1.1	1.2	1.2	0.8	0.3	0.0
Au RC	M US$	**4.4**	0.0	0.0	0.0	0.4	0.5	0.7	0.7	0.6	0.5	0.4	0.4	0.2	0.0	0.0
Ag RC	M US$	**0.1**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Concentrate Transportation	M US$	**33.0**	0.0	0.0	0.0	2.2	4.2	4.0	4.4	4.1	3.5	3.6	3.5	2.5	0.9	0.0
Cu Conc NSR	**M US$**	**1,188.5**	**0.0**	**0.0**	**0.0**	**94.1**	**135.5**	**175.1**	**175.4**	**158.5**	**135.5**	**116.2**	**109.6**	**70.3**	**18.2**	**0.0**
NSR Royalty	US$	**14.9**	0.0	0.0	0.0	1.2	1.7	2.2	2.2	2.0	1.7	1.5	1.4	0.9	0.2	0.0
NSR After-Royalties	**US$**	**1,173.6**	0.0	0.0	0.0	93.0	133.8	172.9	173.2	156.5	133.8	114.7	108.3	69.4	18.0	0.0
	US$/t milled	**151.68**	0.00	0.00	0.00	151.43	146.62	189.51	189.78	171.53	146.63	125.73	118.65	128.59	91.83	0.00
OPEX																
Mining	US$/t milled	**29.60**	0.00	0.00	0.00	28.55	31.75	26.41	22.70	26.28	28.11	32.02	32.60	29.91	45.03	0.00
	US$	**229.0**	0.0	0.0	0.0	17.5	29.0	24.1	20.7	24.0	25.7	29.2	29.7	20.3	8.8	0.0
Processing	US$/t milled	**15.53**	15.53	15.53	15.53	15.53	15.53	15.53	15.53	15.53	15.53	15.53	15.53	15.53	15.53	15.53
	US$	**120.2**	0.0	0.0	0.0	9.5	14.2	14.2	14.2	14.2	14.2	14.2	14.2	8.4	3.0	0.0
Tailings Management Facility	US$/m^3	**8.15**	8.15	8.15	8.15	8.15	8.15	8.15	8.15	8.15	8.15	8.15	8.15	8.15	8.15	8.15
	m^3	**2,510,107**				281,471	286,953	282,492	286,846	377,877	358,739	270,104	211,655	137,267	16,702	0
	US$	**20.5**	0.0	0.0	0.0	2.3	2.3	2.3	2.3	3.1	2.9	2.2	1.7	1.1	0.1	0.0
G&A	US$/t milled	**5.00**	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
	US$	**38.7**	0.0	0.0	0.0	3.1	4.6	4.6	4.6	4.6	4.6	4.6	4.6	2.7	1.0	0.0
Total OPEX	**US$/t milled**	**52.78**	**0.00**	**0.00**	**0.00**	**52.82**	**54.84**	**49.47**	**45.80**	**50.19**	**51.85**	**54.97**	**55.03**	**60.19**	**66.26**	**0.00**
	US$	**408.3**	**0.0**	**0.0**	**0.0**	**32.4**	**50.0**	**45.1**	**41.8**	**45.8**	**47.3**	**50.2**	**50.2**	**32.5**	**13.0**	**0.0**
Net Operating Income	**US$**	**765.2**	**0.0**	**0.0**	**0.0**	**60.5**	**83.7**	**127.8**	**131.4**	**110.7**	**86.5**	**64.6**	**58.1**	**36.9**	**5.0**	**0.0**
Au Cash Cost	**US$/oz**	**685**	**0**	**0**	**0**	**642**	**765**	**496**	**485**	**579**	**674**	**879**	**941**	**1,015**	**1,862**	**0**
Au Cash Cost (Net of BP)	**US$/oz**	**170**	**0**	**0**	**0**	**221**	**175**	**101**	**41**	**103**	**201**	**274**	**306**	**263**	**610**	**0**
CAPEX																
Mining	US$	**76**	0.0	0.0	14.9	27.2	3.9	3.7	1.8	3.2	5.5	5.1	6.0	3.9	1.2	0.0
Site Development	US$	**10**	0.0	2.0	7.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Crushing & Handling	US$	**7**	0.0	0.0	7.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Processing Plant	US$	**36**	0.0	0.0	35.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
On-Site Infrastructure	US$	**26**	0.0	12.8	13.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Tailings Management Facility	US$	**9**	0.0	0.0	2.6	0.8	0.4	0.4	0.4	0.5	0.5	0.6	0.8	0.9	0.2	1.1
Indirect Costs	US$	**10**	0.0	1.8	8.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
EPCM	US$	**13**	0.0	2.3	10.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Owner's Costs	US$	**3**	0.0	0.6	2.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Closure	US$	**19**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	10.0	9.0
Subtotal	**US$**	**209**	**0.0**	**19.4**	**102.3**	**28.0**	**4.3**	**4.1**	**2.2**	**3.7**	**6.0**	**5.7**	**6.8**	**4.9**	**11.4**	**10.1**
Contingency	US$	**26**	0.0	3.9	17.5	0.2	0.1	0.1	0.1	0.1	0.1	0.1	0.2	0.2	2.0	2.0
Total CAPEX	**US$**	**235**	**0.0**	**23.2**	**119.8**	**28.1**	**4.4**	**4.2**	**2.3**	**3.8**	**6.1**	**5.9**	**7.0**	**5.0**	**13.4**	**12.1**
Pre-Production	*US$*	*143*	*0.0*	*23.2*	*119.8*	*0.0*	*0.0*	*0.0*	*0.0*	*0.0*	*0.0*	*0.0*	*0.0*	*0.0*	*0.0*	*0.0*
Production	*US$*	*92*	*0.0*	*0.0*	*0.0*	*28.1*	*4.4*	*4.2*	*2.3*	*3.8*	*6.1*	*5.9*	*7.0*	*5.0*	*13.4*	*12.1*
Working Capital	**US$**	**0**			**5.4**										**-5.4**	
Royalty Buyout Option	**US$**	**0**			**0.0**											
Net Cash Flow	US$	**530**	0.0	-23.2	-125.2	32.4	79.4	123.6	129.1	106.9	80.4	58.7	51.1	31.9	-3.0	-12.1
Cumulative Cash Flow	US$		0.0	-23.2	-148.5	-116.1	-36.7	86.9	216.0	323.0	403.4	462.1	513.1	545.0	542.0	529.9
Taxes	**US$**	**187**		**0.1**	**0.6**	**10.2**	**18.1**	**32.3**	**34.1**	**28.5**	**22.8**	**16.3**	**14.5**	**8.3**	**1.2**	
Net After-Tax Cash Flow	US$	**343**	0.0	-23.3	-125.8	22.2	61.2	91.2	95.0	78.4	57.6	42.4	36.5	23.6	-4.2	-12.1
Cumulative After-Tax Cash Flow	US$		0.0	-23.3	-149.2	-127.0	-65.7	25.5	120.5	198.9	256.6	298.9	335.5	359.0	354.8	342.7

