NI 43-101 Pre-Feasibility Study Technical Report For The Romero Gold Project, Dominican Republic

Prepared for:

GOLDQUEST
TSXV:GQC

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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 Feasibility Study (PFS or 2016 PFS) 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.

Three previous technical reports were prepared for the Romero 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 Preliminary Economic Assessment (PEA) in 2014. All technical reports were filed on SEDAR.

This Technical Report summarizes the results of the 2016 PFS study and was prepared following the guidelines of NI 43-101.

1.2 Project Description

The project concept in this PFS is to develop the Romero deposit as an underground mine utilizing long hole and drift and fill mining methods with cemented paste backfill. The mined mineralized rock would be trucked to surface and fed to a nominal 2,800 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 8 years with approximately 7 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 PFS but remains as a significant Mineral Resource.

Life of mine (LOM) concentrate production is estimated to be 448 kt (dry) of a bulk Cu-Au-Ag concentrate and will be shipped through the port of Puerto Viejo near Azua, Dominican Republic for smelting and refining off-shore.

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

1.3 Location, Access 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 Romero 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. GoldQuest has since applied for a mining permit of the exploration concession previously known as La Escandalosa, which is now called Romero. There are ten granted exploration concessions and four exploration concession applications, and one exploitation concession application 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 Romero. The exploitation concession request is in place and once granted is in place 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).

1.4 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 Romero 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 nine phases of drilling from 2006 to 2015 totaling 170 holes and 46,992.58 m on the Romero Trend. Holes details can be found in Table 10.2.

1.5 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.

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.

1.6 Metallurgical Testing and Mineral Processing

Metallurgical test programs were completed in 2011, 2013, 2014, 2015 and 2016 by ALS Metallurgical Laboratories, Kamloops, B.C. (ALS) on metallurgical composites selected by GoldQuest. The most recent 2016 tests, KM5085, 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 13% copper concentrate grade with a 94.6% copper recovery can be achieved for Romero. The average LOM gold and silver recovery with gravity is approximately 78.1% and 58.6% respectively.

This technical report is based predominantly on the results from program KM4601 and confirmatory test work results from KM5085, 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 produce a relationship between head grade and overall recovery. The resulting LOM average recoveries for a 13% copper concentrate are presented in the Table 1.1.

Table 1.1: Projected Metallurgical Balance

Product	Wt%	Cu (%)	Au (g/t)	Cu Rec (%)	Au Rec (%)
Copper Concentrate	6.4	13	45.3	94.6	78.1
Tailings	93.6	0.05	0.87	5.4	21.9
Feed	100	0.88	3.72	100	100

Source: JDS 2016

1.7 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)	Au-Eq (g/t)	Au Ounces (x1,000)	Au-Eq Ounces (x1,000)
Indicated	Romero	18,390	2.57	0.65	0.31	4.2	3.43	1,520	2,028
	Romero South	1,840	3.69	0.25	0.18	1.6	4.01	218	237
Total Indicated Resources		**20,230**	**2.67**	**0.61**	**0.30**	**4.0**	**3.48**	**1,738**	**2,265**
Inferred	Romero	2,120	1.80	0.39	0.36	3.2	2.32	123	158
	Romero South	900	2.57	0.20	0.21	2.1	2.84	74	82
Total Inferred Resources		**3,020**	**2.03**	**0.33**	**0.32**	**2.9**	**2.47**	**197**	**240**

Note: AuEq g/t = (Au g/t)+(Ag g/t)/92.261)+(Cu %)/0.605)
Source: Micon (2016)

The present report and Mineral Resource estimates are based on exploration results and interpretation current as of November 9, 2015. The effective date of the Mineral Resource estimate is January 14, 2016 for Romero and October 29, 2013 for Romero South.

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. The Mineral Resources presented herein are not Mineral Reserves, however, a portion of the resources have been classified as Mineral Reserves and are detailed herein. The remaining Mineral Resources 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.

1.8 Mineral Reserves Estimate

The Mineral Reserves identified in Table 1.3 comply with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification of National instrument (NI) 43-101 resource and reserve definitions and standards. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified.

The economic viability of the project is presented in Sections 21 and 22, and confirms that the proven and probable reserve estimates meet and comply with CIM definitions and NI 43-101 standards, including the main assumptions used in the definition of the reserves (i.e., metal prices, dilution, operating costs and recoveries).

This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimates of the Mineral Reserves or potential production.

Table 1.3: Mineral Reserve Estimate

Mine Reserves $70 NSR Cut-off) (2)	Tonnes	Au		Ag		Cu		Au-Eq (1)	
		(g/t)	(oz)	(g/t)	(oz)	(%)	(M lb)	(g/t)	(oz)
Total Probable	**7,031,000**	**3.72**	**840,000**	**4.33**	**980,000**	**0.88**	**136**	**4.9**	**1,117,000**

1. Gold equivalent metal prices $1,300/oz Au, $20.00/oz Ag and $2.50/lb Cu
2. Cut-off NSR metal prices: Cu $2.50/lb - Au $1,250/oz - Ag $17.00/oz;
   Recovery: Cu-96.8% Au-71.7% Ag-54.4%, Payable: Cu-96.5 Au-90.0 Ag-95.0,
   Treatment Charges, Refining Charges (TCRC): $257.83/dmt, Cu concentrate 20%

Source: JDS (2016)

1.9 Mining

The proposed underground mine will extract 2,800 tonnes of ore per day by way of primary-secondary transverse sub-level long hole stoping (LH) and mechanized underhand and overhand cut and fill (MCF). A 5.0 m wide by 4.5 m high decline driven at a maximum grade of 15% will provide access for rubber tired mobile equipment and personnel. Raise bore holes, 3.0 m diameter developed and equipped with electric fans, will provide fresh and exhaust air ventilation. The fresh air raise will also function as a secondary egress for personnel. Electric hydraulic two boom jumbos will develop all capital and operating lateral development, as well as 4.0 m wide x 4.0 m high mechanized cut and fill (MCF) stopes. Electric hydraulic long hole drills will develop LH stopes 15 m wide x 20 m high x 30 m long. Bulk ammonium nitrate fuel oil (ANFO) explosives will be used to blast ore and waste twice per day at the end of each 12-hour shift. Rubber tired diesel load-haul-dump (LHD) equipment will be used to load broken ore and waste into trucks for haulage to surface, where surface equipment will re-handle ore to the processing plant and waste to the temporary storage facility.

Paste backfill comprised of process tailings and cement binder will be pumped into mined voids for permanent storage and to provide structural fill for mining and pillar extraction. All potential acid generating (PAG) mined waste will be placed back underground in stopes not requiring structural fill.

Table 1.4 through Table 1.6 outlines the Romero mine production plan, development and waste placement schedules. Highest grade reserves, where possible, will be mined first to maximize project economics.

Table 1.4: Annual Mine Production Schedule

Mine Production	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Mined Waste	kt	940	101	324	271	81	25	34	70	34	0
Mined Ore	kt	7,031	-	818	1,008	1,008	1,008	1,008	1,008	1,008	165
Gold Grade	g/t	3.72	-	4.54	4.85	4.06	3.96	3.66	3.23	2.18	1.80
Silver Grade	g/t	4.33	-	4.97	3.83	3.52	5.33	5.31	3.85	3.90	2.82
Copper Grade	%	0.88	-	0.86	0.83	0.96	0.96	0.89	0.80	0.86	0.78
Zinc Grade	%	0.26	-	0.18	0.36	0.36	0.31	0.20	0.24	0.19	0.12
NSR Value	$/t	121	-	140	146	132	130	120	106	84	72
Gold Equivalent	g/t	4.88	-	5.78	6.04	5.43	5.35	4.95	4.37	3.42	2.91
	k oz	1,126	-	152	196	176	173	160	142	111	15

Gold equivalent metal prices: Cu $2.50/lb, Au $1,250/oz, Ag $17.00/oz
Source: JDS (2016)

Table 1.5: Annual Mine Development

Mine Development	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Ore Development	km	33.6	-	2.2	2.9	3.6	6.5	6.7	5.8	4.1	1.8
Waste Development	km	15.3	1.6	5.1	4.4	1.4	0.4	0.6	1.2	0.6	0.0
Total Development	km	48.9	1.6	7.3	7.3	5.0	6.9	7.3	7.0	4.7	1.8
Lateral Advance Rate	m/day	14.9	4.5	20.0	20.0	13.6	18.9	20.0	19.1	12.9	5.0
Raise Development	km	1.2	0.1	1.0	0.1	-	-	-	-	-	-

Source: JDS (2016)

Table 1.6: Annual Backfill Placement

Mine Backfill	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Paste Backfill	k m^3	1,819	-	180	217	160	299	309	292	309	53
Waste Rock Backfill	k m^3	453	-	84	109	166	27	16	34	16	1

Source: JDS (2016)

Initial capital development will be conducted by contract miners, who will provide the labour, equipment, and materials required to establish a portal and develop 6.8 km of underground ramp, access, footwalls, and infrastructure drifts. Contract mining will ensure highly trained professional miners are available to develop the most critical mine infrastructure in a safe and timely fashion, as well as help train the owner operated labour force. Contracted supervision will further oversee mine operations for the first four years of operation. Mine supervision will include mine management, training officers, maintenance planners, development and production leads, and shift supervisors. Contracted supervision will be reduced over time as the local workforce is adequately trained.

1.10 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 2,800 dry tonnes per day (dt/d). Total annual concentrate production will be approximately 64,000 t.

The mineral processing facility will be located south of the Romero mine site. Listed below are the major process unit operations at Romero:

- Primary jaw crusher;
- Crushed stockpile (live capacity 2,800 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 process plant. Mineralized material will be delivered by truck from the underground portal and rehandled by a front-end loader into a jaw crusher. Feed will be crushed to a nominal product size of approximately150 mm at 80% passing (P_{80}) and conveyed to a 2,800 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 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 the 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.

1.11 Infrastructure

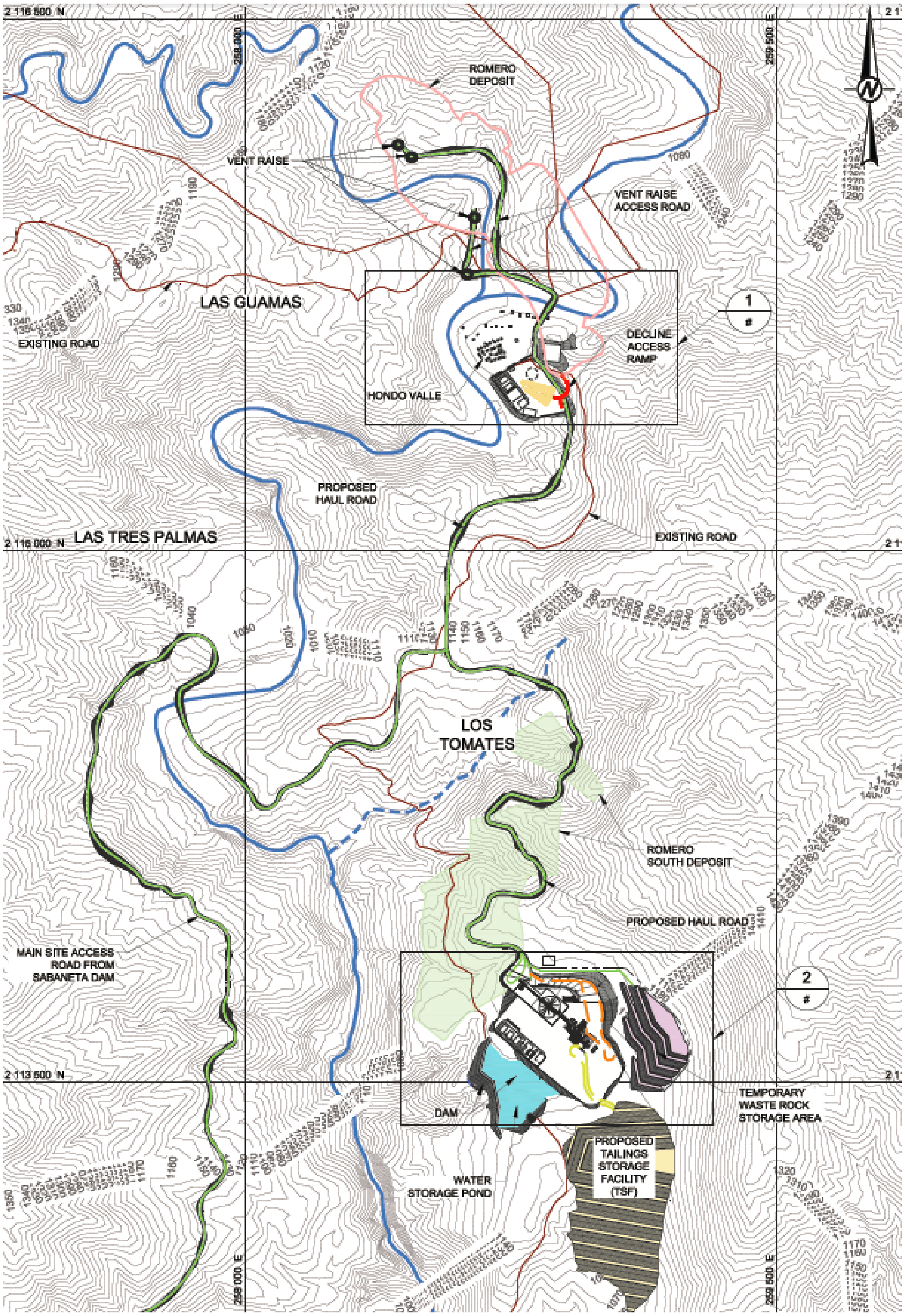
The Romero Project infrastructure and services are designed to support the operation of a 2,800 t/d underground mine and processing plant, operating on a 24-hour per day, 7-day per week basis. It is designed for the local conditions and rugged topography.

The main infrastructure for the project consists of the following facilities:

- A 23.5 km access road between the existing municipal road network at Sabaneta Dam and leading to the site;
- A 2.8 km haul road connecting the underground workings with the processing facilities;
- Gold and copper processing plant with security, administration, and personnel facilities;
- Dry Stack Tailings Storage Facility (DSTSF);
- Paste backfill plant for providing cemented paste to the underground workings;
- Mine support facilities including mobile equipment maintenance, mine personnel facilities, and shotcrete mixing plant;
- Bulk emulsion storage area;
- Utility infrastructure for the site: water, sewer, fire protection and communications;
- 69 kV power transmission line connected to the national electricity grid at Sabaneta Dam;
- 5 kV distribution from on-site stepdown transmission substation to the underground mine;
- Water storage pond for process make-up water;
- Emergency water storage pond for the management excess water during the wet seasons;
- Runoff settling ponds; and
- Surface water diversion infrastructures to manage local streams and runoff from the facilities.

The overall site layout, showing location of the mining portals, processing plant, tailings storage facility (TSF) and other major facilities, is shown in Figure 1.1 below.

Figure 1.1: Overall Romero Site Layout



1.12 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 Bermúdez 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. For these reasons, it is a key design criterion for the Romero Project that the site water management system be independent from the San Juan River and from the regional aquifer. The San Juan River will not be used for water supply during project operations, nor be used for discharge of liquid effluents. This PFS demonstrates that this key design criterion is fulfilled.

The project does not anticipate a design for a tailings pond, a tailings dam, or any anticipated discharge of tailings pond water. Tailings management will be carried out by returning the tailings to the underground mine as paste backfill, and by safe surface storage of some of the dry stacks tailings. As the project will be an underground mine operation, it will not impact the profile of the landscape, as opposed to open pit mining. Since the project will not rely on diesel power generation, but will generate all its power from the national electricity grid, it will have a minimal carbon footprint on the environment.

The project proposed in this PFS is not expected to require any resettlement. Some land acquisitions will however likely be necessary for some of the project facilities.

Permitting of a new mine carries some risk due to the proximity of the project to a national park and the San Juan and La Guama Rivers. As the project plans will 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 with emphasis on local communities.

1.13 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%. The capital cost and operating cost estimate summaries are listed in Table 1.7 and Table 1.8 below.

Table 1.7: Summary of Capital Cost Estimate

Description	Pre-Production (US$M)	Sustaining/Closure (US$M)	Total (US$M)
Underground Mining	15.7	57.4	73.1
Site Development and Roadworks	13.5	4.0	17.5
Process Facilities	32.4	5.2	37.6
On-Site Infrastructure	8.8	4.1	13.0
Off-Site Infrastructure	21.5		21.5
Indirect Costs	11.8		11.8
EPCM	23.2		23.2
Owner's Costs	10.2		10.2
Closure		15.5	15.5
Salvage		-4.5	-4.5
Subtotal	**137.3**	**81.7**	**219.0**
Contingency (15%)	21.3	10.6	32.0
Total Capital Costs	**158.6**	**92.3**	**250.9**

Source: JDS (2016)

Table 1.8: Summary of Operating Cost Estimate

Operating Costs	$/t milled	LOM (US$M)
Mining	27.67	194.5
Processing	11.58	81.4
Re-handle	1.28	9.0
G & A	5.44	38.3
Total	**45.97**	**323.2**

Source: JDS (2016)

1.14 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.

1.14.1 Main Assumptions

Main economic and smelter return assumptions are summarized in Table 1.9 and Table 1.10.

Table 1.9: Economic Assumptions

Item	Unit	Value
Net Present Value (NPV) Discount Rate	%	5
Corporate Income Tax Rate	%	27
Asset Tax	%	0.5
Export Withholding Tax	%	5
Local Community Tax	%	5
Capital Contingency (Overall)	%	15

Source: JDS (2016)

Table 1.10: Net Smelter Return Assumptions

NSR Parameters	Unit	Value
Concentrate Grade	**%**	13
Smelter Payables		
Cu Payable	%	96.5
Au Payable	%	97.5
Ag Payable	%	90.0
Cu Minimum Deduction	%	1.0
Au Minimum Deduction	g/t	0.0
Ag Minimum Deduction	g/t	0.0
TC/RCs		
Treatment Charge	$/dmt concentrate	85.00
Cu Refining Charge	US $/payable lb	0.085
Au Refining Charge	US $/payable oz	5.00
Ag Refining Charge	US $/payable oz	0.50
Transport Costs		
Moisture Content	%	8
Transport to Port	US$/wmt conc	88.93

Source: JDS (2016)

1.14.2 Results

The economic results for the Romero PFS are shown below in Table 1.11.

Table 1.11: Economic Results*

Results	Unit	Value
Gross Revenues	US$M	1,137
LOM Pre-Tax Free Cash Flow	US$M	458
Average Annual Pre-Tax Free Cash Flow	US$M/a	64
Pre-Tax $NPV_{5\%}$	US$M	317
Pre-Tax Internal Rate of Return (IRR)	%	39
Pre-Tax Payback	Years	1.9
NPV to Pre-Production Capital Cost Estimate (CAPEX)	Times	2
Taxes	US$M	149
LOM After-Tax Free Cash Flow	US$M	309
Average Annual After-Tax Free Cash Flow	US$M/a	43
After-Tax $NPV_{5\%}$	US$M	203
After-Tax IRR	%	28
After-Tax Payback	Years	2.5
Pre-Tax Break-Even Au Price‡	US$/Au oz	640
Cash Cost*	US$/Au oz	669
Cash Cost Net of By-Products**	US$/Au oz	191

(‡) Based on constant Cu price of US$2.50/lb, Ag price of US$20/oz

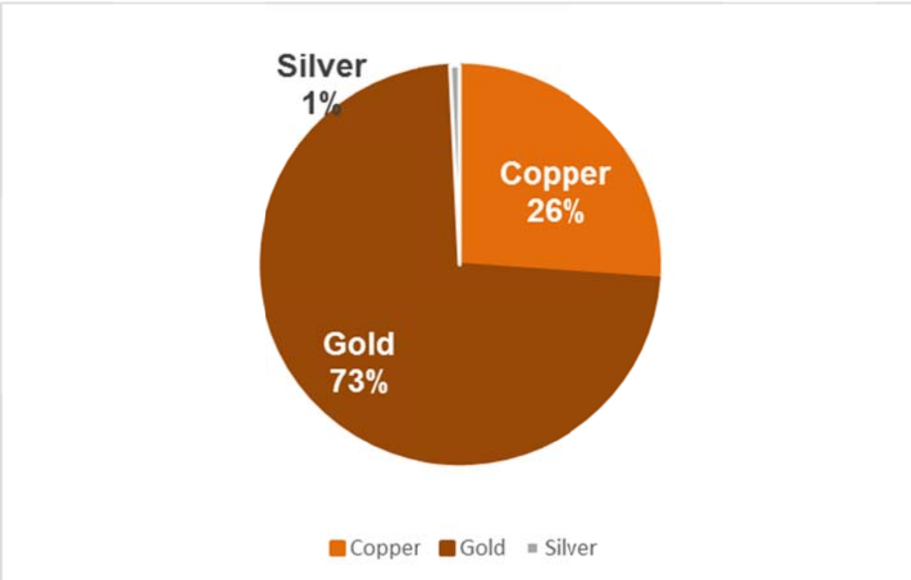
(*) Cash Cost = (Treatment Charges + Refining Charges + Concentrate Handling and Shipping + Royalties + Operating Costs + Sustaining and Closure Capital Costs)/Payable Au oz

(**) Cash Cost Net of By-Products = ((Treatment Charges + Refining Charges + Concentrate Handling and Shipping + Royalties + Operating Costs – (Payable Cu lbs * 2.50/lb) – (Payable Ag oz * $20/oz)) / Payable Au oz

Source: JDS (2016)

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

Figure 1.2: LOM Payable Metal by Value



Source: JDS (2016)

1.14.3 Sensitivities

Sensitivity analyses were performed using metal prices, head grade, capital cost estimate (CAPEX), and operating cost estimate (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 on the after-tax results are shown in Table 1.12.

Table 1.12: After-Tax Sensitivity

After-Tax NPV$_{5\%}$ (US$M)					
Variable	80%	90%	100%	110%	120%
Metal Prices	84	145	203	260	318
Head Grade	87	146	203	259	316
OPEX	236	219	203	186	169
CAPEX	235	219	203	186	170

Source: JDS (2016)

1.15 Conclusions

The PFS indicates that the Romero Project, based on a Proven and Probable Reserve of 7.03 Mt grading 0.88% Cu, 3.72 g/t Au and 4.33 g/t Ag, can support a 2,800 t/d underground mine and concentrator.

Mineralized material will be sent to a process plant designed to achieve copper, gold and silver recoveries of 94.6% 78.1%, and 58.6%, respectively. It is anticipated that, over a mine life of eight years, approximately 119 M lbs of copper, 640 koz. of gold and 434 koz. of silver in concentrate will be produced.

The initial capital cost of the project is estimated to be $158.6 M and the sustaining capital (including the development of the underground mine) is estimated to be $92.3M. The all-in sustaining LOM total cost is US$ $595 /oz. Au, including copper and silver credits and royalty payments.

The project NPV (pre-tax) is estimated to be $317.2M and the project NPV (after-tax) is estimated to be $202.7M using a discount rate of 5%.

The project internal rate of return (IRR) (pre-tax) is estimated at 38.7% and the project IRR (after-tax) is estimated to be 28.2%. The simple payback period (after-tax) is 2.5 years.

Based on the assumptions made in this analysis, it is JDS's opinion that the Romero Project is sufficiently robust to warrant advancing to the next Feasibility Study stage of development and its supporting technical studies.

1.16 Recommendations

It is recommended that the Romero Project proceed to the Feasibility Study stage in line with GoldQuest's desire to advance the project. Several technical programs, including baseline environmental studies, are required to de-risk the project and provide the level of detail necessary to a feasibility level evaluation. It is also recommended that the company continue with its efforts with respect to community engagement and project permitting.

It is estimated that a Feasibility Study, technical studies and supporting field work would cost approximately $4.8 M. A breakdown of the key components of the next study phase is listed in Table 1.13.

Table 1.13: Cost Estimate to Advance Romero to Feasibility Study Stage

Component	Estimated Cost (M$)	Comment
Resource Drilling and Updated Resource	1.0	Conversion of Inferred resources to indicated within and immediately adjacent to the proposed mine. Definition drilling will include holes for combined resource, geotechnical and metallurgical purposes
Metallurgical Testing	0.3	Variability test work including expanded comminution, grinding, flotation and filtration test work as well as multi-element ICP tailings and concentrate analysis for smelter interest and pricing
Access Road	0.3	Reconnaissance, test pitting, borrow source identification and road design
Backfill Testing	0.2	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.), and process plant area
Engineering and Design	2.0	FS-level mine, infrastructure, tailings storage, paste backfill and process design, cost estimation, scheduling and economic analysis
Environment	0.5	Baseline environmental investigations including, water quality, fisheries, wildlife, weather, traditional land use and archaeology
Total	**4.8**	Excludes corporate overheads and future permitting activities

Source: JDS (2016)

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 2016 PFS study and was prepared following the guidelines of Canadian Securities Administrator's National Instrument (NI) 43-101.

2.2 Scope of Study

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 Energy and Mining Inc. (JDS) scope of work included:

- Compile the technical report, including 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 2016 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 International Ltd. (Micon) scope of study included:

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

MineFill Services Inc. (MineFill) scope of study included:

- Backfill and Paste Plant

Golder Associates Ltd. (Golder) scope of study included:

- Hydrology, site water balance, and Pre-Feasibility design of site water infrastructures;
- Pre-feasibility design of DSTSF and Temporary Waste Rock Storage Facility; and
- Geotechnical and Hydrogeology.

2.3 Qualifications, Responsibilities and Site Visits

The results of this PFS 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 qualified persons (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
Garett Macdonald, P.Eng.	JDS Energy and Mining Inc.	1, 2, 3, 18, 19, 20, 21, 22, 24, 25, 26, 27, 28, 29	May 20th, 2016
Indi Gopinathan, P.Eng.	JDS Energy and Mining Inc.	23	May 20th, 2016
Michael Makarenko, P.Eng.	JDS Energy and Mining Inc.	15, 16 (except 16.5 and 16.9.6)	April 6-18, 2015
Kelly McLeod., P.Eng.	JDS Energy and Mining Inc.	13, 17	Did not visit site
Marcel Pineau, Ph.D., M.Sc.,P.Eng.	JDS Energy and Mining Inc.	16.3, 20	May 20th, 2016
B. Terrence Hennessey, P.Geo.	Micon International Limited	4, 5, 6, 7, 8, 9, 10, 11, 12, 14	January 9-12, 2013
Alan San Martin, MAusIMM (CP)	Micon International Limited	14	Did not visit site
Luiz Castro, P.Eng.	Golder Associates Ltd	16.5	January 20-22, 2016
Ken Bocking, P.Eng.	Golder Associates Ltd	18.3	Did not visit site
Luis Vasquez, P.Eng.	Golder Associates Ltd.	18.2	Did not visit site
David Stone, P.Eng.	MineFill Services Inc.	16.9.6	Did not visit site

Source: JDS (2016)

The Romero Project is in an exploration stage and a site visit by Kelly McLeod, P. Eng. was not necessary to complete this PFS. 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, the 2014 Micon PEA, and the 2016 JDS 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 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 updated from the 2014 Micon PEA Report (Preliminary Economic Assessment for the Romero Project, Tireo Property, 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 Concession

Source: GoldQuest (2016)

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 Romero Exploitation concession which has an area of 3,997.0 ha and is shown on a map in Figure 4.2. It was originally granted on November 9, 2010, and the exploitation permit application is currently under review. 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, which changed in title from Las Tres Palmas to La Escandalosa and finally to Romero, has been reapplied for as an exploitation concession. Exploitation concessions in the Dominican Republic, once granted, are valid for 75 years.

The concession is part of the Tireo Property in San Juan owned by GoldQuest. It comprises 14 exploration concessions or applications and one exploitation application. The exploration concessions or applications are titled Valentin, Descansadero, Los Lechones, Loma el Cachimbo, La Tres Veredas, Loma Los Comios, Aguita Fria, La Tachuela, Toribio, Los Gajitos, La Guinea, Piedra Dura, La Pelada, and Tocon de Pino. The exploitation application is called Romero (See Table 4.1 and Figure 4.3).

Every time a concession is granted, its title must change, which is why there are multiple titles per concession.

Table 4.1: Description of Tireo Property Exploration and Exploitation Concessions

Name	Status	Area (ha)	Application Date	Title Date	Mining Registry Date	Resolution Number	Expiry Date
Exploitation Concessions							
La Escandalosa/Romero	Under Application	3,997	23-Oct-15			Exploitation	
Exploration Concessions							
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	Granted	1,311.50	5-Jul-13	12-Sep-16	16-Sep-16	R-MEM-CM-045-2016	12-Sep-21
Loma Viejo Pedro / Loma El Cachimbo	Granted	3,514	21-Dec-09	15-Apr-16	3-May-16	R-MEM-CM-014-2016	15-Apr-21
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-14	7-Nov-14	III-14	15-Oct-19
El Barrero/Bartola/ Valentin	Granted	300	25-Oct-12	25-Jul-16	2-Aug-16	R-MEM-CM-0024-2016	25-Jul-21
Tocón de Pino	Under Application	744	17-Nov-08				
Las Tres Veredas	Granted	781	20-Jun-12	1-Dec-14	8-Jan-14	I-15.	1-Dec-19
Patricio / La Guinea	Under Application	2,768	12 -Feb-14				
Piedra Dura	Under application	362	21-Apr-14				
La Tachuela/ La Fortuna	Granted	335.5	21-Apr-14	31-Mar-16	11-Apr-16	R-MEM-CM-008-2016	31-Mar-21
Toribio	Granted	2,351.45	29-May-14	3-Jun-15	1-Jul-15	R-MEM-CM-0004-2015	3-Jun-20
La Pelada	Under application	625	29-May-14				
Total		**20,762.45**					

Source: GoldQuest, 2016

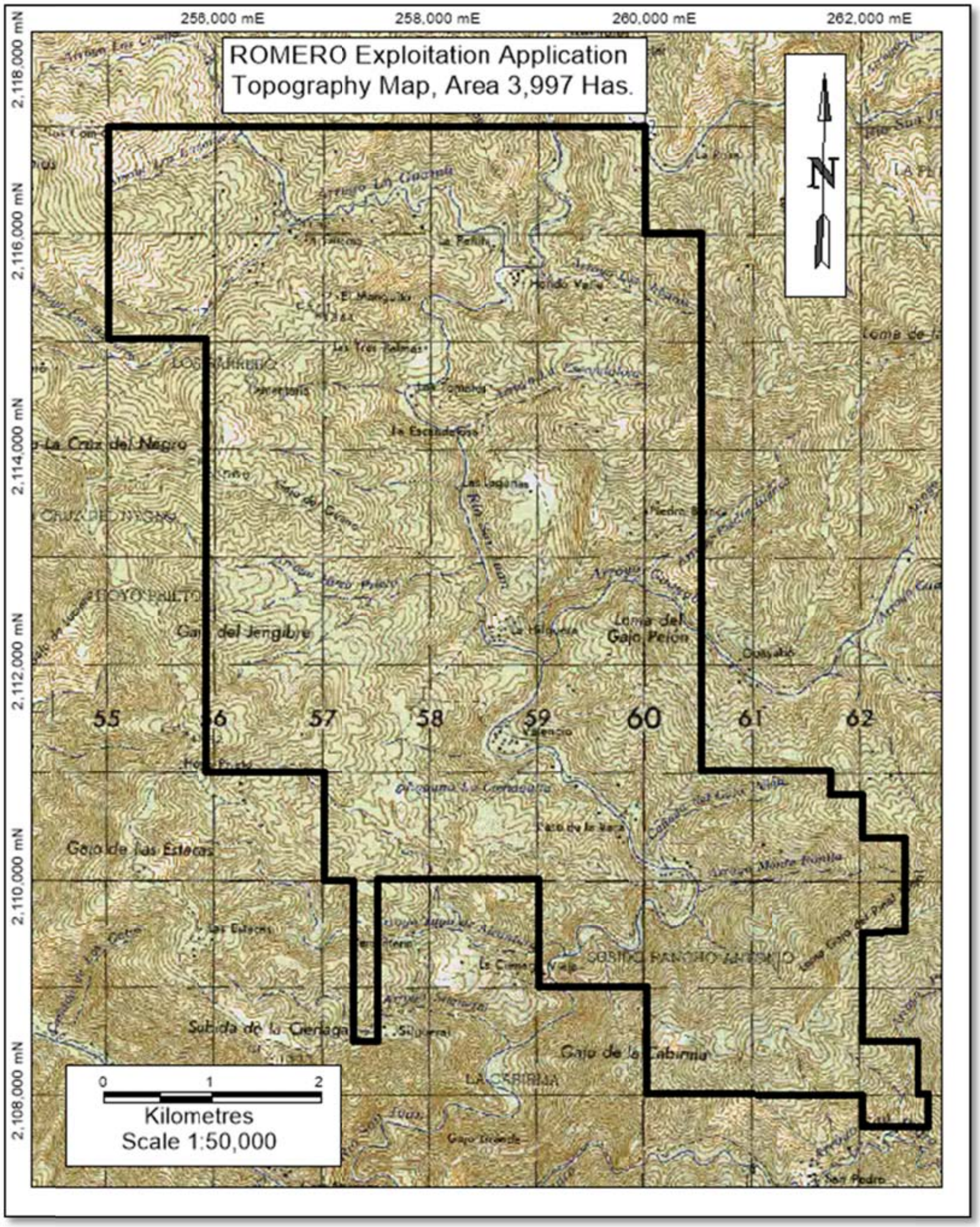
Concession taxes are RD$0.20 (twenty Dominican centavos equal to about US$0.0044 or 0.44 US cents at the current exchange rate of RD$45 to US$1.00) per hectare per 6-month period, equivalent to about US$20 per year for Romero. An exploitation concession may be requested at any time during the exploration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

Exploitation properties are subject to annual surface fees and a NSR 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%.

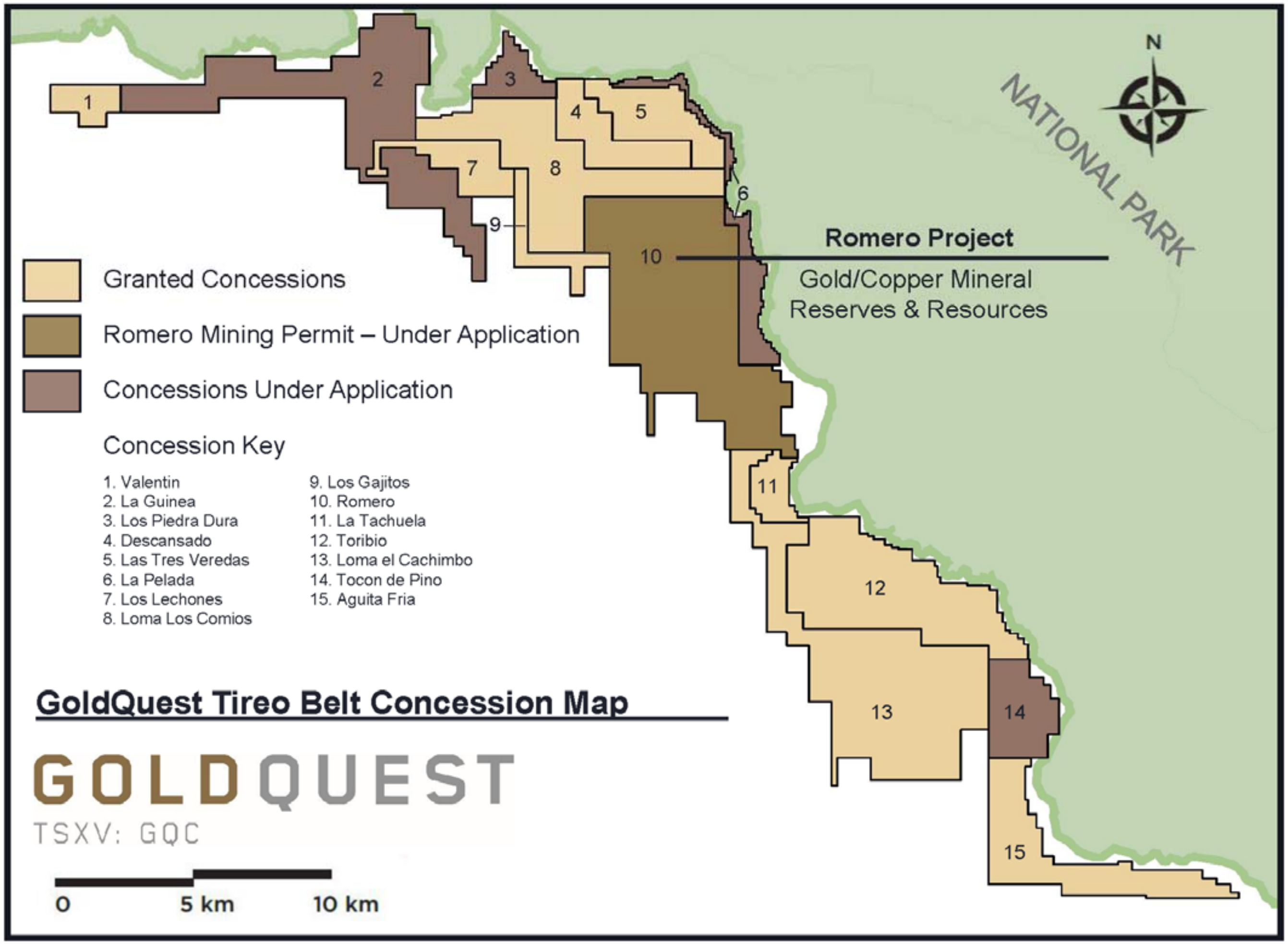
The concession is also subject to a 1.25% NSR royalty in favour of Gold Fields Limited. More detail on taxes and royalties is provided below.

Figure 4.2: Map of Romero Exploration Concession



Source: GoldQuest (2016)

Figure 4.3: Map of the Tireo Property, Including Romero Concession



Source: GoldQuest (2016)

4.2.2 Property Legal History

GoldQuest's subsidiary company Exploration & 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 Romero (now known as the Romero South deposit) in late 2003.

The Las Tres Palmas exploration concession (now known as the Romero 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 November 12, 2010 the concession, then named La Escandalosa, was granted again to the company as an exploration concession until November 12, 2015. GoldQuest then applied for the exploitation permit of the concession, which is now known as Romero, on October 23, 2015.

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 Veredas, Loma Los Comios, Descansadero, Los Lechones, Los Gajitos, Loma El Cachimbo and Aquita Fria. 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$5M 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$5M 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$5M 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.

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 Industry and Commerce (formerly called the Secretary of State of Industry and Commerce until 2010).

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, annually 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 properties are simply known and recorded in their respective property name under a Licence of Metallic Exploitation Concession. Title is valid for seventy-five years. Concession taxes are 50 Dominican centavos (RD$ 0.50) per hectare, annually for concessions between 1,000 and 5,000 ha in size, equivalent to about US$0.01 per hectare per year (at the current exchange rate of RD$45 to US$1.00).

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 (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 (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; and
- 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 NSR 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 Sub-secretary 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);
- 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 exploration concession, now the Romero exploitation concession.

GoldQuest carried out trenching by hand. The trenches were back filled and revegetated. 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 revegetated 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 revegetated.

Water Management Consultants Ltda., of Santiago, Chile carried out a hydrological and hydrochemical baseline survey at La Escandalosa in 2006 (Water Management Consultants, 2006). The company worked with AMEC to monitor ongoing baseline studies from 2012 until 2015 and is currently conducting monitoring work with its employees.

An archaeological survey has not been carried out.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

This section was taken from the 2014 Micon PEA Report and updated where applicable.

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 country, and 35 km north of San Juan de la Maguana, the provincial capital and nearest large town (urban population 169,032 in 2012, see Figure 4.1). The geographical coordinates of GoldQuest's field camp at the village of Hondo Valle on the Romero concession are 19° 07' 00" north, 71° 17' 30" west, and the UTM 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. The PFS contemplates upgrades for the unpaved portions of the roads and a rerouting from Boca de Los Arroyos to Hondo Valle.

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 two 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 (2016)

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. The entire length from Boca de los Arroyos to Hondo Valle is being contemplated and a new road route is being designed for permit applications.

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

Red ellipse shows approximate location of Romero deposit. Yellow dotted ellipse shows location of the camp. The village is behind the exploration camp.
Source: GoldQuest (2016)

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 Romero South 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 (2016)

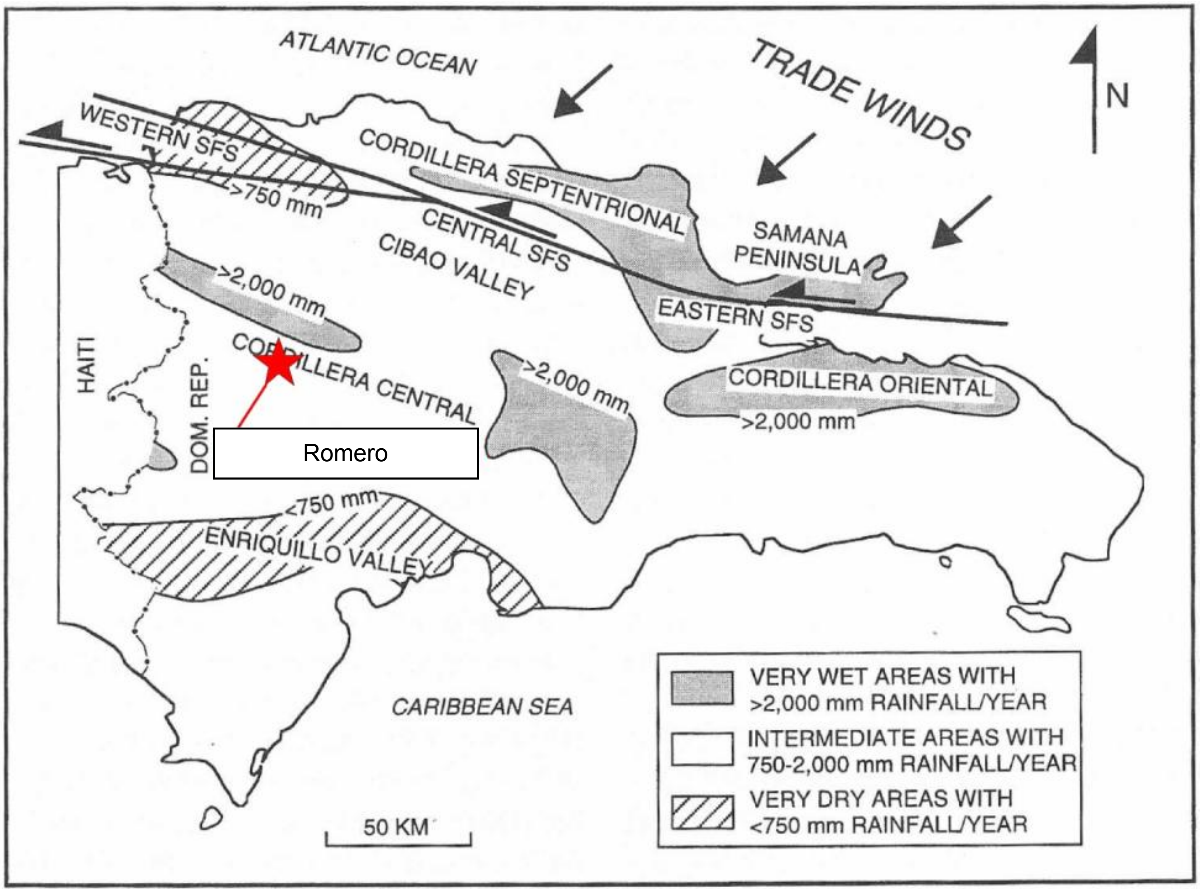
5.2 Climate

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 severity of hurricanes is often measured using the Saffir Simpson Hurricane Wind Scale, with five being the most intense. Severe hurricanes in the Dominican Republic in the recent past have been hurricane Matthew in 2016 (Category 5), Georges in 1998 (Category 3), 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 Romero 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, 25 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. Handheld 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 throughout the property. For the PFS it is contemplated that the mine site will be connected to the national grid and telecommunications network via fibre optic cable.

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 and updated where applicable.

6.1 Historical Mining

Hispaniola was first occupied by the indigenous Taino people 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 people 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 Romero 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. Taino 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 Taino people. 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 Taino 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 Taino people) 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 Romero 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; Bernárdez 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 and 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.

In 2013, GoldQuest announced a Mineral Resource in accordance with NI-43 101, for the Romero deposit and an update for Romero South, formerly known as Escandalosa (Hennessey et al. 2013)

In 2016, GoldQuest updated the Mineral Resources at Romero and Romero South deposits. Details of this estimate are presented within this report.

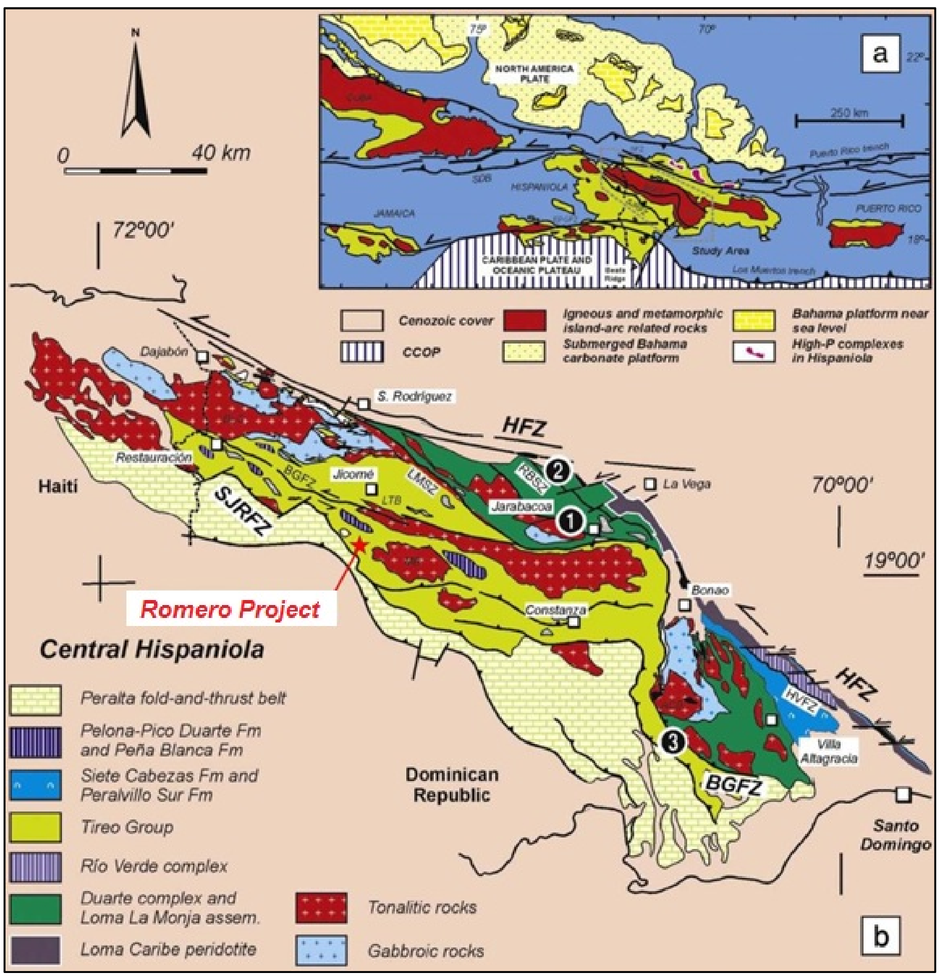
7 Geological Setting and Mineralization

This section was taken from the 2014 Micon PEA Report and updated where applicable.

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).

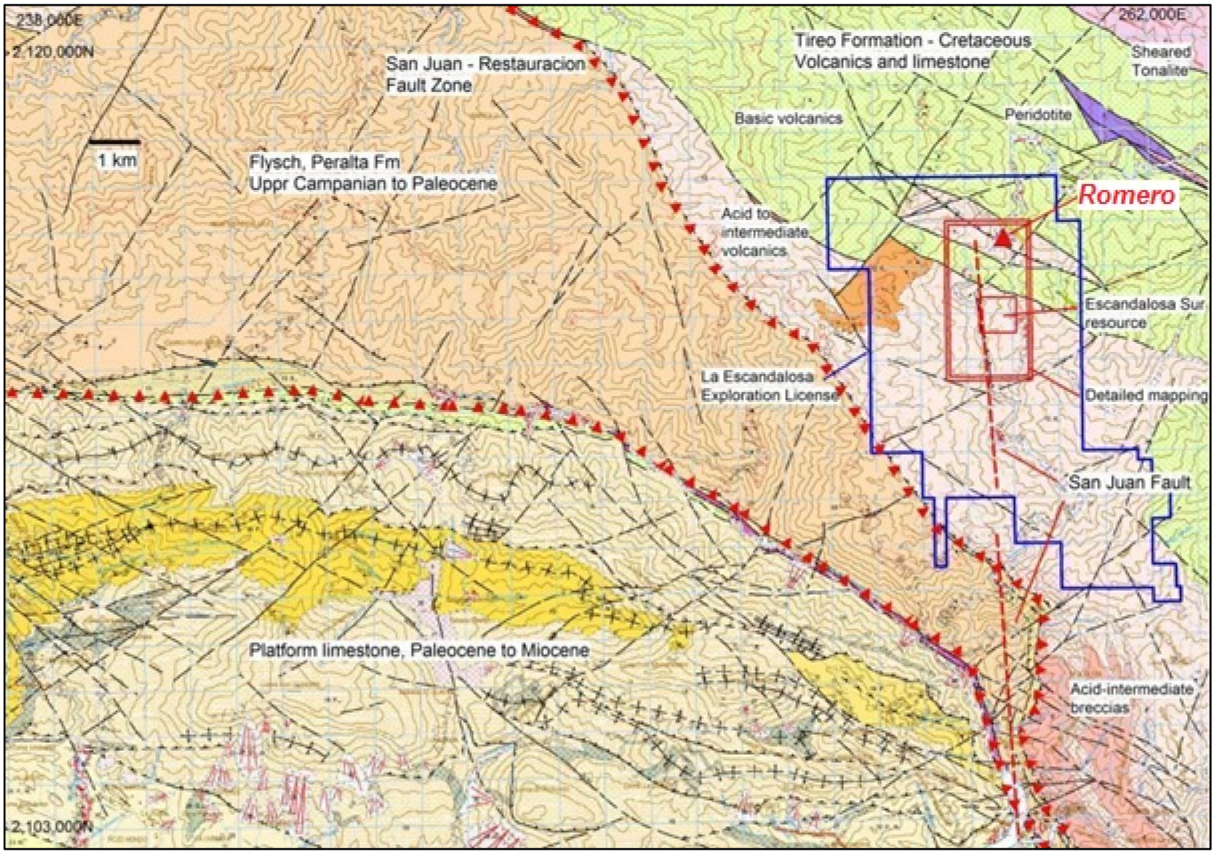
Primitive island arc volcanic rocks of the Early Cretaceous Los Ranchos and Maimón Formations in the Eastern Cordillera are interpreted to be related to northward subduction (Lebron and Perfit, 1994). Cessation of subduction in the mid Cretaceous was marked by accretion of the Loma del Caribe peridotite 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 Rivières 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 Rivières-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 Rivières 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 Romero 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

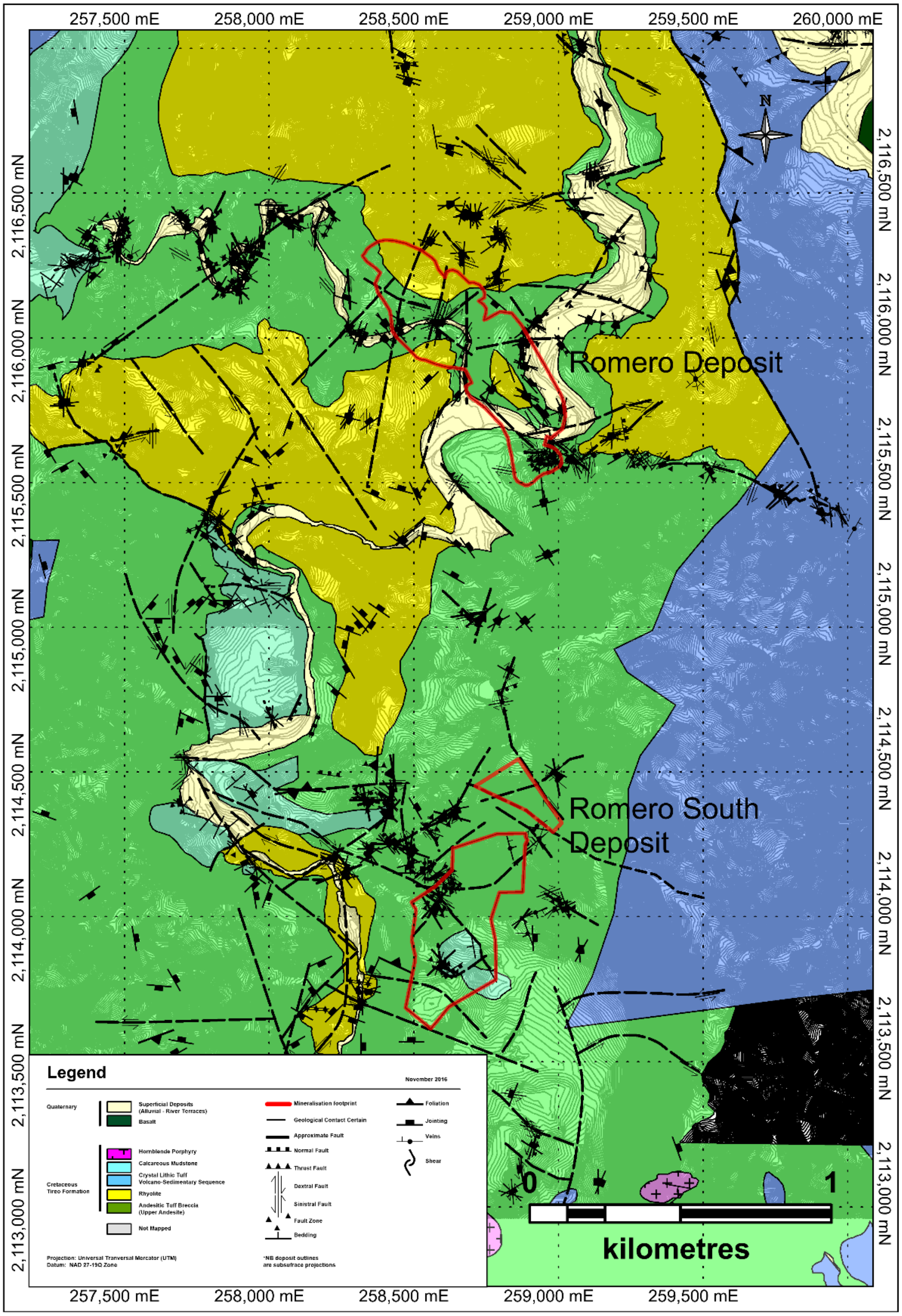
Rhyolite outcrops sporadically for at least 2,000 m of strike length on the west side of the altered horizon from north of Romero to Romero South. There are two apparent rhyolite centres at Romero and Romero South defined by thick rhyolite outcrops, and in between these the flows are thinner with more 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.

Figure 7.3: Geological Map of Romero



Source: GoldQuest (2016)

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 near the deposits 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 appear 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.

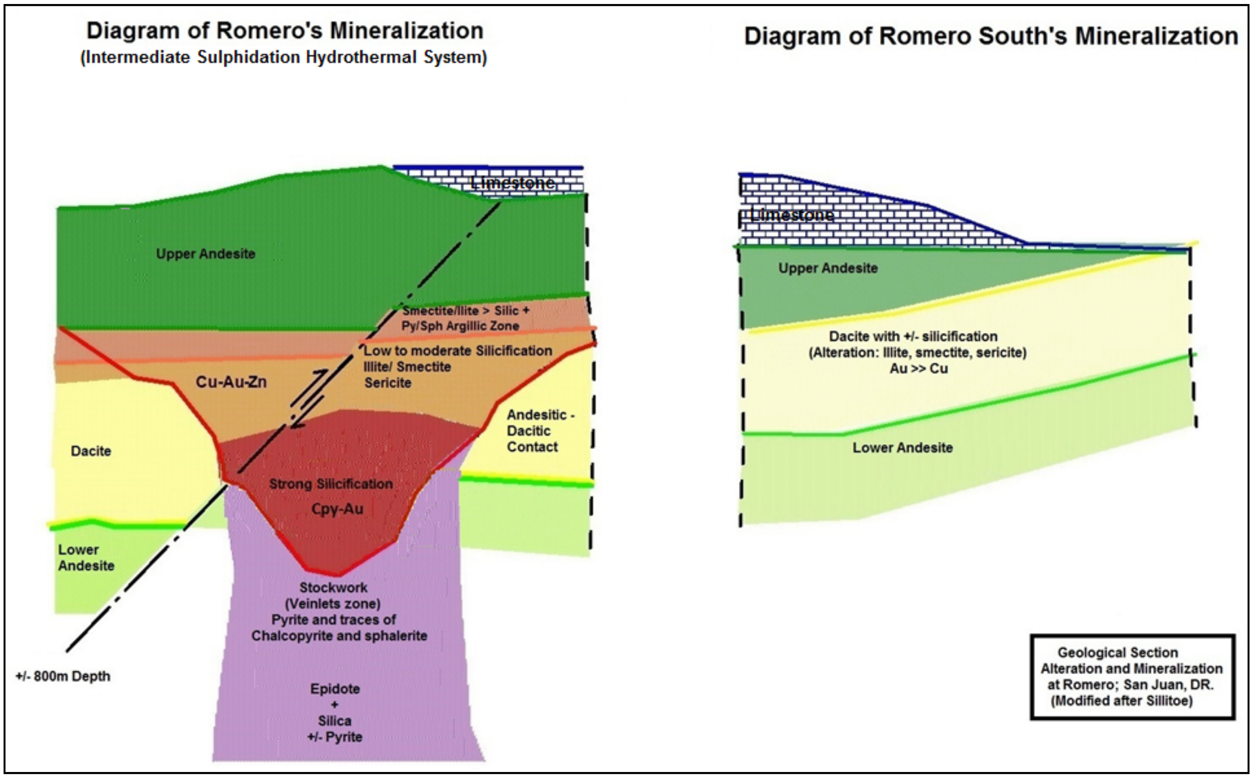
Unconsolidated Quaternary overburden deposits mapped are active river bed alluvium, river terraces, landslides and colluvium. Landslides are common especially in the canyon phase topography.

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 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.

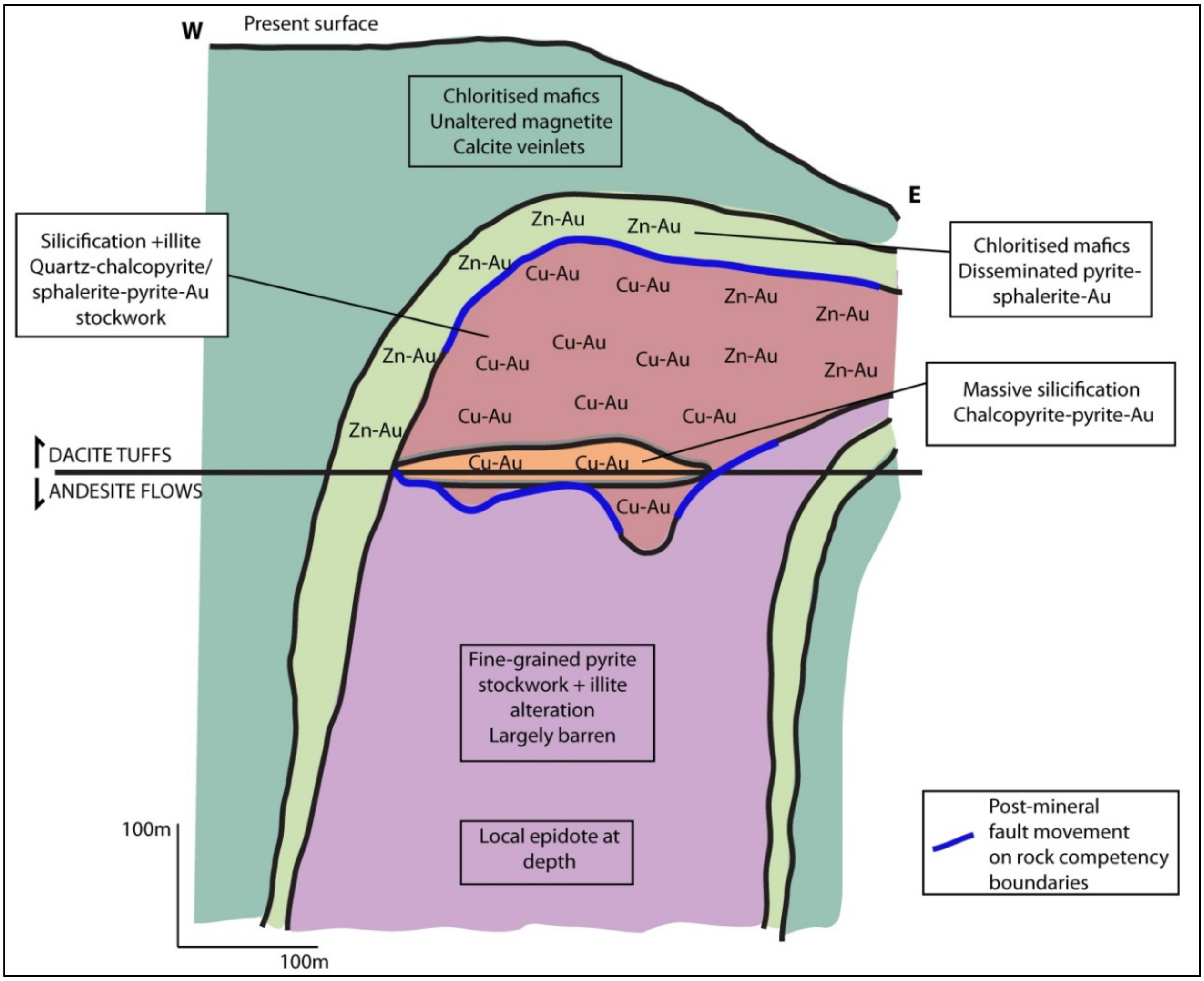
The features of the geological model for alteration and precious/base metals mineralization at Romero are as follows:

- 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;
- 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;
- 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 oxidizing 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 updated 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 2015/16 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

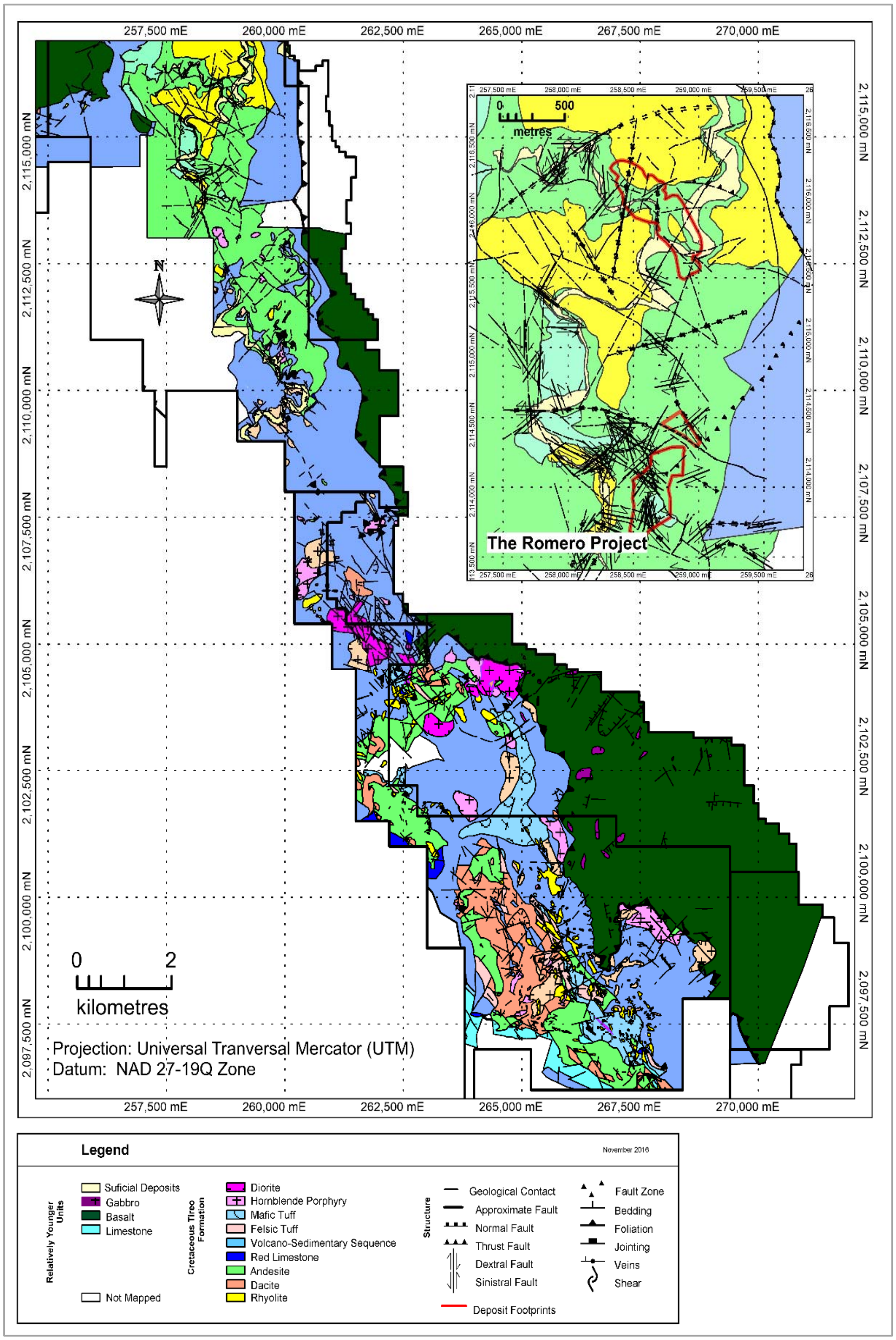
One of the main exploration techniques used in early exploration at Romero has been geochemistry. Within the Romero Concession, GoldQuest has taken 31 fine fraction stream sediment samples (minus 200 mesh), 1,587 soil samples, and 1,192 rock samples, including channel samples.

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. In 2014, a 50 m by 200 m grid was established over the northern portion of the mineralization footprint, and continued up to the northern concessions in an approximate 5.0 km north-south by 5 km east-west grid. A 50 by 50 m spaced grid that has a 200 m north-south dimension by 500 m east- west was established in the southern reaches of the concession in 2015. A total of 1587 soil samples have been taken within the Romero Concession.

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

Within GoldQuest's Tireo Concessions, the company has taken a total of 177 fine fraction stream sediment samples (minus 200 mesh), 5,798 soil samples, and 3,438 rock samples, including channel samples.

Figure 9.1: Compilation of GoldQuest Mapping in the Tireo Project with Romero Inset.



Source: GoldQuest (2016)

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-west lines, spaced at 200 and 100 m (depending on the priorities of the zones), with reading stations at 50 m 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-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 being 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 Figure 9.1to Figure 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 Induced Polarization (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. 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.

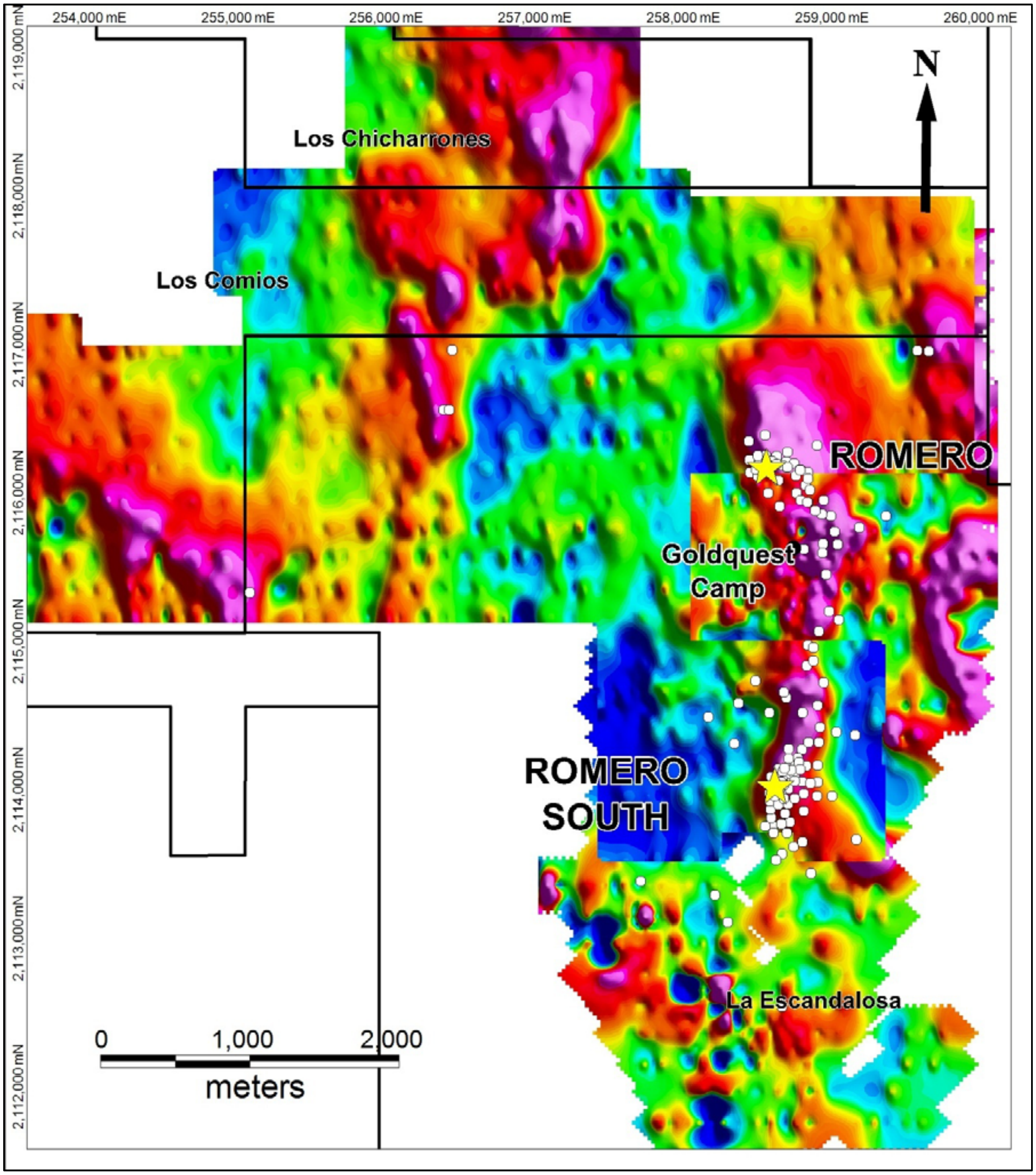
The chargeability map from the 2012-2013 surveys is shown in Figure 9.2, along with the drill hole locations for the Romero and Romero South drilling.

9.4.3 2014 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 a minimum of 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 9.2: 2012-2013 IP Chargeability Results



White dots are drill hole collars
Source: GoldQuest (2016)

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 9.3.

9.4.5 2016 Ground IP Survey

Following the mapping from 2015/16, in March 2016 an extensive ground IP survey with Insight Geophysics was initiated. The survey was designed to cover all areas with surface expressions of hydrothermal alteration with Gradient array IP and resistivity in a sweep from north to south at 200 m-spacing on 2-km long lines in 5-line blocks across the areas of alteration. Where possible the survey was designed such that the area of alteration was in the centre of the lines to obtain optimal coverage. A total of 22 blocks of gradient IP data were collected and areas of high chargeability were examined more closely. A detailed Insight Section IP/Resistivity program immediately followed.

The Insight IP/Resistivity Section program began in the southern part of the 2016 survey area and consisted of 49 sections of either 1 km or 500 m in length. The sections were used to develop drill targets for the 2016 exploration drilling program. The 2016 gradient IP chargeability data can be seen in Figure 9.4 and the 2016 gradient resistivity data in Figure 9.5. Both maps have locations of detailed Insight section shown as black lines.

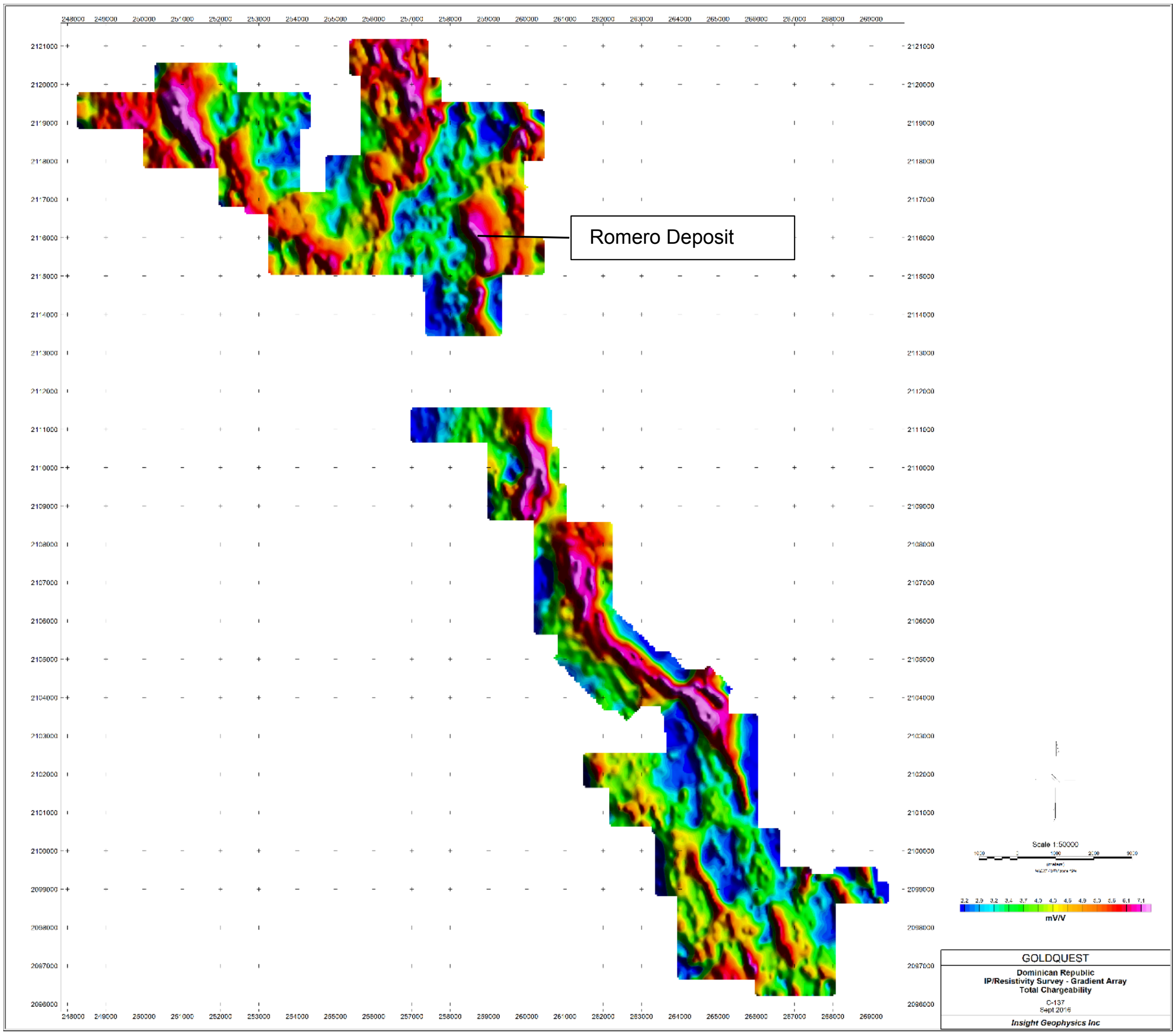
9.5 Deposit Model Confirmation

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

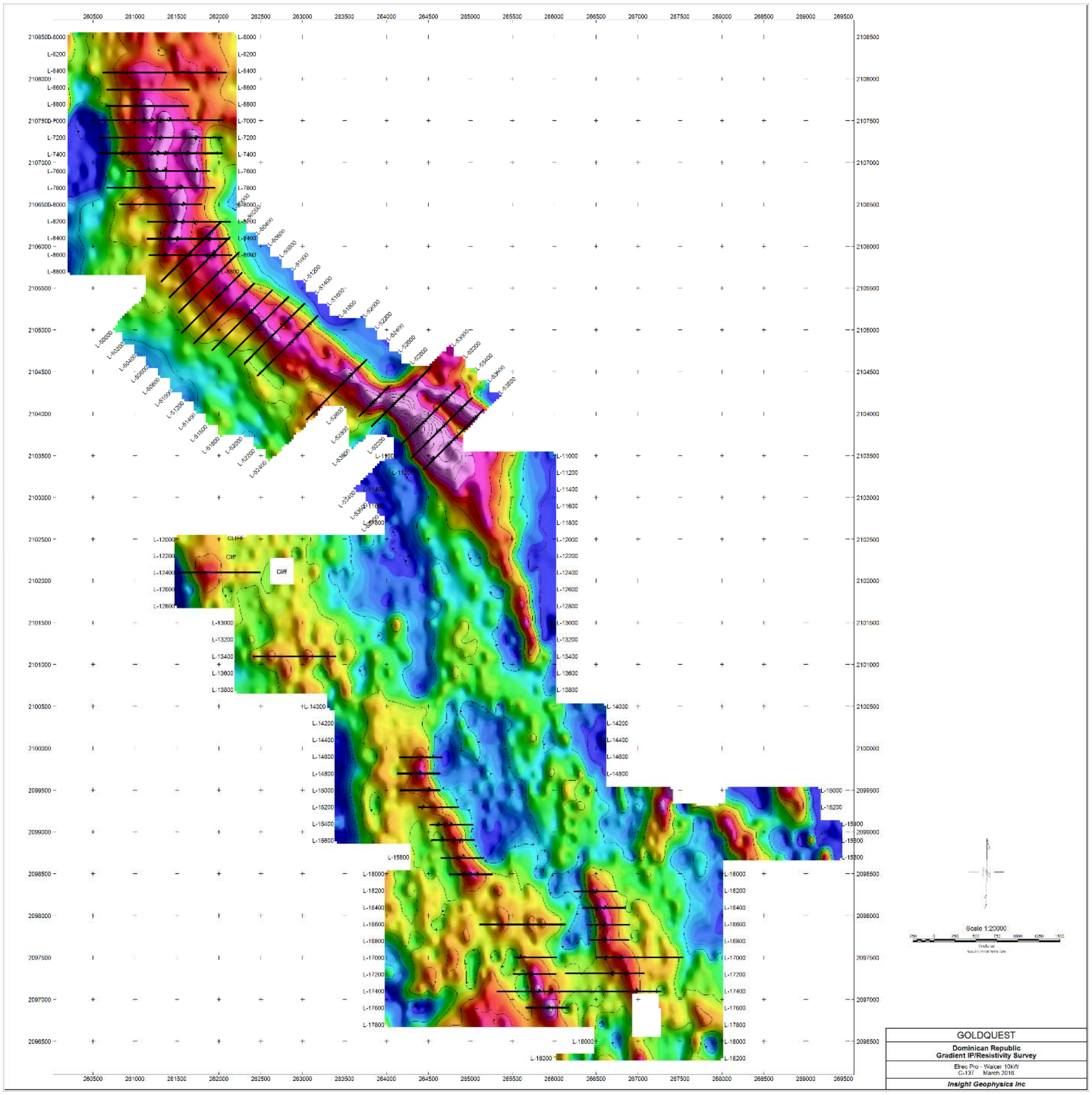
Geological mapping, stream sediment and soil geochemistry and geophysics have confirmed a broad zone of gold and base metal mineralization over a strike length of about 2.2 km, with geophysical anomalies extending over 3.0 km. Several targets for further exploration were identified in the area by geophysics,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

Figure 9.3: 2012-16 Ground IP Gradient Chargeability Compilation



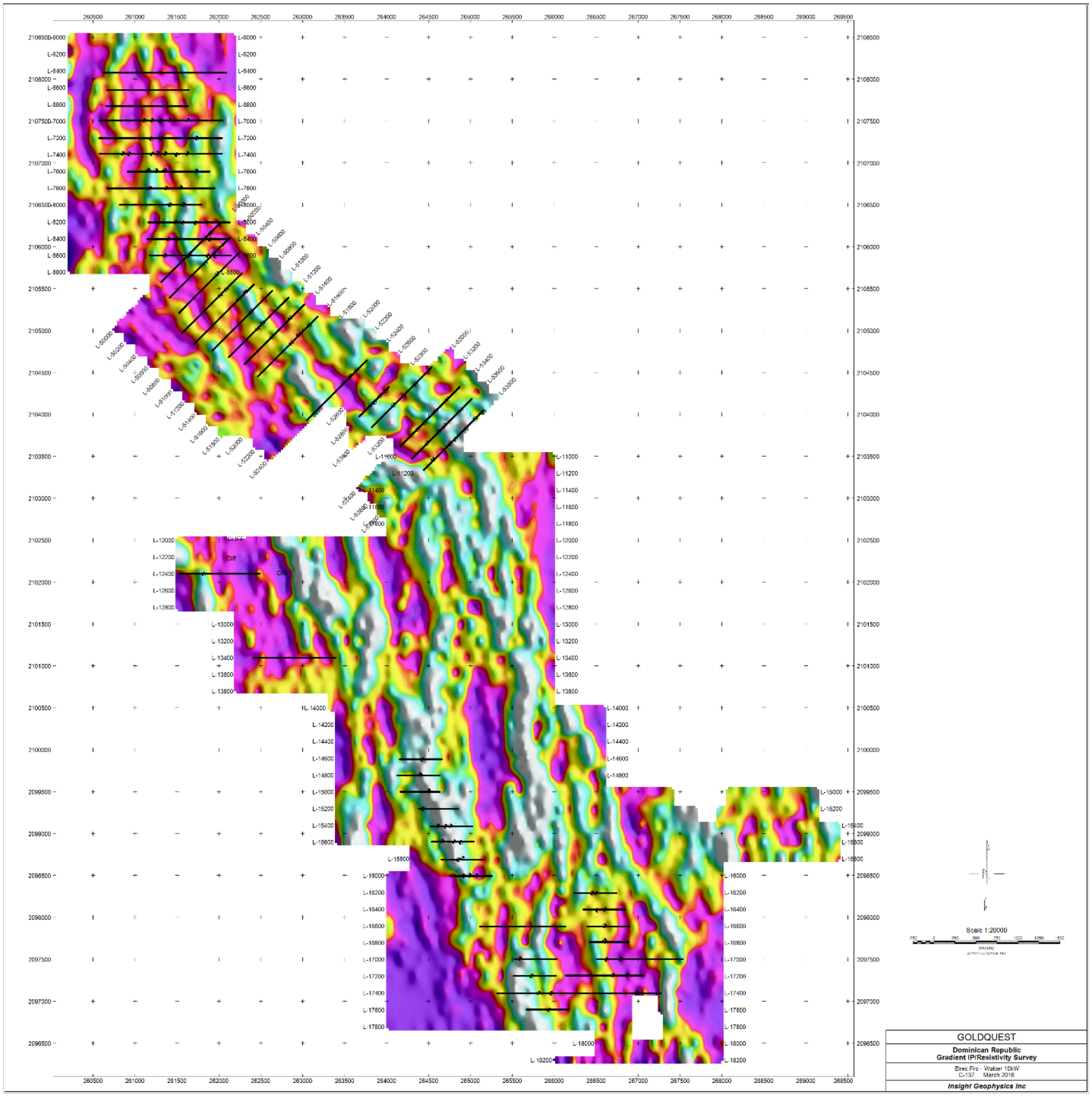
Source: GoldQuest (2016)

Figure 9.4: 2016Ground IP Gradient Chargeability Survey Area



Source: GoldQuest (2016)

Figure 9.5: 2016Ground IP Gradient Resistivity Survey Area



Source: GoldQuest (2016)

10 Drilling

This section was updated from the 2014 Micon PEA, which contained information amended from Steedman and Gowans (2012).

10.1 Romero Trend Drilling

Nine programs of diamond drilling (Table 10.1) have been completed 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 46,992.58 m in 170 holes. The average hole length was 276.43 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 9 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
9	LTP-165 to LTP-170	May-July, 2015

* - Only results up to hole 170 were available for the Mineral Resource estimate.
Source: Micon (2016)

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.

Drilling in Phase 9 was exclusively Pre-Feasibilty drilling at Romero and utilized oriented core equipment. Phase 9 was designed to move inferred material to the measured and indicated categories, which is now included in feasibility level economic studies, as well as to gather material for advanced metallurgical test work and gather data for geotechnical studies. The thickness and grade of the mineral intervals met expectations set by the existing mineral resource block model.

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	Romero Trend
LTP-12	258321	2114527	1114.16	123.48	270	-65	Romero Trend
LTP-13	258434	2114677	1121.8	67.5	270	-60	Romero Trend
LTP-14	258929	2115143	1137.69	187.5	0	-90	Romero Trend
LTP-15	257660	2113326	1190.65	126.7	0	-90	Romero Trend
LTP-16	258246	2113051	1042.09	52.29	0	-90	Romero Trend
LTP-17	258161	2113232	1055.57	45.72	0	-90	Romero Trend
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	Romero Trend
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	Romero Trend
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

Hole-ID	Easting	Northing	Elevation	Length	Az	Dip	Zone Intercept
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	Romero Trend
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	Romero Trend
LTP-81	258854	2114510	1135.33	216.41	0	-90	Romero South
LTP-82	258779	2114780	1175.57	202.69	0	-90	Romero Trend
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	Romero Trend
LTP-86	258894	2114664	1159.04	211.84	0	-90	Romero South
Hole-ID	Easting	Northing	Elevation	Length	Az	Dip	Zone Intercept

Hole-ID	Easting	Northing	Elevation	Length	Az	Dip	Zone Intercept
LTP-87	258826	2114811	1200.82	109.73	0	-90	Romero Trend
LTP-88	258787	2114918	1216.03	109.73	0	-90	Romero Trend
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	2115995	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	2115995	1116.85	509.03	270	-90	Romero
LTP-115	258733.5	2116098	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	Romero Trend
LTP-135	258997	2115087	1182.84	450.4	180	-65	Romero Trend

Hole-ID	Easting	Northing	Elevation	Length	Az	Dip	Zone Intercept
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	Romero Trend
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 Trend
LTP-164	257351	2118873	1428	252.98	190	-70	Romero Trend
Phase 9							
LTP-165	258533.00	2116170.00	1128.16	391.97	236	-76	Romero
LTP-166	258603.00	2115942.00	1099.61	340.16	25	-50	Romero
LTP-167	258533.00	2116170.00	1128.16	390.14	171	-74	Romero
LTP-168	258603.00	2115942.00	1099.61	345.34	40	-55	Romero
LTP-169	258647.92	2116116.72	1098.95	301.91	180	-55	Romero
LTP-170	258793.00	2115842.00	1121.93	230.43	220	-85	Romero

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 (2016)

The drill contractor for all ten 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 (see Figure 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).

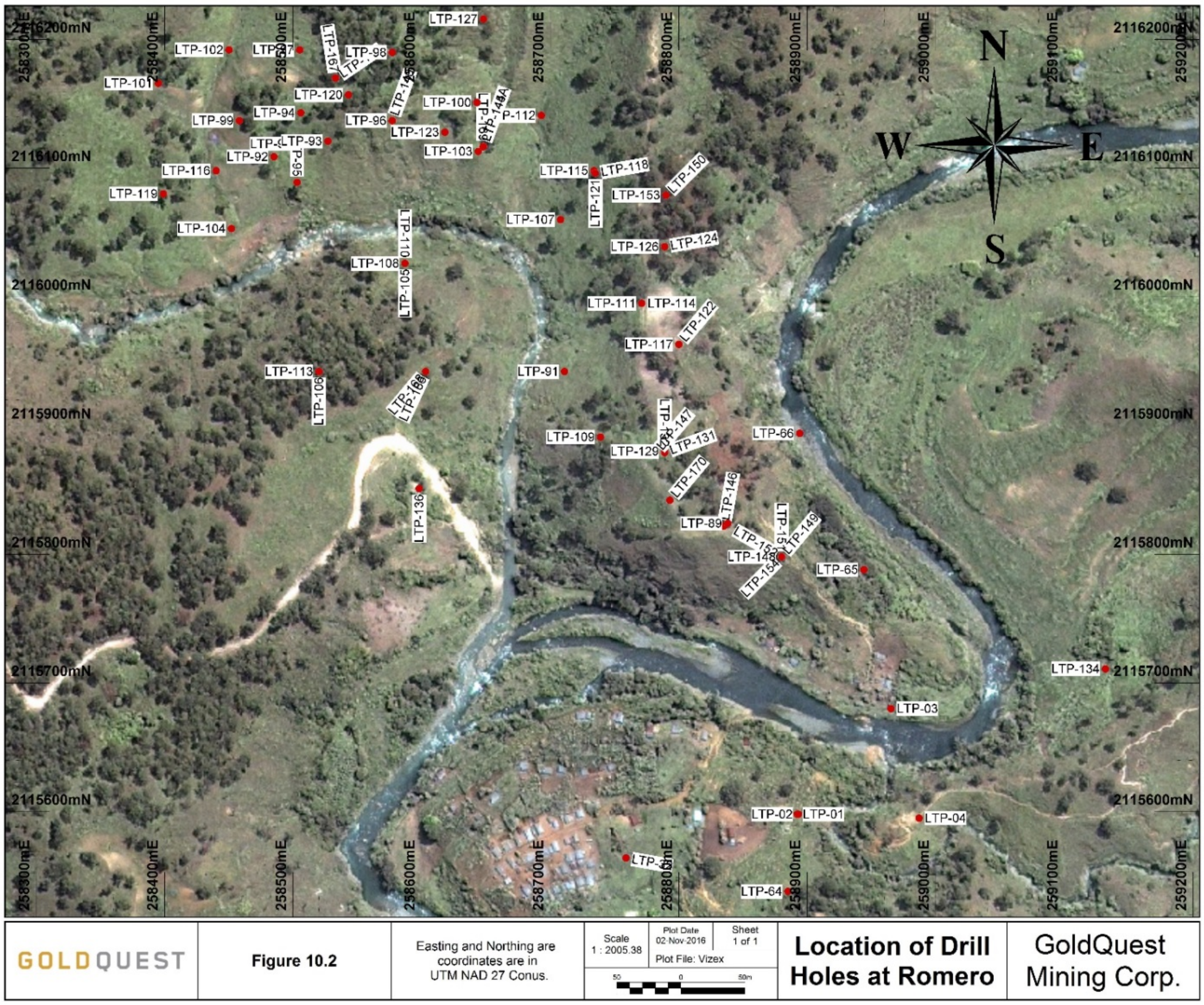
The Phase 8 program comprised seven holes in the Romero Trend for a total of 2,494.70 m (holes LTP-158 to LTP-164). The drilling was carried out between May and October, 2014. All holes targeted new mineralization at geophysical targets outside of the Romero and Romero South deposits.

The Phase 9 program comprised of 6 holes at Romero for a total of 1999.95 m (holes LTP-165 to LTP-170). The drilling was carried out between May and July, 2015. The holes targeted Romero mineralization and were designed to improve resource classification, to gather geotechnical data, to provide material for metallurgical test work and to conduct packer tests.

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 10.2and Figure 10.3, respectively.

Figure 10.2: Location of Drill Holes at Romero



Source: GoldQuest (2016)

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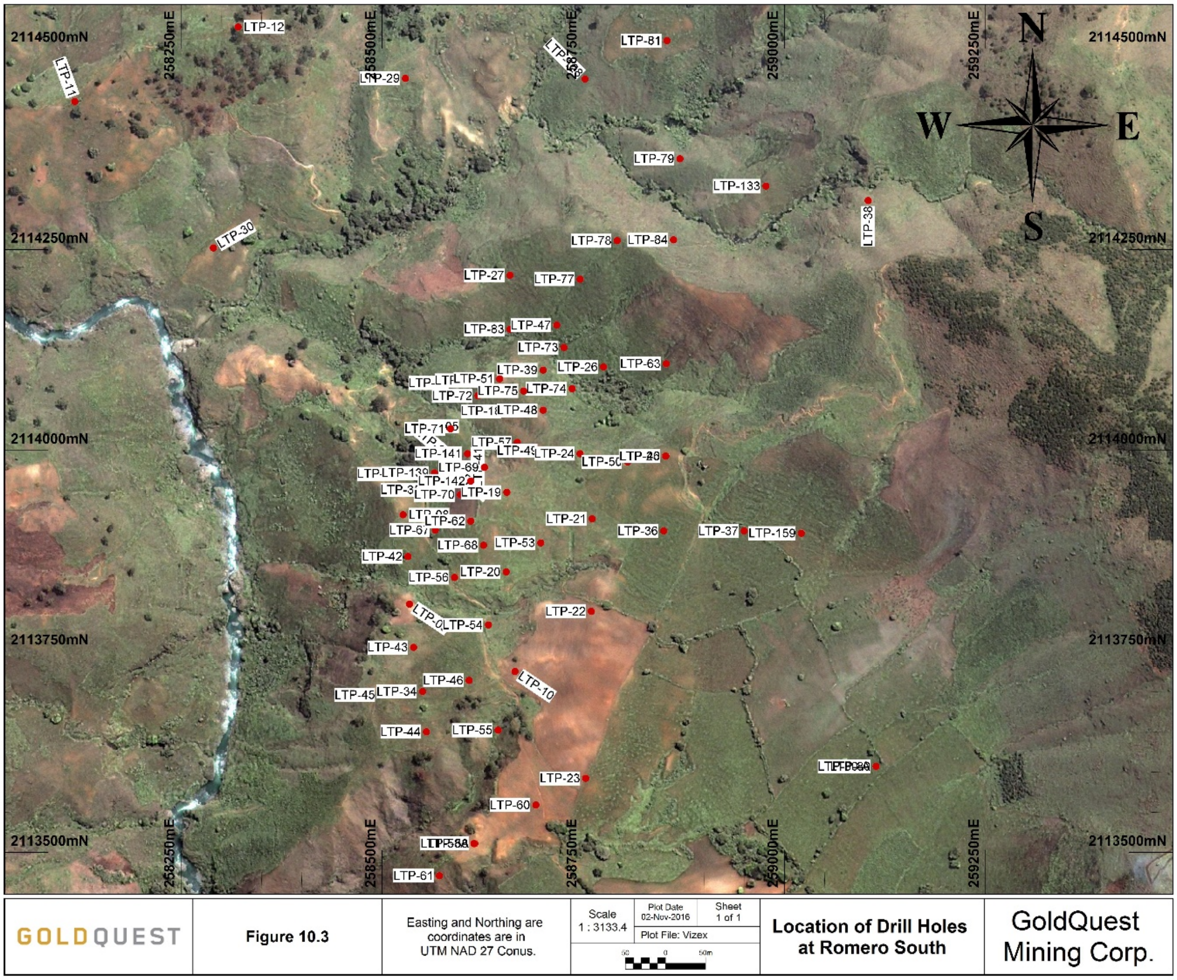
Figure 10.3: Location of Drill Holes at Romero South



Source: GoldQuest (2016)

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

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 10.3and Table 10.4 below. Table Table 10.3 shows those results available as of the 2012 mineral resource estimate (Steedman and Gowans, 2012). Table 10.4shows 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 Phase 6

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	*	
LTP-75	85.78	102	16.22	5.5	*	Escandalosa Sur
including	88	99.68	11.68	7.51	*	

Hole No.	From (m)	To (m)	Interval (m)	Au (g/t)	Cu (%)	Location
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	Hondo Valle
and	177	205	28	0.67	0.13	Hondo Valle
including	195	205	10	1.27	0.12	Hondo Valle

* = no value reported, nsv = no significant values

Source: GoldQuest (2016)

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

Hole_ID	From (m)	To (m)	Interval (m)	Uncut Gold Grade (g/t)	Copper (%)	Gold Grade (cut to 50 g/t)
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
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	

Hole_ID	From (m)	To (m)	Interval (m)	Uncut Gold Grade (g/t)	Copper (%)	Gold Grade (cut to 50 g/t)
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
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

Hole_ID	From (m)	To (m)	Interval (m)	Uncut Gold Grade (g/t)	Copper (%)	Gold Grade (cut to 50 g/t)
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 (2016)

Table 10.5: Significant Gold Intersections from the Romero Project – Late Phase 8 and Phase 9

Hole_ID	From (m)	To (m)	Interval (m)	Uncut Gold Grade (g/t)	Copper (%)	Gold Grade (cut to 50 g/t)
IMP-02	276.7	277.4	0.7	0.25	1.17	Imperial
IMP-03	289.6	303.4	13.8	0.92	0.03	Imperial
and	318.5	335.3	16.8	0.46	0.01	Imperial
IMP-04	195.0	197.0	2.0	1.80	0.02	Imperial
and	217.0	227.0	10.0	0.02	0.15	Imperial
IMP-05	268.48	277.37	7.52	0.41	-	Imperial
IMP-06	227.45	237.9	10.45	1.71	-	Imperial
including	227.45	229.8	2.35	6.65	-	Imperial
IMP-07	388.46	402.34	13.88	0.14	-	Imperial
LB-08	3.1	15.2	12.1	0.53	0.01	Imperial
LTP-165	78.03	90.22	12.19	2.68	0.02	Romero
and	140.21	364.0	223.79	3.03	1.22	Romero
LTP-166	131.27	262.0	130.73	2.08	0.65	Romero
LTP-167	72.0	104.0	32.0	1.87	0.04	Romero
And	147.85	328.0	180.15	1.15	0.98	Romero
LTP-168	124.0	209.0	85.0	2.39	0.41	Hondo Valle
and	252.0	300.0	48.0	1.17	0.46	Hondo Valle
LTP-169	142.0	185.0	43.0	10.10	1.41	Hondo Valle
including	163.0	171.0	8.0	41.23	2.16	Hondo Valle
Including	163.0	165.0	2.0	119.70	5.24	Hondo Valle
LTP-169	233.0	289.0	56.0	0.38	0.10	Hondo Valle
LTP-170	132.3	168.0	35.7	3.66	0.35	Hondo Valle

Source: GoldQuest (2016)

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-170 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 flooded 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), Imperial (IMP-01 to IMP-08) and Loma el Cachimbo (TIR-16-01 to TIR-16-08). 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.

At the time of preparation of the report GoldQuest is actively drilling exploration targets in the Tireo Project, approximately 20 km south of the Romero Deposit. 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 updated 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 and Phase 9 was reviewed for the 2016 Mineral Resource estimate.

11.1 Sampling Method and Approach

The initial indications of mineralization on the Romero 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. Within the Romero concession, a total of 1,587 soil samples were taken in several programs between 2005 and 2015 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 over the two deposits, and 100 m by 100 m, and 50 m by 200 m, and done 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. In 2014 and 2015, sampling was expanded in the north of the deposit and up into the neighbouring concession to the north in a k km north-south by 5 km east-west grid. A smaller 200 by 500 m grid was carried out in the southern part of the concession as well.

Rock sampling was carried out as grab samples of outcrop and float, and channel samples from hand-dug pits and trenches. A total of 1,192 rock samples were collected in the Romero concession. 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 = 4,403 samples captured in the Romero mineralized solid and 600 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. Phase 8 drilling occurred outside of the Romero deposits. Phase 9 included 720 analyses for core as well as 15 blanks, 11 pulp and 9 field duplicate samples, as well as 45 standards were 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 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

Within the Romero concession, there are a total of 1192 rock sample analyses, 1587 soil sample analyses and 14,611 drill core analyses, excluding QC samples.

ALS Chemex analyzed 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 analyzed 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-analyzed by fire assay on a 50 g sample with gravimetric analysis and reported in g/t (method G6Gr-50). Multi-elements were analyzed 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-analyzed 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 analyzed core samples from holes LTP-43 to LTP-157 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-analyzed 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.

Acme analyzed core samples from holes LTP-158 to LTP-170 at its laboratory in Vancouver 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-analyzed by fire assay on a 30 g sample with gravimetric analysis and reported in g/t (method G6Gr-30). Multi-element requests were analyzed 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 7TD). The gold fire assay was used for resource estimation rather than the ICP gold result.