24 Adjacent Properties

There are no adjacent properties whose description directly or materially affects the opinion offered in this Technical Report. Unigold Inc.'s Neita project is found approximately 45 km along strike from Romero to the west-northwest. Unigold recently announced a mineral resource estimate for the project.

25 Other Relevant Data and Information

There is no other relevant data or information for this report.

26 Interpretations and Conclusions

Industry standard mining and processing methods were used in this report. Sufficient information and data was available to the QPs for a PEA-level study and the goal of producing a PEA study, prepared in accordance with 43-101 guidelines, was achieved. The preliminary economic results, based on the assumptions highlighted in this report, show a positive outcome.

It is important to note that this result is only preliminary and could change significantly as more information is gathered and market conditions change. This assessment includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

26.1 Risks

As with almost all mining ventures, there are a large number of risks and opportunities that can influence the outcome of the Romero project. Most of the risks are based on a lack of scientific information (test results, drill results, etc.) or the lack of control over external drivers (metal price, exchange rates, etc.). The following section identifies the most significant potential risks currently known for the Romero project, almost all of which are common to mining projects at this early stage of project development.

Subsequent higher-level engineering studies would be needed to further refine these risks and opportunities, identify new ones, and define mitigation or opportunity implementation plans. While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the Romero project.

Table 26.1 identifies what are currently deemed to be the most significant internal project risks, potential impacts, and possible mitigation approaches. The most significant potential risks associated with the Romero project are the ability to convert inferred resources to indicated and measured, unplanned dilution, lower metal recoveries than those projected, operating and capital cost escalation, ability to attract and retain competent mining and technical personnel, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal prices. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

External risks are, to a certain extent, beyond the control of the project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the project region, smelting and refining terms, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource estimates.