Acme analyzed 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-analyze 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

There were 21 standards used for the 9 drilling phases, as shown in Table 12.1.Table 12.1 The results were evaluated using performance gates. The results were accepted if they were 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 were 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

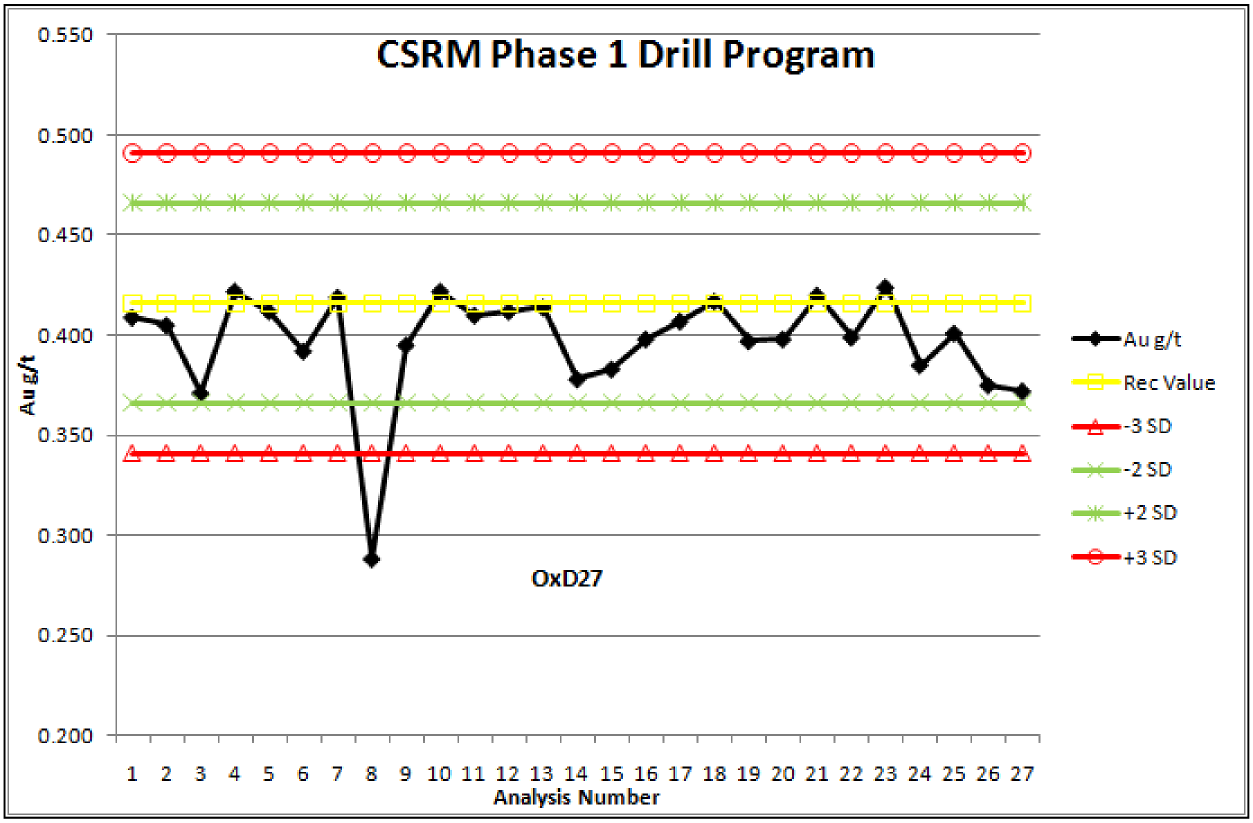
Table 12.1: Standard Reference Material Utilized by GoldQuest

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
CDN-CM-18	5.28 ± 0.35		2.42 ± 0.22			2	Used in Phase 7, 8, 9
CDN-CM-24	0.521 ± 0.056	4.1 ± 0.4	0.365 ± 0.02			2	Used in Phase 7, 8, 9
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, 8
CDN-CM-30	1.30 ± 0.12	15.9 ± 1.3	0.730 ± 0.034	0.273 ± 0.014		2	Used in Phase 8, 9
CDN-ME-1301	0.473 ± 0.044	26.1 ± 2.2	0.299 ± 0.016	0.188 ± 0.010	0.797 ± 0.038	2	Used in Phase 9

Source: GoldQuest (2016)

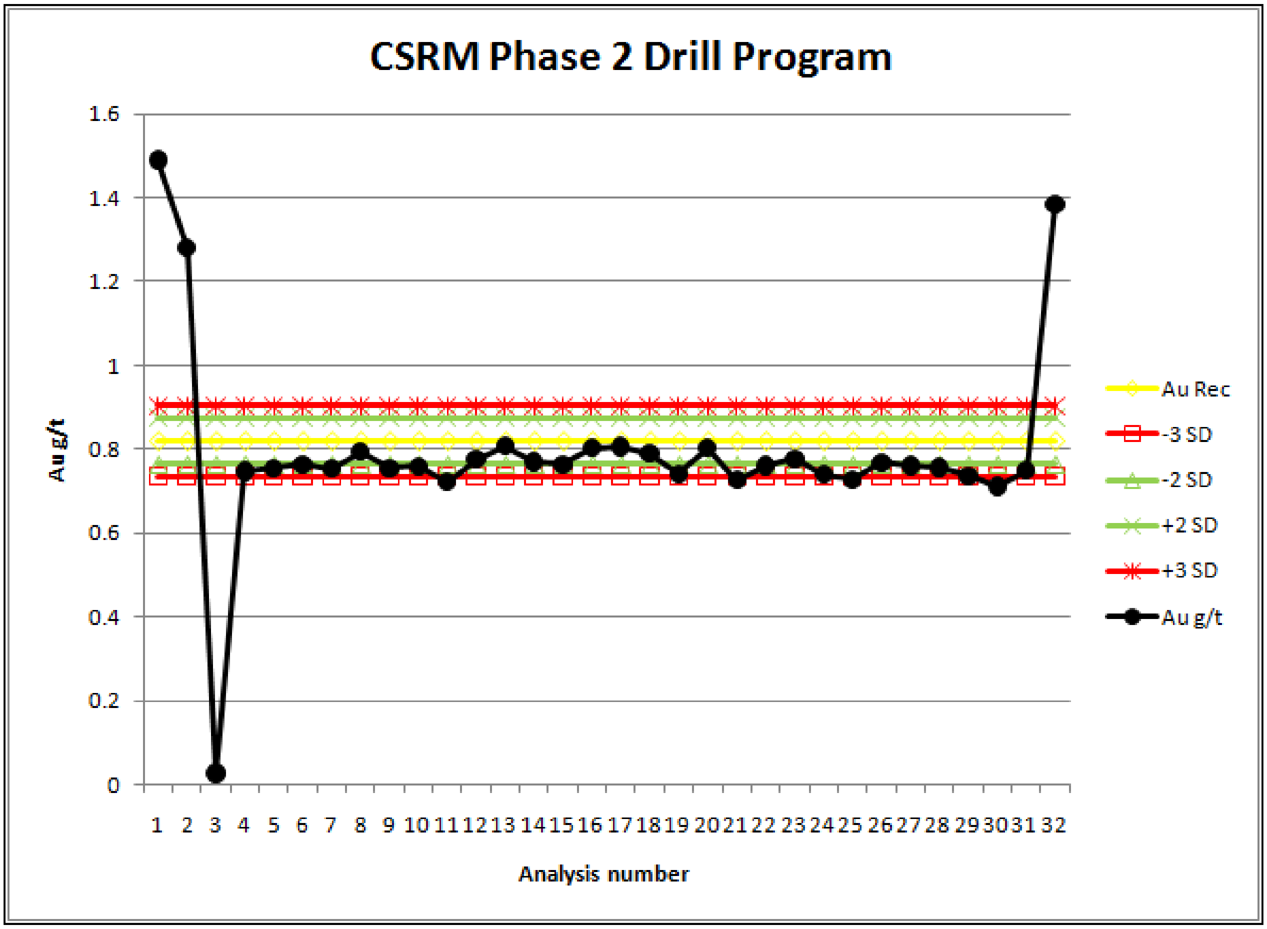
Gold results for the CSRMs for Phase 1 to 3 are shown in Figure 12.1 to Figure 12.3, respectively. There is one exception in the Phase 1 drill program, and four exceptions from the Phase 2 drill program where Au is ± 3 SD.

Figure 12.1: CSRM Plot for Phase 1 Drill Program



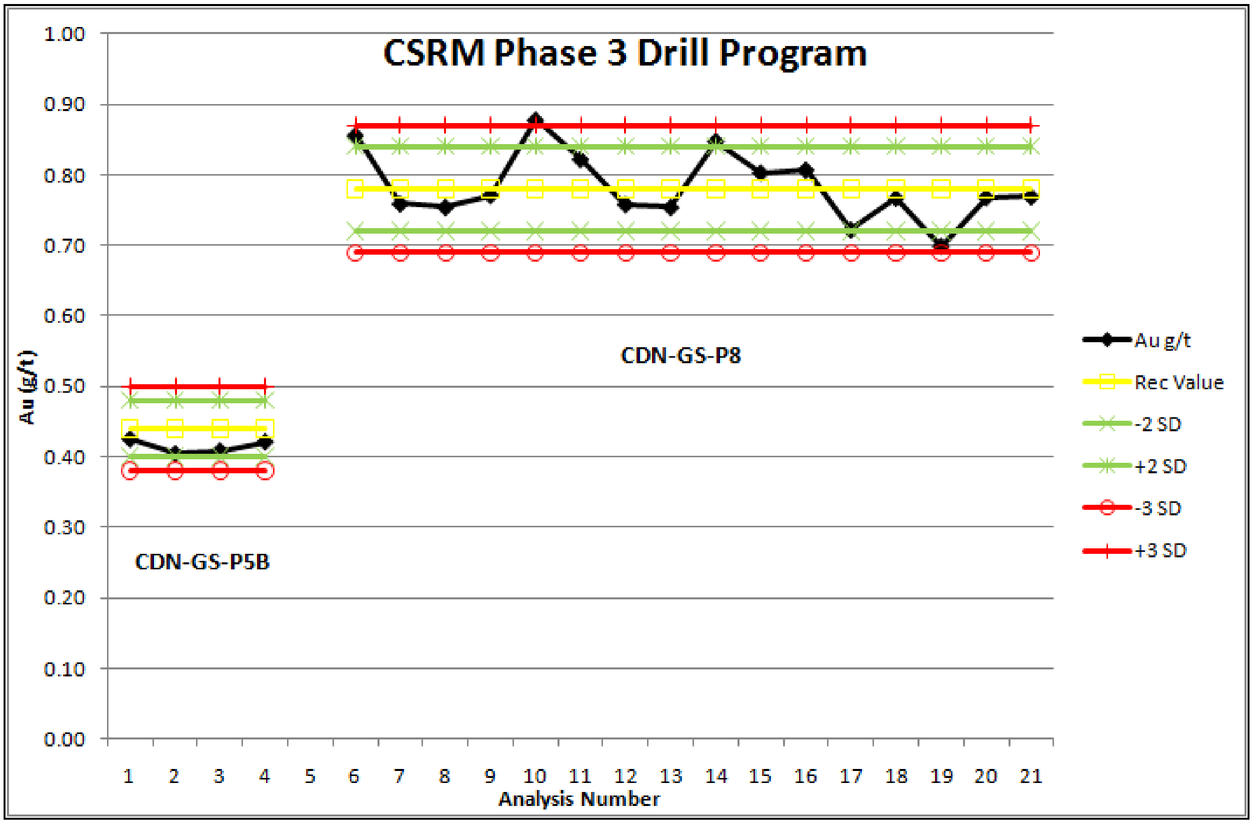
Source: Micon (2016)

Figure 12.2: CSRM Plot for Phase 2 Drill Program



Source: Micon (2016)

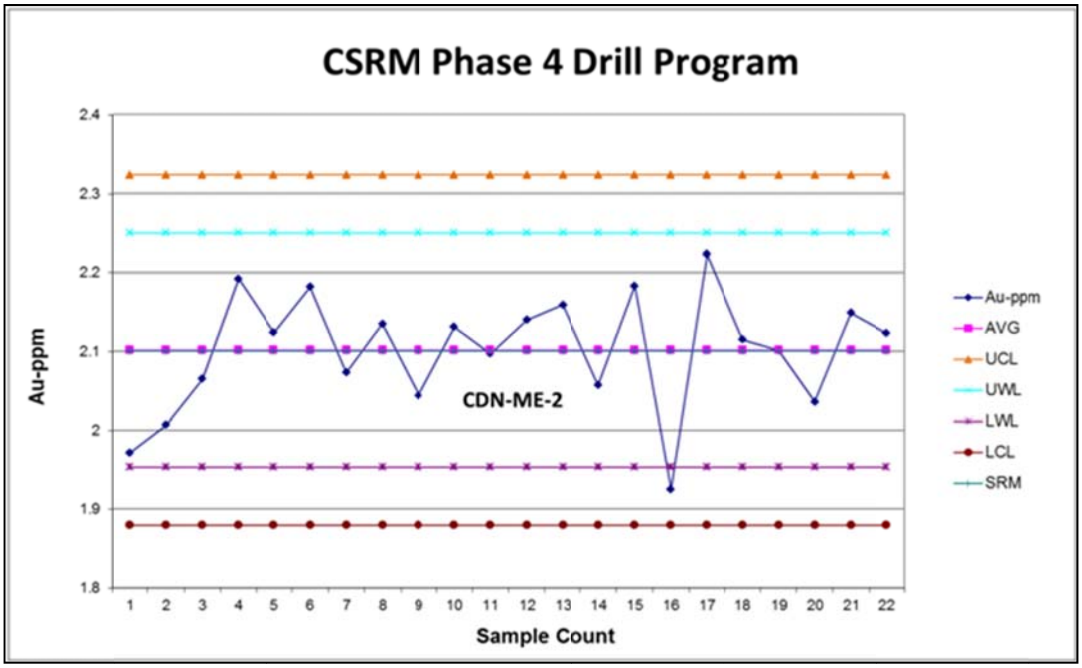
Figure 12.3: CSRM Plot for Phase 3 Drill Program



Source: Micon (2016)

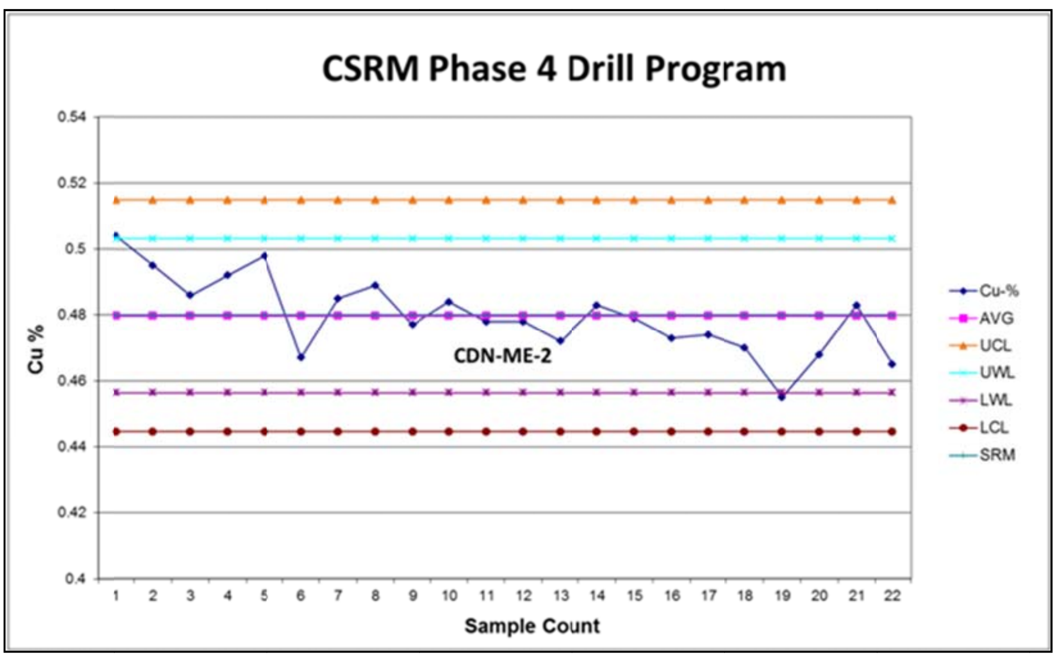
In Phase 4 of drilling, GoldQuest introduced four multi-metal reference standards to monitor the laboratory's analytical performance on both gold and base metals. The more widely used of these is CDN-ME-2 for which the results are shown in Figure 12.4and Figure 12.5. These results demonstrate the laboratory's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

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



Source: Micon (2016)

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

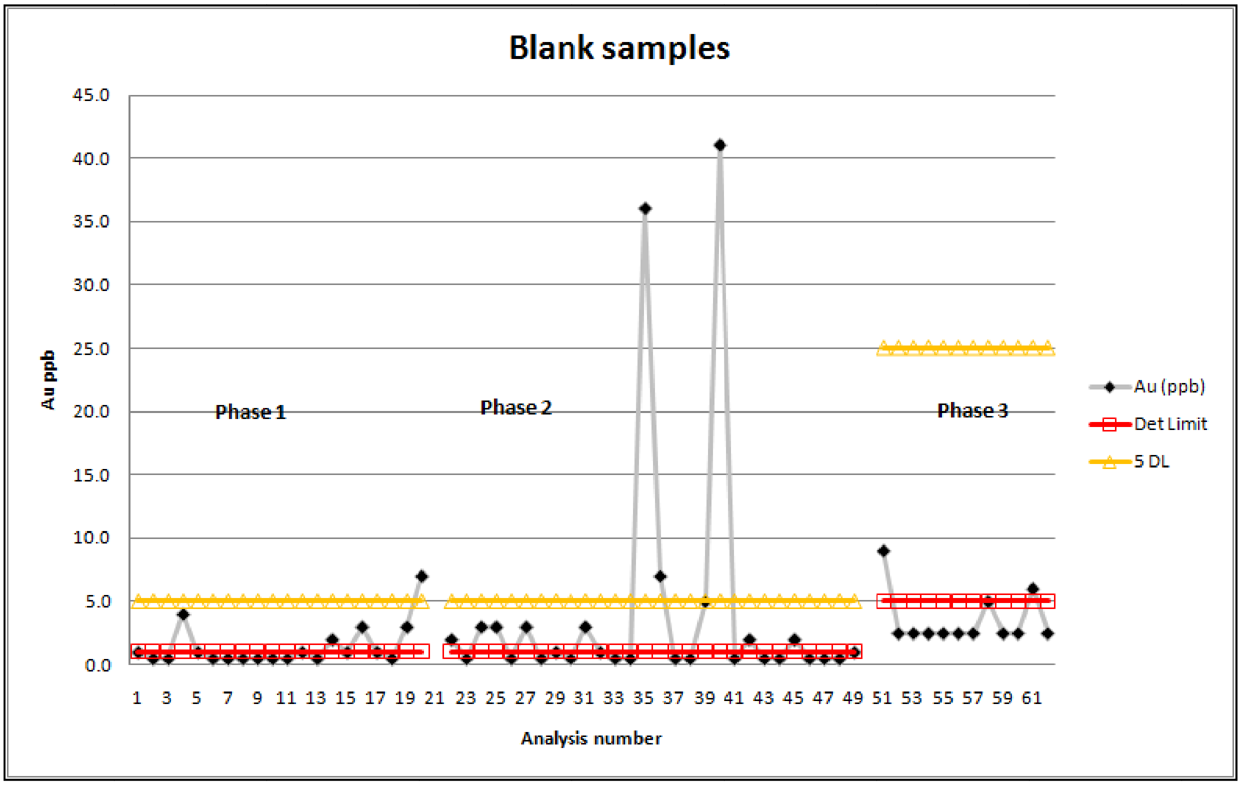


Source: Micon (2016)

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 five 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 reanalyze the intervals at the time.

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



Source: Micon (2016)

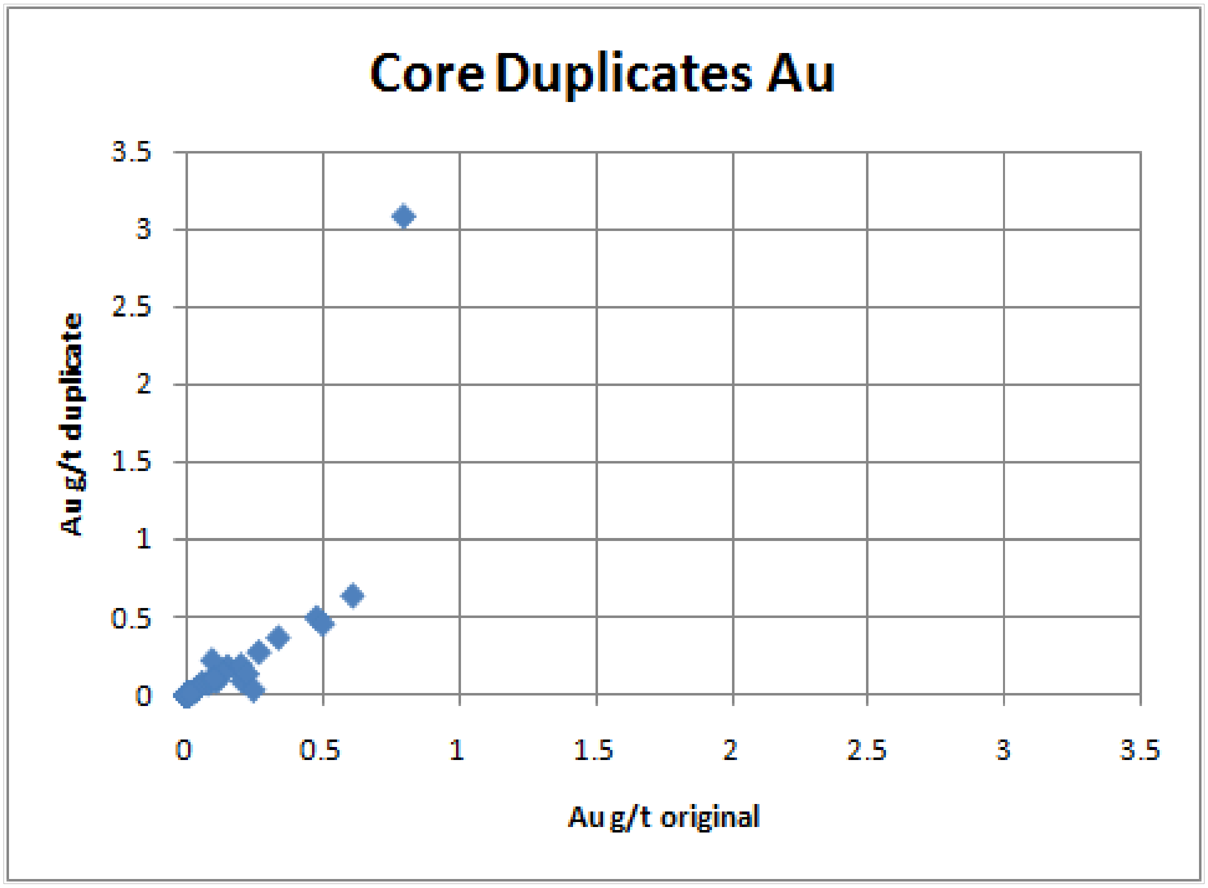
Values below detection were replaced by half the detection limit to avoid negative numbers.

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.

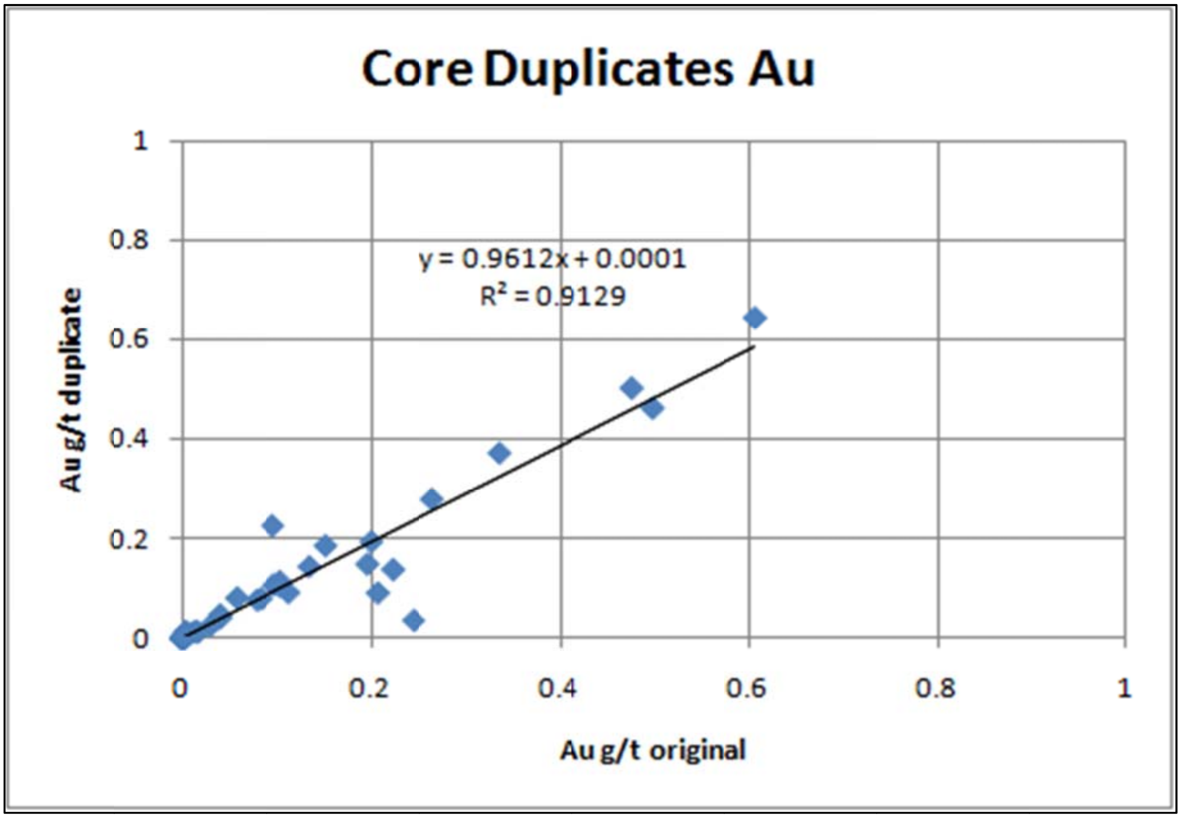
Although there appears to be good repeatability, in 2012, Micon did not recommend continued use of core duplicates due to the inherent geological variability.

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



Source: Micon (2016)

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



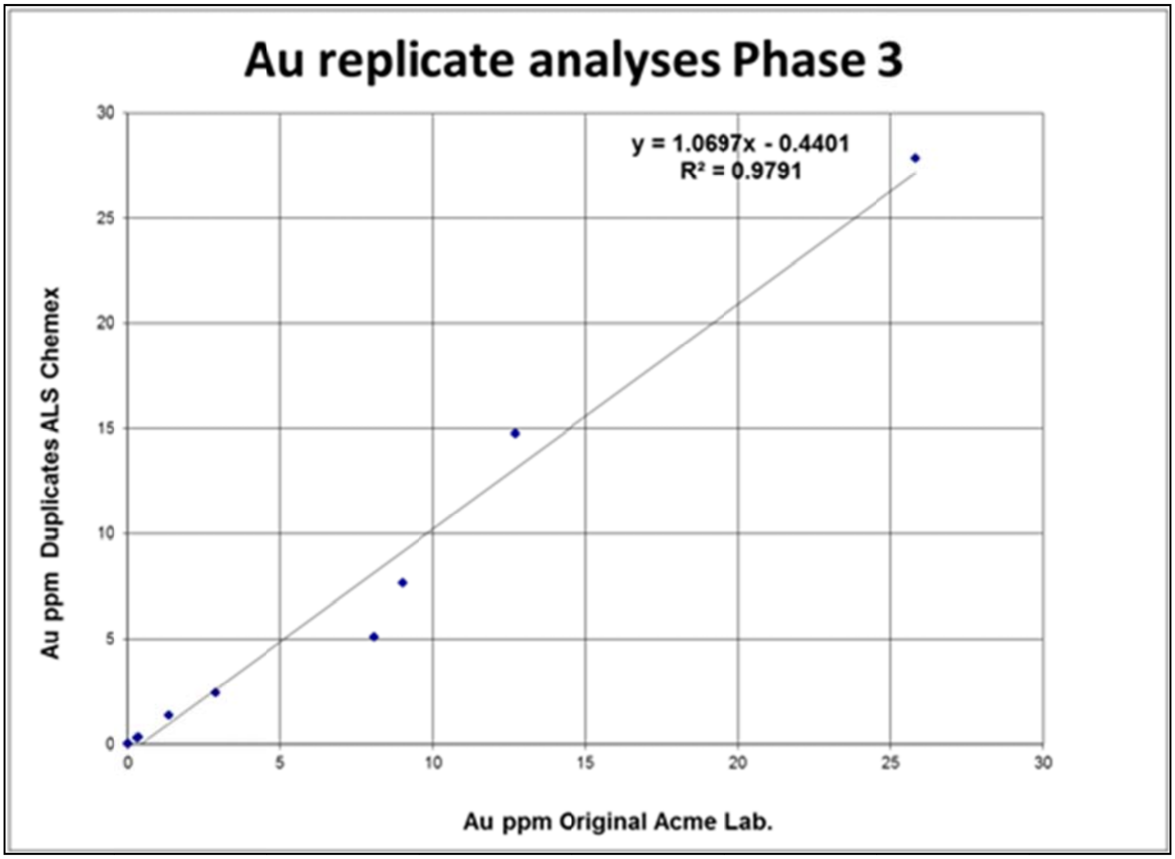
(with one outlier removed)
Source: Micon (2016)

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 gold 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 Figure 12.9 and show a very good correlation between the two laboratories.

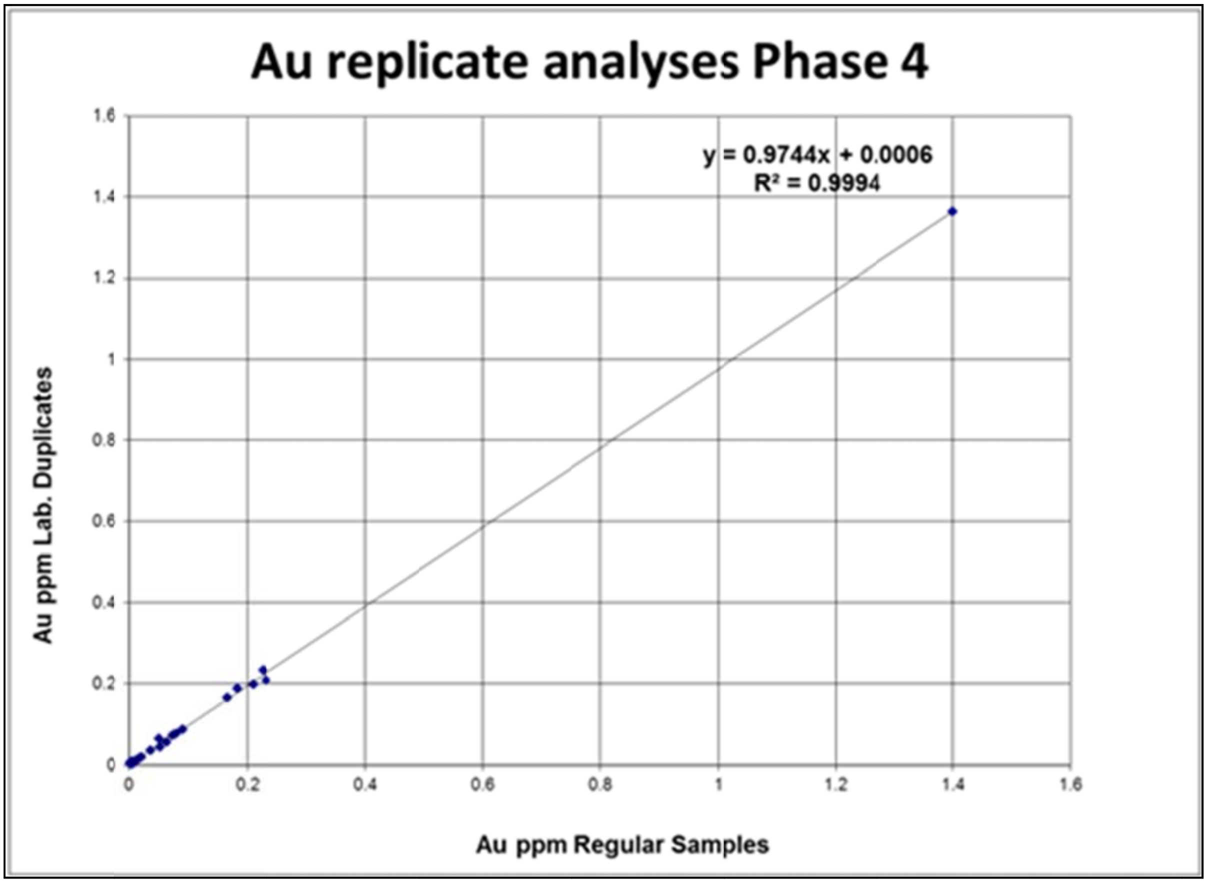
Figure 12.9: Plot of Replicate Analyses for Phase 3 of the Drill Program



Source: Micon (2016)

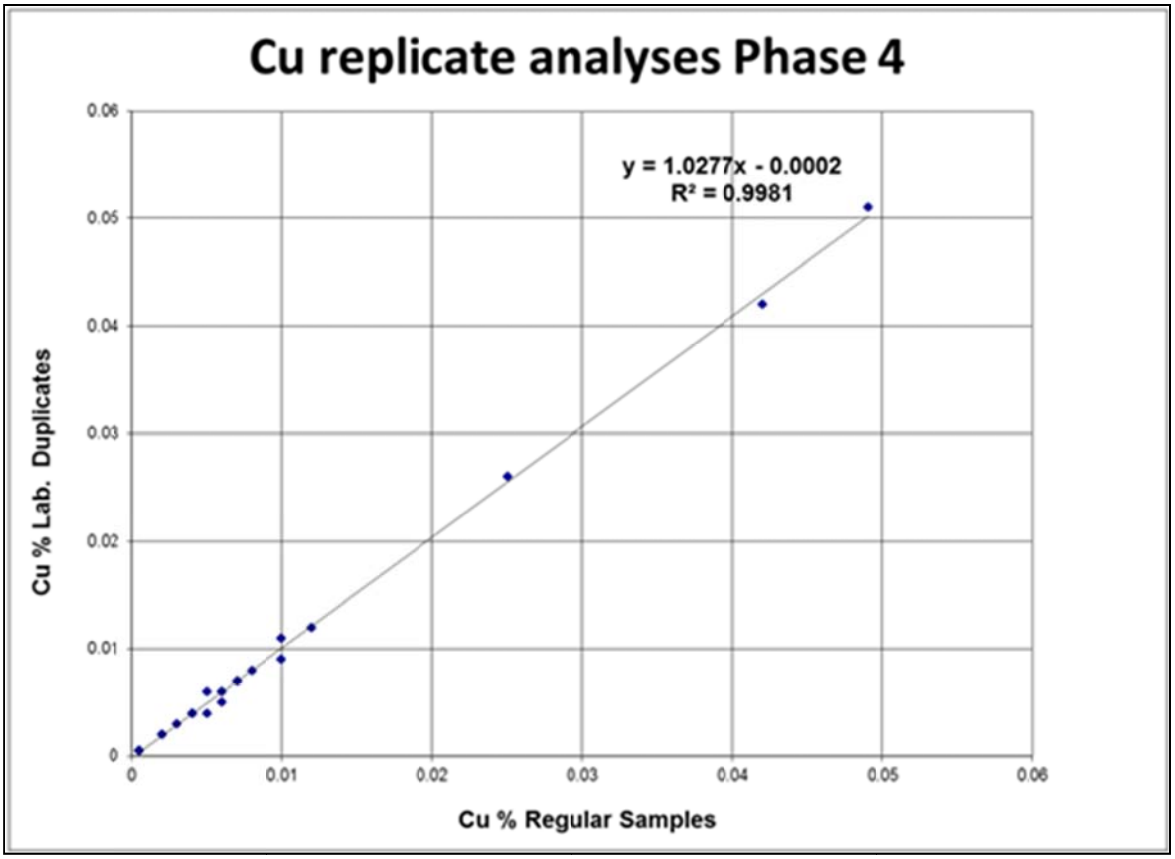
In Phase 4 drilling, replicate analyses were conducted for both gold and base metals. The correlation for all elements (i.e. Au, Ag, Cu, Pb and Zn) is good. Only one sample replicate (i.e. sample number 16978) appeared as an outlier and this is most likely due to a sample switch. The scatter plots for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

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



Source: Micon (2016)

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



Source: Micon (2016)

Later QA/QC plots for phases 5, 6, 7, 8 and 9 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 the 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 the 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 be employed so that contamination can be monitored during this process as well.

Overall, there was a steady improvement noted in the QA/QC protocols from Phases 1 to 3, and 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; and
- 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 Table 12.2

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 (2016)

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:

- The correction of the drill hole collar surveys; some updated collar locations were adjusted using the topographic surface grid provided by GoldQuest;
- A detailed review of down hole surveys, assay data, density measurements; a 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 12.3for 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 (2016)

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 2016 test program completed at the ALS Metallurgical laboratories (ALS) in Kamloops, BC was managed primarily by Met Chem with follow-up work coordinated by JDS. The KM4923 test program was designed to develop a flowsheet that would produce both a copper concentrate, and a gold/silver-rich pyrite concentrate. An economic analysis of the results indicated that a single copper concentrate with increased gold/silver credits would produce a higher return on investment. Subsequent test work at ALS, designated KM5085, was performed on the remaining composites to confirm this conclusion. These results were used to support the metallurgical design criteria developed in the 2015 PEA.

13.1 Summary of Metallurgical Testing

Between 2011 and 2014, GoldQuest conducted grinding and flotation tests on drill core and bulk samples generated by the Romero underground exploration program. This series of test programs investigated the feasibility of producing a copper concentrate, as well as a pyrite concentrate for gold recovery. 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 testing. 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, test program KM4601 was completed on six samples from the previous ALS test program. The work focused on producing a single copper concentrate and included tests aimed at improving recovery of gold and silver.

In 2016, test programs KM4923 and KM5085 were completed in support of a PFS. Five composite samples were constructed and a full suite of test work was performed in KM4923, including comminution, mineralogy, gravity and flotation. A two day pilot plant campaign was also run on a master composite to assess the copper concentrate / pyrite concentrate flowsheet. Products from this campaign were sent for dewatering and environmental testing. After results from an economic analysis determined that a single flotation concentrate was more profitable, follow-up flotation and gravity test work was conducted on the remaining composites (KM5085).

13.2 Historical Test Work

Metallurgical test programs were completed in 2011, 2013, 2014 and 2015 on metallurgical composites selected by GoldQuest. The following list of historical metallurgical test reports was 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);
- ALS Metallurgy Kamloops, KM4076 "Metallurgical Flowsheet Development Testing on Three Composite Samples from the Romero Deposit", June 16, 2014 (ALS, 2014); and
- ALS Metallurgy Kamloops, KM4601 "Metallurgical Evaluation of Samples from the Romero Deposit", April 8, 2015 (ALS, 2015).

13.2.1 Metallurgical Test work, 2011 - 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 Labs 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 PFS flowsheet.

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

Program	Sample	Bwi (kWh/Tonne)	P80 (µm)	Close Screen Size (µm)	Ai	SMC (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 (2016)

13.2.2 Metallurgical Test Work, 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.

13.2.2.1 Composite Characteristics

The six composite samples previously used in KM3650 and KM4076 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

Composite	Assay Results							
	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)

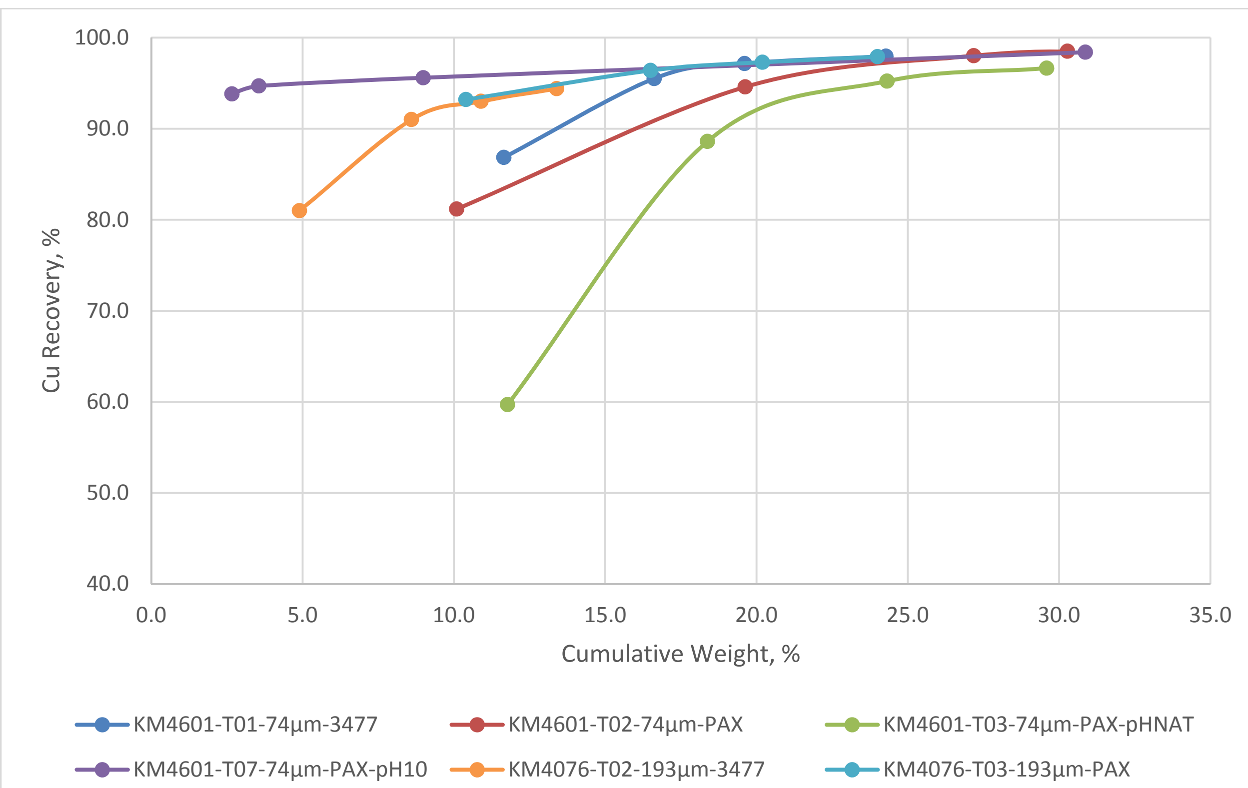
13.2.2.2 Rougher Flotation Tests

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

- Primary grind K80 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 Potassium Amyl Xanthate (PAX) and 3477; and
- Lime addition for pH control.

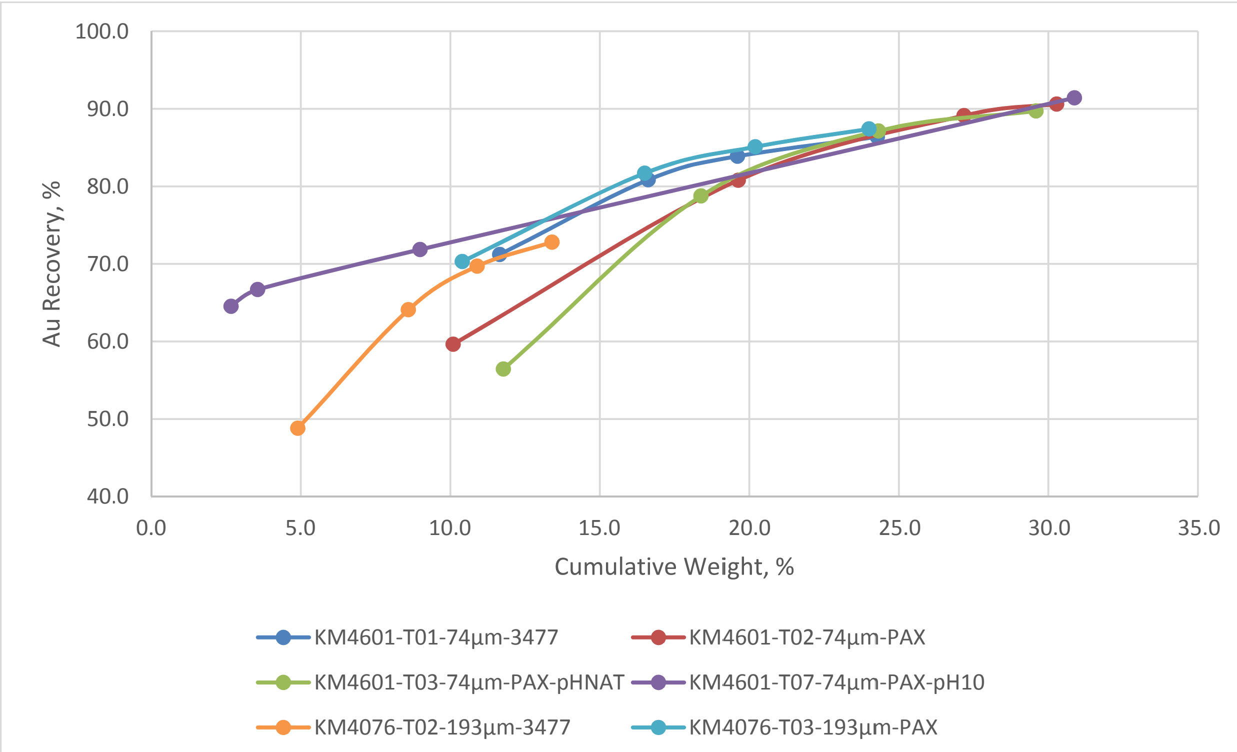
The rougher optimization tests identified that approximately 98% of the copper and 88% of the gold can be recovered with an aggressive mass pull of 30%. This required mass pull was directly correlated to the slow flotation kinetics associated with the gold-bearing mineral particles. A primary grind of 74 µm using PAX at a pH of 10 was selected as the optimal conditions. Figure 13.1 and Figure 13.2 show a comparison of the optimization tests of KM4601, together with relevant historical results, comparing copper and gold recoveries to rougher flotation concentrate mass pull.

Figure 13.1: Rougher Optimization Copper Recoveries versus Mass Pull



Source: ALS KM4076 and KM4601 Test Programs

Figure 13.2: Rougher Optimization Gold Recoveries versus Mass Pull



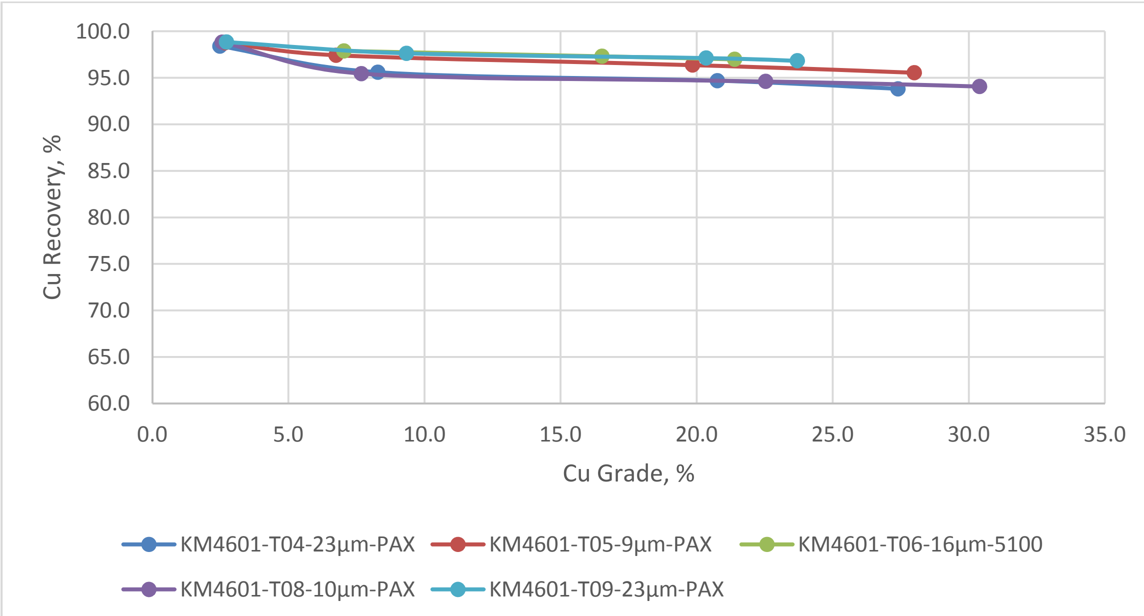
Source: ALS KM4076 and KM4601 Test Programs

13.2.3 Cleaner Flotation Results

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

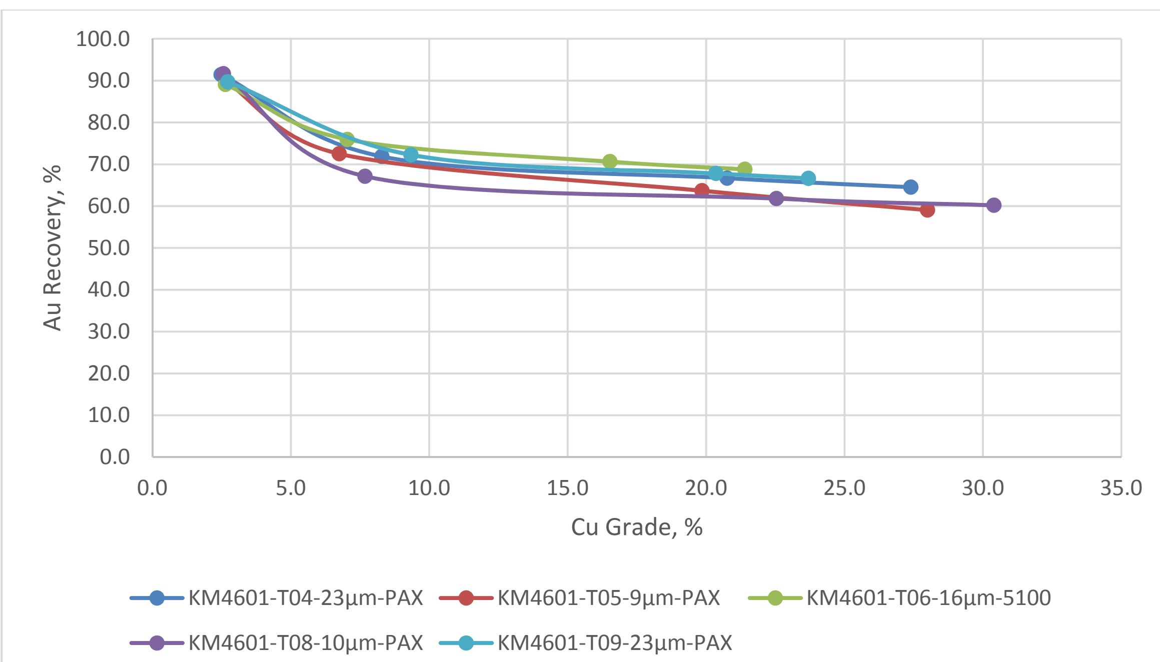
The results shown in Figure 13.3 indicate that an average of 95% of the copper could be recovered to a potentially saleable concentrate of 25% copper. An average of 63% of the gold reported to the final concentrate as shown in Figure 13.4.

Figure 13.3: Batch Cleaner tests - Copper Recoveries



Source: ALS KM4076 and KM4601 Test Programs

Figure 13.4: Batch Cleaner Tests - Gold Recoveries



Source: ALS KM4076 and KM4601 Test Programs

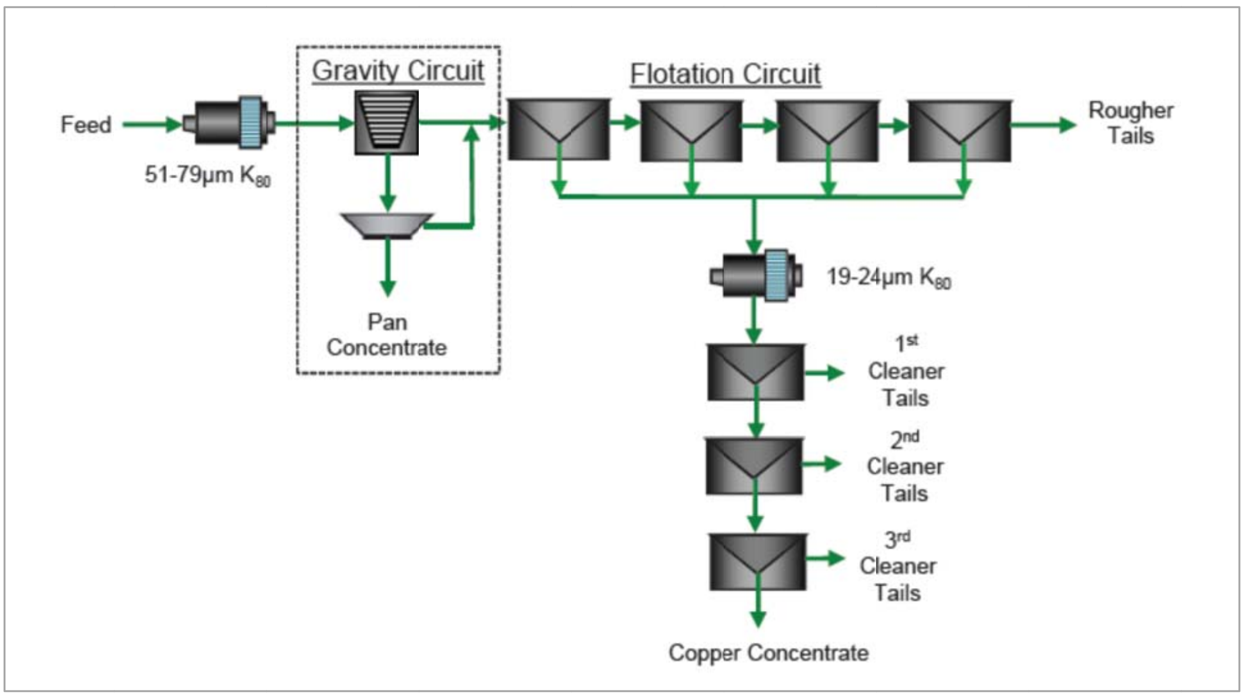
13.2.3.1 Gravity and Flotation Results

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

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

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

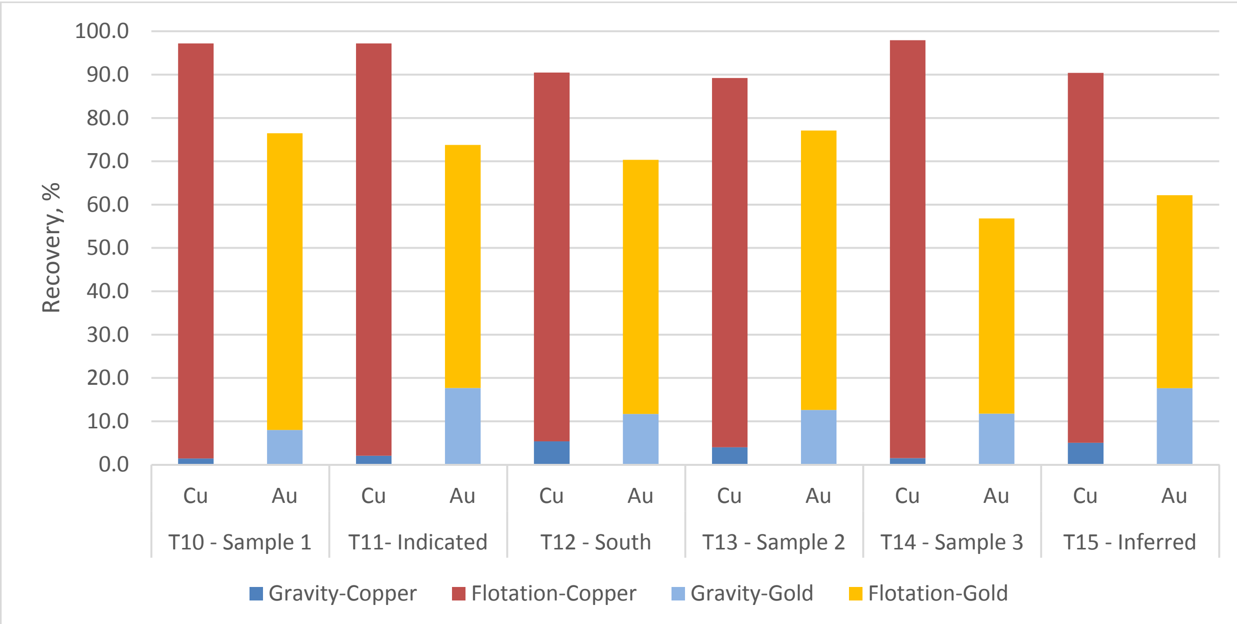
Figure 13.5: Batch Cleaner Flowsheet, taken from ALS Report KM4061



Source: ALS (2015)

The addition of the gravity circuit resulted in up to 17.7% of gold in the feed reporting to a gravity concentrate. Copper recovery was unaffected by the introduction of the gravity circuit. Three tested composite samples did not attain a copper concentrate greater than 20% of copper, due to dilution by pyrite and sphalerite (zinc). The combined gravity and flotation recovery results are presented in Figure 13.6.

Figure 13.6: Combined Gravity and Flotation Recovery Results



Source: ALS (2015)

13.2.3.2 Filtration Results

Filtration testing was conducted on concentrate and tailings composite samples to assess the amenability of the samples to vacuum filtration. The results are presented in Table 13.3. These results are not considered reliable for preliminary filter design purposes owing to the inadequate preparation of the samples.

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 SG	-	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)

13.3 Latest Test Work – KM4923

This technical report is based predominately on the work completed in KM4601 and the Pre-Feasibility Study test work completed in KM4923 and KM5085:

- ALS Metallurgy Kamloops, KM4923 "Metallurgical Evaluation of Five Composites from the Romero Project to Support a Feasibility Study", June 24, 2016 (ALS, 2016a); and
- ALS Metallurgy Kamloops, KM5085 "Further Metallurgical Evaluation of Composites from the Romero Project", August 11, 2016 (ALS, 2016b).

The objective of the 2016 metallurgical test program KM4923 was to continue the development of the Romero flowsheet by conducting Pre-Feasibility level mineralogical and metallurgical test work. The study focused on a copper concentrate / pyrite concentrate flowsheet and included comminution testing, gravity recoverable gold tests, batch rougher and cleaner flotation tests, locked-cycle flotation analysis and pilot plant testing. The results from an economic analysis found that a copper concentrate / pyrite concentrate produced a lower NPV than generating a single copper concentrate with increased gold/silver credits. The objective of KM5085 was to confirm these results with batch gravity / cleaner flotation tests.

13.3.1 Composite Characteristics

Five metallurgical composites representing various areas of the deposit were constructed for flotation test work: Composite 1, Composite 2, Composite 3, Composite 4, and Composite 5. These composites were constructed by combining crushed samples that had been received passing 6-mesh. Each sample had been individually bagged and ranged in weight between 1 to 5 kg.

Comminution composites were constructed with half HQ drill core samples and were named: Comminution Composite 1, Comminution Composite 2, Comminution Composite 3, Comminution Composite 4, and Comminution Composite 5, and reportedly represented similar zones of the deposit as their namesake metallurgical composites.

A single Master composite, Master Composite 1, was constructed by combining 20 kg each of metallurgical composites 1 through 3. The Master composite was used for limited flotation testing.

For the pilot plant campaign, remaining useable comminution composite material, material from metallurgical composites 1 through 5, and additional drill core interval segments were combined to produce the Pilot Plant Composite. In addition, approximately 500 kg of unused crushed drill core intervals were combined to prepare the pilot plant commissioning composite.

The chemical compositions for all twelve composites are presented in Table 13.4 and the mineral content for composites 1 – 5 are shown in Table 13.5.

Table 13.4: Chemical Composition of 2016 Metallurgical Composites

Composite	Assay Results					
	Cu (%)	Zn (%)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)
Composite 1	1.13	0.12	6.3	5.93	3.39	4
Composite 2	1.23	0.18	6.7	7.07	1.57	3
Composite 3	0.76	0.4	7.5	9.05	4.21	8
Composite 4	0.56	0.11	6.8	7.66	1.36	3
Composite 5	0.32	1.33	6.7	8.13	3.5	9
Master Composite	1.09	0.23	6.7	7.34	2.95	5
Pilot Plant Composite	0.9	0.28	6.9	6.92	3.11	5
Comminution Composite 1	1.24	0.07	5.7	5.39	3.54	6
Comminution Composite 2	1.27	0.44	7.2	7.03	2.12	5
Comminution Composite 3	1.34	0.15	8.1	9.89	10.5	10
Comminution Composite 4	0.59	0.34	5.5	6.58	1.01	2
Comminution Composite 5	0.34	1.26	5.5	6.9	2.98	2

Source: ALS (2016a)

Table 13.5: Mineral Content of 2016 Metallurgical Composites

Mineral	Mineral Content (%)				
	Composite 1	Composite 2	Composite 3	Composite 4	Composite 5
Copper Sulphides	3.4	3.8	2.3	1.6	1
Galena	<0.1	<0.1	<0.1	<0.1	0.1
Sphalerite	0.2	0.3	0.7	0.2	2.1
Pyrite	8.5	10	13.5	12.8	12.1
Iron Oxides	0.3	0.4	0.3	0.2	0.2
Quartz	76.5	65.7	63.7	67.7	53.7
Muscovite	3.2	6	9	8.5	10.3
Feldspars	0.4	0.7	1.6	1.3	1.1
Chlorite	6.8	12.1	5.6	7	15.2
Biotite/Phlogopite	0.2	0.1	0.3	0.1	0.1
Barite	<0.1	<0.1	1.9	<0.1	2.8
Calcite	0.2	0.4	0.2	0.1	0.2
Rutile/Anatase	<0.1	<0.1	0.2	<0.1	0.3
Apatite	0.1	0.1	0.1	0.1	0.1
Others	0.3	0.4	0.6	0.4	0.9

Source: ALS (2016a)

13.3.2 Comminution Results

Comminution testing was completed on Comminution Composite 1 – 5. The results are summarized in Table 13.6. The Bond ball mill work index and abrasion index are similar to the values generated in KM3650 and KM4076, while the Axb value was higher than results obtained in KM3650, indicating a softer ore.

Table 13.6: 2016 Metallurgical Testing – Comminution Results

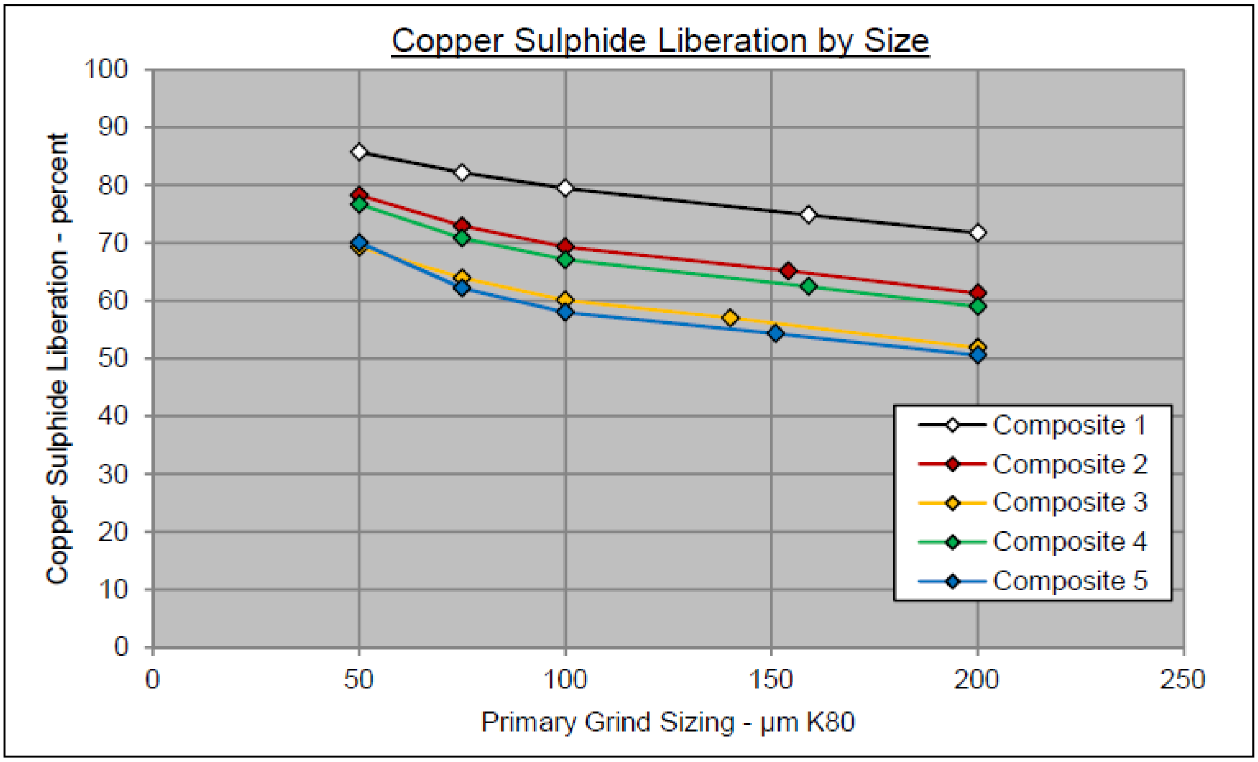
Sample	Bond Crusher Work Index (kWh/Tonne)	Bond Ball Mill Work Index (kWh/Tonne)	BWI Close Screen Size (µm)	Bond Abrasion Index (g)	SMC (A X b)	SMC (ta)
Comminution Composite 1	7.15	15	212	0.22	50.8	0.48
Comminution Composite 2	4.86	15.4	212	0.135	43.5	0.4
Comminution Composite 3	5.3	15	212	0.306	38.7	0.35
Comminution Composite 4	5.57	15.8	212	0.155	45.5	0.42
Comminution Composite 5	5.55	15	212	0.163	53.9	0.51

Source: ALS (2016a)

13.3.3 Mineral Liberation

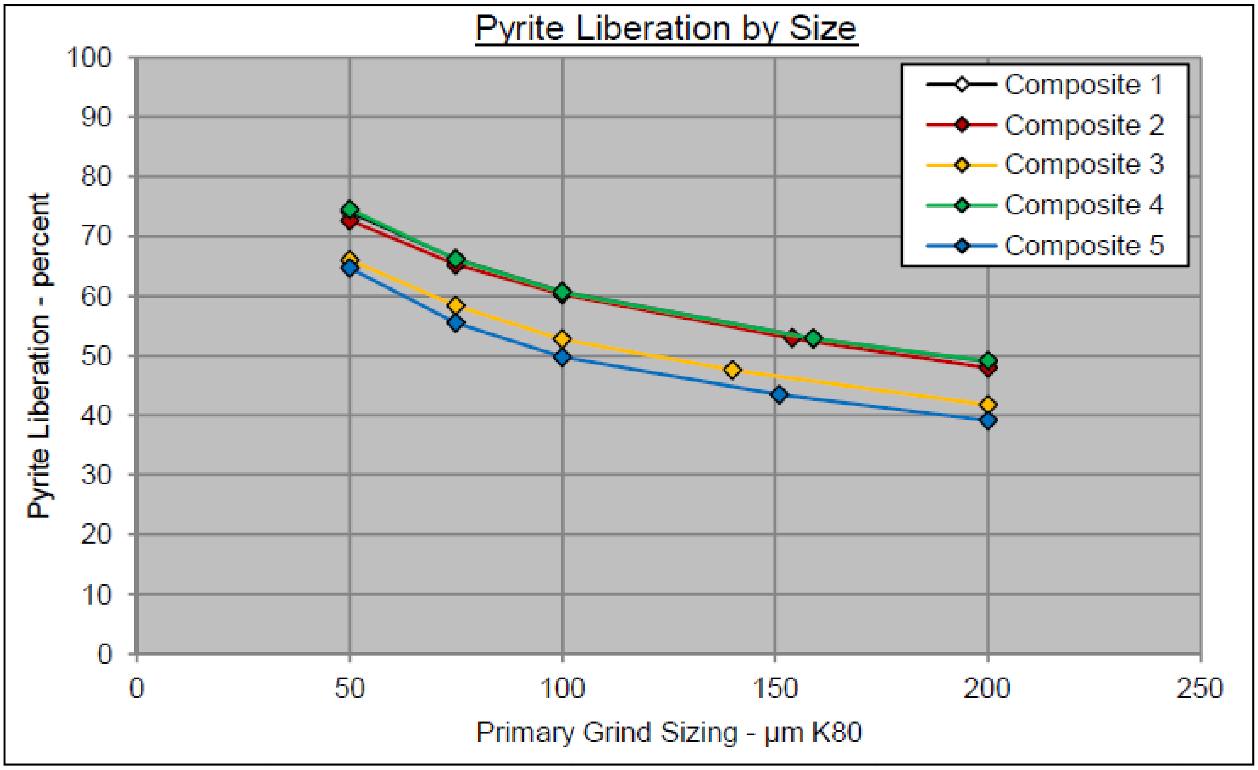
The liberation and fragmentation characteristics of composites 1 – 5 were analyzed using QEMSCAN at a P80 grind size of 150 µm. Copper sulphide minerals were well liberated at 54 to 75%, depending on the composite; while pyrite requires a finer grind size with liberation measuring only 43.5 to 53%. Liberation characteristics were extrapolated for grind sizes between 50 and 200 µm. The results are presented in Figure 13.7 and Figure 13.8. Unliberated copper sulphide and pyrite minerals were primarily associated with non-sulphide gangue minerals such as quartz.

Figure 13.7: Copper Sulphide Liberation Projections



Source: ALS (2016a)

Figure 13.8: Pyrite Liberation Projections

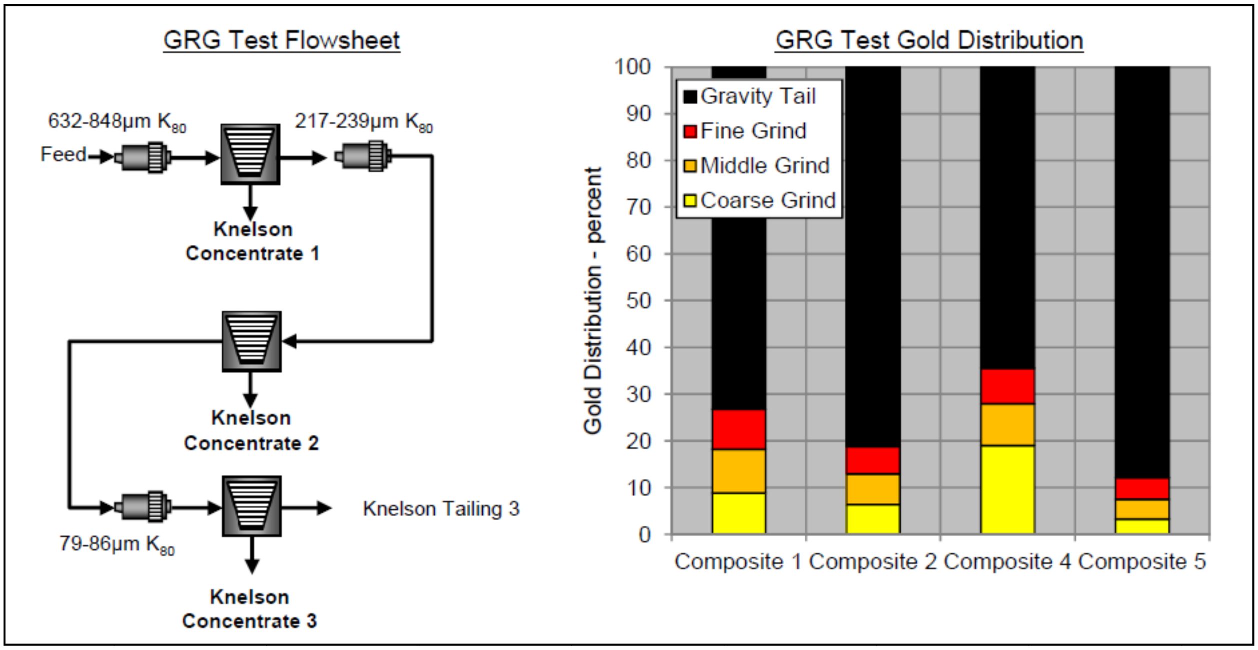


Source: ALS (2016a)

13.3.4 Gravity Recovery

Both batch and gravity recoverable gold (GRG) testing was carried out to assess the validity of including a gravity circuit in the flowsheet. Master Composite sample was processed through a lab-scale Knelson gravity concentrator and the concentrate panned to produce a final product. Five percent of the gold was concentrated into 0.5% of the feed mass. GRG testing was then performed on composites 1, 2, 4 and 5. The results from GRG testing are summarized in Figure 13.9. The results varied between composites. Composites 1 and 4 exhibited higher recoveries, while composites 2 and 5 were considerably less. A size analysis of the products found that a majority of the gold recovered was below 38 µm. Installing a gravity concentrator in the regrind circuit could potentially recover this fine free gold.

Figure 13.9: GRG Results

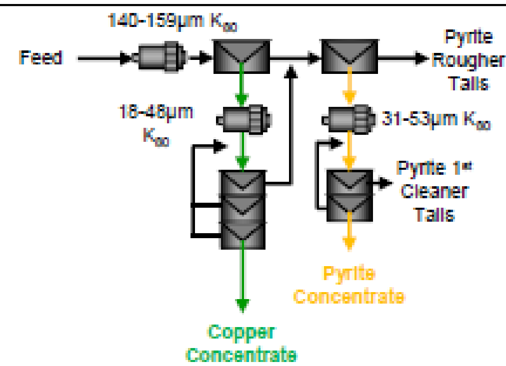


Source: ALS (2016a)

13.3.5 Locked-Cycle Flotation

After batch rougher and cleaner flotation tests were used to optimize conditions, locked-cycle testing was done on composites 1 – 5 to test circuit stability and predict concentrate grades and recoveries. The flowsheet, key test conditions and results are presented in Figure 13.10. Copper concentrate grades varied from 15.4 to 23.6% with average copper and gold recoveries of 88.6% and 50.7% respectively. The subsequent pyrite concentrate achieved an average gold recovery of 26.4%.

Figure 13.10: Two Concentrate Locked-Cycle Test Results



Stage	pH	Redox mV	Reagents - g/tonne		
			Lime	3477	PAX
Primary Grind	10.6-11.0	-13 to 17	1000-1600	-	-
Copper Rougher	11.0	-23 to 17	0-150	4-30	-
Copper Regrind	11.5	-33 to -5	400-450	-	-
Copper Cleaner	11.5	-45 to -6	-	4-21	-
Pyrite Rougher	10.4-10.7	-38 to 40	-	-	100
Pyrite Regrind	9.0-9.3	85 to 116	-	-	-
Pyrite Cleaner	8.8-9.3	-10 to 133	-	-	60

Result Summary

Product	Mass %	Assay - percent or g/t					Distribution - percent			
		Cu	Zn	S	Au	Ag	Cu	Zn	S	Au
Test 27 - Composite 1 - Cycles IV + V										
Flotation Feed	100.0	1.20	0.10	5.94	3.47	-	100.0	100.0	100.0	100.0
Copper Con	5.7	20.4	0.95	41.2	42.9	64	96.8	54.7	39.5	70.4
Pyrite Con	7.2	0.33	0.28	44.2	6.98	-	1.9	20.3	53.2	14.4
Pyrite 1st Clnr Tail	14.2	0.03	0.02	0.79	0.49	-	0.4	2.9	1.9	2.0
Pyrite Ro Tail	73.0	0.01	0.03	0.44	0.63	-	0.9	22.1	5.4	13.2
Test 28 - Composite 2 - Cycles IV + V										
Flotation Feed	100.0	1.29	0.17	6.96	1.52	-	100.0	100.0	100.0	100.0
Copper Con	5.8	21.5	1.26	40.1	17.1	29	95.9	43.5	33.3	65.0
Pyrite Con	8.3	0.37	0.53	50.0	4.45	-	2.4	26.5	59.9	24.4
Pyrite 1st Clnr Tail	12.8	0.06	0.05	1.02	0.22	-	0.5	3.8	1.9	1.8
Pyrite Ro Tail	73.1	0.02	0.06	0.47	0.18	-	1.2	26.3	4.9	8.7
Test 34 - Composite 3 Cycles IV + V										
Flotation Feed	100.0	0.80	0.41	9.11	4.30	-	100.0	100.0	100.0	100.0
Copper Con	2.9	23.6	5.53	38.3	60.5	56	83.7	38.2	12.0	40.2
Pyrite Con	11.8	0.64	0.78	51.7	14.5	-	9.5	22.3	67.2	39.9
Pyrite 1st Clnr Tail	5.3	0.07	0.21	4.51	1.55	-	0.5	2.7	2.6	1.9
Pyrite Ro Tail	80.0	0.06	0.19	2.07	0.97	-	6.3	36.8	18.2	18.0
Test 29 - Composite 4 Cycles IV + V										
Flotation Feed	100.0	0.55	0.11	7.44	1.41	-	100.0	100.0	100.0	100.0
Copper Con	3.1	15.4	1.55	43.6	21.1	24	87.0	44.4	18.1	46.0
Pyrite Con	9.3	0.42	0.17	52.0	4.50	-	7.2	14.7	64.8	29.5
Pyrite 1st Clnr Tail	10.1	0.09	0.05	4.12	0.46	-	1.7	4.7	5.6	3.3
Pyrite Ro Tail	77.5	0.03	0.05	1.10	0.38	-	4.1	36.2	11.5	21.1
Test 35 - Composite 5 Cycles IV + V										
Flotation Feed	100.0	0.31	1.34	7.96	3.48	-	100.0	100.0	100.0	100.0
Copper Con	1.1	21.4	8.86	37.9	96.8	119	79.5	7.6	5.5	32.0
Pyrite Con	10.2	0.34	2.34	48.5	8.08	-	11.2	17.8	62.2	23.7
Pyrite 1st Clnr Tail	6.8	0.03	0.86	2.55	0.98	-	0.7	4.4	2.2	1.9
Pyrite Ro Tail	81.8	0.03	1.15	2.93	1.80	-	8.6	70.2	30.2	42.4

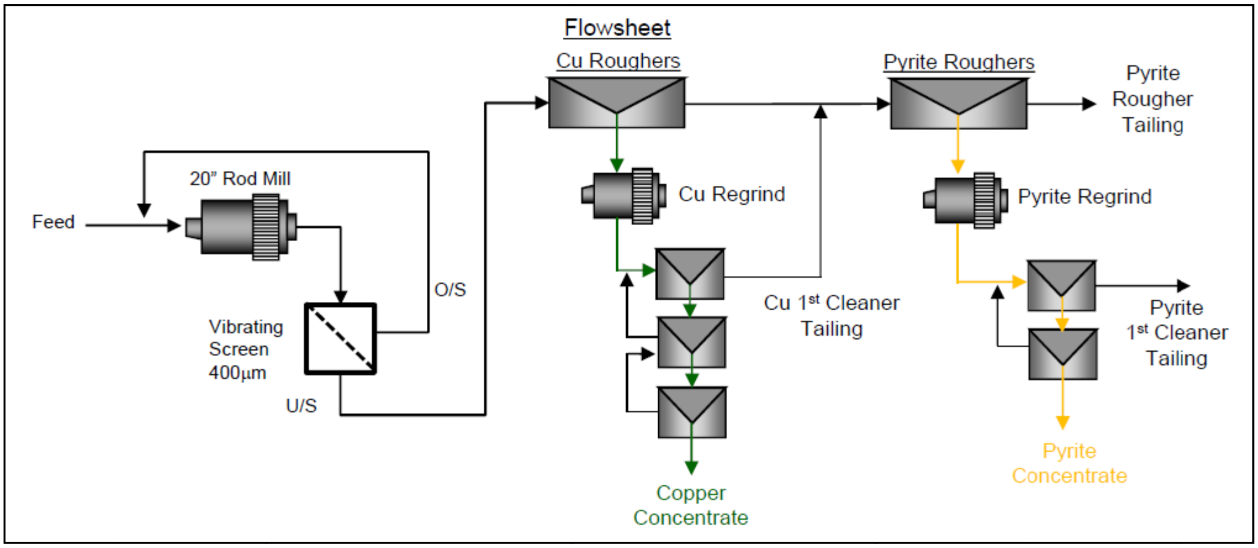
Notes: a) Gold and silver assays are reported in g/tonne; all others are reported in percent.
b) Complete conditions and results can be located in Appendix II.

Source: ALS (2016a)

13.3.6 Pilot Plant Testing

Pilot plant testing was performed over a two day span to evaluate circuit performance and generate samples for dewatering and environmental testing. The pilot plant flowsheet and summary details are shown in Figure 13.11 and Table 13.7respectively. The pilot plant performed well, with results similar to those obtained during batch and locked-cycle testing.

Figure 13.11: Pilot Plant Configuration



Source: ALS (2016a)

Table 13.7: 2016 Metallurgical Testing – Pilot Plant Summary

<table>
<tr><th>Unit Operation</th><th>Equipment</th><th>Grinding Media</th><th>Results</th></tr>
<tr><td rowspan="3">Primary Grinding</td><td rowspan="2">20” Rod Mill</td><td rowspan="2">166-178 kg of mild steel rods</td><td>Average Feed Rate = 104 kg/hr</td></tr>
<tr><td>Total Operating Time = 12.2 hrs</td></tr>
<tr><td>400 µm</td><td>-</td><td>Average P_{80} = 146 µm</td></tr>
<tr><td>Cu Roughers</td><td>4 x 15.8 L Cells</td><td>-</td><td></td></tr>
<tr><td>Cu Rougher Regrind</td><td>1 x 2 L Stirred Mill</td><td>1,800 g of 3.58mm ceramic beads</td><td>Average P_{80} = 29 µm</td></tr>
<tr><td>Cu 1st Cleaners</td><td>8 x 1.5 L Cells</td><td>-</td><td></td></tr>
<tr><td>Cu 2nd Cleaners</td><td>4 x 1.5 L Cells</td><td>-</td><td></td></tr>
<tr><td rowspan="3">Cu 3rd Cleaners</td><td rowspan="3">1 x 1.5 L Cells</td><td rowspan="3">-</td><td>Average Cu Grade = 20.6%</td></tr>
<tr><td>Average Cu Recovery = 88.6%</td></tr>
<tr><td>Average Au Recovery = 54.2%</td></tr>
<tr><td>Pyrite Roughers</td><td>8 x 15.8 L Cells</td><td>-</td><td></td></tr>
<tr><td>Pyrite Regrind 1</td><td>1 x 2 L Stirred Mill</td><td>1,600 g of 5mm ceramic beads</td><td rowspan="2">Average P_{80} = 46.5 µm</td></tr>
<tr><td>Pyrite Regrind 2</td><td>2 x 2 L Stirred Mills</td><td>1,800 g of 3.5mm ceramic beads</td></tr>
<tr><td>Pyrite 1st Cleaner</td><td>2 x 15.8 L Cells</td><td>-</td><td></td></tr>
<tr><td rowspan="3">Pyrite 2nd Cleaners</td><td rowspan="3">2 x 15.8 L Cells</td><td rowspan="3">-</td><td>Average Au Grade = 9.64 g/t</td></tr>
<tr><td>Average Au Recovery = 32.9%</td></tr>
<tr><td>Average Cu Recovery = 8.7%</td></tr>
</table>

Source: ALS (2016a)

13.3.7 Dewatering Test Work

Concentrate and tailings samples from the pilot plant were shipped to Outotec for thickening and filtration testing. The results are summarized in the following two reports:

- Outotec, 115944-T1 “Filtration Test Report – Filtration Testing”, June 24, 2016 (Outotec, 2016a); and
- Outotec, 115944-T1 “Filtration Test Report – Thickening Testing”, June 24, 2016 (Outotec, 2016b).

The results from the study are summarized in Table 13.8 and Table 13.9.

Table 13.8: 2016 Metallurgical Testing – Filtration Results

Sample	pH	Air Drying Time (min)	Filtration Rate (kg/DS/m^{2}h)	Filter Cake Moisture (%w/w)	Filter Cake Thickness (mm)	Pumping Pressure (Bar)	Pressing Pressure (Bar)	Air Pressure (Bar)
Cu Final Concentrate	10	2 – 3	313 - 363	7.5 - 10.6	23.7 - 28.2	6	12	8-Jul
Py Rougher Tailings	10	3-Feb	100 - 151	12.8 - 16.2	23 - 36	6	12	7 - 9.5

Source: Outotec (2016a)

Table 13.9: 2016 Metallurgical Testing – Thickening Results

Sample	pH	Particle Size (P_{80}, µm)	Solids Loading Rate (t/m^{2}h)	Rise Rate (m/h)	Flocculant Dosage (g/t)	Achievable Underflow Density (%w/w)	Achievable Overflow Clarity (ppm TSS)	Maximum Unsheared Underflow Yield Stress (Pa)
Cu Final Concentrate	10	37	0.3 - 1.2	1.27 - 5.08	10-20	67.7 - 71.5	23 - 76	70
Py Rougher Concentrate	10	208	0.15 - 1.2	0.65 - 5.19	10-30	53 - 69	22 - 151	169
Py Rougher Tailings	10	216	0.4 - 1.2	1.76 - 5.27	20 - 40	63 - 71	36 - 103	209

Source: Outotec (2016b)

13.3.8 Cyanidation Testing

Cyanidation bottle roll leach tests were conducted on pyrite rougher concentrate and pyrite rougher tails from pilot plant samples and composites 1 – 5 batch flotation tests. The results are summarized in Table 13.10.

Table 13.10: 2016 Metallurgical Testing – Cyanidation Results

Sample	CN Feed Size (F_{80}, µm)	Leach Time (hrs)	Recovery to CN Feed (%)			Leach Extraction (% of Leach Feed)		
			Mass	Au	Cu	Au	Ag	Cu
Pilot Plant Composite Day: 2 Pyrite Rougher Concentrate	125	48	38.1	39	9	36.8	23.6	25.6
Pilot Plant Composite Day: 2 Pyrite Rougher Tailings	167	48	58.1	7	2	63.6	53.3	19.4
Composite 1 Flotation Test: 27 Pyrite Rougher Concentrate	41	48	21.3	16.4	2.3	71	51.7	51.1
Composite 1 Flotation Test: 27 Pyrite Rougher Tailings	151	48	73	13.2	0.9	86.5	58.9	20.9
Composite 2 Flotation Test: 28 Pyrite Rougher Concentrate	37	48	21.1	26.2	2.9	42.2	38.8	43.1
Composite 2 Flotation Test: 29 Pyrite Rougher Tailings	154	48	73.1	8.7	1.2	83.3	19.7	20
Composite 3 Flotation Test: 34 Pyrite Rougher Concentrate	31	48	17.1	41.8	10	33	26.4	37.2
Composite 3 Flotation Test: 34 Pyrite Rougher Tailings	140	48	80	18	6.3	50.1	20.7	14.5
Composite 4 Flotation Test: 29 Pyrite Rougher Concentrate	33	48	19.4	32.8	6.9	55.8	22.4	41.6
Composite 4 Flotation Test: 29 Pyrite Rougher Tailings	159	48	77.5	21.1	4.1	81.3	19.3	17.3
Composite 5 Flotation Test: 35 Pyrite Rougher Concentrate	37	48	17	25.6	11.9	69.3	40.8	43.4
Composite 5 Flotation Test: 35 Pyrite Rougher Tailings	151	48	81.8	42.4	8.6	64.2	31.9	14.1

Source: ALS (2016a)

13.4 Latest Test work – KM5085

Upon completion of KM4923, an economic analysis was done to determine whether the copper / pyrite concentrate process flowsheet was more favourable than the single concentrate option investigated in KM4601. The results indicated that a single copper concentrate, with high gold/silver credits, would be more economically favourable.

13.4.1 Flotation Testing

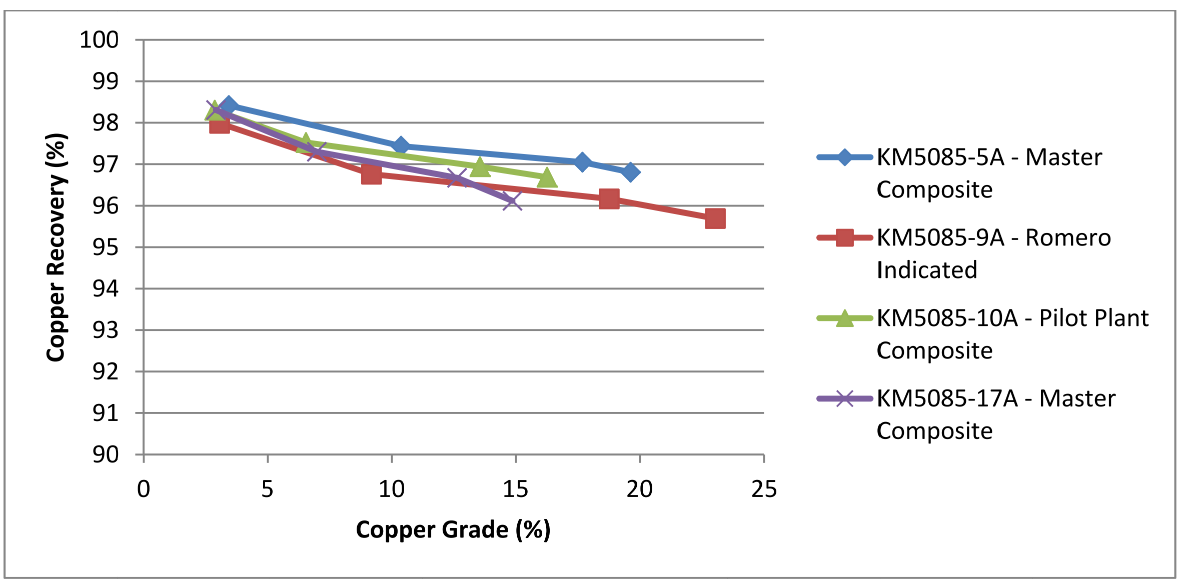
The flowsheet and test conditions developed in KM4601 were applied to the remaining composites from KM4923. The results for each composite are summarized in Table 13.11, Figure 13.12 and Figure 13.13.

Table 13.11: Single Copper Concentrate Flotation Results – KM5085

Sample	Grind Size (P_{80}, µm)	Gravity Recovery (%)	Rougher Flotation Time (min)	Rougher pH	Concentrate Grade		Concentrate Recovery	
					(Cu,%)	(Au, g/t)	(Cu,%)	(Au,%)
Master Composite ((KM5085-05)	66	11.2	10	10	19.6	39.3	96.8	66.2
Pilot Plant Composite (KM5085-10)	67	9.7	10	10	16.3	43.6	96.7	70.1
Romero Indicated (KM5085-09)	74	9.1	10	10	23	57.7	95.7	65.5
Master Composite (KM5085-17)	70	14.6	10	10	14.9	38.3	96.1	74.5

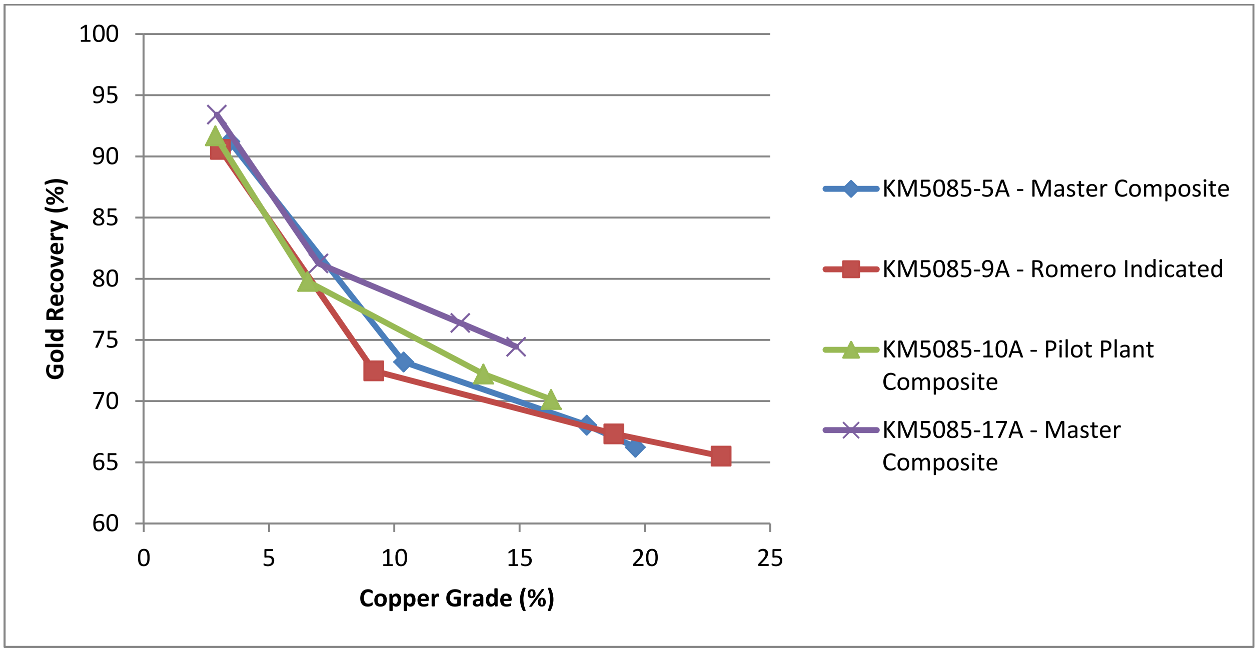
Source: ALS 2016b

Figure 13.12: Copper Grade vs. Copper Recovery – KM5085



Source: ALS 2016b

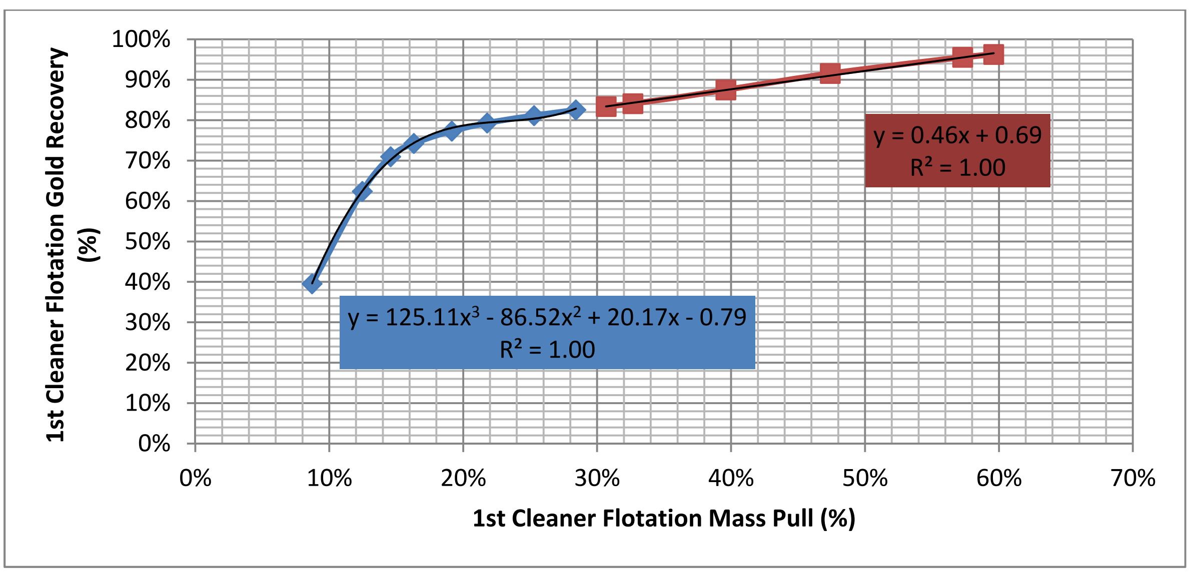
Figure 13.13: Copper Grade vs. Gold Recovery – KM5085



Source: ALS 2016b

The batch cleaner tests indicated that the main source of gold loss was in the first cleaner tails. A first cleaner flotation kinetic test was completed to identify what mass pull should be targeted to reduce these loses. The kinetic profile is presented in Figure 13.14.

Figure 13.14: First Cleaner Flotation Gold Kinetics – KM5085-21A



Source: ALS 2016b

13.4.2 Dewatering Test Work

Bulk rougher and first cleaner tailings samples were generated during KM5085 to conduct 3rd party thickening, filtration and rheology testing. Flotation tests were performed at a primary P_{80} of 75 µm and a regrind P_{80} of 23 µm using the test procedure discussed in Section 13.4.1. The samples were then shipped to FLSmidth for testing. The results are summarized in the following report: FLSmidth, "GoldQuest Romero Project" Oracle Project Reference #9232502977, August 25, 2016 (FLS, 2016).

The results of the thickening test work showed that an anionic polyacrylamide flocculant with a very high molecular weight and medium charge density produced the best overflow clarity and settling velocities. Flux testing showed the optimum feedwell suspended solids concentration for flocculation to be 8-wt% for both samples. The thickening results are summarized in Table 13.12.

Table 13.12: FLS Dewatering Test work – Thickening Results

Sample	Solids Loading Rate ($t/d/m^2$)	Design Unit Area ($m^2/t/d$)	Min. Mud Residence Time Required (hrs)	Flocculant Dosage (g/t)	Design Underflow Density (%w/w)	Design Overflow Clarity (ppm TSS)	Yield Stress for Rake Design (Pa)
Rougher Flotation Tailings	20	0.05	0.6	25	60	<100	<50
1st Cleaner Flotation Tailings	20	0.05	2	25	60	<100	<50

Source: FLS 2016

Pressure and vacuum filtration tests were performed on the rougher tailings, first cleaner tailings, and a 50/50 split of the tailings samples on a dry solids basis. Testing used a feed solids concentration of approximately 60-wt% for the rougher tailings and 50\50 split fraction, and 55-wt% solids for the cleaner tailings, as prescribed by thickening and rheology test work. The results are presented in Table 13.13.

Table 13.13: FLS Dewatering Test work – Filtration Results

Sample	Pressure Filtration (Filter Press)			Vacuum Filtration (Disc Filter)		
	Cake Moisture (%w/w)	Chamber Thickness (mm)	Filtration Rate ($kg/m^2/hr$)	Cake Moisture (%w/w)	Cake Thickness (mm)	Filtration Rate ($kg/m^2/hr$)
Rougher Flotation Tailings	13.1 - 19.6	50	123 – 405	21 – 23	8	211 – 388
1st Cleaner Flotation Tailings	20.2	50	183	26.5	8	168
50\50 Split	14.8 – 17.4	50	158 - 218	22.5	8	152

Source: FLS (2016)

13.5 Process Design

The process design criteria and proposed flowsheet were based on test 11 of ALS test program KM4601 (KM4601-11GCl) and the results of test program KM5085. Results from previous Romero test programs, industry standards, and vendor recommendations were also used where test work was not available. The flowsheet includes crushing, grinding, gravity, flotation, dewatering and filtration unit operations.

13.5.1 Comminution Circuit

Based on the mineralized material hardness noted from the grinding comminution test work, it was determined that a jaw crusher would be suitable for the primary crushing stage to reduce the underground material in one stage from 80% passing 600 mm to 150 mm. 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 LOM daily tonnage, with an assumed plant availability of 90% and targeting a final P80 particle size of 75 µm. A summary of mill sizing data is shown in Table 13.14.

The diameter, length and motor size for the mills were confirmed by selected vendors.

Table 13.14: Process Design Criteria

Mill Process Design Parameters	Unit	Value	Mill Operating Parameters	Unit	Value
Operating Parameters			**SAG Mill Specifics**		
Daily Dry Tonnage	t/d	2,800	Number of SAG Mills	-	1
Availability	%	92	Mill Outside Diameter	ft	18 (5.5 m)
Hourly (Instantaneous) Throughput	t/h	126.8	Mill Length-EGL	ft	8 (2.4 m)
Ore Specific Gravity	-	2.94	Percent of Critical Speed (VS)	%	72
Ball Mill Work Index	kWh/t	15	Mill Speed	rpm	13.2
Abrasion Index	-	0.195	Percent Volume Total Charge	%	25
Feed Size,K80	µm	120,000	Percent Volume Steel Charge	%	12
Final Grind Size, P80	µm	75	Tonnes of Steel Charge	t	35.4
SAG Mill Parameters			Ore Specific Gravity	-	2.94
Final Grind Size	µm	1,000	Slurry Pulp Density	% sol	72
SAG Efficiency Factor	-	1.5	Slurry Specific Gravity	-	1.91
Transmission Loss Factor	-	1.05	Charge Specific Gravity	-	4.12
Unit Power Consumption	kWh/t	6.8	Charge Density	lb/ft^3	257
Power Required	kW	862	Mill Power Draw	kW	933
	hp	1,156	Mill Power Draw	hp	1,252
Installed Power	hp	1,250			
% Power Utilized	%	92	Ball Mill Specifics		
Ball Mill Parameters			Number of Mills	-	1
Discharge Size P80	µm	75	Mill Diameter	ft	14 (4.3 m)
EF1 - Dry/Wet Grind	-	1	Mill Length	ft	23 (7.0 m)
EF2 - Open/Closed Circuit Grinding Factor	-	1	Mill Diameter Inside Liners	ft	13.5
EF3 - Diameter Efficiency Factor	-	0.914	Mill Length Inside Liners	ft	21.5
EF4 - Oversized Feed Factor	-	1	Volume Inside Mill	ft^3	3,077
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.	156
EF7 - Low Ratio of Reduction Factor	-	1.011	Percent of Mill Critical Speed	%	70
EF8 - N/A - Rod Mill Only	-	1	Mill Speed	rpm	15
Transmission Loss Factor	-	1.05	Bulk Density of Ball Charge	lb/ft^3	290
Unit Power Consumption	kWh/t	12.22	Make-up Ball Size	in	3
Power Required	kW	1,549	Ball Size Factor	-	0.49
	hp	2,077	Kilowatts per tonne Balls	kW/t	10.4
Installed Power	hp	2,250	Mill Power Draw	kW	1,627
Power Utilized	%	92	Mill Power Draw	hp	2,182

Source: JDS (2016)

13.5.2 Gold Recovery

The results from KM5085 indicated that the inclusion of a gravity concentrator would recover 9.1 to 14.6% of the gold (Tests KM 5085-05, -09, -10, and -17). Based on the gravity test results for Composite 1 and 4, it is recommended that a gravity circuit should be included in the process flowsheet. The location should be either in the primary grinding circuit, the regrind circuit or both. More test work is required to determine the most economical option.

13.5.3 Flotation

The flotation circuit design criteria were based on ALS KM4601-GCI11 flowsheet, reagent dosages, concentrate mass pulls and flotation times. The test parameters for KM4601-GCI11 are shown in Table 13.15. These conditions were used as the standard flowsheet in KM5085. A primary grind P_{80} of 75 µm was selected as the feed size for the flotation circuit, while a P_{80} of 23 µm was used to design the regrind circuit. The flowsheet included rougher flotation, followed by regrind of the rougher concentrate and three stages of flotation cleaning.

Table 13.15: KM4601-GCI11 Test Parameters

Stage	Reagents Added g/tonne				Time (minutes)		pH	Redox
	Lime	PAX	MIBC	Grind	Cond.	Float		
Natural							7	32
COPPER CIRCUIT:								
Rougher 1	350	5	15		1	2	10	82
Rougher 2	√	4	15		1	2	10	43
Rougher 3	√	3	15		1	2	10	52
Rougher 4	√	2	15		1	4	10	58
Regrind	650			25			11	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, KM5085

13.5.4 Regrind

During the KM4601 metallurgical test program, flotation tests were completed at a range of particle sizes. A P_{80} of 23 µm 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, and this formed the basis for sizing the regrind mill.

13.5.5 Dewatering and Filtering

The tailings dewatering circuit was sized based on FLS (2016) test work. Rougher tailings will be thickened in a high rate thickener and a 25% split will be combined with the first cleaner tailings. The resulting mixture will be pumped to the paste plant for subsequent use underground as paste backfill. The remaining 75% of the rougher tailings will be filtered in a pressure filter, to achieve 13% moisture content, and dry stacked in a tailings facility.

Outotec (2016a) test work was used for sizing the concentrate dewatering circuit. The concentrate will be thickened in a high rate thickener and filtered in a pressure filter. The target moisture content for the copper concentrate will be 8%.

13.6 Metallurgical Predictions

Open circuit cleaner tests performed during ALS test program KM5085 were analyzed to predict the copper and gold recoveries expected at specific head grades. The results were then entered into the overall economic model and a copper concentrate grade of 13% was selected as the design basis.

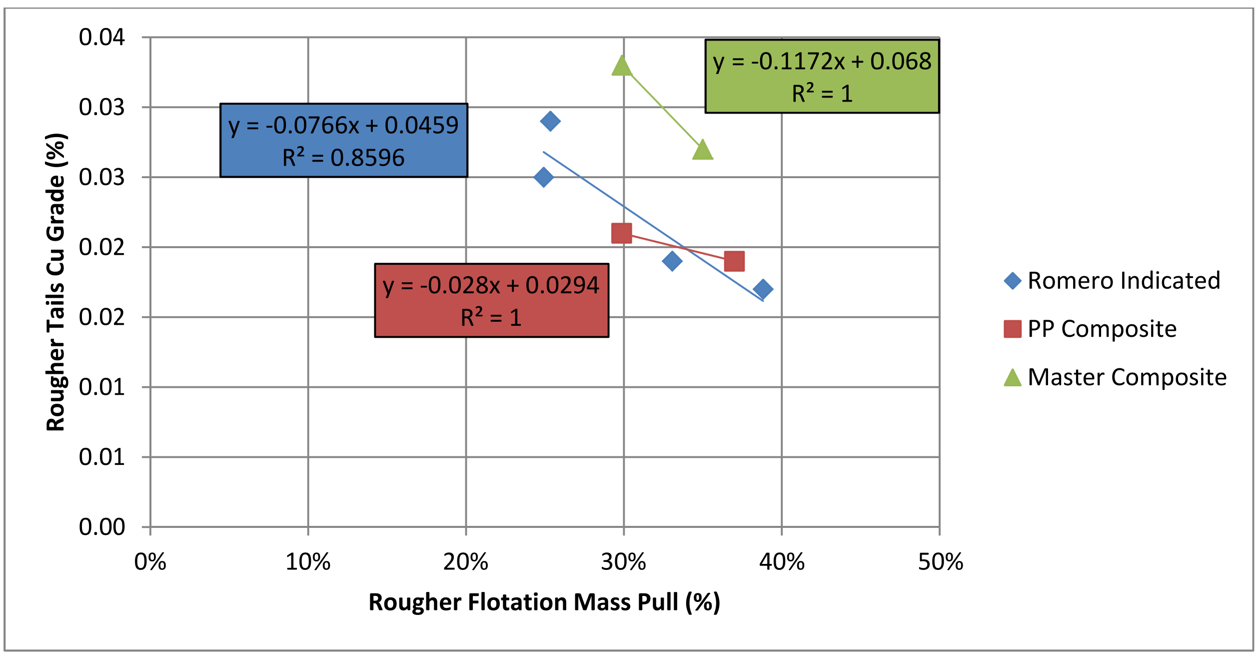
13.6.1 Copper Recovery

Projections for copper recovery were based on the mass pull targets anticipated during operation. For the rougher stage, results from KM5085 were plotted to develop the relationship between rougher flotation mass pull and copper tailings grade. The analysis for each composite and the corresponding correlation equation is presented in Figure 13.15. The following empirical equation for Romero Indicated was selected for use in calculating Cu rougher recovery:

$$\boldsymbol{Cu\ Rougher\ Tailings\ Grade} = -0.08 \times Rougher\ Mass\ Pull + 0.05$$

At a 30% mass pull, the Cu tailings grade is estimated to be 0.026%. Mass balance calculations were then used to calculate a corresponding Cu rougher recovery of 97.9% at the LOM head grade of 0.88% Cu.