Table 26.1 Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Dilution	Higher than expected dilution has a severe impact on project economics. The mine must ensure accurate drilling and blasting practices are maintained to minimize dilution from wall rock backfill and other mineralized zones, minimize secondary breaking and optimize extraction. The ability to segregate higher grade material, early in the mine life, is critical to project economics.	A well planned and executed grade control plan is necessary immediately upon commencement of mining.
Resource Modelling	All mineral resource estimates carry some risk and are one of the most common issues with project success. 15% of the resources in the mine plan are Inferred.	Infill drilling may be recommended in order to provide a greater level of confidence in the resource.
Metallurgical Recoveries	Negative changes to metallurgical assumptions could lead to reduced metal recovery, increased processing costs, and/or changes to the processing circuit design. If LOM metal recovery is lower than assumed, the project economics would be negatively impacted.	Additional sampling and test work is needed at the next level of study.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success. If OPEX increases then the NSR cut-off would increase and, all else being equal, the size of the mineable resource would reduce yielding fewer mineable tonnes.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
Permit Acquisition	The ability to secure all of the permits to build and operate the project is of paramount importance. Failure to secure the necessary permits could stop or delay the project.	The development of close relationships with the local communities and government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required. Maintain direct control with a clear solution.
Development Schedule	The project development could be delayed for a number of reasons and could impact project economics. A change in schedule would alter the project economics.	If an aggressive schedule is to be followed, PFS field work should begin as soon as possible.
Ability to Attract Experienced Professionals	The ability to attract and retain competent, experienced professionals, especially UG miners is a key success factor for the project. High turnover or the lack of appropriate technical and management staff at the project could result in difficulties meeting project goals.	The early search for professionals as well as competitive salaries and benefits identify, attract and retain critical people.

Source: JDS 2015

26.2 Opportunities

Table 26.2 identifies what are currently deemed to be the most significant opportunities for the Romero project and their potential benefit. The most significant potential opportunities associated with the Romero project are extended mine life through the inclusion of the Romero South deposit, run of river hydroelectric power generation, improved metallurgical recoveries, and reduced cement content required for paste backfill.

Table 26.2: Identified Project Opportunities

Risk	Explanation/Potential Impact	Potential Benefit
Expansion of the Mine	The mineral resource has not been fully delineated and there is an opportunity to expand the mineable resource. The Romero South deposit is not in the mine and contains a combined Indicated & Inferred resource of over 3.6Mt at $50/t NSR cut-off	Increased mine life
Run of River Hydro-electric Power Generation	Several potential run of river hydro-electric power generation sites have been identified on the property	Hydro-electric power generation has the potential to reduce operating costs during high river flow periods. Capex may be able to be offset by tax deductions and their may be a potential to sell excess power by to the Dominican grid
Metallurgical Recoveries	Improvements could potentially be made to process recoveries and/or concentrate grade and marketability	The NPV of the project may be improved with optimization of metallurgical recoveries and concentrate grade. The sensitivity of the project with respect to changes to process recovery is similar to the project's sensitivity to changes in processed head grades which has been included in the sensitivity analysis of this PEA
Backfill Cement Content	Paste backfill testing may reduce the 4% cement content assumption	Reduce mining costs

Source: JDS 2015

27 Recommendations

It is recommended that Romero proceed to the preliminary feasibility study stage in line with GoldQuest's desire to advance the project. It is also recommended that environmental and permitting continue as needed to support Romero project development plans.

It is estimated that a PFS and supporting field work would cost approximately $3.1 M. A breakdown of the key components of the next study phase is as follows in Table 27.1.

Table 27.1: Cost Estimate to Advance Romero to PFS Stage

Component	Estimated Cost (M$)	Comment
Resource Drilling & Updated Resource	0.6	Conversion of inferred resources to indicated within and immediately adjacent to the proposed mine. Drilling will include holes for combined resource, geotech and metallurgical purposes
Metallurgical Testing	0.2	Variability test work including expanded comminutionun, grinding, flotation and filtration testwork as well as multielement ICP tailings and concentrate analysis
Access Road	0.1	Reconnaissance, test pitting, borrow source indentification and road design
Backfill Testing	0.1	Paste backfill testing including tailings characterization, rheology, strength tests
Geotechnical/ Hydrology/Hydrogeology	0.5	Mine and surface facilities geotechnical investigations (logging, test pitting, sampling, lab tests, etc.)
Engineering & Design	1.5	PFS-level mine, infrastructure, tailings storage, paste backfill & process design, cost estimation, scheduling & economic analysis
Environment	0.1	Other investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology
Total	**3.1**	Excludes corporate overheads and future permitting activities

Source: JDS 2015

Further details on recommendations not mentioned in Table 27.1 are found in the next sections.