Figure 13.15: Rougher Flotation Mass Pull versus Cu Rougher Tailings Grade



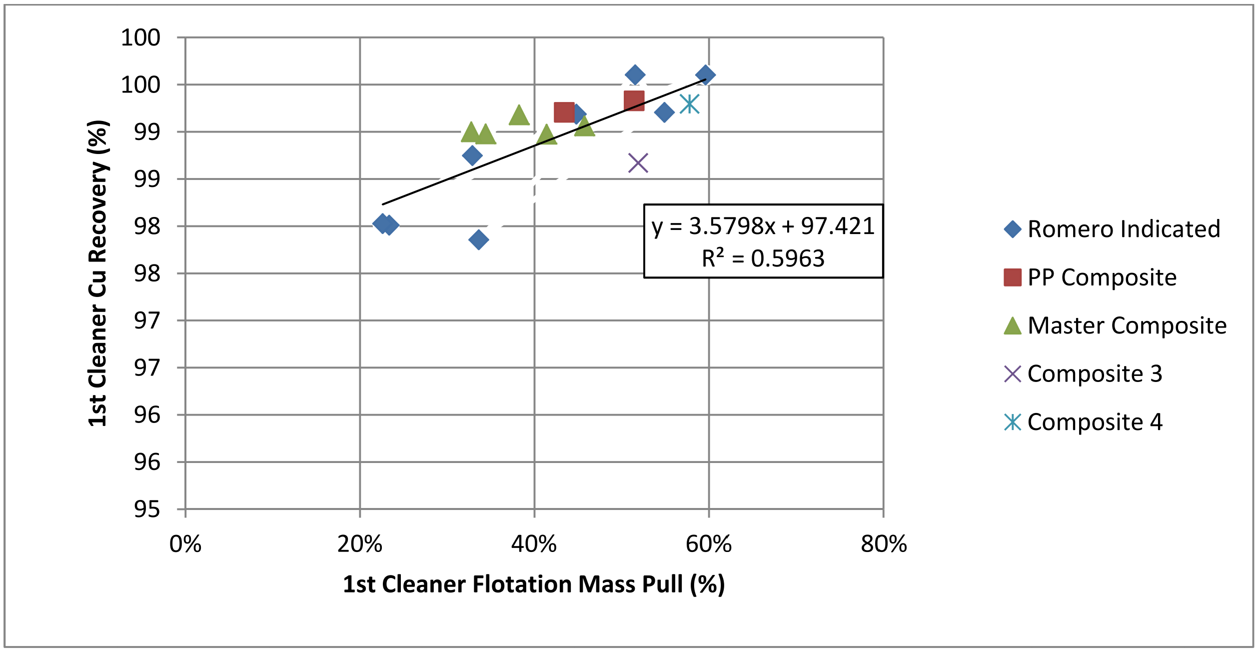
Source: JDS (2016)

For the first cleaner stage, a relationship between mass pull and Cu recovery was developed using the results from KM5085. The analysis included all test results so a range of mass pulls could be analyzed. The resulting graph is shown in Figure 13.16. The following correlation equation was used to predict Cu recovery in the first cleaner stage.

$$\boldsymbol{Cu\ 1st\ Cleaner\ Recovery} = 3.58 \times 1st\ Cleaner\ Mass\ Pull + 97.42$$

At a 40% mass pull, the Cu recovery in the first cleaning stage is estimated to be 98.85%. Assuming a 2% Cu loss in the second and third stages of cleaning, the overall Cu recovery is estimated to be 94.9%.

Figure 13.16: First Cleaner Flotation Mass Pull versus Cu Recovery



Source: JDS (2016)

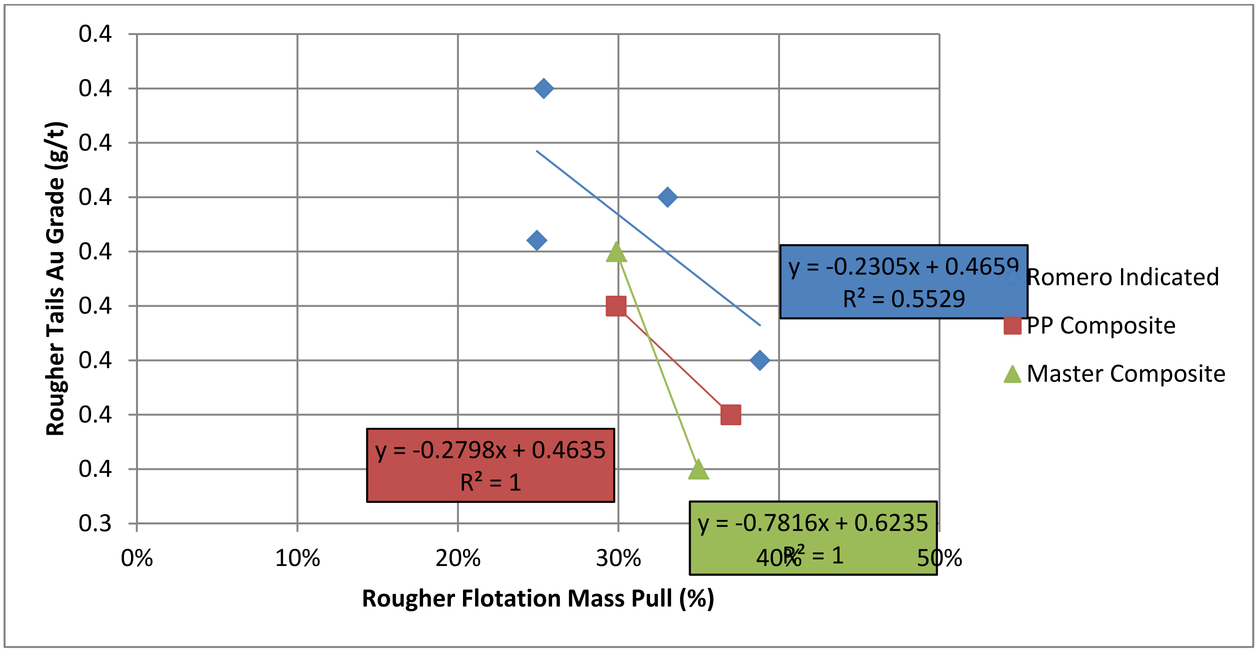
13.6.2 Gold Recovery

Projections for gold recovery were based on the mass pull targets discussed in Section 13.6.1. For the rougher stage, results from KM5085 were plotted to develop the relationship between rougher flotation mass pull and Au tailings grade. The analysis for each composite and the corresponding correlation equation is presented in Figure 13.17. The following empirical equation, representing a combination of all three composites, was selected for use in calculating Au rougher recovery:

$$\boldsymbol{Au\ Rougher\ Tailings\ Grade} = -0.25 \times Rougher\ Mass\ Pull + 0.465$$

At a 30% mass pull, the Au tailings grade is estimated to be 0.39 g/t. Mass balance calculations were then used to calculate a corresponding Au rougher recovery of 92.7% at the LOM head grade of 3.72 g/t Au.

Figure 13.17: Rougher Flotation Mass Pull versus Au Rougher Tailings Grade



Source: JDS (2016)

For the first cleaner stage, the cleaner kinetics test presented in Figure 13.14 formed part of the basis for recovery predictions. The kinetic curve was split into two portions. From a 0 to 30% mass pull, the trend line was fitted using a polynomial equation, while a linear trend line was fitted for mass pulls between 30% to 60%:

$$\boldsymbol{1st\ Cleaner\ Au\ Recovery} = 125.11 \times Mass\ Pull^3 - 86.52 * Mass\ Pull^2 + 20.17 \times Mass\ Pull - 0.79$$

$$\boldsymbol{1st\ Cleaner\ Au\ Recovery} = 0.46 \times 1st\ Cleaner\ Mass\ Pull + 0.69$$

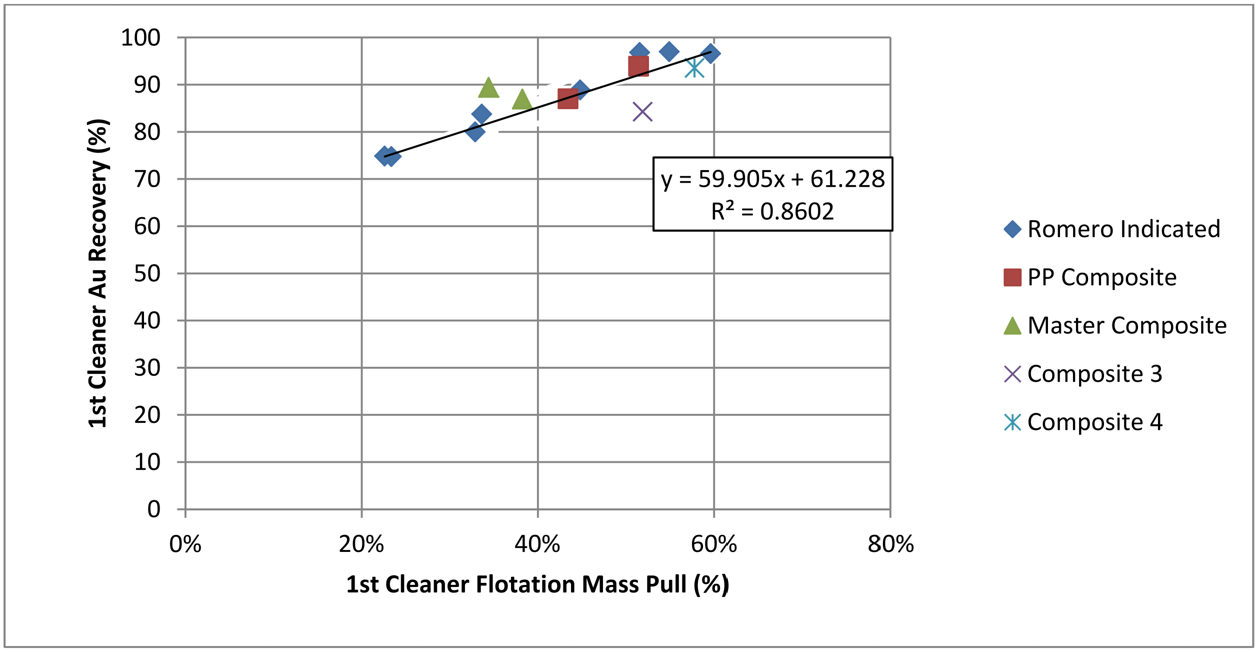
At a 40% mass pull, the linear equation predicts a 1st cleaner Au recovery of 87.4%. To increase confidence in this prediction, a relationship between mass pull and Au recovery was also developed using the other results from KM5085. The resulting graph is presented in Figure 13.18 and the correlation equation is reproduced below:

$$\boldsymbol{Au\ 1st\ Cleaner\ Recovery} = 59.9 \times 1st\ Cleaner\ Mass\ Pull + 61.23$$

At a 40% mass pull, this method predicts a 1st cleaner Au recovery of 85.2%.

For recovery predictions, an average of the above two methods was applied. Assuming a 2% Au loss in the second and third stages of cleaning, the overall Au recovery is estimated to be 78.2%.

Figure 13.18: First Cleaner Flotation Mass Pull versus Cu Recovery



Source: JDS (2016)

13.6.3 Summary of Results

Based on the test work analysis of KM5085 outlined in Sections 13.6.1 and 13.6.2, the following equations were developed to summarize the relationship between head grade, mass pull and overall flotation recovery:

$$\boldsymbol{Cu} = \left(\frac{\left(HG - \left(\left(-0.08 \times MP_R + 0.05\right) \times \left(1 - MP_R\right)\right)\right)}{MP_R}\right) \times \left(\frac{MP_R}{HG}\right) \times \left(\frac{\left(3.58 \times MP_C + 97.42\right)}{100}\right) \times (1 - CL)$$

$$\boldsymbol{Au} = \left(\frac{\left(HG - \left(\left(-0.25 \times MP_R + 0.465\right) \times \left(1 - MP_R\right)\right)\right)}{MP_R}\right) \times \left(\frac{MP_R}{HG}\right) \times \left(\frac{\left(0.6 \times MP_C + 61.23\right) + 100 \times \left(0.46 \times MP_C + 0.69\right)}{200}\right) \times (1 - CL)$$

Where:

HG	=	Metal Head Grade
MPR	=	Rougher Flotation Mass Pull
MPC	=	1st Cleaner Flotation Mass Pull
CL	=	2nd / 3rd Cleaner Losses

The resulting recovery projections of copper and gold, at head grades of 0.8% Cu and 4.7 g/t Au, are summarized in Table 13.16.

Table 13.16: Predicted LOM Metallurgical Recoveries of the Romero Deposit

Product	Wt%	Cu (%)	Au (g/t)	Cu Rec (%)	Au Rec (%)
Copper Concentrate	6.4	13	45.3	94.9	78.2
Tailings	93.6	0.05	0.87	5.1	21.8
Feed	100	0.88	3.72	100	100

Source: JDS (2016)

13.7 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 contained no deleterious elements and will not encounter smelter penalties. The results are presented in Table 13.17 below.

Table 13.17: Multi-element ICP Scan Results of Copper Concentrates – KM4601

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

Source: ALS (2015)

13.8 Opportunities and/or Future Investigations

The following opportunities were identified for future investigation should the project advance to a Feasibility Study:

- A gravity circuit to process the 1st cleaner tailings, potentially recovering gold lost to tailings;
- A gravity concentrator in the regrind circuit to recover freshly liberated gold particles.

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) and updated in 2014 for the Micon PEA. For that PEA, a new Mineral Resource at Romero was estimated. The Romero Mineral Resource estimate presented in this report supersedes the 2014 estimate.

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 May 10, 2014 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 solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.”

“The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.”

“Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.”

“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 Modifying Factors. The phrase ‘reasonable prospects for eventual economic extraction’ implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs.” Based on the CIM definitions the Mineral Resource estimate was carried out as described below.

Figure 14.1: Relative Location of the Romero Project Mineralized Zones

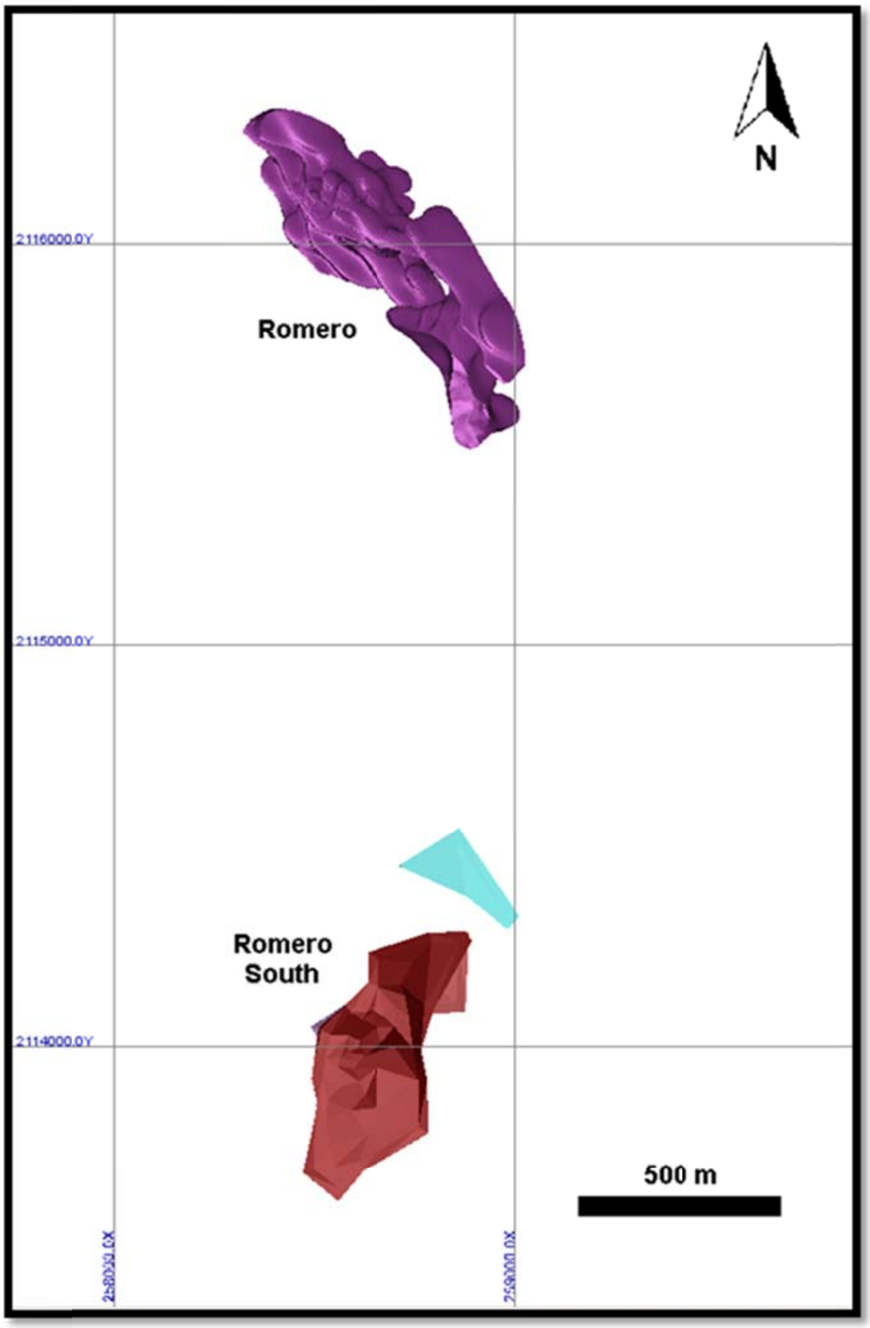


Figure supplied by Micon (2016).

14.2.1 Supporting Data

The Romero Project database provided to Micon comprises 170 drill holes with a total of 46,992.58 m of drill core and containing 17,130 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 drill holes and samples used in the estimate for Romero were 70 holes and 4,403 samples, totaling 8,585 m and for Romero South were 57 holes and 600 samples, totaling 1,182 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. The interpretation of this mineralization, 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, 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 (2016)

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 in Table 14.2 below.

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	11,834	11,834	11,834	11,834	4,184	4,184	4,184	4,184
Minimum value*	0	0	0	0	0.00025	0.002	0.0004	0
Maximum value	288.6	186	21.941	20.02	68.5	98	2.714	3.87
Mean	0.655	2.423	0.18	0.159	0.346	0.902	0.031	0.041
Median	0.091	1	0.012	0.02	0.014	0.262	0.007	0.01
Variance	22.942	21.898	0.397	0.282	4.286	6.062	0.01	0.025
Standard Deviation	4.79	4.68	0.63	0.531	2.07	2.462	0.101	0.158
Coefficient of variation	7.312	1.931	3.505	3.327	5.975	2.73	3.231	3.829

* - Zero value means missing assays assumed to be zero

Source: Micon (2016)

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 in 2014 to account for the additional drilling completed since 2011. The Romero South interpretation has not changed for this report.

Given that the 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 and copper assays. Romero South also used zinc. The metal prices assumed for this calculation were; Au = US$1,400/oz, Ag = US$20.00/oz and Cu = US$2.50/lb. For the 2016 Romero Mineral Resource presented in this report the Zn price was not considered as its contribution is not economically significant. These metal price assumptions were supplied to Micon by GoldQuest.

An in-situ metal value was calculated for each sampled interval in the holes used for the Mineral Resource estimate at Romero. 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)

Gold and Silver units are in ppm and copper and zinc prices are in weight percent. Applying unit adjusting factors to the prices gives the following formula:

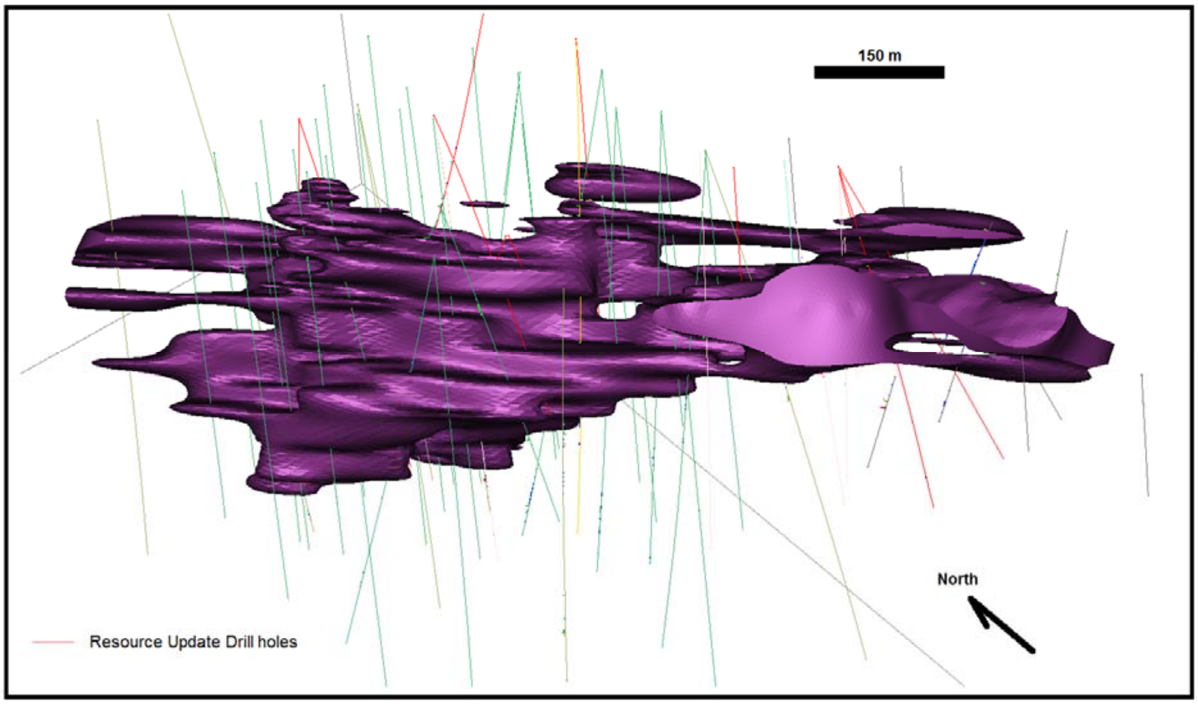
- Metal Value in-situ = (Au g/t x US$45.01) + (Ag g/t x US$0.643) + (Cu % x US$55.12)

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.

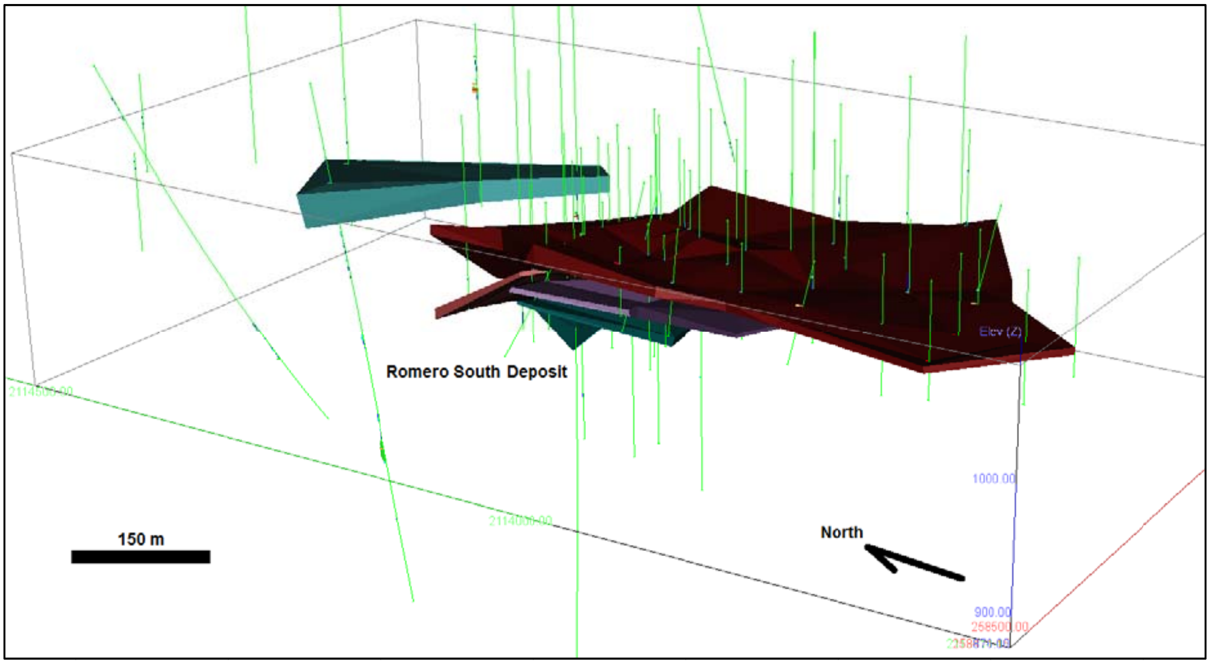
The Romero South deposit is simple set of stacked, flat lying lenses. The mineralized envelope was updated in 2014 using a US$15 cut-off metal value and the wireframe was constructed by conventional manual triangulation methods. Figure 14.2 and Figure 14.3 show 3D isometric views of the final interpreted mineralization lenses and intersecting drill holes.

Figure 14.2: Romero Deposit Resulting Wireframe



(Looking down dip to the north-east)
Source: Micon (2016)

Figure 14.3: Romero South Deposit Resulting Wireframes



(Looking down dip to the north-east)
Source: Micon (2013)

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.

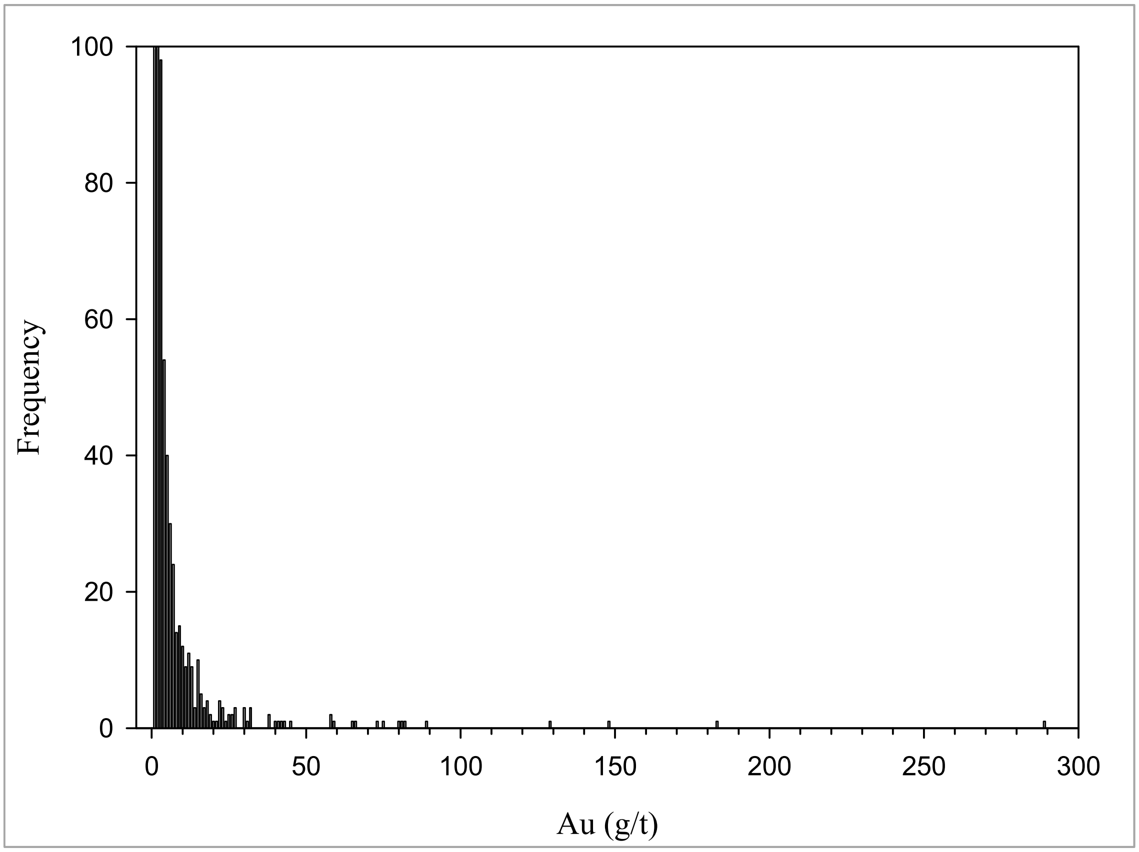
14.2.7 Data Processing

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

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.

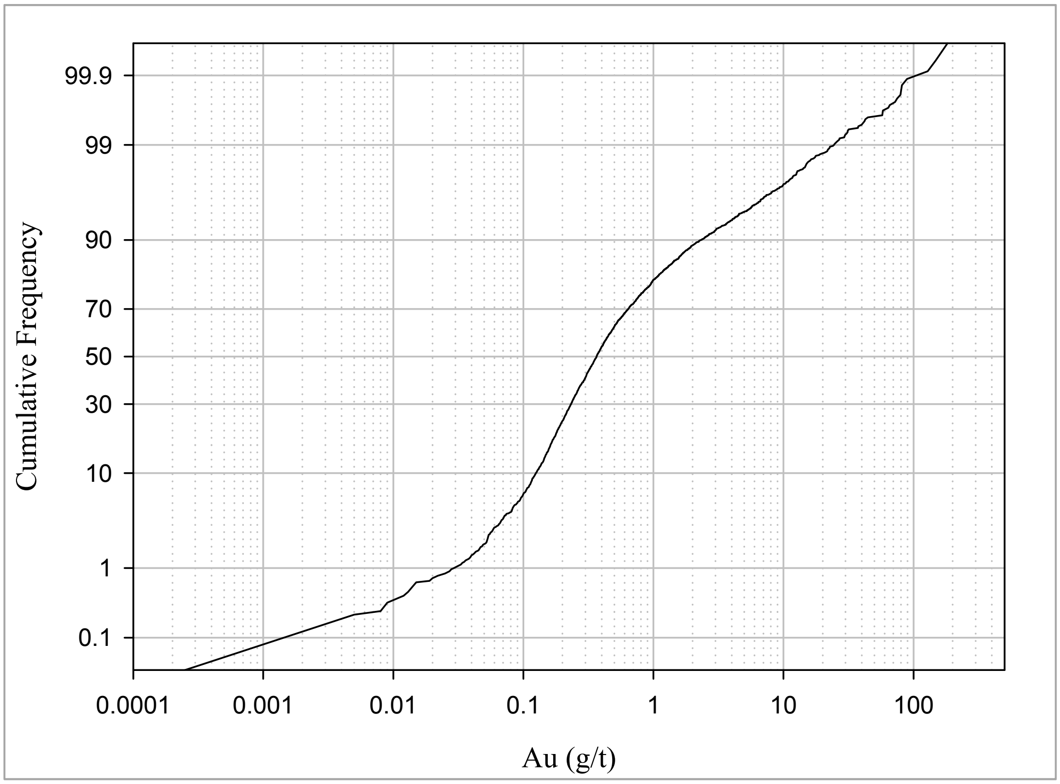
An example histogram and probability plot are shown Figure 14.4 and Figure 14.5. Log normal populations plot as straight lines on probability plots. The upper point at which the straight line breaks down is often accepted as the capping value.

Figure 14.4: Romero Deposit Gold Histogram



Source: Micon (2016)

Figure 14.5: Romero Deposit Gold Probability Plot



Source: Micon (2016)

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 (2016)

14.2.7.1 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 14.4shows 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 (%)
Romero								
Number of samples	4,268	4,268	4,268	4,268	4,268	4,268	4,268	4,268
Minimum value	0.00025	0.00025	0.01	0.01	0.001	0.001	0.001	0.001
Maximum value	218.2	72.2	97	64	13.969	6.37	16.259	5.82
Mean	1.594	1.505	3.857	3.789	0.439	0.428	0.331	0.317
Median	0.385	0.385	2	2	0.134	0.134	0.11	0.11
Geometric Mean	37.992	21.794	39.015	30.997	0.714	0.548	0.543	0.333
Variance	6.164	4.668	6.246	5.567	0.845	0.74	0.737	0.577
Standard Deviation	3.866	3.103	1.619	1.469	1.923	1.73	2.229	1.824
Coefficient of variation	4,268	4,268	4,268	4,268	4,268	4,268	4,268	4,268
Romero South								
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	0
Maximum value	68.5	20.5	86.17	15	1.398	1.25	3.547	1.65
Mean	2.19	2.006	2.233	1.882	0.156	0.155	0.17	0.161
Median	0.473	0.473	1.19	1.19	0.09	0.09	0.04	0.04
Geometric Mean	25.103	13.499	27.522	6.605	0.036	0.035	0.118	0.078
Variance	5.01	3.674	5.246	2.57	0.189	0.186	0.343	0.28
Standard Deviation	2.288	1.832	2.35	1.366	1.21	1.196	2.018	1.74
Coefficient of variation	591	591	591	591	591	591	591	591

Source: Micon (2016)

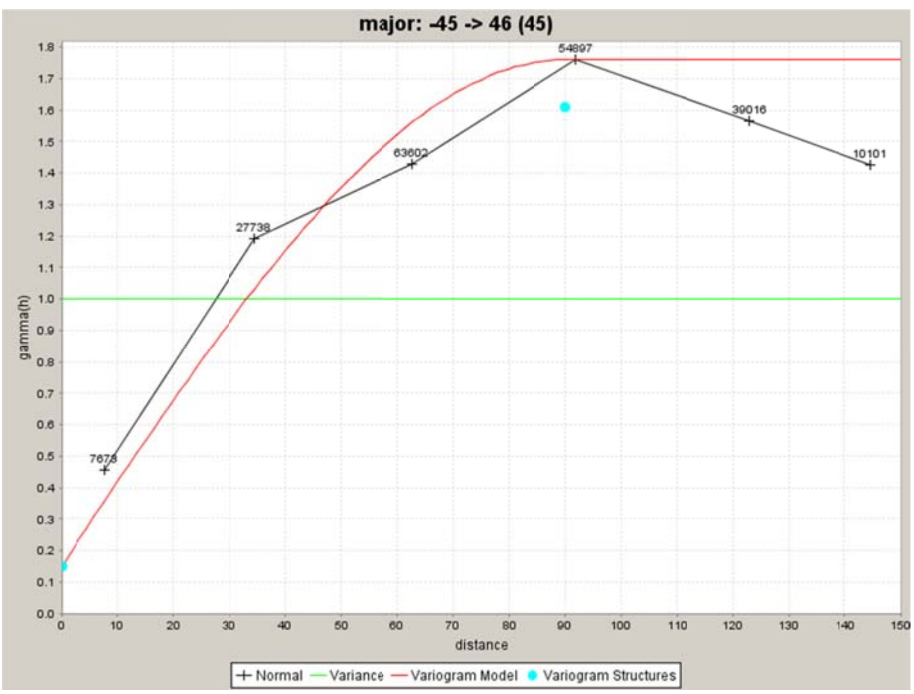
14.2.8 Variography

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. For the current update of the Romero deposit Micon decided to subdivide the composites into two separate domains for the purpose of variography and the setting of search parameters. These domains are called “Core” and “Layered”. Based on the geometry and geology of the deposit the separation was done to better assess the variography and to respect the interpreted fluid flow directions of the geological model. Mineralizing fluids are interpreted to have flowed sub-vertically in the core and out laterally into the layered domain. As representative examples, Figure 14.6, Figure 14.7 and Figure 14.8 show the resulting major axis variograms for gold in both zones.

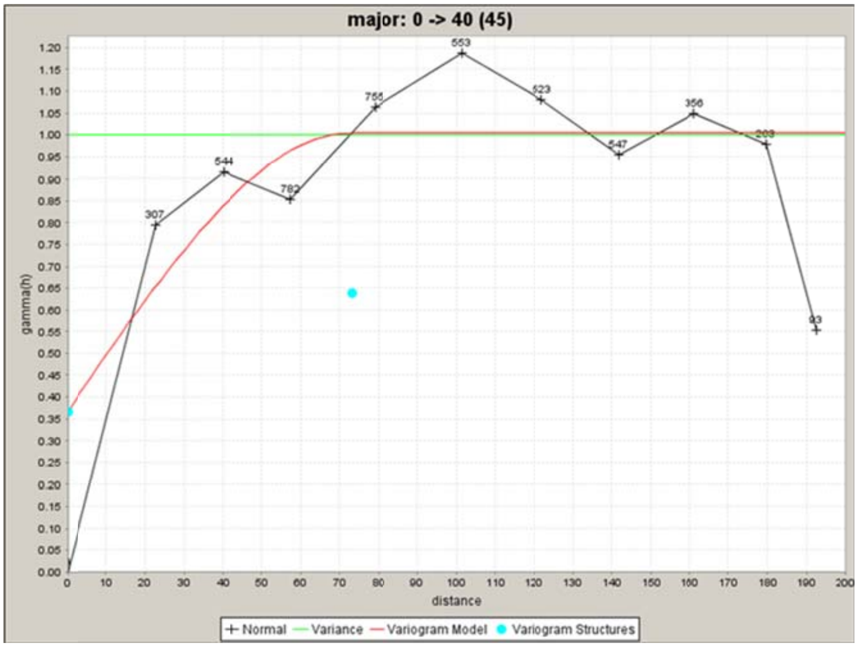
Variography should be performed on data from regular, coherent mineralized shapes with geological support. Romero South presented four different mineralized layers (see Section 14.2.6) and five zones where variograms were tested. Variograms could be modelled only for Zones 1 (upper) and Zone 2 North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

Figure 14.6: Romero - Major Axis Variogram for Gold



Source: Micon (2016)

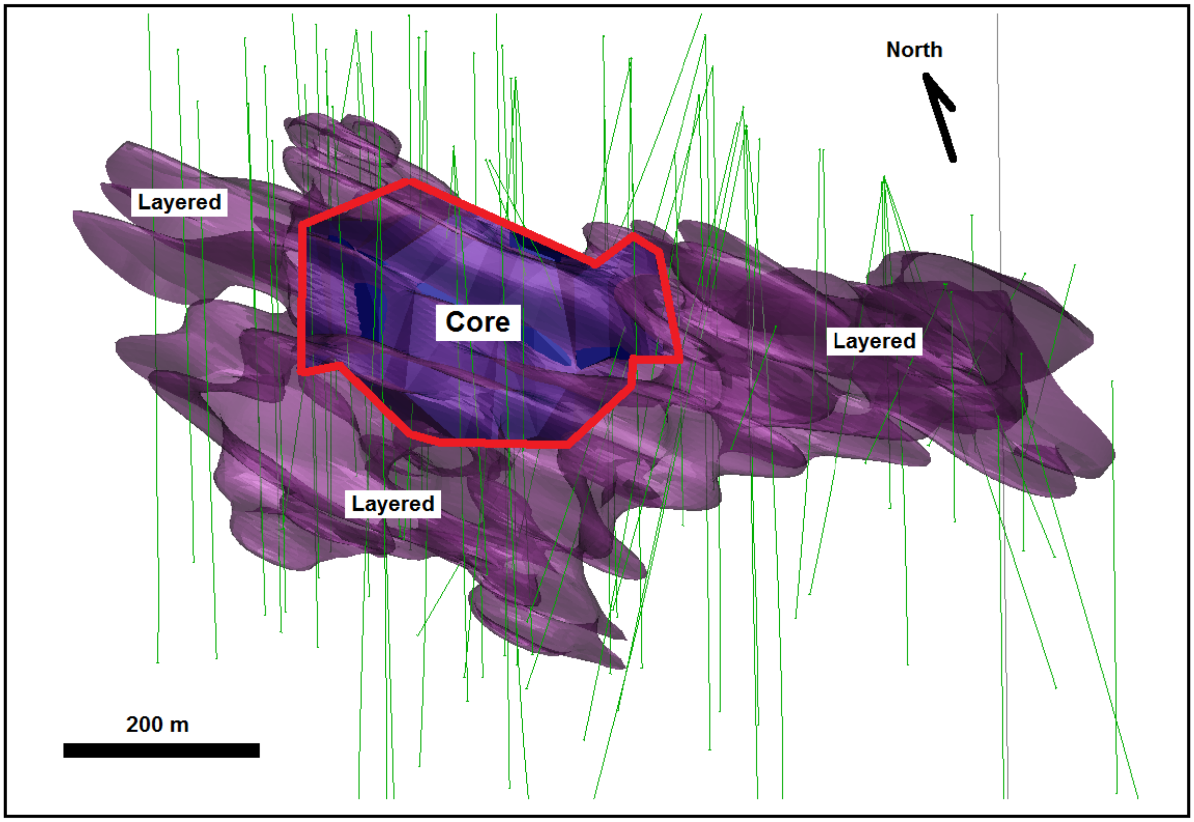
Figure 14.7: Romero South - Major Axis Variogram for Gold



Source: Micon (2016)

Figure 14.8 shows the interpreted core and layered zones of the Romero deposit.

Figure 14.8: Romero – Core and Layered Domains Location



Source: Micon (2016)

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 are 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 (2016)

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	160 / 46	75 / -45	0.5	0.100 / 0.150	1.096 / 1.610	100 / 90	75 / 90	50 / 60	6	12	2
	ROM6	2	160 / 46	75 / -45	0.5	0.100 / 0.150	1.096 / 1.610	200 / 180	150 / 180	100 / 120	4	8	2
	ROM6	3	160 / 46	75 / -45	0.5	0.100 / 0.150	1.096 / 1.610	200 / 180	150 / 180	100 / 120	2	8	2
Ag	ROM6	1	115 / 136	0 / 0	50 / 50	0.060 / 0.140	0.900 / 0.670	100 / 60	75 / 60	50 / 30	6	12	2
	ROM6	2	115 / 136	0 / 0	50 / 50	0.060 / 0.140	0.900 / 0.670	200 / 120	150 / 120	100 / 60	4	8	2
	ROM6	3	115 / 136	0 / 0	50 / 50	0.060 / 0.140	0.900 / 0.670	200 / 120	150 / 120	100 / 60	2	8	2
Cu	ROM6	1	140 / 46	40 / -45	#DIV/0!	0.100 / 0.060	1.001 / 1.442	75 / 85	50 / 85	50 / 85	6	12	2
	ROM6	2	140 / 46	40 / -45	#DIV/0!	0.100 / 0.060	1.001 / 1.442	150 / 170	100 / 170	100 / 170	4	8	2
	ROM6	3	140 / 46	40 / -45	#DIV/0!	0.100 / 0.060	1.001 / 1.442	150 / 170	100 / 170	100 / 170	2	8	2
Zn	ROM6	1	110 / 192	0 / 40	70 / 25	0.100 / 0.050	0.684 / 1.149	85 / 100	50 / 60	50 / 50	6	12	2
	ROM6	2	110 / 192	0 / 40	70 / 25	0.100 / 0.050	0.684 / 1.149	170 / 200	100 / 120	100 / 100	4	8	2
	ROM6	3	110 / 192	0 / 40	70 / 25	0.100 / 0.050	0.684 / 1.149	170 / 200	100 / 120	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 (2016)

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	Sub-level Open Stoping	Room and Pillar
Au price US$/Oz	1,400.00	1,400.00
Ag price US$/Oz	20	20
Cu price US$/lb	2.5	2.5
Zn price US$/lb	N/A	N/A
Au recovery %	71.7	71.7
Ag recovery %	54.4	54.4
Cu recovery %	96.8	96.8
Zn recovery %	90	90
Price Weighted Avg. Recovery %	71.5	71.5
Mining Cost US$/t	28	24
Mill Cost US$/t	11.5	11.5
GandA Cost US$/t	5	5
Overall Cost US$/t	44.5	40.5

Source: Micon (2016)

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.

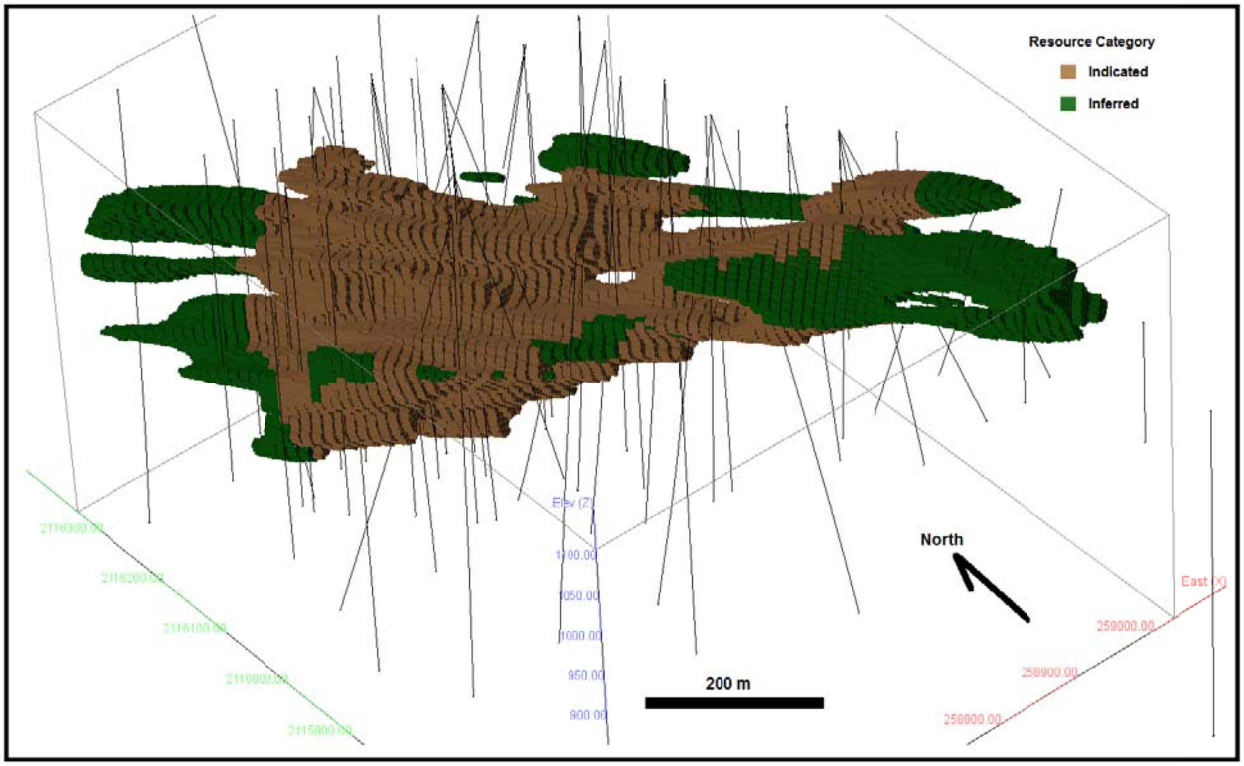
14.3.4 Mineral Resource Categorization

The mineral resource estimates for Romero and Romero South have been categorized into the indicated and inferred categories (Figure 14.9 and Figure 14.10). No measured resources have been determined at this time. The criteria for classification are 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).

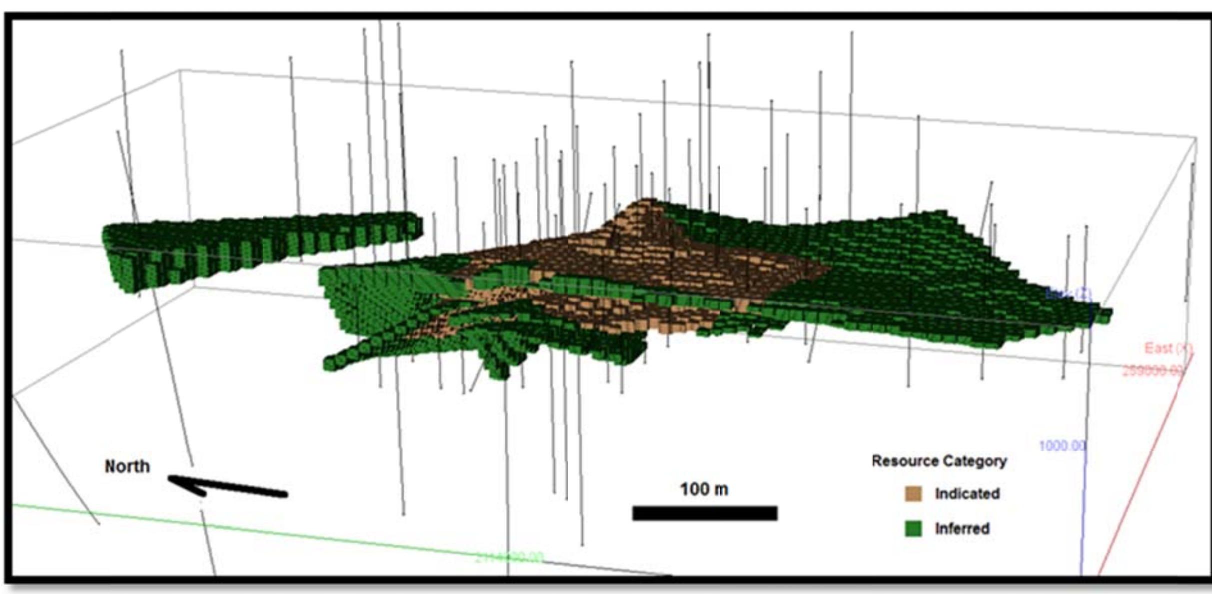
These rules were combined with a visual check of the model to make certain that the Indicated resource has a regular, continuous shape and is not broken up creating the “spotted dog effect” (scattered isolated islands of Indicated resource). Some small clusters of Indicated blocks were downgraded in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

Figure 14.9: Romero Block Model Isometric View - Resource Category



Source: Micon (2016)

Figure 14.10: Romero South Block Model Isometric View - Resource Category



Source: Micon (2016)

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)	Au-Eq (g/t)	Au Ounces (x 1,000)	Au-Eq Ounces (x 1,000)
Indicated	Romero	18,390	2.57	0.65	0.31	4.2	3.43	1,520	2,028
	Romero South	1,840	3.69	0.25	0.18	1.6	4.01	218	237
Total Indicated Resources		**20,230**	**2.67**	**0.61**	**0.3**	**4**	**3.48**	**1,738**	**2,265**
Inferred	Romero	2,120	1.8	0.39	0.36	3.2	2.32	123	158
	Romero South	900	2.57	0.2	0.21	2.1	2.84	74	82
Total Inferred Resources		**3,020**	**2.03**	**0.33**	**0.32**	**2.9**	**2.47**	**197**	**240**

Note: Au-Eq g/t = (Au g/t)+(Ag g/t)/92.261)+(Cu %)/0.605)

Source: Micon (2016)

The present report and Mineral Resource estimates are based on exploration results and interpretation current as of November 9, 2015. The effective date of the Mineral Resource estimate is January 14, 2016 for Romero and October 29, 2013 for Romero South.

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 in Table 14.8 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.

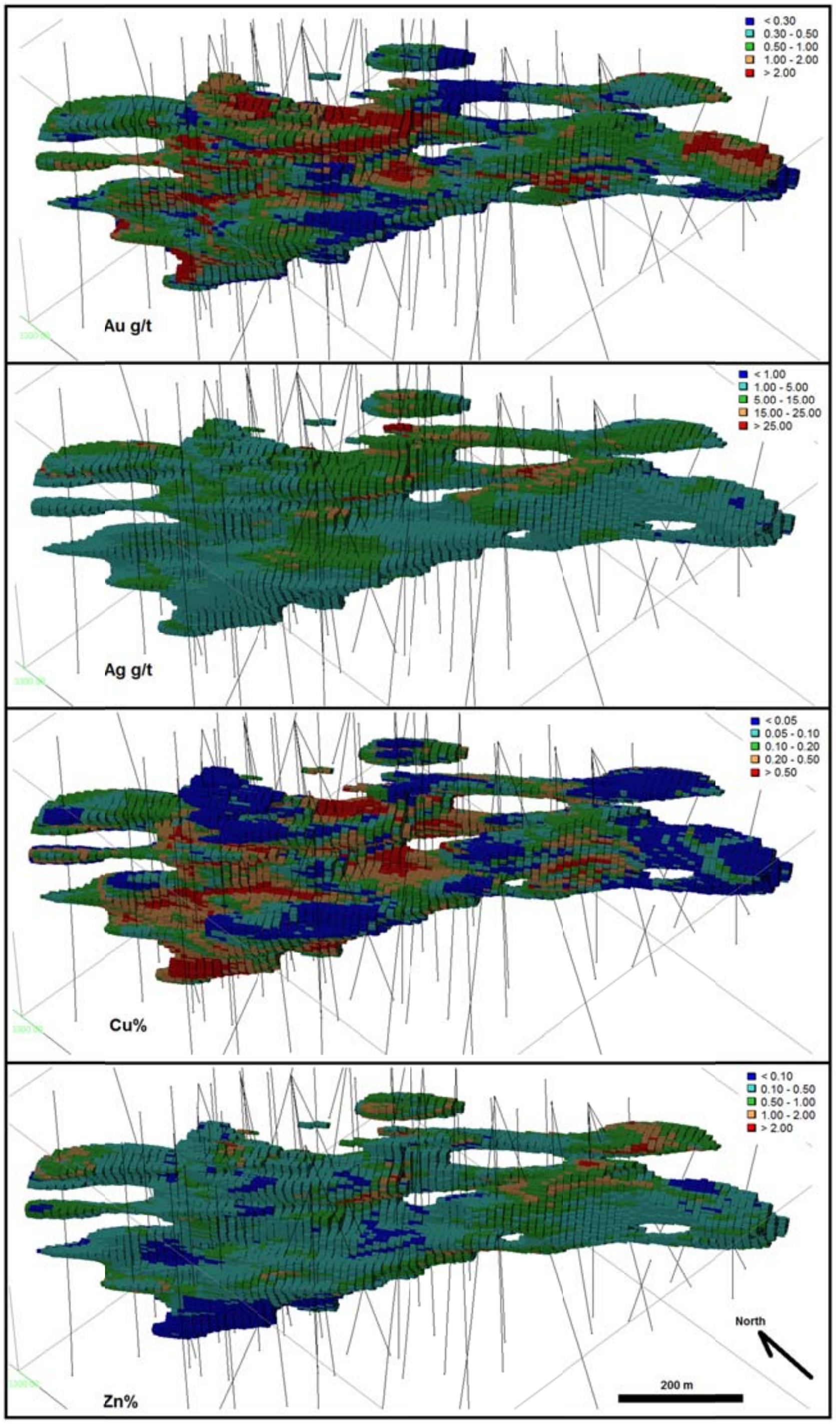
14.4.1 Responsibility for Estimation

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, 2014. 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

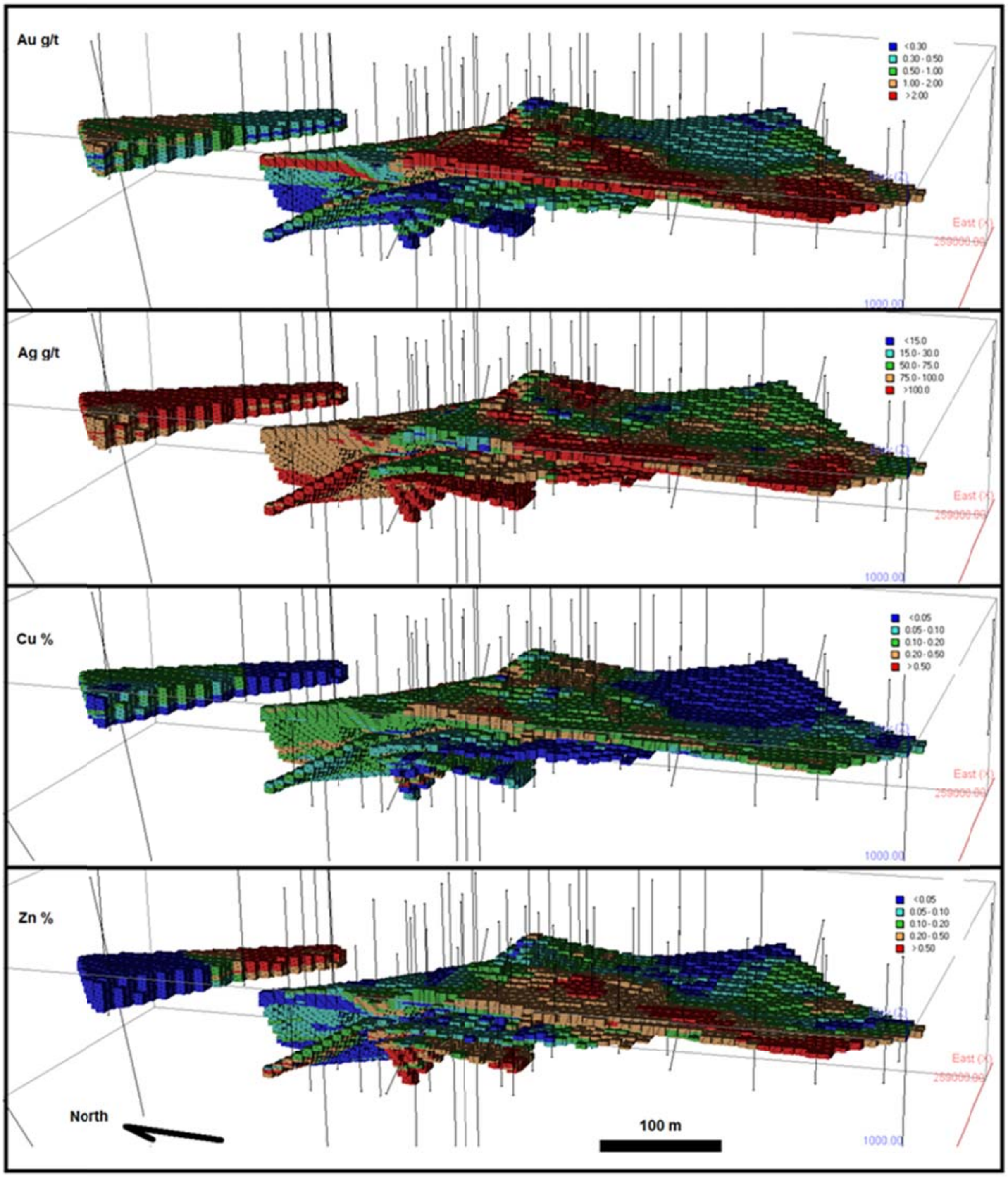
Figure 14.11 and Figure 14.12 graphically show the grade of the Mineral Resources tabulated above as 3D isometric views of the block model.

Figure 14.11: Romero Block Model Isometric View - Grade Distribution



Source: Micon (2016)

Figure 14.12: Romero South Block Model Isometric View - Grade Distribution



Source: Micon (2016)

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 14.12 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	5,680,000	5.35	4.9	0.9	0.34	6.52	978,000	1,191,000
Indicated	140	6,280,000	5.06	4.9	0.88	0.33	6.2	1,021,000	1,253,000
Indicated	130	7,000,000	4.75	4.8	0.86	0.33	5.88	1,069,000	1,322,000
Indicated	120	7,840,000	4.45	4.8	0.84	0.33	5.54	1,121,000	1,398,000
Indicated	110	8,820,000	4.15	4.7	0.82	0.33	5.21	1,176,000	1,478,000
Indicated	100	10,020,000	3.84	4.7	0.79	0.33	4.87	1,236,000	1,568,000
Indicated	90	11,430,000	3.53	4.6	0.76	0.32	4.53	1,297,000	1,664,000
Indicated	80	13,210,000	3.21	4.5	0.73	0.32	4.17	1,365,000	1,772,000
Indicated	70	15,510,000	2.89	4.4	0.7	0.31	3.8	1,439,000	1,894,000
Indicated	**60**	**18,390,000**	**2.57**	**4.2**	**0.65**	**0.31**	**3.43**	**1,518,000**	**2,028,000**
Indicated	50	22,300,000	2.25	4.1	0.6	0.3	3.04	1,611,000	2,180,000
Indicated	40	27,630,000	1.93	4	0.54	0.3	2.65	1,715,000	2,351,000
Indicated	30	34,820,000	1.63	3.9	0.47	0.3	2.26	1,825,000	2,529,000

(reported cut-off in bold)
Source: Micon (2016)

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	320,000	3.75	2.9	0.62	0.46	4.55	39,000	47,000
Inferred	140	380,000	3.54	2.9	0.61	0.44	4.33	43,000	53,000
Inferred	130	440,000	3.41	2.9	0.59	0.44	4.17	48,000	59,000
Inferred	120	530,000	3.18	2.8	0.56	0.43	3.91	54,000	67,000
Inferred	110	630,000	3.01	2.8	0.54	0.41	3.71	61,000	75,000
Inferred	100	770,000	2.8	2.8	0.51	0.4	3.46	69,000	86,000
Inferred	90	960,000	2.56	2.8	0.48	0.39	3.18	79,000	98,000
Inferred	80	1,220,000	2.32	2.9	0.45	0.38	2.9	91,000	114,000
Inferred	70	1,580,000	2.07	3	0.42	0.37	2.62	105,000	133,000
Inferred	**60**	**2,120,000**	**1.8**	**3.2**	**0.39**	**0.36**	**2.32**	**123,000**	**158,000**
Inferred	50	3,060,000	1.51	3.6	0.34	0.37	1.98	148,000	195,000
Inferred	40	4,640,000	1.23	3.9	0.29	0.39	1.64	183,000	245,000
Inferred	30	7,470,000	0.97	3.8	0.23	0.39	1.31	232,000	314,000

(reported cut-off in bold)
Source: Micon (2016)

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	890,000	5.52	1.5	0.292	0.181	5.9	158,000	169,000
Indicated	140	960,000	5.34	1.51	0.288	0.184	5.71	165,000	176,000
Indicated	130	1,050,000	5.11	1.51	0.281	0.186	5.48	173,000	185,000
Indicated	120	1,140,000	4.91	1.51	0.277	0.187	5.27	180,000	193,000
Indicated	110	1,220,000	4.74	1.52	0.273	0.187	5.1	186,000	200,000
Indicated	100	1,320,000	4.55	1.54	0.268	0.187	4.9	193,000	208,000
Indicated	90	1,440,000	4.33	1.56	0.263	0.186	4.67	200,000	216,000
Indicated	80	1,570,000	4.11	1.56	0.256	0.184	4.44	207,000	224,000
Indicated	70	1,710,000	3.88	1.56	0.251	0.18	4.21	214,000	232,000
Indicated	**60**	**1,840,000**	**3.69**	**1.55**	**0.245**	**0.175**	**4.01**	**218,000**	**238,000**
Indicated	50	2,000,000	3.48	1.51	0.238	0.17	3.79	224,000	244,000
Indicated	40	2,210,000	3.22	1.46	0.229	0.168	3.53	229,000	250,000
Indicated	30	2,430,000	2.99	1.45	0.218	0.163	3.27	233,000	256,000

(reported cut-off in bold)
Source: Micon (2016)

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	210,000	5.32	1.96	0.205	0.192	5.6	36,000	38,000
Inferred	140	240,000	5.03	2.06	0.213	0.216	5.32	39,000	41,000
Inferred	130	280,000	4.73	2.15	0.221	0.238	5.03	43,000	45,000
Inferred	120	320,000	4.47	2.23	0.229	0.256	4.78	46,000	49,000
Inferred	110	360,000	4.24	2.21	0.227	0.267	4.55	49,000	53,000
Inferred	100	420,000	3.92	2.16	0.22	0.271	4.22	53,000	57,000
Inferred	90	470,000	3.71	2.13	0.219	0.269	4.01	56,000	61,000
Inferred	80	540,000	3.46	2.11	0.214	0.265	3.75	60,000	65,000
Inferred	70	660,000	3.08	2.06	0.206	0.247	3.36	65,000	71,000
Inferred	**60**	**900,000**	**2.57**	**2.06**	**0.196**	**0.211**	**2.84**	**74,000**	**82,000**
Inferred	50	1,320,000	2.07	2.11	0.187	0.174	2.33	88,000	99,000
Inferred	40	1,830,000	1.71	2.16	0.177	0.147	1.96	101,000	115,000
Inferred	30	2,520,000	1.4	2.34	0.162	0.139	1.63	113,000	132,000

(reported cut-off in bold)
Source: Micon (2016)

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	1.109	1.505
	Ag g/t	3.6	3.789
	Cu %	0.313	0.439
	Zn %	0.339	0.317
Romero South	Au g/t	1.467	2.006
	Ag g/t	2	1.882
	Cu %	0.147	0.155
	Zn %	0.149	0.161

Source: Micon (2016)

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 ID2 Grades for Gold and Copper

Category	Zone	Tonnes (x 1,000)	Au (g/t)		Cu (%)	
			OK	ID^2	OK	ID^2
Indicated	Romero	18,390	2.57	2.57	0.65	0.67
	Romero South	1,840	3.67	3.69	0.24	0.24
Inferred	Romero	2,120	1.8	1.8	0.39	0.41
	Romero South	900	2.56	2.62	0.19	0.19

Source: Micon (2016)

Table 14.15: Comparison of OK and ID2 Grades for Zinc and Silver

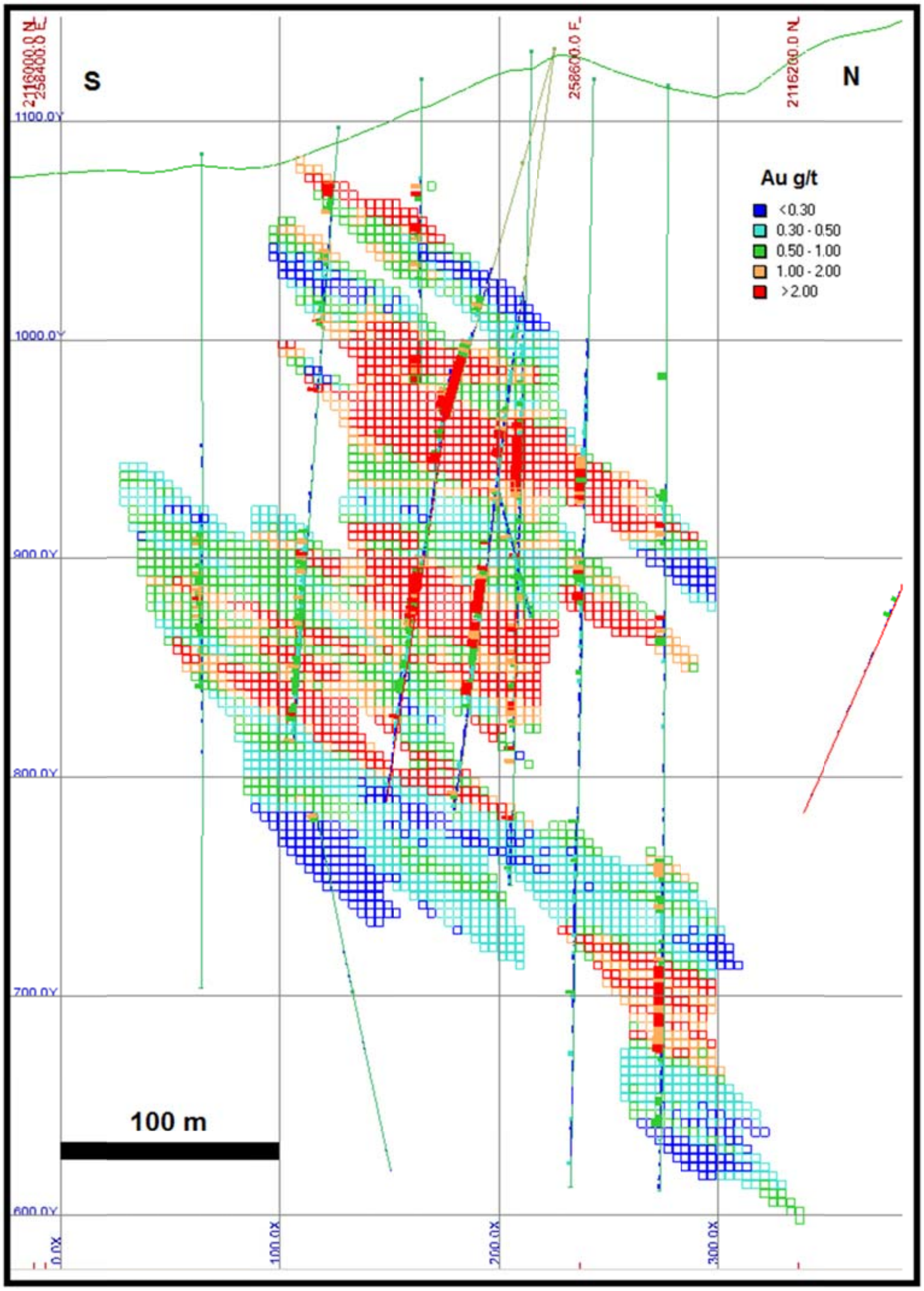
Category	Zone	Tonnes (x 1,000)	Zn (%)		Ag (g/t)	
			OK	ID^2	OK	ID^2
Indicated	Romero	18,390	0.31	0.3	4.2	4.3
	Romero South	1,840	0.17	0.17	1.5	1.5
Inferred	Romero	2,120	0.36	0.34	3.2	3.2
	Romero South	900	0.21	0.22	2	2.1

Source: Micon (2016)

14.6.3 Visual Inspection

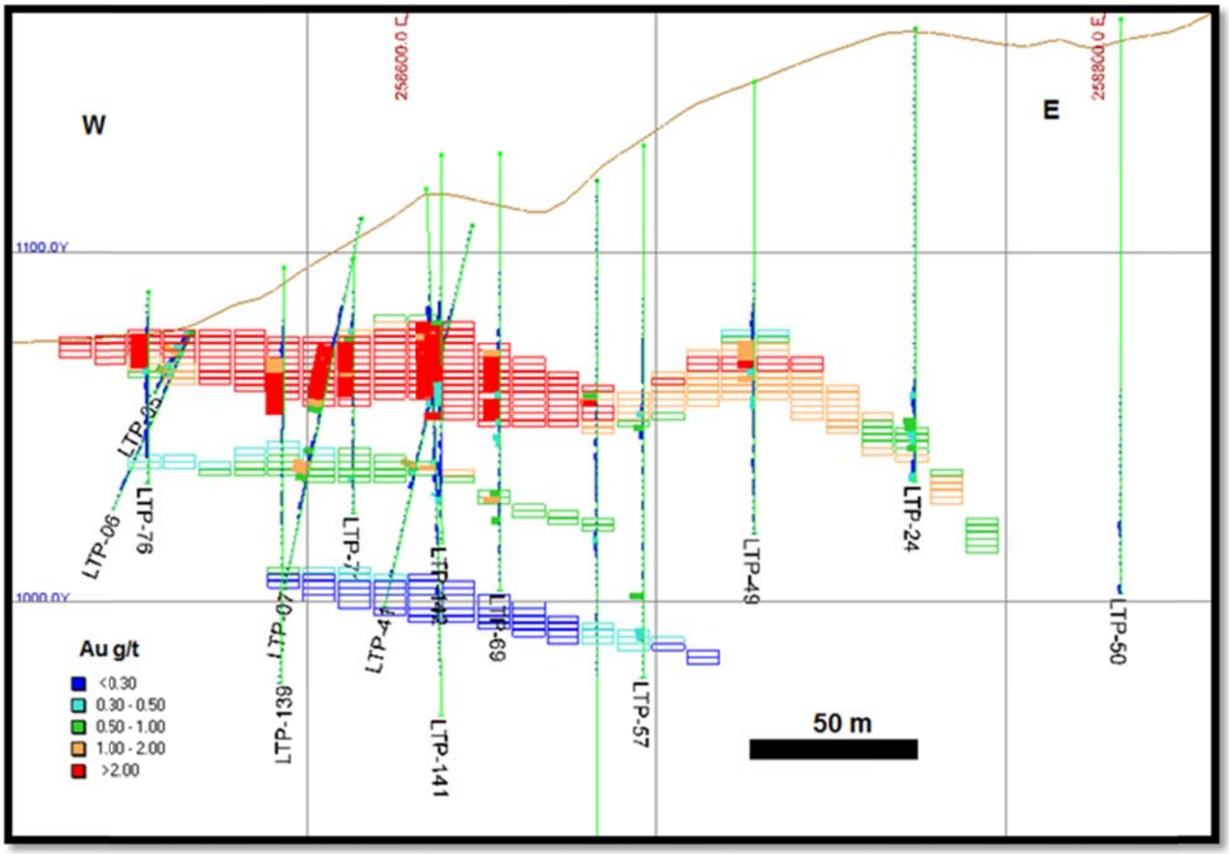
The block models and drill holes were reviewed on section to ensure that the grade distribution in the blocks honoured the neighbouring drill hole data. Figure 14.13 and Figure 14.14 show typical results.

Figure 14.13: Romero Typical Vertical Section



Source: Micon (2016)

Figure 14.14: Romero South Typical Vertical Section



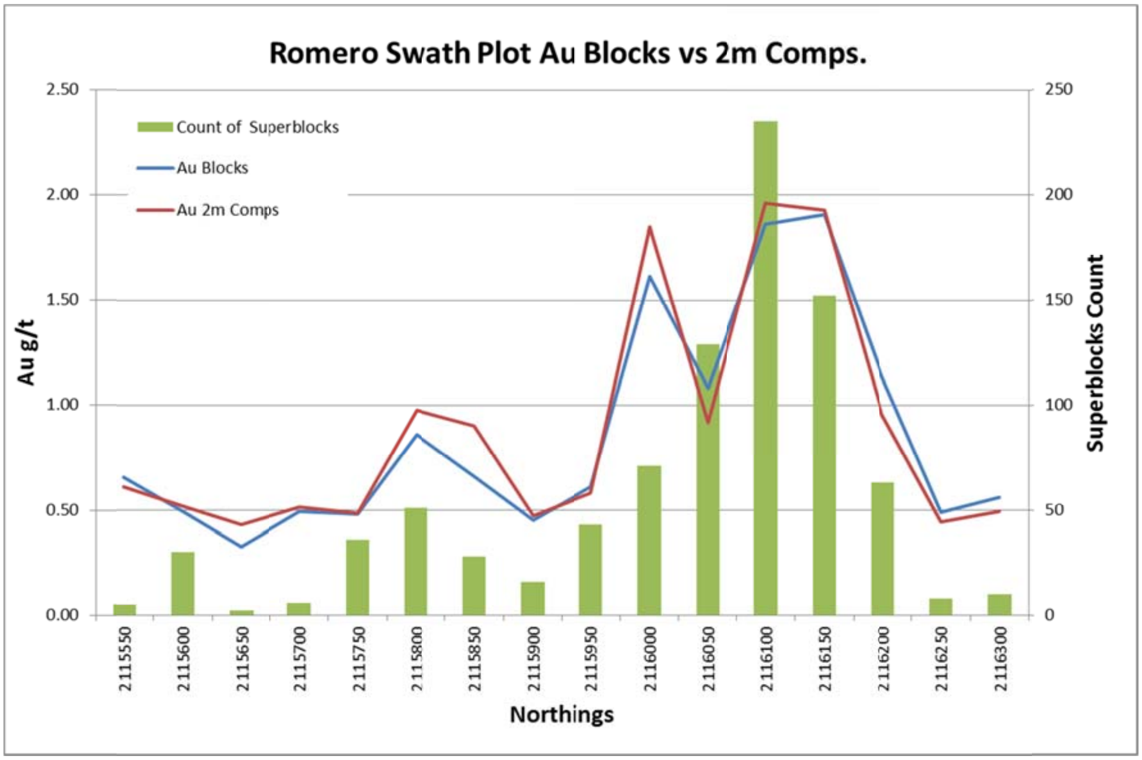
Source: Micon (2016)

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.

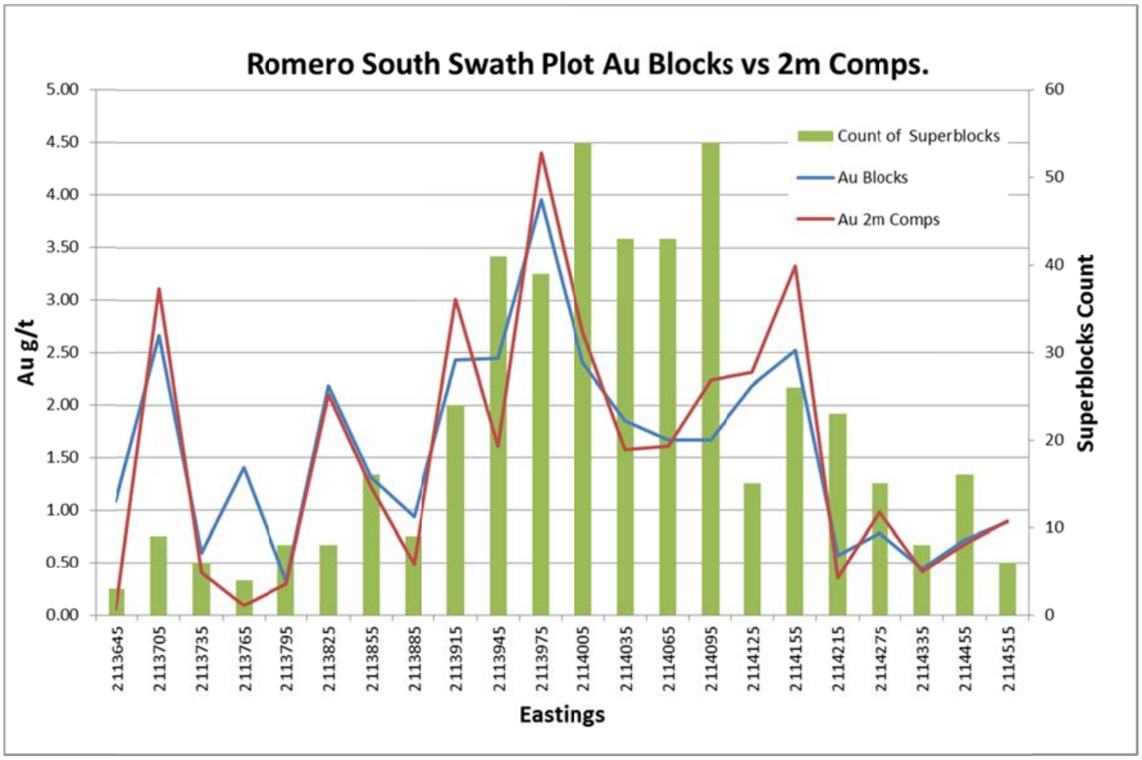
The gold swath plots for Romero and Romero South are shown in Figure 14.15 and Figure 14.16. Reasonable agreement with minor smoothing of extremes can be seen.

Figure 14.15: Romero Trend Analysis Chart for Gold



Source: Micon (2016)

Figure 14.16: Romero South Trend Analysis Chart for Gold



Source: Micon (2016)

15 Mineral Reserve Estimates

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource demonstrated by at least a PFS. This PFS includes adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Mineral Reserves are those parts of Mineral Resources, which, after the application of all mining factors, result in an estimated tonnage, and grade that is the basis of an economically viable project. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the economic mineralized rock and delivered to the treatment plant or equivalent facility. The term “Mineral Reserve” need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

The reserve classifications used in this report conform to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification of NI 43-101 resource and reserve definitions and Companion Policy 43-101CP. These are listed below.

A “Proven Mineral Reserve” is the economically mineable part of a Measured Mineral Resource demonstrated by at least a PFS. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

A “Probable Mineral Reserve” is the economically mineable part of an Indicated Mineral Resource, and in some circumstances a Measured Mineral Resource, demonstrated by at least a PFS. The study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

15.1 Cut-off Grade Criteria

Mining reserve values were calculated from block model tonnes and grades to define a net smelter return (NSR) cut-off to determine the mineable portions of the Romero deposit. The parameters used for the calculation were based on the data shown in Table 15.2 and Table 15.2.

Table 15.1: NSR Calculation Metal Prices

Commodity	Unit	Price (US$)
Copper Price	US$/lb	2.50
Gold Price	US$/oz	1,250
Silver Price	US$/oz	17.00
Exchange Rate	US$:C$	0.93

Source: JDS (2016)

Table 15.2: NSR Copper Concentrate Smelter Terms

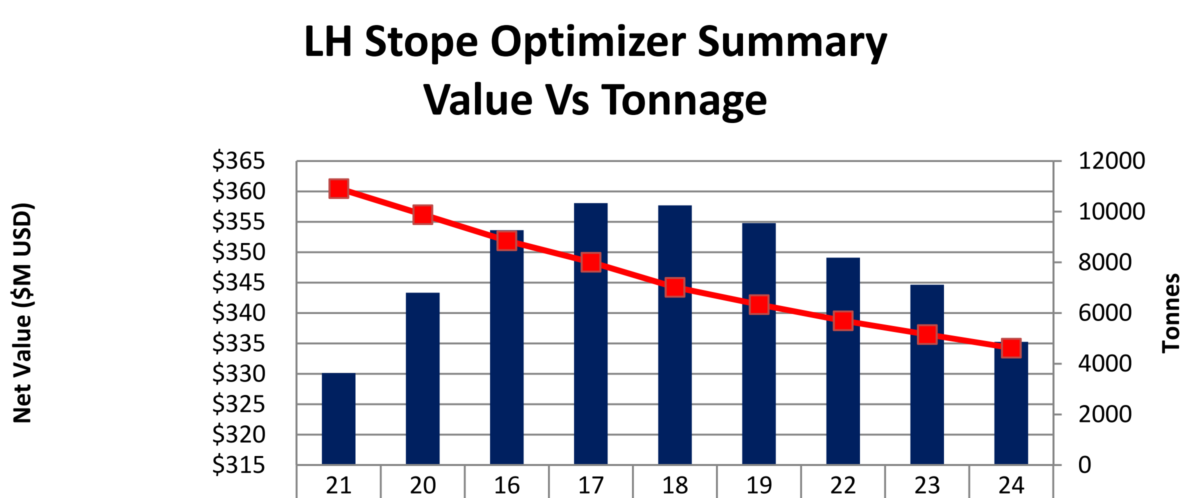
NSR Assumptions	Unit	Cu Concentrate
Recoveries		
Cu	%	96.8
Au	%	71.7
Ag	%	54.4
Concentrate Grade	**%**	**20.0**
Moisture Content	%	8.0
Smelter Payables		
Cu Payable	%	96.50
Au Payable	%	90.00
Ag Payable	%	95.00
Minimum Deduction in Concentrate	%	1.0
Au Minimum Deduction	g/t	0.0
Ag Minimum Deduction	g/t	0.0
TC/RCs		
Treatment Charge	US$/dmt concentrate	90.00
Refining Charge		
Cu	US $/lb	0.10
Au	US $/oz	6.00
Ag	US $/oz	0.96
Deleterious Element Penalties		
As	US $/dmt concentrate	0.00
Transport Costs		
Ocean Freight	US$/wmt concentrate	100.00
	US$/dmt concentrate	108.00
Royalty	%NSR	1.25
Insurance	US$/$1K value	0.495

Source: JDS (2016)

Mineable blocks, stopes and drifts were defined based on NSR values greater than $70/t. Some lower value or incremental material, greater than $50/t is also included in the mining reserve. The incremental material is predominately development ore that had to be taken to mine the stope in its vicinity.

Cut-off grade was selected by evaluating the net value of multiple stope optimization trials weighted against an estimated operating cost of $50/t, using the calculation tonnes x (NSR – OPEX). The results of this exercise are depicted in Figure 15.1. Stope optimizations were performed using Maptek Vulcan© software at $5.00 NSR increments. The results were plotted together to form a bell curve of net value and identify the optimum cut-off grade.

Figure 15.1 Cut-off Grade Bell Curve



	21	20	16	17	18	19	22	23	24
NET VALUE ($M)	$330	$343	$354	$358	$358	$355	$349	$345	$335
Recovered ktonnes	10,864	9,829	8,791	7,940	6,944	6,259	5,615	5,066	4,528
Recovered Au (g/t)	2.64	2.78	2.96	3.11	3.32	3.49	3.67	3.86	4.04
NSR Cutoff	50	55	60	65	70	75	80	85	90

Source: JDS (2016)

Trial 18 represents the highest net value optimization scenario with an NSR cut-off of $70, which was selected as the cut-off for Romero detailed mine design.

15.2 Dilution

Two types of dilution were applied to the stope designs:

- External dilution – In-situ material that falls into the stope from the geometry of the stope shape; and
- Fill dilution – run of mine waste, and/or paste back fill expected to fall into the stope being mined from adjacent stopes and/or inadvertently scraped off the stope floors during mucking.

The modes of dilution were estimated by mining method and stope type, based on the stope design tonnages, and are summarized in Table 15.3.

Table 15.3: Dilution by Mining Type

Mining Type	External Dilution%	Fill Dilution%	Total Dilution%
Cut and Fill	5.4	4.4	9.8
Long Hole	8.4	2.9	11.3

Source: JDS (2016)

Dilution was calculated from equivalent linear over-break/slough (ELOS) estimated for each mine method in two different ground conditions; good and poor. Good ground conditions exist within the silicified geologic zone, while poor ground conditions exist within the argillic geologic zone, as further explained in Section 7 of this report. Preliminary mine designs were used to query the resource model for approximate ratios of good and poor ground conditions for each mine method, which were then used to estimate the anticipated ELOS in the walls, floor, back, and ends of stopes. Separate dilution calculations were prepared for primary and secondary drifts and stopes, as ELOS material densities vary between in-situ over-break and back fill over-break. The dilution estimation for cut and fill and long hole stopes is shown in Table 15.4.

Table 15.4: Mine Dilution Parameters and Calculation

Mine Development Dimensions	Unit	Cut and Fill	Long Hole
Height	m	4.0	20.0
Width	m	4.0	15.0
Design Tonnes	t/m or t	45.8	23,814
RMR Good (calc from mine plan)	%	53	100
RMR Poor (calc from mine plan)	%	47	0
ELOS Back - Good RMR	m	0.10	0.50
ELOS Back - Poor RMR	m	0.50	4.00
ELOS Back - Average	m	0.13	0.78
ELOS Back - Dilution	tonnes	1.7	1066.3
ELOS Walls - Good RMR	m	0.10	0.25
ELOS Walls - Poor RMR	m	0.50	0.25
ELOS Walls - Average	m	0.13	0.25
ELOS Walls - Dilution	tonnes	1.5	343.6
ELOS Floor - Good RMR	m	0.10	0.25
ELOS Floor - Poor RMR	m	0.50	1.00
ELOS Floor - Average	m	0.13	0.31
ELOS Floor - Dilution	tonnes	1.3	340.4
ELOS HW/FW - Good RMR	m	0.00	1.00
ELOS HW/FW - Poor RMR	m	0.00	2.00
ELOS HW/FW - Average	m	0.00	1.08
ELOS HW/FW - Dilution	tonnes	0.0	924.4
ELOS Total	**tonnes**	**4.5**	**2674.7**
ELOS Factor (Total Dilution)	**%**	**9.8**	**11.3**
External Dilution	**%**	5.4	8.4
Fill Dilution	**%**	4.4	2.9

Source: JDS (2016)

Both the quality and condition of the walls and long hole drilling deviation are considered as key to minimizing wall and adjacent stope dilution. The dilution is within the sulphide envelope and is assumed to carry the grades shown in Table 15.5.

External dilution grades adjacent to the planned stopes is calculated by querying the block model within a 0.7 m dilution envelope constructed around the long hole stope wire frames, to represent the average estimated ELOS. The metal content contained in the envelope is divided over the envelope tonnes to estimate an average dilution grade. The stope dilution grade is then applied to the designed external dilution tonnes and combined with the Mineral Resource tonnes and grade for the final stope grade. Dilution grades are shown in Table 15.5.

Table 15.5: Dilution Grade Values

Metal	Dilution Grade
Au	2.2 g/t
Ag	3.7 g/t
Cu	0.68%

Source: JDS (2016)

Fill dilution is assumed to carry zero metal grades.

Additional sources of dilution include planned or internal dilution and Inferred resource dilution. Planned dilution is comprised of waste material that carries no metal value and is unavoidable in the stope design shape. Any Inferred resource class tonnage within the mining reserve stope shapes have been treated as waste and have been assigned zero metal grades. Planned and Inferred dilution comprises approximately 2.4% of the total reserve respectively.

The total external fill, planned and Inferred dilution is approximately 13.9% of the total mining reserve.

15.3 Mining Recovery

Mining or extraction recovery is a function of mineralized material left behind due to operational constraints typical in the mining process.

The long hole mining method is largely dependent on accuracy of long hole drilling and explosive detonation to properly fracture the ore. Where holes deviate from the ore limits, some material will remain hung up and may never report to the stope floor for recovery.

Lesser factors considered to affect recoveries in long hole mining include ragged mucking floors and limited visibility for remote mucking.

Secondary stopes recognize higher recoveries due to improved probability of blasted mineralization making its way to the stope floor for mucking.

A mining recovery of 95% was assigned based on industry norms as well as JDS operational experience for remote mucking stopes of similar size and dip.

15.4 Mineral Reserve Estimates

The mining stope and sub-level designs with dilution and ore recovery factors applied determined the Mineral Reserve estimate shown in Table 15.6.

Table 15.6: Mineral Reserve Estimate

Category	Tonnes[3]	Au		Ag		Cu		Au-Eq[1]		NSR[2]	
	(x1000)	(g/t)	(koz)	(g/t)	(koz)	(%)	(M lb)	(g/t)	(koz)	US$/t	$M US$
Probable	7,031	3.72	840	4.33	980	0.88	136	5	1,126	121	851

Source: JDS (2016)

(1) Gold equivalent metal prices $1,300/oz Au, $20.00/oz Ag and $2.50/lb Cu
(2) Cut-off NSR metal prices: Cu $2.50/lb Au $1,250/oz Ag $17.00/oz; Recovery: Cu-96.8 Au-71.7 Ag-54.4, Payable: Cu-96.5 Au-90.0 Ag-95.0, TCRC: $257.83/dmt, Cu concentrate 20%

The Mineral Reserves identified in Table 16.6 comply with CIM definitions and Standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified.

This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimates of the Mineral Reserves or potential production.

16 Mining Methods

16.1 Introduction

The mine design and planning for Romero is based on the resource model completed by Micon in 2016, as detailed in Section 14 of this report. The mine design and plan considers Indicated Mineral Resources of the Romero North deposit only. Inferred resources have been excluded from mine planning for this study. Where Inferred resources fall within the stope designs they have been assigned a zero waste grade. Inferred Mineral Resources are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is also no certainty that these inferred Mineral Resources will be converted to Measured and Indicated categories through further drilling, or into Mineral Reserves, once economic considerations are applied.

16.2 Mine Planning Criteria

Mine planning criteria are listed below:

- Pre-production period is approximately nine months, with three months of surface preparations and portal construction, and six months of underground ramp and infrastructure development. Ore is extracted in the first quarter of year one and ramps up quarterly from 50%, 75%, to 100% of the full 2,800 t/d production rate;
- Underground mining and maintenance carried out by Owner, supplemented by contracted supervision and training staff;
- Contract raise bore development will be utilized;
- Conventional, trackless diesel-electric mining equipment will be utilized; and
- Mined voids will be filled with paste fill and mine development waste.

Other key mine planning criteria are summarized in Table 16.1.

.

Table 16.1 Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	365
Shifts per Day	Shifts	2
Hours per Shift	Hour	12
Work Rotation	Four weeks in/Four weeks out	4x4
Nominal Ore Mining Rate	t/d	2,800
Annual Ore Mining Rate	t/a	~1,008,000
Ore Density	t/m^3	variable, from block model. 2.94 average
Waste Density	t/m^3	2.7
Swell Factor		1.35

Source: JDS (2016)

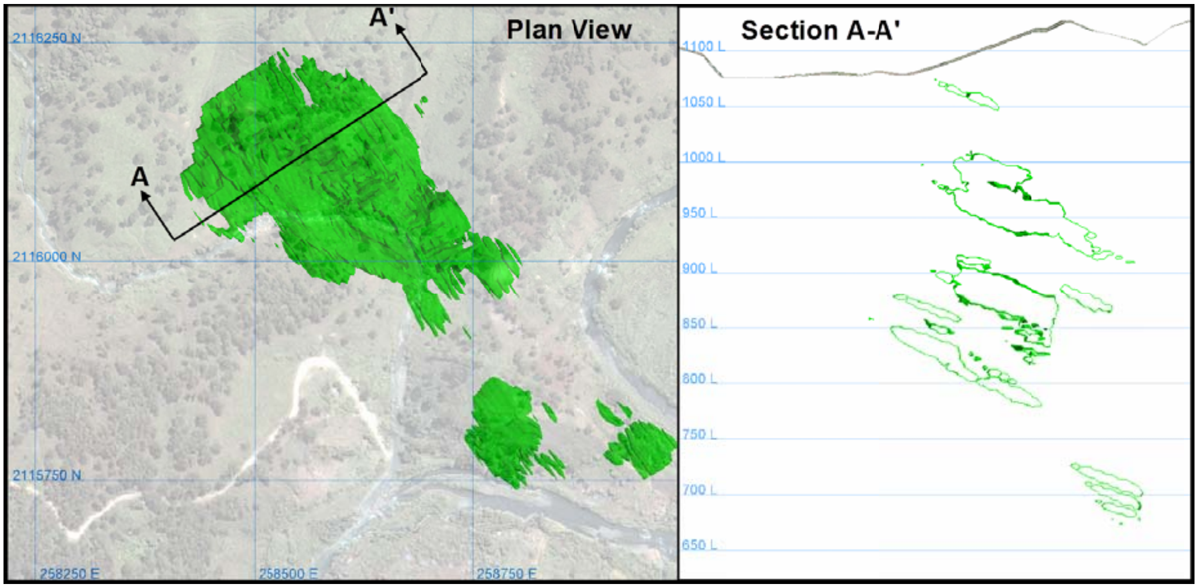
Cut-off NSR value, dilution and mining ore recovery criteria have been defined previously in Sections 15.1 to 15.3 of this report.

16.3 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 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 and ranges up to 50 m. Generally, lower grade mineralization surrounds the higher grade lenses.

The strike length of the main portion of the mining resource is 430 m. Two smaller pods of high grade mineralization exist approximately 200 m along strike to the southeast of the main larger main economic body. The deepest mining level is 420 m below surface (680 m level) and the highest mining level is 85 m below surface (1,000 m level), meaning the total vertical extent of the mining resource is 320 m. Perpendicular to strike, the deposit is about 170 m wide. Figure 16.1 depicts a plan and section view of the resource viewed as a grade shell with $70 NSR cut-off.

Figure 16.1: Romero Deposit Geometry



Source: JDS (2016)

16.4 Mining Methods

Two mining methods are proposed for the Romero deposit, sub-level long hole (LH), stoping and mechanized cut and fill (MCF). A combination of paste backfill and development waste rock fill will be used in the mining sequence. LH stoping will be used in areas of competent ground strength and generally thick Mineral Resource. MCF will be utilized in areas of poor ground conditions and/or thin Mineral Resource which does not warrant LH stoping.

Approximately 85% of the total mining resource will be mined with LH stoping (including ore sub-levels) and the remaining 15% with MCF stopes.

16.4.1 Sub-level Long Hole Stoping

Long hole (LH) stoping provides high productivity at low mining costs from a small number of working faces. All stopes will be filled with a mixture of paste fill and/or development waste.

Geotechnical design have determined stope sizes of 30 m along strike, with widths up to 15 m wide and sub-level to sub-level intervals of 20 m. Stope extraction sequencing is planned to be in a primary-secondary fashion with the lower stopes leading the stopes above. Primary and secondary stopes are sized equally at 15 m wide. After the primary stopes are mined, they will be filled with cemented paste backfill of adequate strength to allow exposure of a 20 m high x 30 m long fill wall adjacent to the secondary stopes that will be mined alongside. Two lifts of primary stopes will be mined before the first secondary stopes are started to allow the drilling drifts to be reused as mucking drifts for the next sub-level above and to minimize the stoping span.

LH stopes will be developed by driving a central ore drift up to mineralization thickness to a maximum 5 m by 4 m high access drift central to the stope.

A slot raise will be developed at one end of the stope by LH drilling and short stage blasting from the bottom up using drop-raise blasting techniques. The slot raise will be enlarged to form a slot across the full width of the stope.

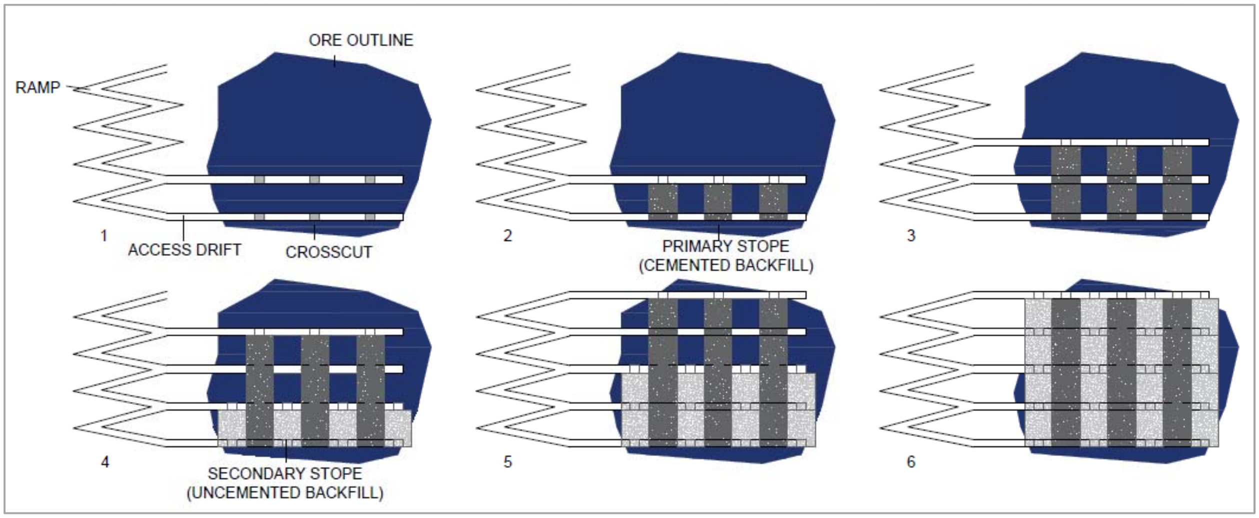
Vertical rings of drill holes will be blasted into the open stope and mineralized material will be mucked from the bottom of the stope by load-haul-dump (LHD) machines with remote control.

The sub-level mining sequence in the ore lenses will be from the bottom up where possible to avoid leaving sill pillars. When mining cannot begin at the bottom of an ore body, the bottom of the first mined stope will be filled with higher strength backfill to facilitate underhand mining for the stope below. Sill pillars have been designed based on a minimum backfill strength of 2 MPa (Golder 2016) and thickness 0.5 x the span of the pillar. After the stopes at the bottom sub-level in a mining block are mined out, it will be backfilled to form the mucking level for the stope above. This sequence will ensure availability of multiple stopes on different sub-levels.

No rib pillars were planned and the stoping sequence with the paste backfilling will allow 100% extraction in the LH stoping blocks.

Illustrative, sub-level stoping diagrams are shown in Figure 16.2.

Figure 16.2: Transverse Long Hole Stoping



Source: JDS (2016)

16.4.2 Mechanized Cut and Fill

Mechanized cut and fill mining will be utilized in thinner areas where LH stopes are not economic. MCF will also be used in areas of poor ground conditions where larger stope are not geotechnically possible. MCF is a lower productivity, higher cost mining method than LH stoping, but provides highly selective mining with minimal dilution. Stopes can be sized with irregular backs and walls to match the ore boundaries.

A two boom, electric hydraulic jumbos will drill 4.88m (16 foot) long rounds on a standard development heading pattern. The drilled holes will be charged with high explosive primers and ANFO and initiated with non-electric caps. After blasting, the heading will be washed, scaled and blasted ore will be mucked with LHDs into trucks and hauled to surface. Ground support will then be installed with a mechanized bolter as required.

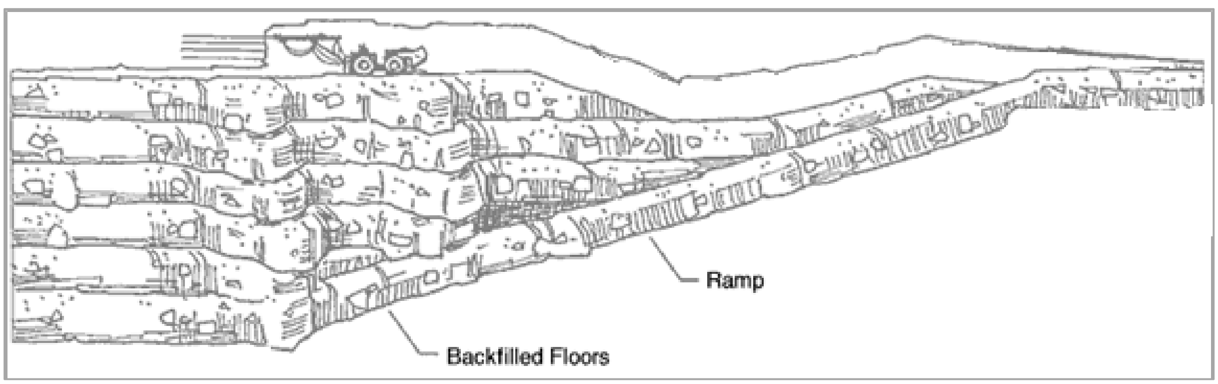
Two types of MCF will be utilized at Romero. Overhand MCF accounts for 73% of MCF mining, and 11% of all production. In overhand MCF each mining block is accessed by an attack ramp and mined in 4 m high lifts. Stopes are developed on the lowest level first, and each subsequent stope or 4 m lift is developed above the depleted and backfilled stope. Mining direction is bottom up.

Underhand MFC accounts for 27% of MCF mining, and 3% of all ore production. In underhand cut and fill an MCF stope is mined out and backfilled with a high strength structural paste fill, after it has been prepared with additional ground support on the floor. Once the structural paste fill has cured the next 4 m lift will commence underneath the filled stope. Mining direction is top-down.

The requirement of both mine methods is a byproduct of optimizing the LH stope production schedule, and avoiding the requirement to mine through backfilled areas to gain access to MCF zones.