27.1 Recommended Work Programs

27.1.1 Metallurgical and Processing

The flowsheet developed from recent testwork is based on primary grind of 75 microns with a regrind P80= 23 microns to produce a 20% copper concentrate with a recovery of approximately 97%. With gravity and flotation the final concentrate will include 75% gold and 49.8% silver.

In the next phase of study, the number of metallurgical samples required to better define the Romero property should include composites from the first 3-years of operation by rock type and variability samples of varying grades. From this testwork, Goldquest can proceed with some confidence towards a full-scale pre-feasibility or feasibility level study.

Engineering work should include:

- Trade-off study for tailings disposal methods;
- Updated design criteria based on testwork to confirm flowsheet with new samples;
- Detailed mass and process water balance calculations;
- Equipment sizing and specifications;
- Detailed operational and capital cost estimates; and
- Flowsheets for each unit operation.

Further studies should include:

- Perform additional tests at a coarser primary grind at higher K80 to investigate opportunities to reduce power requirements with higher rougher recovery;
- Complete trade-off study to compare gold recovery vs the cost higher regrind energy at the higher rougher mass pull;
- Adjust rougher flotation conditions to try and improve recovery with lower masspulls to reduce equipment sizes;
- Look at regrind energy requirements with further test work to confirm the results for variability samples and composites representative of the first three years of the LOM;
- Complete trade off study to determine the best recovery vs. concentrate grade ratio economically (logistics/trucking and smelting terms);
- Investigate opportunities to recovery more gold in the later stages of flotation circuits using more selective reagents regimes; and
- Carry out additional test work on Romero South to better define a flowsheet with improved grade and recoveries to produce a saleable concentrate and investigate alternate recovery methods.

27.1.1.1 Recommended Testwork

The following metallurgical testing programs are recommended:

- Grinding and work crushing indices;
- SMC, CWi, abrasion testing;
- Gravity concentration testing with Knelson or Falcon benchtop machines;
- Lock-cycle Flotation testwork;
- Mineralogical studies and primary grind characterizations;
- Flotation optimization;
- Regrind studies;
- Thickening, filtration, rheology parameters; and
- Tailings analysis for paste backfill and dry stack.

27.1.2 Underground & Surface Geotechnical and Hydrogeology

As the Romero Project advances to the pre-feasibility study (PFS) level of design, geotechnical specific drilling, test pitting and rock testing and engineering will be required to support stope design and drift ground support requirements. The following geotechnical work is recommended:

- A full geotechnical characterization program including geotechnical specific core drilling and discontinuity orientation, laboratory strength testing of core samples and engineering to support PFS stope and ground support;
- Geotechnical characterization of shallow foundation materials beneath the proposed facilities and tailings management facility;
- Measurement of static water levels while drilling; and
- Packer and injection testing and possibly 1-2 monitoring wells.

27.1.3 Paste Backfill

- Tailings material characterization including particle size distribution, specific gravity, mineralogy, pH and solids concentration;
- Rheology testing;
- Floculant screening;
- Thickener feed dilution characterization;
- Vacuum and disc filtration testing; and
- Backfill strength testing to determine optimum cement content for the required backfill strength.

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29 Units of Measure, Abbreviations and Acronyms

above mean sea level	amsl
actual cubic feet per minute	Acfm
ampere	A
annum (year)	a
bed volumes per hour	BV/h
billion	B
billion tonnes	Bt
billion years ago	bya
British thermal unit	BTU
centimetre	cm
centipoise	cP
Concentration of hydrogen in ion (level of scidity)	pH
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 metres per hour	$m^{3/}h$
cubic metres per second	m^3/s
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)	C$
dry metric ton	dmt
foot	ft
gallon (US)	gal
gallons per minute (US)	gpm
Gigajoule	GJ
gigapascal	GPa
gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
hectare (10,000 m2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk

hours per year	h/a
hydraulic conductivity	K
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m3
kilograms per hour	kg/h
kilograms per square metre	kg/m2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
litre	L
litres per minute	L/min
litres per second	L/s
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above mean sea level	mamsl
metres below ground surface	mbgs
metres below sea level	mbsl
metres per minute	m/min
metres per second	m/s
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
Normal cubic metres per hour	Nm^3/h
parts per billion	ppb
parts per million	ppm
pascal	Pa
pounds per square inch	psi
percent by weight	wt%
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
standard cubic feet per minute	Scfm
tonne (1,000 kg) (metric ton)	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
Troy ounce	oz
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt