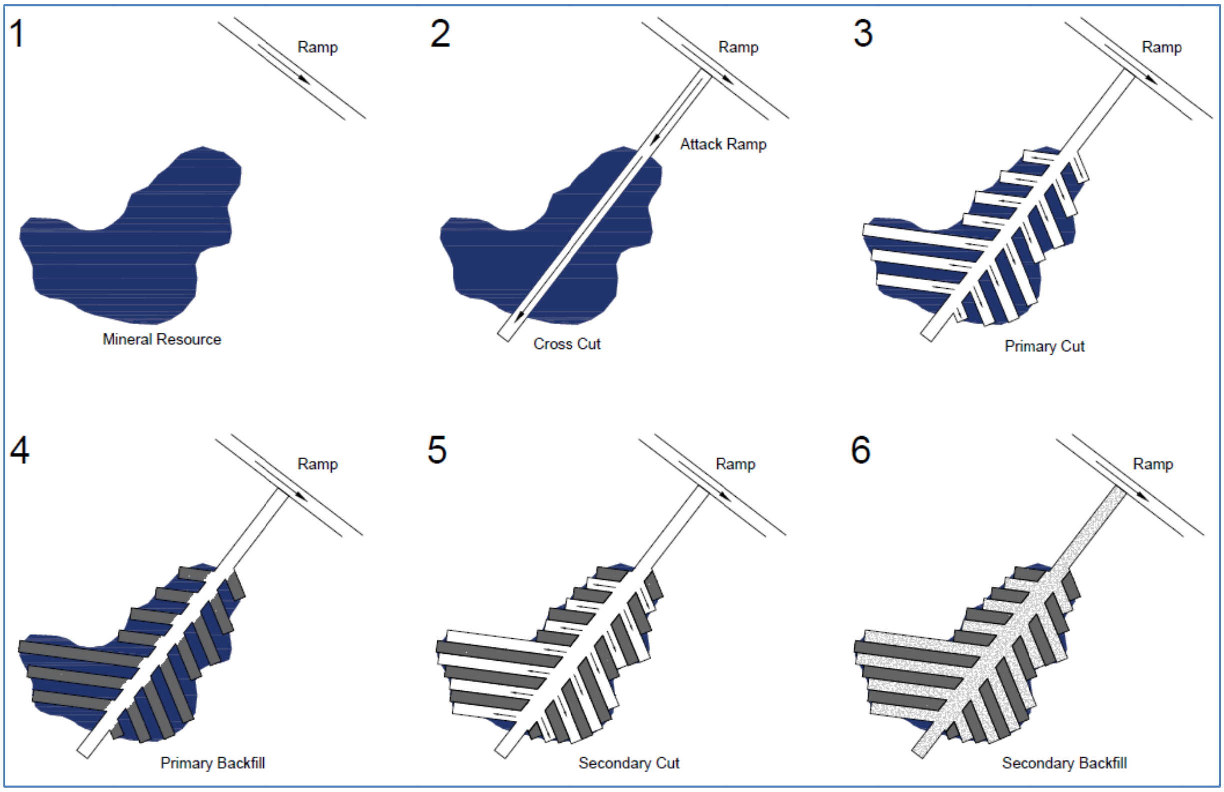
An illustration of overhand MCF level access is shown in Figure 16.3. MCF stope sequencing is illustrated in Figure 16.4

Figure 16.3: MCF Level Access



Source: JDS (2016)

Figure 16.4: MCF Mine Sequence



Source: JDS (2016)

16.5 Geotechnical Criteria

A geotechnical evaluation of the 2015 Romero PEA mine design was conducted by Golder Associates in 2016 (Golder 2016). This evaluation was done in order to prepare ground support criteria and recommendations to be utilized in the development of the 2016 PFS mine design. The results of this evaluation are summarized below.

16.5.1 Alteration

There are four main types of alteration within the Romero deposit area:

Propylitic Alteration – This alteration is regional and usually occurs in the upper andesite, with the tendency to cause the partial chloritization of the iron-magnesium minerals. It is characterized mainly by calcite veins with occasional silica veins. The altered rocks contain magnetite and are therefore magnetic, and the feldspar and amphibolite minerals are still present. This type of regional alteration is not related to the hydrothermal alteration that formed the Romero deposit and does not appear to alter the strength of the host rock. For simplification, carbonate alteration is grouped with the propylitic alteration for the assessments in this report.

Chloritic Alteration – Chloritization was characterized independently of the propylitic alteration within the geotechnical holes and was therefore grouped as its own type of primary alteration, although the strength parameters for the two alteration types are found to be similar. The presence of chlorite as an alteration mineral can be associated with the other types of alteration – propylitic or argillic, but on its own does not appear to influence the strength of the rock mass.

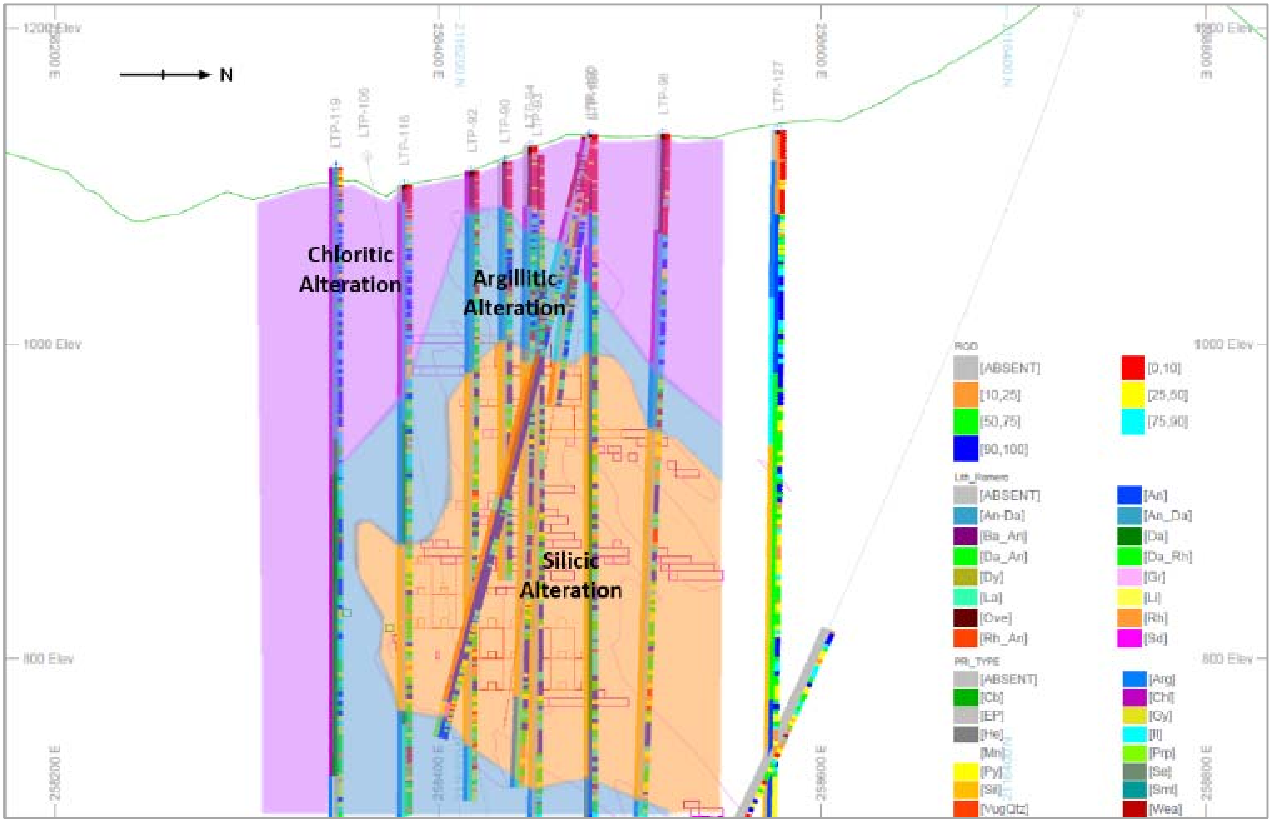
Argillic Alteration – The argillic alteration changes the feldspar minerals into clays and the amphibolite into chlorite. This argillic alteration tends to form a halo of alteration around the silicified andesite/dacite in the centre of the deposit, which was likely formed by the cooling of hydrothermal fluids.

- Based on the geotechnical holes, this alteration occurs above the mineralization and is normally bounded at its base by the semi-massive and massive sulphides;
- It shows a thickness of about 30 m on top of the silicified dacite and can extend to about 50 m laterally;
- Within the zone of moderate to strong argillic alteration, there is essentially a complete change of the feldspar into clay minerals, such as illite and smectite (which is a swelling clay mineral). This zone also contains variable amounts of silica and pyrite and may be slightly mineralized with gold, but not copper;
- This type of alteration reduces the rock strength;
- Within this zone, there is an almost complete destruction of the magnetite, which has been replaced by pyrite. It constitutes the interface between the zones with high magnetic susceptibility (propylitic zones) and the zone with low magnetic susceptibility with higher hydrothermal alteration and higher mineralization; and
- For simplification, illite, argillic and smectite altered zones have been grouped together.

Silicic (or Silicification) Alteration – This type of alteration likely involved higher temperature and/or pressure from the hydrothermal alteration, which caused the replacement / intrusion of silica, and introduced the sulphide minerals. The silicified alteration zone forms the core of the Romero deposit showing high content of copper and gold. The silicic alteration tends to occur in the breccia and dacitic tuff rock units.

Faulting – In order to adequately characterize the rock mass at Romero, intervals that were logged within the geotechnical boreholes by GoldQuest geologists as having weak, moderate or strong degrees of faulting, were grouped as a fault unit.

Figure 16.5: Generalized Alteration Model Based on Exploration Borehole Alteration Data, showing silicic alteration



Source: Golder (2016)

Based on the above descriptions and assumptions, Table 16.2 presents the alteration groupings used for characterization of the Romero rock masses.

Table 16.2 Primary Alteration Grouping[1]

Silicic Alteration [2]	Argillic Alteration [3]	Propylitic Alteration [4]	Chloritic Alteration	Faulting
Sil 1	Arg 1	Prp 1	Chl[5]	Faulting 1
Sil 2	Arg 2	Prp 2	Chl 1	Faulting 2
Sil 3	Arg 3	Prp 3	Chl 2	Faulting 3
VugQtz 2	Il 1	Cb 1	Chl 3	
	Il 2	Cb 2		
	Il 3	Cb 3		
	Smt 3	Py 2		
	Gy 1			

1) Note: the number from 1 to 3 after the alteration type indicates the degree of intensity, as logged by GoldQuest geologists for the geotechnical boreholes (1 – weak, 2 – moderate, 3 – strong).
2) Vuggs (filled with quartz crystals) were included in the silicic alteration due its presence within the silicified dacite / andesite upon review in the field.
3) Argillic, illite (Il), smectite (Smt) and gypsum (Gy) alteration were grouped together as they are all associated with argillic alteration, making the rock mass weaker.
4) Carbonate alteration (Cb) was grouped with the propylitic (regional) alteration, along with pyrite (Py) alteration (13.10 m encountered during drilling).
5) No intensity logged.

Source: Golder (2016)

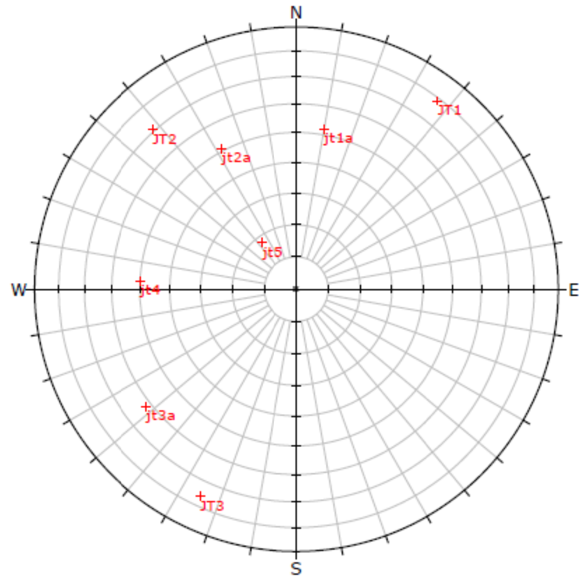
16.5.2 Structural Data

The structural features used in the rock mass characterization were logged by GoldQuest from the six oriented core boreholes LTP-165 to LTP-170. The discontinuity data was analysed statistically using the software DIPS©, distributed by RocScience. The major and minor discontinuity sets were obtained from stereographic projections (stereonets), which provide a 2D representation of the structural data. Details on the structural assessment can be found in Appendix B. For the majority of stereonets, the data was sorted by alteration type.

The sets “JT” and “jt” refer to major and minor joint sets respectively. Major joint sets (JT) were defined when observed in multiple boreholes and therefore assumed pervasive. Where the sets show variations in dip or dip direction, the variations are labelled as subsets such as ‘a’ or ‘b’ with lower case letters to help define the subsets.

Table 16.3 presents a summary of the selected sets with the average dip and dip directions that were used in the stability assessments.

Table 16.3: Summary of Selected Discontinuity Sets

Set	Dip	Dip Direction	Stereonet
Major Sets			
JT1	79	217	
JT2	71	138	
JT3	76	025	
Minor Sets			
JT1A	52	190	
JT2A	51	152	
JT3A	62	052	
JT4	50	093	
JT5	18	144	

Source: Golder (2016)

16.5.3 Rock Strength

Rock strength for the Romero Project was assessed based on field strength estimates, point load testing and laboratory testing from the six geotechnical boreholes.

Intact rock strength envelopes following Hoek-Brown failure criterion (Hoek et al., 2002) have been fitted per alteration type. Hoek-Brown parameters (m_i and σ_{ci}) were derived in RocData (RocScience, v 4.012) based on the simplex best-fit Hoek-Brown curve through laboratory data. A summary of Golder's interpretation of the Hoek-Brown intact rock strength envelopes per alteration type is shown in Table 16.4.

Table 16.4 Summary of intact rock strength parameters

Alteration Type	UCS (MPa)		Hoek-Brown	
	Average	Count	σ_{ci} (MPa)	mi
Argillic	27	9	32	8
Silicic	136	9	128	20
Propylitic	42	2	42	12
Chloritic	83	3	83	23

UCS = uniaxial compression testing

Source: Golder (2016)

16.5.4 Rock Mass Classification

Rock mass classification systems are used for rock engineering projects to provide a quantitative index of rock mass quality based on measurements and observations of rock mass characteristics. An assessment of the overall quality of the rock mass for the Romero Project area has been prepared using the RMR_{76} (Bieniawski, 1976) and the NGI-Q (Barton et al., 1974) rock mass classification systems.

The rock mass classification was calculated on a per run basis for each of the geotechnical boreholes. A general trend of higher rock mass quality ratings (Q' and RMR) associated with silicic alteration and lower ratings with faulting and argillic alteration was observed from these estimations on a per run basis. However, upon review of the average numbers by alteration grouping that the per drill run analysis yielded, Golder considers these rock mass ratings too optimistic and not representative of the core observed during the 2016 site visit (with the understanding that this core has been split and subject to degradation by exposure). The average values obtained from the per drill run assessment were therefore considered more representative of the best encountered conditions for each alteration type and downgraded based on engineering judgement of expected ground conditions at Romero (based on discussions with GoldQuest on site and core observations). The ranges in RMR_{76} and Q' ratings by alteration type are provided in Table 5. Approximately 194 m of argillic altered rock was logged, 245 m of propylitic altered rock was logged, 184 m of chloritic altered rock was logged and 312 m of faulted rock was encountered in the 2015 geotechnical drilling program. Approximately 735 m of silicic altered core was intercepted. Of this total, about 100 m of core was observed to have secondary argillic (or illite) alteration that appeared to have more of an effect on the rock mass quality than the silicic alteration and consequently, presented separately in Table 16.5.

Table 16.5: RMR_{76} Rating and Q' Rating for Alteration Groups

Alteration Type	Average RMR_{76} (range)	RMR_{76} Quality	Average Q' (range)	Q' Quality
Silicic	76 (63 – 85)	Good	20 (6 – 47)	Good
Silicic and Argillic	58 (43 – 68)	Fair	6 (1 – 14)	Fair
Propylitic	59 (42 – 73)	Fair	5 (1 – 9)	Fair
Chloritic	65 (50 – 76)	Good	4.2 (1 – 10)	Fair
Argillic	57 (37 – 64)	Fair	4 (1 – 9)	Poor-Fair
Fault Zones	28 (13 – 31)	Poor	0.3 (0.03 – 0.7)	Very Poor

Source: Golder (2016)

16.5.5 Stope Sizing Assessment

The methodology and parameters used for this assessment are explained as follows. The dilution of the stope walls at the Romero site was estimated and provides ranges of potential over-break depths that are difficult to define prior to the start of mining. Due to the unknown nature of the rock excavated at depth and stress regime of the area before and during excavation, five (5) different stope assessments were completed for different stope depths and orientations, which affect the induced stresses acting on the stope walls and structural geology influences. Stopes excavated in silicic altered rock, argillic altered rock and silicic altered rock with secondary argillic alteration, were considered for this assessment. Table 16.6 and Table 16.7 outline the five different stope assessments.

Table 16.6: Shallow Transverse Stope Stability Assessment (depth of 150 m)

Stope Type	Case 2A Argillic Hanging Wall (HW)		Case 2B Silicified HW	
Stope Depth	150 mbgs			
Stope Azimuth	55			
Stope Width	12 m		15 m	
Stope Strike Length	12 m		12 m	
Stope Height	20 m			
Stope Wall Dip	90			
Stope Wall	Stability	ELOS	Stability	ELOS
Back	Unsupported Transition (With ground support)	4 m (Assume 1 m)	Stable	<0.5 m
End Wall	Stable	<0.5 m	Stable	<0.5 m
Hanging Wall	Stable	<0.5 m	Unsupported Transition (With ground support)	4 m (Assume up to 2 m)
Footwall	Stable	<0.5 m	Stable	<0.5 m

Source: Golder (2016)

Table 16.7: Deep Longitudinal Stope Stability Assessment

Stope Type	Case 2A Argillic HW		Case 2B Silicified HW		Case 2C Silicified and Argillic HW	
Stope Depth	400 mbgs					
Stope Azimuth	145					
Stope Width	15 m		15 m		15 m	
Stope Strike Length	12 m		30 m		15 m	
Stope Height	20 m					
Stope Wall Dip	90					
Stope Wall	Stability	ELOS	Stability	ELOS	Stability	ELOS
Back	Stable	1 m	Stable	1 m	Stable	1 m
End Wall	Stable	1 m	Stable	1 m	Stable	1 m
Hanging Wall	Unsupported Transition	>4 m	Stable	1 m	Unsupported Transition	1 m
Footwall	Stable	<0.5 m	Stable	<0.5 m	Stable	<0.5 m

Source: Golder (2016)

For Case 1A, stope dimensions of 12 m (L) x 12 m (W) x 20 m (H) would present mineable hanging wall (HW), footwall (FW) and end walls with no support. However, the stope back would require cable bolts installed from the overcut drift. For Case 1B, the stability curves show that for a 12 m (L) x 15 m (W) x 20 m (H) stope, would present mineable back, footwall and end walls with no support

For Case 2A, the stope strike length in this orientation was reduced from 30 m to 12 m due to the weak argillic altered rock present in the hanging wall and a larger stope will not be possible at this stope depth and rock strength.

For Case 2B, the stope dimensions as measured in the existing 3D mine plan model (30 m length x 15 m width x 20 m height) are mineable where stopes exist within the silicified ore body with no major structural or weak alteration influences. Case 2B would represent the majority of the stopes within the silicified mineralized zone. Reduced stope dimensions and/or additional ground support may be required when crossing or in close proximity to a fault zone.

Case 2C shows the potential for wall failure of the hanging walls due to the reduced intact rock strength from the secondary argillic alteration and orientation of the stope within the ore body.

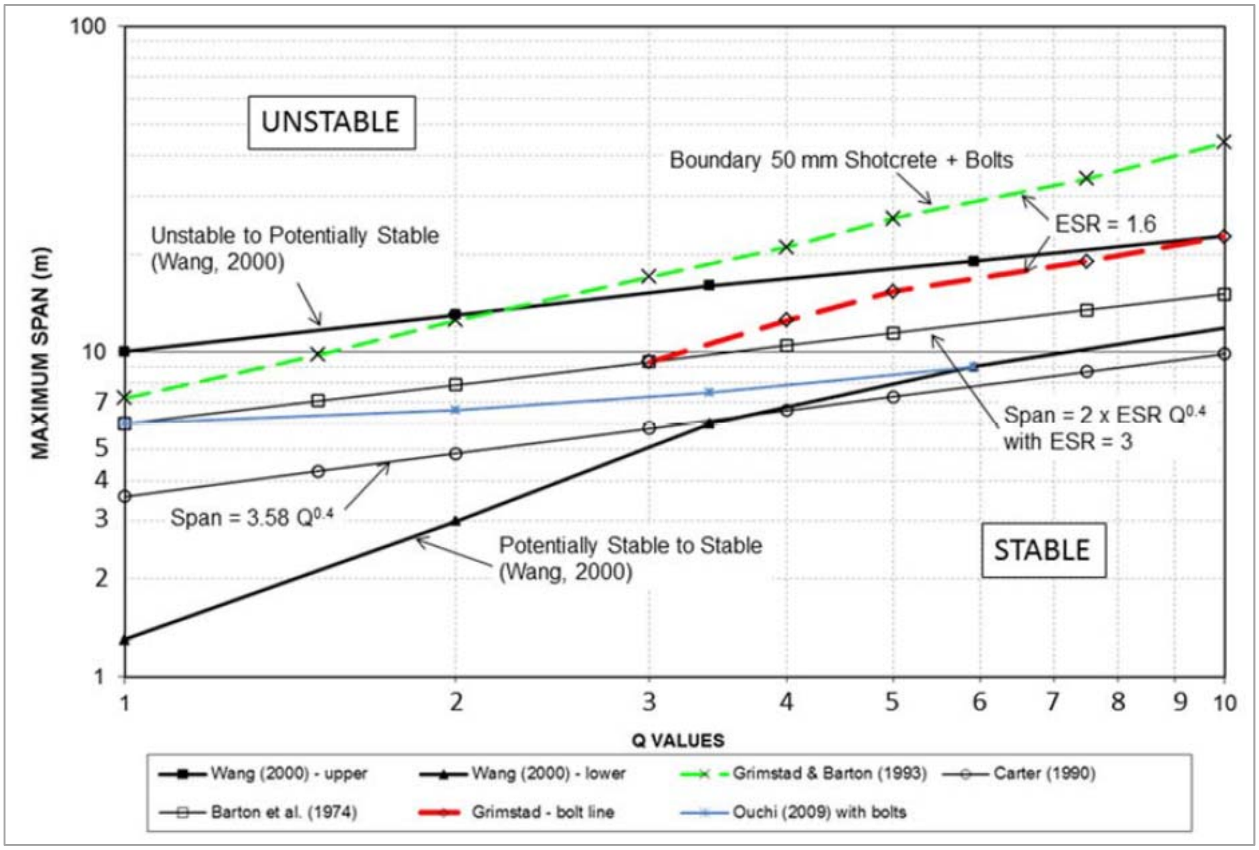
The results of the five stope assessments indicate the need for additional support and reduced excavation dimensions when operating within the argillic alteration zones. For this reason the Romero mine plan contains LH stopes of dimension 20m tall x 15 m wide x 30 m long, situated solely in the silicified zone. The remainder of the economic resource is to be mined using smaller excavation methods in the form of underhand and overhand cut and fill.

Figure 16.6 shows the maximum spans as a function of the rock mass quality using the Q-rock mass classification system for these empirical methods (Barton et al. 1974, Carter 1990, and Wang et al 2000).

Plate 15 also includes an upper limit span, estimated using the support line that divides the areas with shotcrete and bolts and fibre- or mesh- reinforced shotcrete and bolts in the Grimstad and Barton (1993) support chart. Above this limit, heavy ground support would be required, which could impact unfavourably on the economics of drift and fill mining.

Based on the rock mass quality presented in Table 16.5, considering Jw = 1 (dry) and SRF = 2 (moderate stress level), Figure 16.6 and Table 16.8 provide estimates of the maximum unsupported and supported spans for the drift and fill areas.

Figure 16.6: Maximum Unsupported and Supported Spans versus Rock Mass Quality



Source: Golder (2016)

Table 16.8 Unsupported and Supported Spans for Drift and Fill Stopes

Alteration Type	Rock Mass Quality	Q Values	Q' Values with SRF = 2	Maximum Unsupported Span (Requires Minimum Support)			Maximum Supported Spans		
				Barton et al. (1974) 2 x ESR x Q^0.4 with ESR = 3	Carter (1990) 3.58 x Q^0.44	Recommended Unsupported Span	Ouchi et al. (2009)/Wang (2000)	Grimstad (1993) – Bolt Line	Recommended Supported Span [1 to 3]
Silicified	Good Quality (Avg Condition)	20	10	15.1	9.9	10	12	11.2	11
	Fair Quality (Lower Bound Condition)	6	3	9.3	5.8	6	7.3	9	8
Silicified and Argillic	Fair Quality (Avg Condition)	6	3	9.3	5.8	6	7.3	9	7
	Poor Quality (Lower Bound Condition)	1	0.5	Not Applicable			-	-	44
Argillic, Propylitic, Chloritic	Fair Quality (Avg Condition)	4	2	7.9	4.9	5	6.5	-	5
	Poor Quality (Lower Bound Condition)	1	0.5	Not Applicable			-	-	44
Faults	Extremely to Very Poor Condition	0.3	0.15	Not Applicable			-	-	44

Notes:
1) Openings are assumed temporary, i.e., they are considered to be open for less than 6 months
2) Bolt length should be at least 1/3 the effective span of the opening
3) Standard ground support guidelines are presented in Section 5.0
4) 75 mm to 100 mm of mesh reinforced shotcrete will be required (see Section 5.1.4)
Source: Golder (2016)

Recommended spans exceed 10 m in good ground, and should not exceed 4 m in poor ground conditions. The majority of cut and fill stopes designed for the Romero mine plan target argillic zones where ground conditions are expected to be poor. For this reason the cut and fill stopes at Romero are planned to be 4.0 m tall by 4.0 m wide to 4.0 m tall by 5.0 m wide.

16.5.6 Backfill Strength

In open stope mining, fill is placed to prevent the uncontrolled collapse of the stope walls. Under these circumstances, the main requirement of the cemented fill is for it to remain stable when a fill wall is exposed by mining an immediately adjacent stope. Cement is therefore added to provide cohesion within the fill mass so that it stands unsupported when exposed. Again, the purpose of the cement is not to stiffen the fill mass in a regional sense but rather to provide for the stability of exposed fill walls. Thus the height and width of the planned exposure significantly influence design considerations.

It is assumed that prior to mining against a filled stope, the primary stope has been filled with cemented paste backfill and sufficient time has been allowed for the fill to gain the required strength. To minimize dilution and improve regional ground support after the mining of that area is complete, the backfill must remain self-supporting when exposed. The time for the fill to achieve the required strength will have to be obtained by lab and field trials.

Two equations were used to estimate the strength of the backfill based on Mitchell et al. (1981) considering both the simplified compressive strength requirement (i.e., $UCS = \gamma H / 1+H/L$) and a wedge-type of failure of the fill.

The parameters required are the stope dimensions, density of the fill and the internal angle of friction, as follows:

- Length of the stope: 12 to 15 m;
- Height of the stope: 20 m;
- Stope width: 15 to 30 m;
- Bulk unit weight of the fill (γ): 20 N/m^3; and
- Internal angle of Friction: 30° to 35°.

Table 16.9 shows the strength required by the fill for the stope to be self-supporting at various stope dimensions. A safety factor of three is used to account for variability on the paste fill strengths and maximize the potential for primary stopes to have good fill stability. The table below depicts the highest strength requirements from the two strength equations evaluated in this study.

Table 16.9: UCS Strength Required for Self-Supporting Fill

Length or Width of Exposed Face (m)	Height of Exposed Face (m)	UCS Strength Required (KPa) (Including Safety Factor of 3)
12	20	580
15	20	640
20	20	700
25	20	730
30	20	770

Source: Golder (2016)

The addition of Portland cement as a binding agent will provide the required paste fill strength. Secondary stopes will not require structural backfill; however, binder will be added to the paste fill in order to prevent liquefaction after deposition. The following binder content assumptions in Table 16.10 were utilized for the Romero mine design.

Table 16.10: Backfill Strength and Binder Content

Type	Strength (KPa)	Binder Content (%)	Application
High Strength Backfill	2,000	8%	Underhand MCF and Sills
Medium Strength Backfill	700	6%	Primary LH
Low Strength Backfill	175	2.50%	MCF and secondary LH

Source: Golder (2016) and Minefill 2016

16.5.7 Ground Support Recommendations

Ground support recommendations were designed for the Romero mine plan based on drift type and ground condition. Alteration zones were flagged within the Romero resource block model and the development drifts were queried for percent content within each zone. This information was then used to apply the appropriate ground support materials to the different drift types within the mine plan.

Table 16.11 through Table 16.13 below outline the results of this review, which describes the ground support material type, length, and spacing planned for use in the Romero underground mine. As a minimum requirement 100% of the back and shoulders will be supported with welded wire mesh and resin rebar rock bolts, with split set rock bolts used to supplement pinning mesh tight to the back. Additional wall support, cable bolting, and shotcrete will be used in decreasing ground conditions and in large permanent spans such as intersections and main entries.

Table 16.11 Ground Support Criteria - Cut and Fill and Capital Development

Description	Unit	Cut and Fill Development			Capital Development		
		Good Ground	Fair Ground	Poor Ground	Good Ground	Fair Ground	Poor Ground
Bolt Spacing							
Back							
Resin Rebar 1.8m	m	1.2	1.2		1.2		
Resin Rebar 2.4m	m			1.2		1.2	1.2
Split Set 0.6m	m	1.2	1.2	1.2	1.2	1.2	1.2
Walls							
Resin Rebar 1.8m	m				1.2	1.2	1.2
Split Set 1.5m	m	1.2	1.2	1.2			
Mesh							
Distance From Floor	m	2.5	1.5	0.0	1.5	1.0	0.0
#9 Welded Wire	m^2	21.0	19.0				
#6 Welded Wire	m^2			14.0	11.5	12.5	14.5
Shotcrete							
Thickness	mm	0.0	0.0	100.0	0.0	0.0	100.0

Source: Golder (2016)

Table 16.12 Ground Support Criteria - LH Sub-levels

Description	Unit	LH Sub-level - Primary			LH Sub-level - Secondary		
		Good Ground	Fair Ground	Poor Ground	Good Ground	Fair Ground	Poor Ground
Bolt Spacing							
Back							
Resin Rebar 1.8m	m	1.2	1.2		1.2	1.2	
Resin Rebar 2.4m	m			1.2			1.2
Split Set 0.6m	m	1.2	1.2	1.2	1.2	1.2	1.2
Cable Bolt 25t 8.0m	m				2.0	2.0	2.0
Walls							
Resin Rebar 1.8m	m	1.2	1.2	1.2	1.2	1.2	1.2
Cable Bolt 25t 6.0m	m	1.2	1.2	1.2	1.2	1.2	1.2
Mesh							
Distance From Floor	m	1.5	1.0	0.0	1.5	1.0	0.0
#9 Welded Wire	m^2	13.0	13.0	-	13.0	13.0	-
#6 Welded Wire	m^2			12.0			12.0
Shotcrete							
Thickness	mm	0.0	0.0	100.0	0.0	0.0	100.0

Source: Golder (2016)

Table 16.13 Ground Support Criteria - Intersections

Description	Unit	Intersections		
		Good Ground	Fair Ground	Poor Ground
Bolt Spacing				
Back				
Resin Rebar 2.4m	m	1.2	1.2	1.2
Split Set 0.6m	m	1.2	1.2	1.2
Cable Bolt 25t 3.6m	m	2.0	2.0	2.0
Walls				
Resin Rebar 1.8m	m	1.2	1.2	1.2
Mesh				
Distance From Floor	m	0.0	0.0	0.0
#6 Welded Wire	m^2	14.5	14.5	14.5
Shotcrete				
Thickness	mm	0.0	100.0	100.0

Source: Golder (2016)

When drifting through poor ground conditions two coats of shotcrete will be applied. A flash coat of 25 mm of shotcrete will be applied on the back and shoulders before starting mucking activitites, and a second coat will be applied after the installation of bolts and mesh.

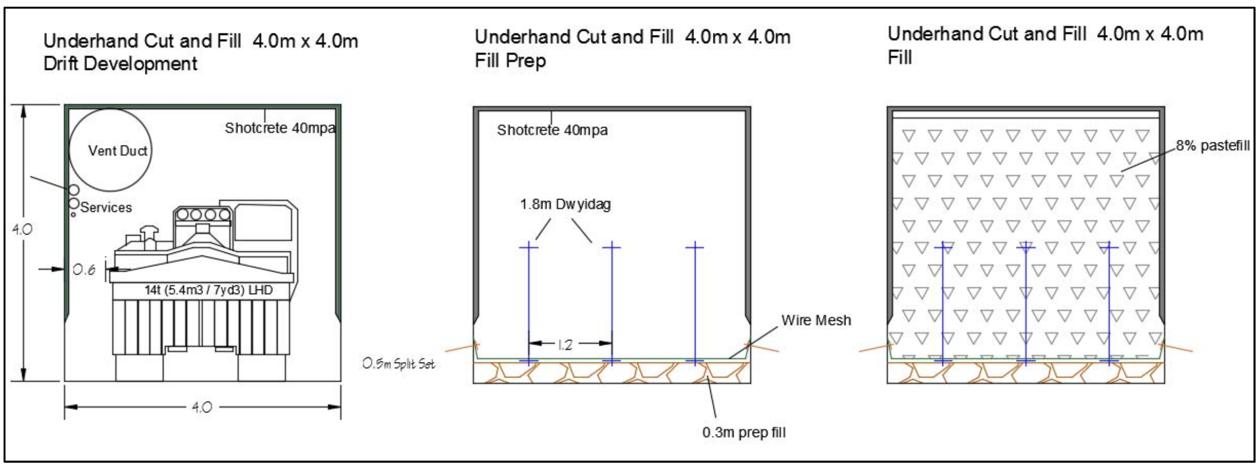
Cable bolting will be required at intersections and along the backs of top cuts in LH stopes.

In addition to the minimum ground support requirements stated above, in areas where mining is to take place beneath a backfilled stope, such as in underhand cut and fill mining, or mining up against a sill pillar, the following sill pillar preparations will first be made.

1. Spread 30 cm of prep fill over the floor to act as a sacrificial blast curtain to protect the structural fill from mining activities in the cut below;
2. Lay 2-4" aperture wire mesh on the floor and pin to the corners of the drift;
3. Erect 1.8 m Dwyidag rock bolts on a 1.2 m spacing throughout the drift, with plates on either end and on either side of the wire mesh placed on the floor;
4. Lay plates on the floor on 1.2 m spacing prior to installing the mesh to ease installation; and
5. Use twine to tie rock bolts standing vertically.

Figure 16.7 below depicts the stope preparations required for mining under a backfilled zone.

Figure 16.7: Stope preparation prior to Underhand Mining



Source: JDS (2016)

16.6 Hydrogeology Criteria

Hydrogeologic reviews of the Romero deposit suggests that very little water inflow is expected in the underground mine, and seasonal inflows from heavy rain events will be minimal.

16.7 Mine Design

16.7.1 Optimization

Mine planning for the Romero Project was completed by using Maptek© Vulcan 3D (Vulcan) and Minemax iGantt software.

Vulcan's stope optimization software was utilized to generate optimum stope shapes within the resource block model. Based on geotechnical criteria provided by Golder (2016), stope dimensions of 15 m wide x 20 m tall x 30 m deep were run through the optimizer at $5 NSR cut-off increments until the optimal cut-off NSR of $70 was realized. Stope optimization parameters for the selected optimization trial are listed below in Table 16.14.

Table 16.14: Stope Optimization Parameters

Parameter	Unit	Value
Block Model		JDS_RBM_20160525_stope_op.bmf
Cut-off Variable		nsr_jds_pfs_2016
Stope Orientation Plane		YZ
Framework Bearing	degrees	55
Framework Plunge	degrees	0
Framework Dip	degrees	0
Rotation Origin X	m	258788.3
Rotation Origin Y	m	2115292
Rotation Origin Z	m	560
Offset X	m	600
Offset Y	m	1200
Offset Z	m	560
Step Y	m	15
Step Z	m	20
Maximum Waste Fraction	%	40
NSR Cut-Off	$/t	70
Minimum Stope depth	m	10
Wall Dilution	m	1
Top to Bottom Max Ratio	#	2.25
Max Strike Deviation	degrees	20 (-10 to +10 w/ max change 5.0)
F/W H/W Max Dip	degrees	60/120
Max Dip Change between stopes	degrees	20

Source: JDS (2016)

The resulting stope optimizer shapes were reviewed in plan and section and adjusted as necessary to improve grade, tonnage, and/or mineability of each stope.

LH stopes were restricted to Indicated resource blocks, as well as blocks outside of the argillic zone. The argillic zone has been identified in geotechnical design (Golder 2016) as a zone of poor ground quality and will not be suitable for LH mining. The argillic zone was, flagged for extraction by MCF.

For the remaining resource inside the argillic zone above $70 NSR, and areas too thin or awkwardly oriented for LH stoping, the block model was reviewed in 5 m sections and MCF stopes were constructed.

iGantt scheduling software 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.

16.7.2 Access

The Romero deposit will be accessed via a single portal developed in the Hondo Valle village. A decline will be driven at -15% grade from the portal entrance and descend northwest towards the deposit, crossing under the San Juan River 40m below surface. The decline will be developed 5.0 m wide by 4.5 m tall to accommodate 40 tonne haul trucks and temporary twin 1.2 m diameter vent ducts.

16.7.3 Development Types

The decline leads to a spiral ramp 125 m below surface, which provides access to each production level spaced 20 m vertically apart. The spiral ramp is driven at -15% grade and 5.0 m wide by 4.5 m tall, with a maximum curvature radius of 25 m. At each operating level the spiral ramp will run at 0% grade for 20 m to provide equipment better visibility and turning abilities on and off the haulage ramp.

Access drifts and footwall drives are developed 5.0 m x 4.5 m to allow truck access and reduce the haul distance of LHDs. Footwall drifts are spaced 20 m away from LH stopes to prevent stability issues as a result of production blasting.

LH stopes are accessed by 5.0 m W x 4.0 m H cross-cuts developed from footwall drives on 15 m spacing. Cross-cuts are used to provide a platform for LH production drills, as well as remote mucking access for blasted material.

MCF zones are accessed by attack ramps from footwall drives or the haulage ramp directly. Attack ramps are driven at a maximum 18% grade and will stack vertically to access multiple production levels from a single access point, as shown in Figure 16.3. Cut and fill drifts are driven at 4.0 m x 4.0 m in order to maintain structural integrity in the lower rock quality areas for which cut and fill is targeted.

Ventilation access drifts are driven on each level to ensure fresh and exhaust air raise connections to the stoping levels. The cross-cuts are approximately 4.0 m wide x 4.0 m high.

Remucks are excavated on the main ramp and footwall drives to help speed up the development mucking cycle. A maximum of 150 m separates the remucks, which are typically driven 5.0 m wide x 4.5 m high x 12 m long.

The back at the intersection of remucks and the connecting drift will require slashing to 6.4 m tall in order to allow full extension and dumping of the LHD bucket, as shown in Figure 16.6.

Water collection sumps are located on every level. Sumps have been sized at 4.5 m high x 5.0 m wide. Three main sumps are planned at 72 m, 140 m, and 300 m below surface. A main sump of 72 m below surface will be used for water storage and reuse as drill water, as well it will discharge water to the surface collection pond for treatment.

There are storage areas for both detonators and explosives underground. These will be placed on the main decline.

Electric power centres will be located outside the access drift on each level in drifts 4.0 m high x 4.0 m wide.

Refuge stations will be on every third level with the first located on the 940 m level. Portable refuge stations will also be moved and located as required throughout the mine.

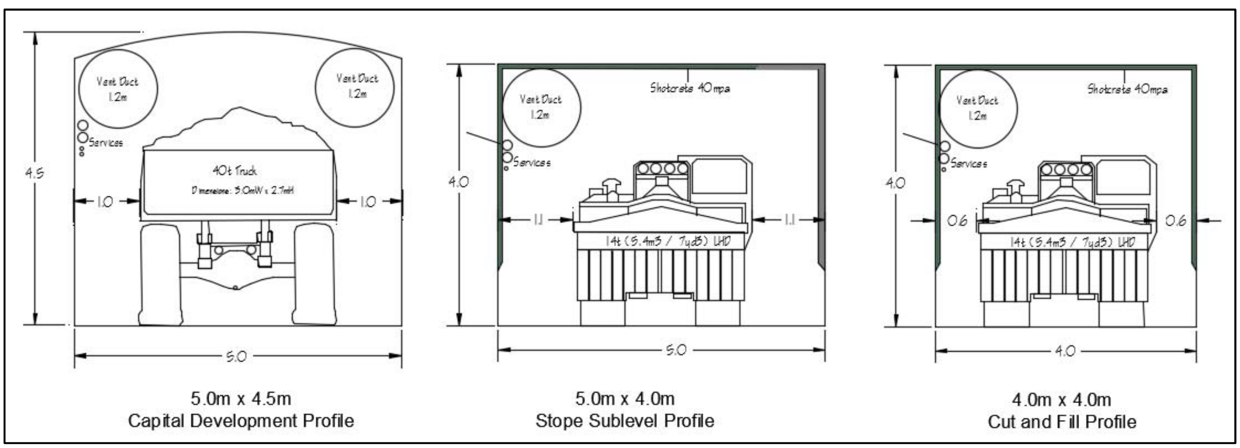
There is no plan to develop drifts dedicated entirely to diamond drilling. Any definition diamond drilling will likely be carried out from the main ramp or the truck load-out zone.

A fresh air raise 3.0 m diameter will be driven to connect the access drift of each level. Two exhaust raises 3.0 m diameter will be developed at the extents of footwall drifts on each level. The raises are driven via raisebore and the fresh air raise will be equipped with ladders for secondary egress. The raises are sequenced in a leapfrog pattern to enable the fresh air to be carried in the direction of the ramp progression.

In general, long term development will receive 2.0 m radius arched back, while all temporary drifts will be driven with a flat back. In areas of poor ground it may be required to drive stope sub-levels with an arched back, as their life span is generally longer than that of a MCF drift.

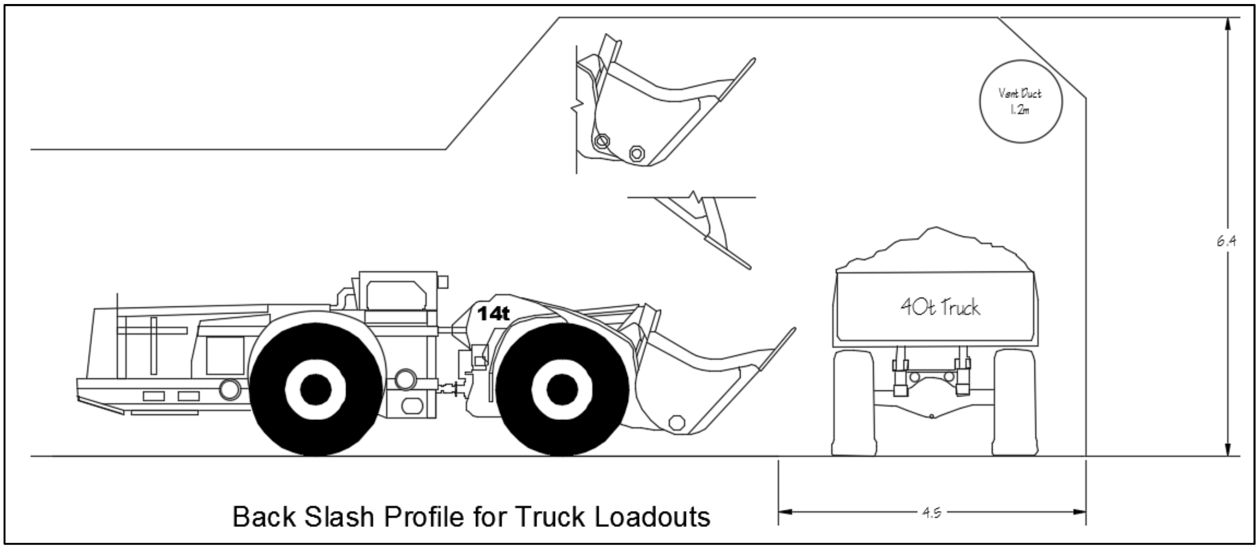
Figure 16.8 and Figure 16.9 depicts the various drift dimensions used in the Romero mine plan.

Figure 16.8: Drift Profiles



Source: JDS (2016)

Figure 16.9: Remuck Back Slash Long Section



Source: JDS (2016)

Figure 16.10 and Figure 16.11 depict the general arrangement of the mine plan in long section and plan view.

Figure 16.10: Mine Design Plan View

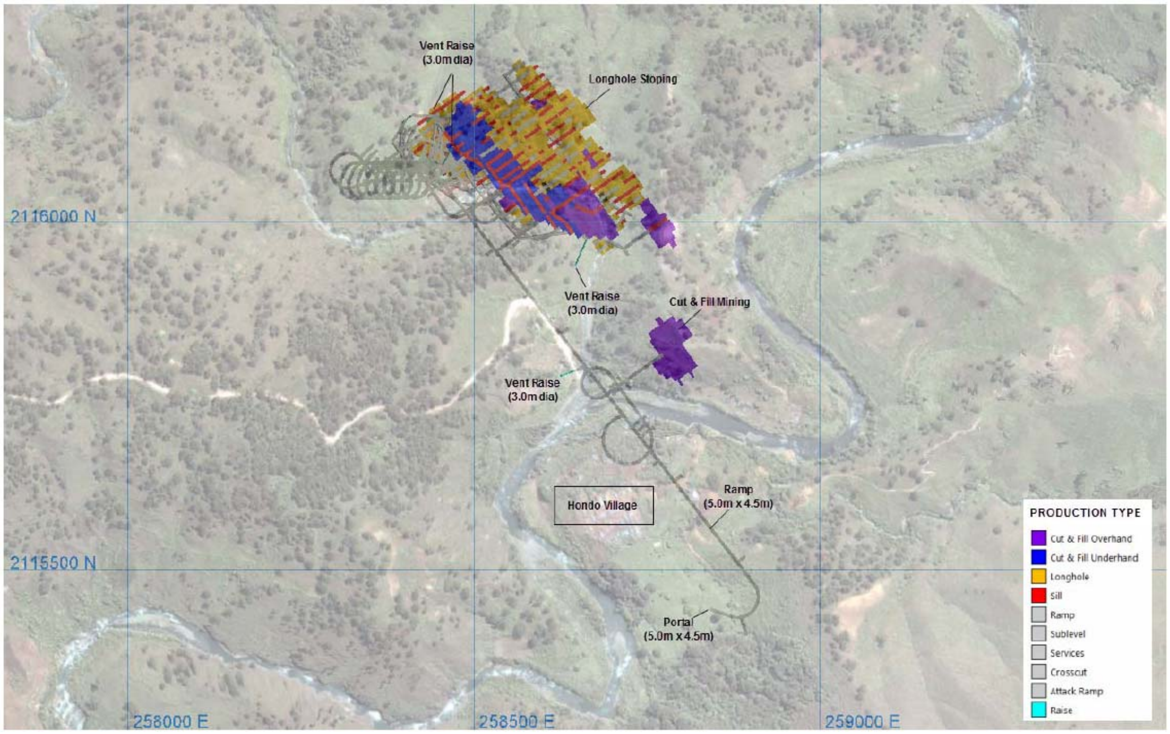
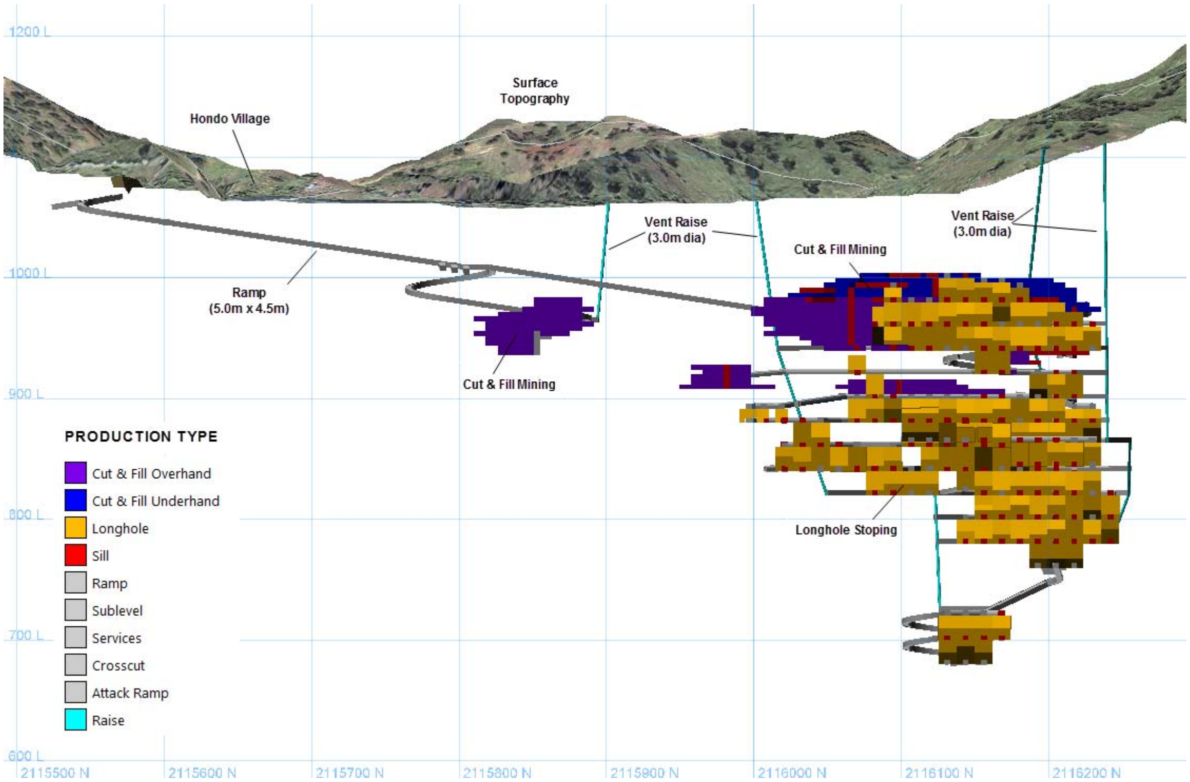


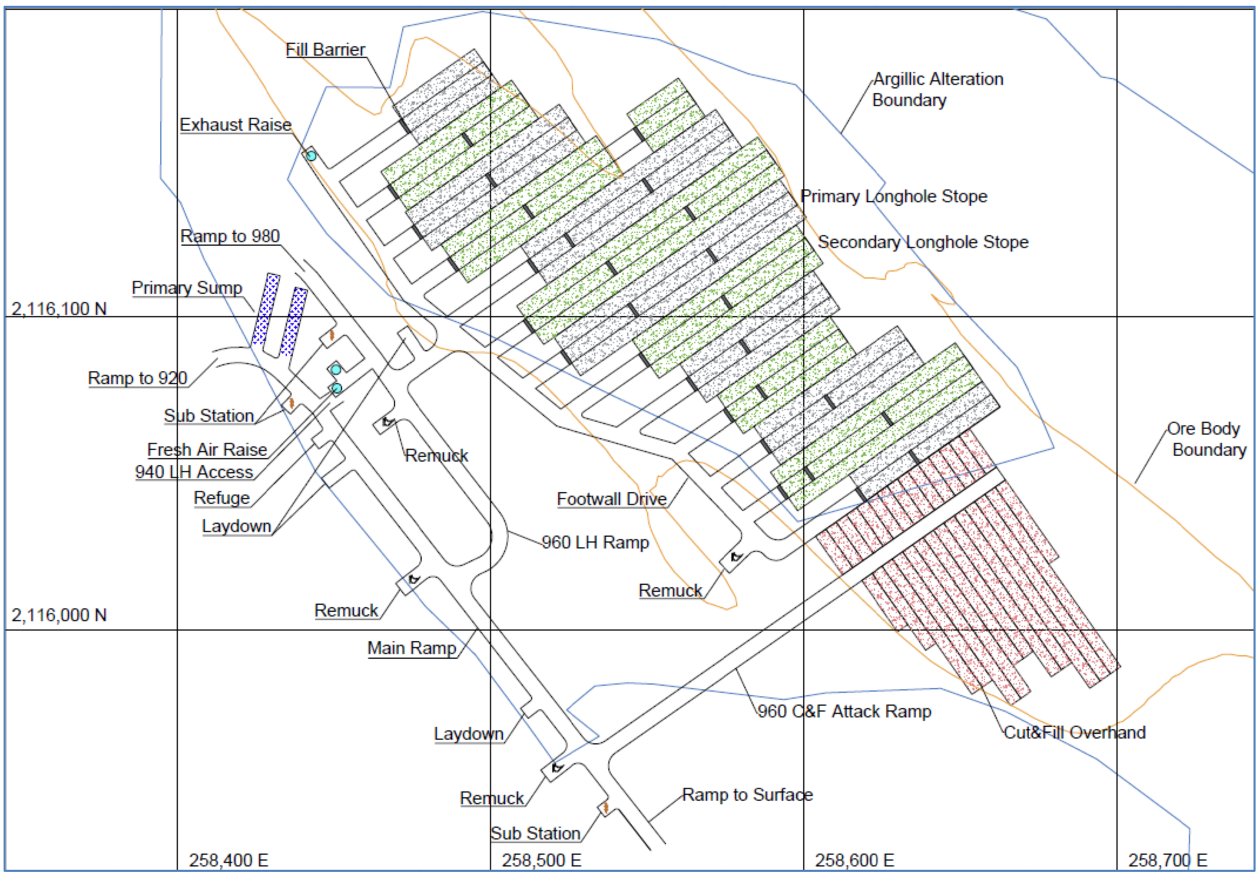
Figure 16.11: Mine Design Long Section



16.7.4 Mine Design Considerations

The geotechnical review prepared by Golder Associates highlighted the potential difficulty and increased support requirements involved in creating large open stopes in the weaker argillic alteration zones in the Romero resource. As a result of this, the PFS mine design has been optimized to restrict LH stoping to silicified alteration zones and extract any remaining ore through MCF within the argillic alteration zones. Figure 16.12 below depicts the 960 m elevation level, where LH stoping is restricted within the silicified resource (top left), and MCF stopes extract the remaining economic resource (bottom right).