APPENDIX A – QP CERTIFICATES

CERTIFICATE OF AUTHOR

I, Michael E. Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report on the Romero Project, Dominican Republic", with an effective date of April 29, 2015, (the "Technical Report") prepared for GoldQuest Mining Corp.;

2. I am currently employed as a Senior Project Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a graduate of the University of Alberta with a B.Sc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

4. I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over seven years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

7. I visited the Romero project April 6-18, 2015;

8. I am responsible for Sections 1, 2, 3, 15, 16, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27, 28 and 29 of this Technical Report;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

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

11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: April 29, 2015
Signing Date: June 2, 2015

(original signed and sealed) "Michael E. Makarenko, P.Eng."

Michael E. Makarenko, P. Eng.

CERTIFICATE OF AUTHOR

I, Kelly Shea McLeod, do hereby certify that:
This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report on the Romero Project, Dominican Republic", with an effective date of April 29, 2015 (the "Technical Report") prepared for GoldQuest Mining Corp.;

1. I am currently employed as a Senior Engineer Metallurgy, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

2. I am a graduate of McMaster University with a Bachelor's of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984;

3. I am a Registered Professional Mining Engineer in British Columbia;

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and engineering and mineral processing design, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

5. I have not visited the Romero Project site;

6. I am responsible for sections 13 and 17, and share responsibility for Sections 23 and 27 of the Technical Report.

7. I have had no prior involvement with the property that is the subject of this Technical Report;

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

9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective date: April 29, 2015
Signing date June 2, 2015

(Original signed and sealed) "Kelly S McLeod, P.Eng."

Kelly S McLeod, P.Eng.

CERTIFICATE

B. Terrence Hennessey, P.Geo.

As co-author of this report on certain mineral properties of GoldQuest Mining Corp. which are located in San Juan province, Dominican Republic, I, B. Terrence Hennessey, P.Geo., do hereby certify that:

1. I am employed as a senior geologist and Vice President by, and carried out this assignment for:

   Micon International Limited
   Suite 900, 390 Bay Street
   Toronto, Ontario
   M5H 2Y2

   tel. (416) 362-5135
   fax (416) 362-5763
   e-mail: thennessey@micon-international.com

2. I hold the following academic qualifications:

B.Sc. (Geology)	McMaster University	1978

3. I am a registered Professional Geoscientist with the Association of Professional Geoscientists of Ontario (membership # 0038); as well, I am a member in good standing of several other technical associations and societies, including:

   The Canadian Institute of Mining, Metallurgy and Petroleum (Member).

4. I have worked as a geologist in the minerals industry for 35 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 7 years as an exploration geologist looking for iron ore, gold, base metal and tin deposits, more than 11 years as a mine geologist in both open pit and underground mines and 18 years as a consulting geologist working in precious, ferrous and base metals as well as industrial minerals.

6. I visited the Dominican Republic and the Romero project during the period January 9 to 12, 2013 to review exploration results and examine drill core and exposures of the Romero and Romero South zones. The property had not previously been visited by me.

7. I am responsible for the preparation of Sections 4 to 12, and 14, and the summaries therefrom in Section 1, of the Technical Report titled “Preliminary Economic

Assessment, Technical Report on the Romero Project, Dominican Republic" and with an effective date of April 29, 2015.

8. I am independent of the parties involved in the transaction for which this report is required, as defined in Section 1.5 of NI 43-101.

9. I was a co-author of the Technical Reports entitled "A Mineral Resource Estimate for the Romero Project, Tireo Property, Province of San Juan, Dominican Republic", dated December 13, 2013 and "Preliminary Economic Assessment (PEA) for the Romero Project, Tireo Property, Province of San Juan, Dominican Republic", dated July 11, 2014. These reports, prepared for GoldQuest, are my only prior involvement with the property.

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

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

Mineral Resource effective date: October 29, 2013
Report Effective date: April 29, 2015

Dated this 2nd day of June, 2015

"B. Terrence Hennessey" {signed and sealed}

B. Terrence Hennessey, P.Geo.
Micon International Limited