Figure 16.12: Level Plan 960 m Elevation



Source: Golder (2016)

16.8 Mine Services

16.8.1 Mine Ventilation

The design basis of the underground ventilation system at Romero is to adequately dilute exhaust gases produced by underground diesel equipment. Air volume was calculated using CANMET (NRCAN 2016) ventilation regulations for each diesel engine operating underground, which state the required ventilation volume in cubic feet per minute (cfm) for an engine operating at 2,200 rpm. In addition to minimum requirements a 5% air leakage was assumed for joins, doors, and damaged bagging and regulators. Table 16.15 lists the air requirements for full production with the total 396,000 cfm (187 m^3/s) air volume required.

Table 16.15: Diesel Equipment Ventilation Requirements

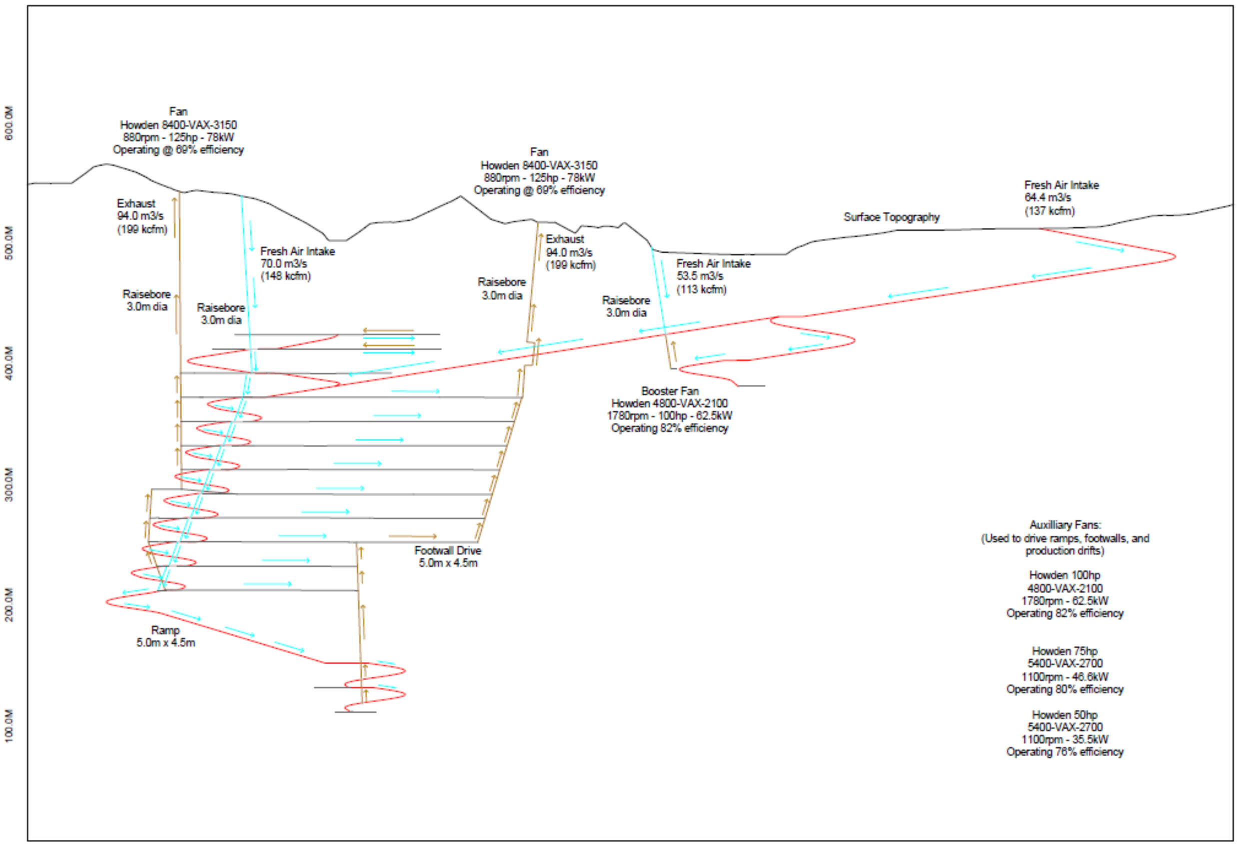
Mobile Equipment	Requirements (ft^3/m)	Utilization (%)	Max Quantity
Truck (40t/19.0m^3)	43,238	85	5
LHD (14t/6.4m^3)	27,855	82	4
Jumbo 2 Boom	2,661	57	2
Bolter	2,445	73	3
ANFO Loader	2,051	87	1
Long Hole	2,877	80	2
Scissor Lift	4,462	66	1
Shotcrete Mobile	4,296	64	1
Personnel Carrier	10,593	26	1
Fuel/Lube	10,593	66	2
Boom Truck	10,593	53	2
Grader	9,512	40	1
Tractor	3,692	40	2
Backhoe	6,125	26	2
Telehandler	4,989	40	2
Mechanics Truck	6,336	53	4
Electrician Truck	6,336	16	2
Supervisor Truck	9,152	33	8
Long Hole	948	8	1
Subtotal Vent Requirements		m^3/s	178
Leakage @ 5%		m^3/s	8.9
Total Vent Requirements		**m^3/s**	**187**

Source: JDS (2016)

The primary ventilation system utilizes two axial vane fans as the prime movers located on surface on the two raises which connect the extents of each mining level footwall drive. These fans are designed to pull exhaust from the mine workings to surface, drawing fresh air into the main decline and the secondary egress raise linked to the spiral ramp. The raises will be initially driven from the 940 m level (NW raise) and the 960 m level (SE raise) and will later be extended in a leapfrog manner as the mine is developed deeper. Exhaust raises are designed to be smooth raise bore without ladder or other infrastructure to minimize friction losses. The fresh air raise will also be a raise bore developed, however a ladder way will be installed which will hinder airflow. This raise is designed mainly for secondary egress and is not the prime fresh air source for the mine.

Ventilation barricades will be utilized to direct airflow to different mine levels during operation and will be constructed in such a manner to allow for quick adjustments to the ventilation network. Figure 16.13 depicts the LOM ventilation network at Romero.

Figure 16.13: Ventilation Network Schematic



The main exhaust fans located on surface will be a Howden model 8400-VAX-3150. The fans will have a 125 hp electric motor and will run at 880 rpm. Each fan will deliver a peak volume of 94 m^3/s (199,000 cfm) at the pressure of 600-700 Pa (2.4-2.8" w.g.).

The fans will be equipped with a variable pitch, adjustable at rest, such that the fans may be adjusted to optimize efficiency as the mine deepens and the pressures increase at the ventilation raise collars. One of the two main exhaust fans will be temporarily installed onto an exhaust raise 500m down the main decline during mine development. This is in order to reduce pressures and power requirements for running bagged ventilation the full 1,000 m distance to the first accessible exhaust raise (NW raise on 940 m level). Once the NW exhaust raise is driven to surface and the second main fan has been installed, the initial exhaust raise will be decommissioned; ladders installed, and will be utilized as an auxiliary fresh air intake and secondary egress.

Two additional booster fans are required in years 6 and 7. The fans selected for this duty are Alphair model 8400 AMF 5000 Arr. #4. Each fan will have 150 hp (112 kW) motor operating at 710 rpm and will deliver 68 m^3/s (144,500 cfm) of the air.

Auxiliary ventilation for ramp, production and level development will be done with 75 kW (100 hp) fans and single or twin 1.22 m (48 inch) and 1.07 m (42 inch) diameter flexible ducting.

16.8.2 Water Supply

The nearby paste plant facility will generate approximately 150 m^3/hr of water from the filtering of paste tails. A sedimentation pond will also be located at Hondo Valle village (refer to Golder, 2016b, Romero Project PFS Water Management Report) to collect surface runoff and receive mine water pumped to surface. The sedimentation pond will provide non-potable water to the underground mine, and be supplemented as needed by the excess water produced at the paste plant facility. Excess water in the sedimentation pond will be pumped, via the paste plant reclaimed water line, to the process water tank and tailings thickener facility at the process plant site.

16.8.3 Dewatering

Given the small amount of groundwater inflow expected at Romero, the largest water management source will be from equipment consumption including drilling, washing of muck piles or ramps, and shotcrete application. Table 16.16 depicts the estimated pumping requirements expected in the Romero mine during peak production.

Table 16.16 Dewatering Requirements

Equipment	Water Use (L/hr)	Peak Annual Equipment Hours	Peak Annual Consumption (Million Litres)
Jumbo 2 Boom	3,960	5,767	22.8
Bolter	4,500	10,888	49.0
Long Hole	9,000	4,928	44.3
Jackleg	3,600	2,087	7.5
Stoper	3,600	2,087	7.5
Shotcrete Mobile	1,800	5,157	9.3
Long Hole	9,000	1,460	13.1
Misc	9,000	400	3.6
Gross Water Consumption			157.2
Water use / penetration factor			70%
Actual Water Consumption			110.1
Groundwater Inflow	14,400		126.1
Main sump requirements			236.2 ML
			7.59 L/s

Source: JDS (2016)

Three main sump stations are designed for the Romero dewatering network.

The mine portal will be developed such that there is positive grade prior descending into the decline. This will prevent the portal from collecting water from the surrounding topography. To further aid this, a catchment ditch will be established around the box cut to direct surface runoff around the portal, rather than entering it.

Forty metres into the decline a small sump station will be installed to catch any water that is collected within the portal box cut, the content of which will be pumped to surface with a submersible electric 1 hp pump. It is anticipated that this sump will only become active during the rainy season.

The first main sump station is designed 400 m down the decline and consists of three sump drifts to act as primary, secondary, and tertiary settling systems installed parallel to one another spaced 10 m on centre down ramp. As the primary sump fills, a borehole drilled 3.5 m from the floor will direct water into the adjacent sump down ramp. This allows for solids to settle to the bottom of the sump prior to entering the adjacent drift. The process will be repeated in the second sump, settling and flowing into a third drift, from which water will be pumped to surface for redistribution underground, or delivery to the water treatment facility. This sump station will be run by one 15 hp submersible pump, with another 15 hp pump as backup.

A second sump station equipped with twin parallel settling drifts is located at the base of the decline on the 940 m level. This will serve as the catchment for all water pumped up the spiral ramp and will be managed by one 5 hp submersible electric pump.

Single drift sumps are located at the entrance of each level on the spiral ramp, established 10 m down ramp from the entrance to catch water from the level and prevent drainage to the level below.

Each sump will be managed by a range of 1 hp to 5 hp submersible pumps, depending on the activity on the level and production schedule.

Level development will be constructed at a positive grade of 0.5 to 1.0% to promote water drainage to the sump located at the entrance of each level.

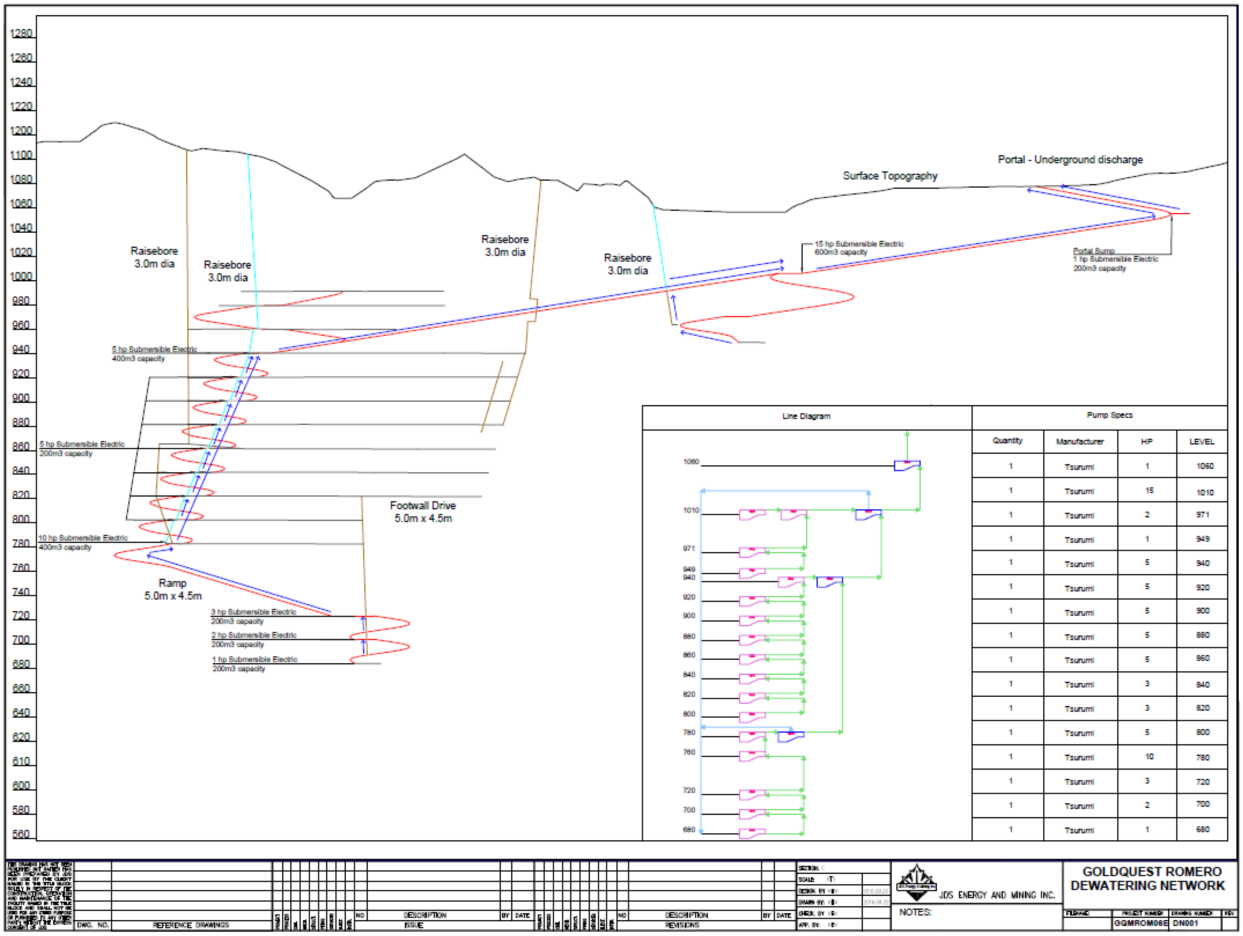
Air driven Wilden face pumps will be utilized during drilling and loading activities to keep water away from the face.

All water will be pumped to surface and deposited into a sedimentation pond located in Hondo Valle village. Details for surface water management are located in section 18 of this report.

Hydrogeological reviews of the Romero deposit (Golder 2016) have suggested that there may be a water bearing fault which becomes charged annually during the rainy season. If the mine operations become hindered by this seasonal flow it will be possible for the cable grouting equipment to be used to install grout curtains around the fault crossings and prevent excessive groundwater inflows.

Figure 16.14 below depicts the dewatering network at the Romero underground mine.

Figure 16.14: Dewatering Schematic



16.8.4 Electrical Distribution

Power will be supplied to the mine portal, underground mine and paste plant via a 3 km, 4160 V overhead line from the process facility substation. The power line will follow the haul road from the process facility and terminate at a 4160 V switchgear line-up in electrical room 3 (ER-3). The 4160 V gear will feed two (2), 2000 kVA, 4160 V to 480 V transformers for the general mine loads and paste plant loads and will also directly feed the raise bore drill, mine air compressor and the disc filter for the paste plant.

Anticipated total connected loads at the mine are listed below in Table 16.17. Power requirements at the mine will steadily increase over the first year of operations from around 650 kW of load in first quarter to 1500 kW by end of the fourth quarter.

Table 16.17 Underground Mine Electrical Power Loads

Operational Area	Connected Load (kW)	Operational Load (kW)
4000 - Underground	2,239	1,108
5000 – Paste Plant	1,336	920
Underground and Paste Plant Totals	**3,575**	**2,028**

Source: JDS (2016)

A temporary power supply may be needed to supply the necessary electrical requirements to pre-develop and prepare the portal and underground portion of the site. This power supply could be a mobile substation tapped off the incoming overhead line or utilization of the emergency power diesel generators at the mine portal and process facility. By start of third quarter year one, the permanent 4160 V switchgear and 480 V MCCs should be installed and fully operational.

For a more detailed breakout of the loads and sub-area totals, see document 16VA0027 Romero Electrical Load List (Allnorth 2016).

16.8.4.1 Underground Loading Timeline

Year One

During the first quarter of the first year, the large jumbo and the bolter will be used to develop the portal and begin the underground works proper. Waste will be stockpiled in preparation for bringing the paste plant online during quarters three and four. Also operational during this time will be the ventilation fans and sump pumps at the portal and ramp. The compressor will run when required, but not continuously until fourth quarter. The raise bore will be utilized to create ventilation shafts. Electrical power loading will climb from the quarter one value of 650 kW to 1,500 kW by end of quarter four.

Year Two

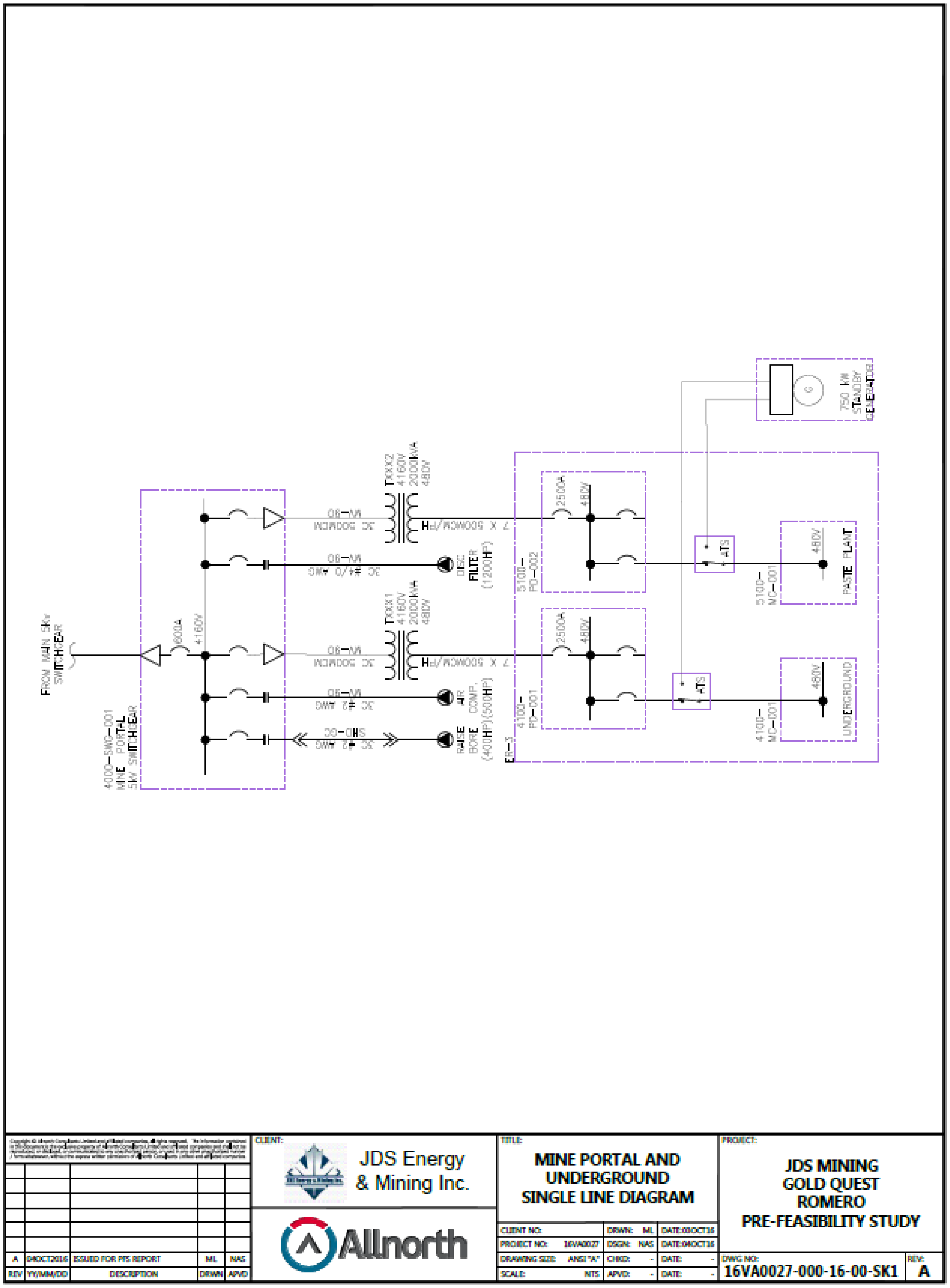
By year two, all infrastructures should be complete to support typical full load operational demands. The underground mobile loads will be fully operational, paste plant online and by fourth quarter of year two almost all underground sump pumps and the ventilation system fans will be running. The compressor will be operating continuously, drills will be fully operational. By quarter three, the raise bore is expected to be non-operational.

Years Three to Eight

Mine portal, underground and paste plant operational loads are expected to stabilize and run steady at around 2,000 kW during the mine's normal production life span, See Table 16.17 above.

Figure 16.15 below illustrates the single line diagram for the underground electrical distribution.

Figure 16.15: Underground Single Line Diagram for Electrical Distribution



16.8.5 Communications

A wireless communication system will connect the mine with surface operations. Wireless access points will be set up throughout the ramp and footwall drives to allow the use of voice, tracking, video, automation and process control applications to enhance mining safety and productivity.

16.8.6 Compressed Air

Compressed air will be used for jumbo, long hole, stoper, jackleg and sinker drilling, secondary pumping, ANFO loading, scissor lift operations, and blast hole cleaning. The underground mine will have a dedicated compressed air system, consisting of three 250 hp compressors (two operating and one backup) providing 2,000 cfm at 125 psi.

16.8.7 Explosives and Detonator Storage

Explosives will be stored underground in permanent magazines, while detonation supplies (NONEL, electrical caps, detonating cords, etc.) will be stored in a separate magazine. Underground powder and cap magazines will be on the main ramp at elevation 1016 m and 972 m, a distance of 300 m from each other. Day boxes will be used as temporary storage for daily explosive consumption.

16.8.8 Fuel Storage and Distribution

A mobile equipment fueling and lube station will be located near the portal to provide fuel for the underground mobile equipment fleet. Additionally there will be one mobile fuel and lubrication vehicle to service equipment underground.

16.8.9 Mobile Equipment Maintenance

Mobile underground equipment will be maintained in the surface maintenance located in Hondo Village, near the mine portal. A mechanics truck will be used to perform emergency repairs underground.

A maintenance supervisor will provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision to maintenance crews. The supervisor will also provide training for the maintenance workforce.

A maintenance planner will schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle. A computerized maintenance system is recommended to facilitate planning.

The equipment operators will provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required.

16.8.10 Mine Safety

Self-contained portable refuge stations will be provided in the main underground work areas. The refuge chambers are designed to be equipped with compressed air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers will be capable of being sealed to prevent the entry of gases. The portable refuge chambers will be move to the new locations as the working areas advance, eliminating the need to construct permanent refuge stations.

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fueling stations, and other strategic areas. Every vehicle will carry at least one fire extinguisher of adequate size. It is recommended that underground heavy equipment be equipped with automatic fire suppression systems.

Primary mine access will be through the main portal and decline. Secondary emergency egress will be through a fresh air raise connecting the spiral ramp outside of each level access. The raise will be equipped with ladders and platforms.

16.8.11 Contract Mining

The Romero mine assumes the use of contract services to develop the Romero underground mine, and provide the first operating year's waste development requirements. A South American mine contractor pricing was utilized, the details of which are included in the capital and operating costs, section 21 and 22, of this report.

Contract miners will provide the labour, equipment, and materials required to establish a portal and develop 6.8 km of underground ramp, access, footwalls, and infrastructure drifts. Contract mining was selected for the Romero mine plan to ensure highly trained professional miners would be available to develop the most critical mine infrastructure in a safe and timely fashion. The presence of contract miners with Romero's in-house labour force will also provide opportunities for training of the local work force and exposure to safe and efficient practices underground. Raise bore contractors will also be utilized for all raise bore development requirements.

16.8.12 Contract Supervision

Contract supervision will oversee mine operations for the first four years of operation. Mine supervision will include mine management, training officers, maintenance planners, development and production leads, and shift supervisors. Contracted supervision will be reduced over time as the local workforce is adequately trained.

16.9 Unit Operations

16.9.1 Drilling

Development, sub-level, and MCF drilling will be conducted by twin boom electric jumbo drills. Jumbos will be equipped with 16' drill steel and will advance an average 9.5 m/d per machine throughout the mine, which equates to approximately 2.2 rounds per day per machine. Typical jumbo drill patterns for Romero development are depicted in Figure 16.16 though Figure 16.18.

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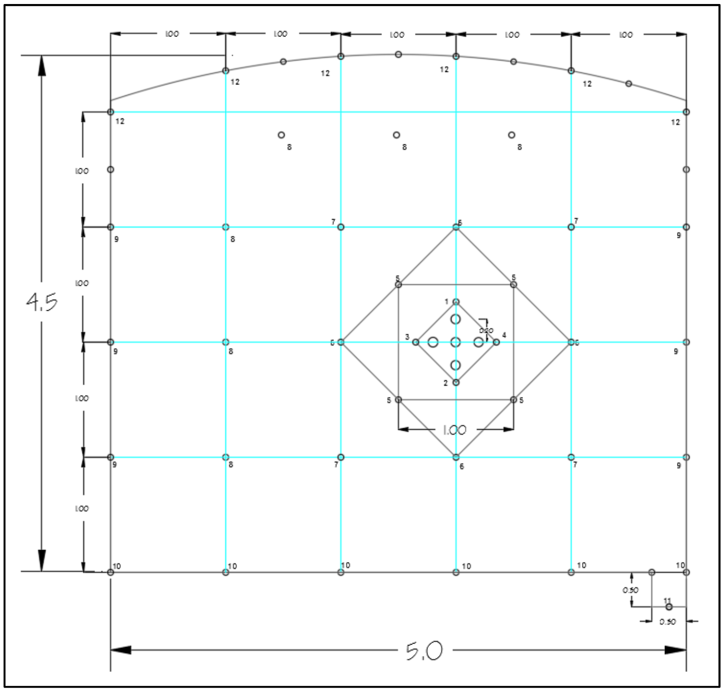
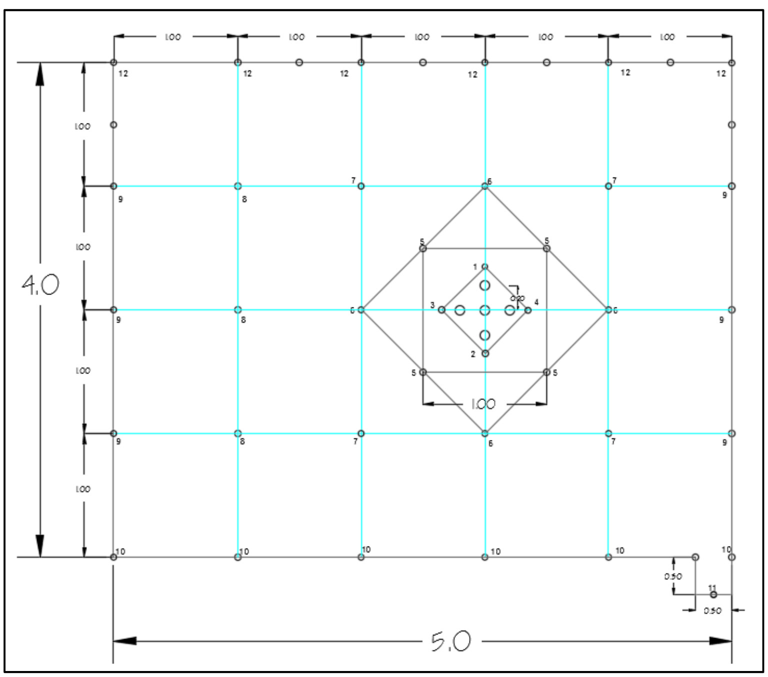
Figure 16.16: Capital Development Drill Pattern



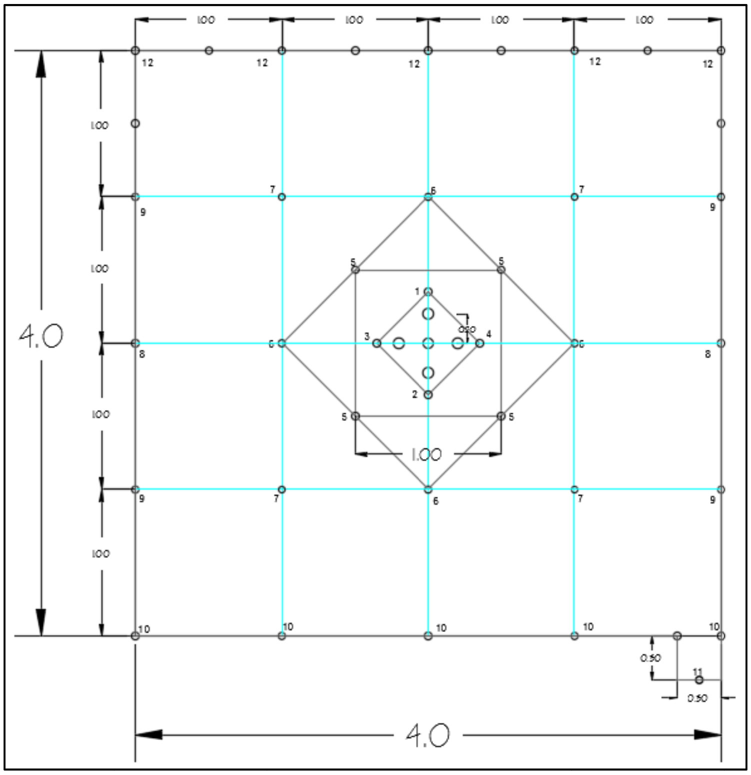
Source: JDS (2016)

Figure 16.17: Sub-level Drill Pattern



Source: JDS (2016)

Figure 16.18: Cut and Fill Drill Pattern



Source: JDS (2016)

16.9.2 Blasting

ANFO will be used as the major explosive for mine development and stoping. Packaged gelatinous dynamite (Geldyne) will be used as a primer and for loading lifter holes in the development headings and for wet LHs. Smooth blasting techniques may be used as required in main access development headings, with the use of trim powder for loading the perimeter holes.

During the pre-production period, blasting in the development headings will be done at any time during the shift when the face is loaded and ready for blast. All personnel underground will be required to be in a designated Safe Work Area during blasting. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes at the end of the shift. A mobile ANFO loader will be utilized to deliver explosives to the face.

16.9.3 Ground Support

Ground support will be conducted by a combination of scissor bolter, LH drill, and mobile shotcrete machine.

Ground support will be conducted post-mucking of the blasted drift. In poor ground conditions a flash coat of shotcrete will be applied prior to mucking the round. No additional development will be commenced in the heading prior to the installation of proper ground support.

Different ground support criteria are recommended for various types of ground conditions, rated from good to poor, and largely associated with the transition from argillic alteration to silicified alteration. Discretion will be made by the development lead as to which ground support is required, with additional review and authority provided by the on-site geotechnical engineer.

Regular pull tests will be conducted on site to ensure adequate installation of resin rebar, split set, and cables bolts are being done. Shotcrete, when required, will also be sampled by use of splatter boards and in-situ coring to be tested for strength and adequacy.

Rock bolts and screen will be installed from a scissor bolter machine. Jackleg and Stoper drills will be available and used in areas the bolter cannot access or during times of maintenance.

Cable bolts will be installed in intersections and along sub-level LH drifts. Cable bolts may be installed shortly after the development behind the development crew as to maintain the advance rate of the drift. In areas of poor ground and intersections are planned, shotcrete and cable bolts will be installed prior to development of the intersection.

16.9.4 Mucking

Mucking will be performed by LHD machines for all MCF and waste development drifts. In LH stopes an LHD equipped with remote control will be utilized in order to keep personnel away from unsupported ground. Muck will be hauled a maximum 150 m to a nearby remuck drift or directly into haul trucks. Where applicable, waste rock will be hauled directly to empty secondary stopes that are ready for non-structural backfill placement.

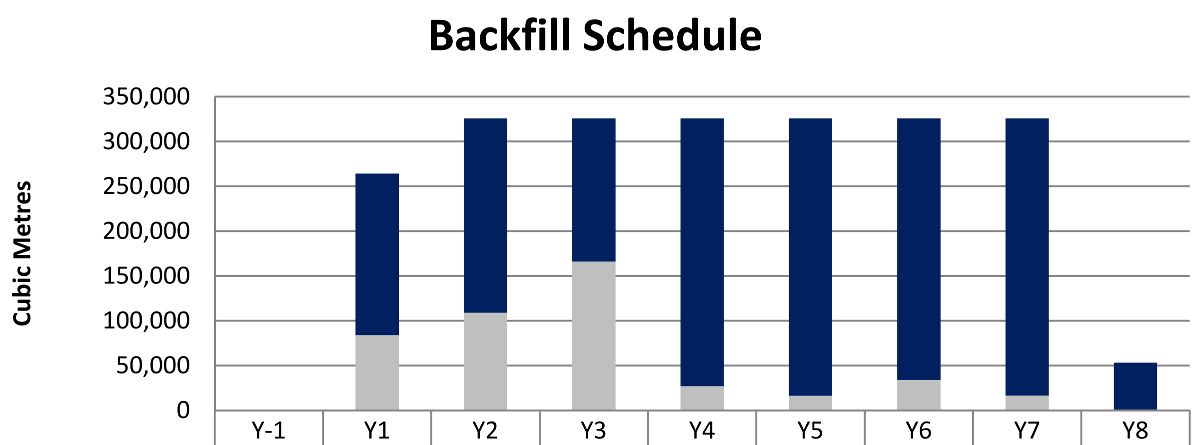
16.9.5 Hauling

Muck will be hauled to surface by 40 tonne haul trucks. Trucks will be loaded at remuck stations by LHD machines. In most cases trucks will be restricted to loading at remuck stations due to the increase back height requirements for LHDs to load over the side of the truck box. Trucks will haul muck to surface and dumped in ore and waste piles, to be rehandled by surface equipment for transport to the mill site.

16.9.6 Backfill and Paste Plant

A key driver for the Romero Project is to limit the environmental impact by placing PAG (sulphide) tailings in the underground backfill. Primary stopes will receive structural backfill in the form of cemented paste, which is comprised of de-watered sulphide tails from the processing facility. Secondary stopes will receive a non-structural backfill in the form of mine waste rock. Where there is insufficient waste rock available paste will be used in secondary stopes. Figure 16.19 below depicts the backfill requirements over the course of the mine life.

Figure 16.19: Backfill Schedule



	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Paste fill to UG	0	180,426	216,868	159,629	298,824	309,407	291,837	309,311	52,836
Rock fill to UG	0	83,840	108,847	166,085	26,890	16,308	33,877	16,403	392

Source: MineFill (2016)

The backfill plant has been located on surface near the mine portal. This location was selected to reduce the area of disturbance on surface and the need to extend tailings pipeline and power cables further than absolutely required.

According to the LOM plan, the Romero mine will operate at an approximate annual production rate of 1,008 kt/a. The plant capacity was designed based on the tailings production rate of 104 t/h. Average operating capacity of the paste fill plant will be 56.6 t/h which is roughly 60% of the design capacity. The lower capacity recognizes that not all of the tailings can be placed underground, and allows for downtime due to maintenance and cleanup. Based on mine plan quantity requirements, the backfill plant is expected to operate at average 56% utilization.

Each LH stope pour will consist of approximately 3,600 m^3 of paste for 90 paste plant operating hours. Cut and fill drifts will consist of approximately 2,500 m^3 for 60 operating hours.

The process plant will produce two types of tailings. A rougher tail that is expected to be inert and cleaner tails, which will contain pyrite and therefore will be potentially acid generating. In efforts to eliminate risks associated with acid rock drainage, it is proposed that all cleaner tails be placed underground as paste backfill. The estimated LOM paste requirement is 2.1 Mt, while cleaner tails is expected to be 1.75 Mt, suggesting that approximately 80% of the paste backfill produced will need to be sourced from cleaner tails.

Table 16.18 below outlines the recommended paste mix designs, as well as recipe for one of the mixes in Table 16.19.

Table 16.18: Recommended Paste Mix Designs

	Unit	200 kpa	700 kpa
Binder Content	%	3.0	6.0
Curing Time	days	4d	28d
Yield Stress	Pa	250 Pa	300 pa
Wt% Solids	%	71.6	72.2
Paste Density	Kg/m^3	1940	1940

Source: MineFill (2016)

Table 16.19: Paste Mix Recipe - 700kpa Mix

	Volume per $1,000m^3$	Weight per $1,000m^3$
Tailings Solids	444.2	1,341
Cement	21.29	67
Water	547.6	440
Total	**1,000**	**1,956**

Source: MineFill 2016

The paste plant system includes the following major components:

- Two 3.8 m diameter vacuum disc filters (one on standby);
- Filter Cake weigh conveyor with belt scale;
- 280 t binder storage silo with screw conveyor, capable of holding four days of cement usage at nominal operating rate;
- One twin shaft pug mill style mixer to mix tails, cement, and water; and
- Paste hopper and distribution pump.

Paste will be delivered underground by gravity. Starting from the paste plant, a 50 m long, near vertical borehole will house a paste delivery line down to intersect the main ramp at about 1,050 m EL. A single paste line will then extend down the main ramp. From the base of the decline the paste will travel down a second blind borehole to a series of stations at pre-determined levels within the mine. At each station a removable pipe spool will allow diversion of the paste to the station level, or direct the paste to a station on a lower level. Once the paste has reached the level being filled, the paste line will revert to Schedule 40 or Schedule 80 mild steel piping. The steel piping will be used to deliver paste from the hangingwall side of each deposit. The final segment of paste piping will be HDPE and will transfer paste from the level piping, through a cross-cut, to the stope being filled.

The proposed distribution system for Romero consists of two main classes of pipe:

- Nominal 5-inch schedule 80 carbon steel piping – ASTM grade B with Victaulic Style 77 grooved fittings and ANSI class 150 flanges; and
- Nominal 6-inch SDR9 HDPE piping with butt fused ends and ANSI class 150 flanges.

The design properties for this piping are shown in Table 16.20 below.

Table 16.20: Backfill Pipe Distribution Design Properties

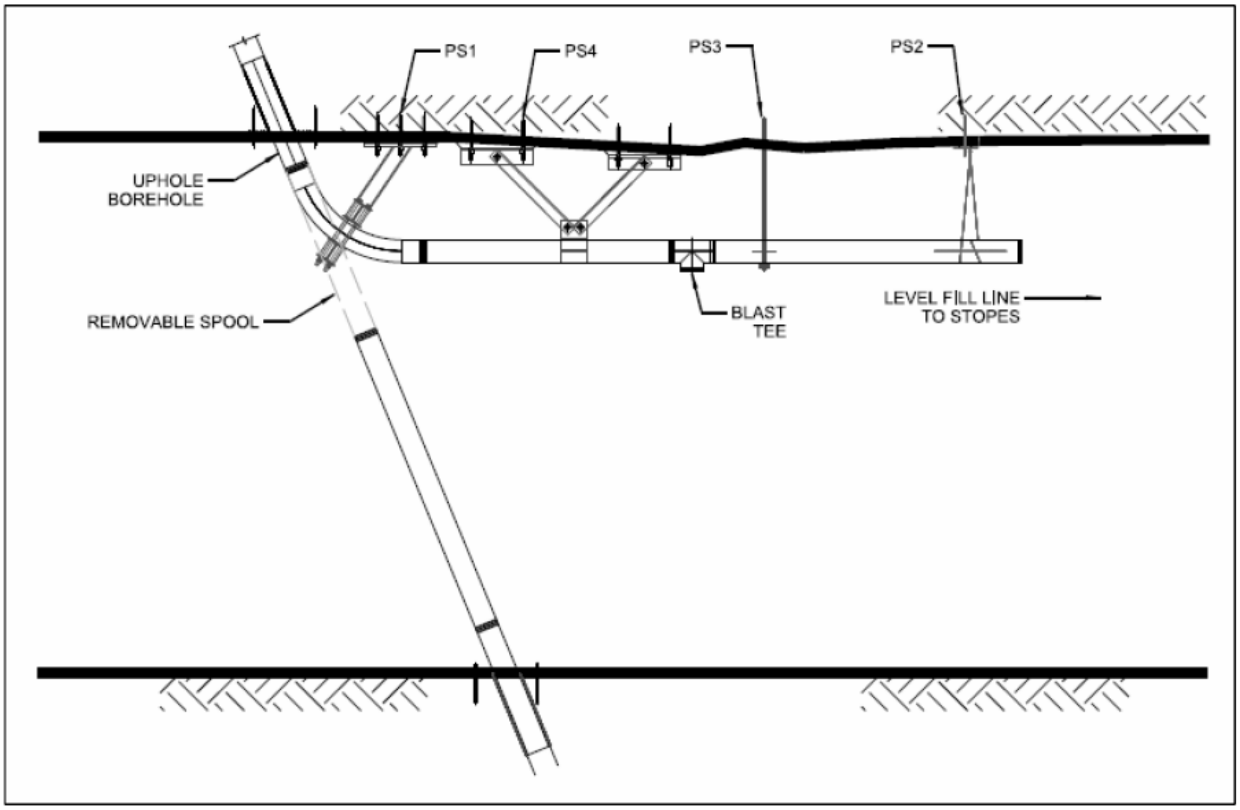
	Unit	CS Steel Schedule 80	HDPE SDR 9
Inside Diameter	mm	122.3	122.2
Design Pressure Rating	kPa	14,272	1,723
Target Flowrate	m^3/h	40.5	40.5
Paste Velocity	m/s	0.96	0.96

Source: MineFill (2016)

The underground reticulation network consists of the following main elements:

- An initial 50 m long borehole from the paste plant at elevation (El.) 1,094 m down to intersect the main ramp at El. 1,050m. This hole will be near vertical and will be fitted with ceramic lined single or dual paste lines of Schedule 80 carbon steel pipe;
- A 750 m long run down the main access ramp from the 1,050 mL to the 940 m Level. This will consist of a single paste line of Schedule 80 carbon steel pipe hung from the back or pinned to the sidewall of the ramp with other mine services;
- At the 940 mL the pipe diverges with one line continuing up a blind bore to the 980 mL which is the top mining level in the mine. This line will also consist of ceramic lined Schedule 80 carbon steel pipe. The other line will continue down a blind bore in the hangingwall of the spiral ramp down to the 680 mL. This line will have removable spools at every second mining level to allow paste to be diverted onto a given mining level (see Figure 16.20); and
- The last leg of piping will consist of up to 300m of level piping to deliver paste to stopes. The last 100m or so of this piping will consist of HDPE piping as specified above. The remainder of the level piping will be either Schedule 40 or Schedule 80 carbon steel pipe.

Figure 16.20: Paste Borehole Transfer Station to Re-direct Paste.



Source: MineFill (2016)

Where applicable, mine waste rock will be deposited underground into secondary stopes not requiring structural backfill. Waste will be placed into stopes by LHD machines either directly from blasted development drifts, remuck drifts, or specified dumping areas. Waste will be delivered by 40- tonne trucks between mining levels where necessary, as well as from a temporary waste stockpile on surface. Ejector beds will be equipped in the trucks to allow dumping into remucks or along the footwall drive near the stope to be filled.

It should be noted that backfilling tight to the back is important for structural integrity, and is often difficult without specified rammer jammer equipment. As such it is recommended that where possible waste rock is used to fill the majority of a stope, and that non-structural pastefill is deposited afterwards to ensure tight fill to the back.

16.10 Mine Equipment

The selection of underground mining equipment is based on mine plan requirements, mining methods, operating drift and stope dimensions. No work was undertaken in this PFS to evaluate alternates or new technology. It is assumed that all mobile equipment will be new to avoid issues with development and production schedules for unplanned maintenance associated with used equipment.

Two boom and diesel/electric jumbos will be used for lateral development and MCF stoping, while production drilling will be completed by diesel/electric LH drills capable of drilling 101.6 mm (4”) diameter production holes and 63. 5mm (2.5”) diameter cable bolt holes. Mucking will be carried out with 7 m^3 LHDs with remote operating capabilities (used for development and stope mucking). Waste and ore will be hauled in 40 t trucks.

The underground equipment fleet is summarized in Table 16.21. Equipment is split between contractor and GoldQuest owned fleets.

Table 16.21: Mobile Equipment Fleet

Equipment Description	Year -1		Year 1		Year 3	
	Contractor	GoldQuest	Contractor	GoldQuest	Contractor	GoldQuest
Truck (40t/19.0m^3)	1	-	1	4	-	5
LHD (14t/6.4m^3)	1	-	1	3	-	4
Jumbo 2 Boom	1	-	1	1	-	2
Bolter	1	-	1	2	-	3
ANFO Loader	1	-	1	1	-	1
LH Large	-	-	1	1	-	2
Jackleg	-	-	1	1	-	1
Stoper	-	-	1	1	-	1
Scissor Lift	1	-	1	1	-	1
Shotcrete Mobile	1	-	1	1	-	1
Personnel Carrier	1	-	1	1	-	1
Fuel/Lube	1	-	1	1	-	2
Boom Truck	1	-	1	1	-	2
Grader	1	-	1	-	-	1
Tractor	1	-	1	1	-	2
Backhoe	1	-	2	-	-	2
Telehandler	1	-	1	1	-	2
Mechanics Truck	1	-	1	1	-	4
Electrician Truck	1	-	1	1	-	2
Supervisor Truck	2	-	2	2	-	8
LH Small	1	-	1	-	-	1

Source: JDS (2016)

16.11 Mine Personnel

The mine will operate on two 12-hour shifts, 365 days per year with four mining and maintenance crews. Two crews will be on site at any one time, one on dayshift and one on nightshift, with the other crew off-site on break. The majority of the mining and maintenance personnel will work a four-week-on, four-week-off (4x4) rotation, while technical staff and management will work on a five-day-on, two-day-off (5x2) schedule.

The underground mine personnel requirement peaks at 205 personnel during full production, with 112 on site at one time. Mining personnel requirements are summarized in Table 16.22 through Table 16.27.

Table 16.22: Mine Management Personnel Summary

Mining Management	Category	Rotation	Qty
Mine Superintendent	Salary	5x2	1
Maintenance Manager	Salary	5x2	1
Technical Services Manager	Salary	5x2	1
Mine Foreman	Salary	5x2	1
Mine Clerk	Salary	5x2	1

Source: JDS (2016)

Table 16.23: Mine Operations Personnel Summary

Mining Operations (Production)	Category	Rotation	Qty
Shift Supervisor	Hourly	4x4	4
Blasting Supervisor	Hourly	4x4	4
Blaster	Hourly	4x4	8
Blasting Helper	Hourly	4x4	8
Development Services/Shotcrete	Hourly	4x4	8
Development Miner / Jumbo Operator	Hourly	4x4	4
Production Miner / Jumbo Operator	Hourly	4x4	4
Long Hole Drill Operator	Hourly	4x4	8
LHD Operator	Hourly	4x4	16
Haul Truck Operator	Hourly	4x4	20
Bolter Operator	Hourly	4x4	12
Grader Operator	Hourly	4x4	4
Nipper/Equipment Operator	Hourly	4x4	16

Source: JDS (2016)

Table 16.24: Contractor Services Personnel Summary

Contractor Services	Category	Rotation	Qty
Expat - Mine Manager	Hourly	4x4	1
Expat - Shift Supervisor	Hourly	4x4	4
Expat - Development Lead	Hourly	4x4	4
Expat - Production Lead	Hourly	4x4	4
Expat - Maintenance Lead	Hourly	4x4	4
Expat - Trainer	Hourly	4x4	4

Source: JDS (2016)

Table 16.25: Mine Services Personnel Summary

Mining Operations (Services)	Category	Rotation	Qty
Paste Plant Operators	Hourly	4x4	8
Backfill - Pipe	Hourly	4x4	12
Backfill - Barricade	Hourly	4x4	8
Mine Electrician	Hourly	4x4	4

Source: JDS (2016)

Table 16.26: Mine Maintenance Personnel Summary

Mine Maintenance	Category	Rotation	Qty
Maintenance Supervisor	Salary	5x2	4
Heavy Equipment Mechanic	Hourly	4x4	8
Mechanic Helper	Hourly	4x4	4
Welder	Hourly	4x4	4
Electric/Hydraulic Mechanic	Hourly	4x4	4

Source: JDS (2016)

Table 16.27 Technical Services Personnel Summary

Mining Technical Services	Category	Rotation	Qty
Senior Mine Engineer	Salary	5x2	1
Geotechnical Engineer	Salary	5x2	1
Chief Geologist	Salary	5x2	1
Ventilation Engineer	Salary	5x2	1
Mine Surveyor	Salary	5x2	2
Surveyor Helper	Salary	5x2	3
Geologist	Salary	5x2	2
Sampler	Salary	5x2	2
Short Term Mine Planner	Salary	5x2	1
Project Engineer	Salary	5x2	1
Long Term Mine Planner	Salary	5x2	1
Draftsman	Salary	5x2	1

Source: JDS (2016)

16.12 Mine Production Schedule

The following factors were considered in the estimation of the underground mine production rate:

- Mining inventory tonnage and grade;
- Geometry of the mineralized zones;
- Amount of required development;
- Stope productivities; and
- Sequence of mining and stope availability.

The underground mine production rate of 2,800 t/d is considered appropriate due to the high degree of mechanization and potential high productivities of the selected stoping methods. Based on the presence of several mineralized zones and ability to have production from different sub-levels, JDS considers the underground production rate to be achievable.

The underground mine life is estimated at eight years in addition to the nine months of pre-production.

16.12.1 Mine Development

Mine development is divided into two periods: pre-production development (prior to commercial production) and ongoing development (during commercial production). The objective of pre-production development is to provide an access to higher grade areas and prepare enough resources to support the mine production rate when access to the lower levels is being established.

Pre-production development is scheduled to:

- Develop ore stopes prior to production;
- Provide access for trackless equipment;
- Provide ventilation and emergency egress; and
- Install mining services.

A development crews will start working on surface to excavate a portal for underground access. Following this a development crew will work to drive a decline towards the Romero deposit and install cross-cuts for ventilation raises, sumps, remuck stations, magazines, shops, electrical cut outs, and lay downs. Vertical raise development will be done with contract mining crews as ventilation drifts become available.

During pre-production, the mining crews will:

- Excavate a portal box cut accommodate a 5.0 m W x 4.5 m H size entrance;
- Develop a 2,775 of ramp to access all mining levels within the mine plan;
- Develop 1,930 m of access and footwall drives to gain access to production zones;

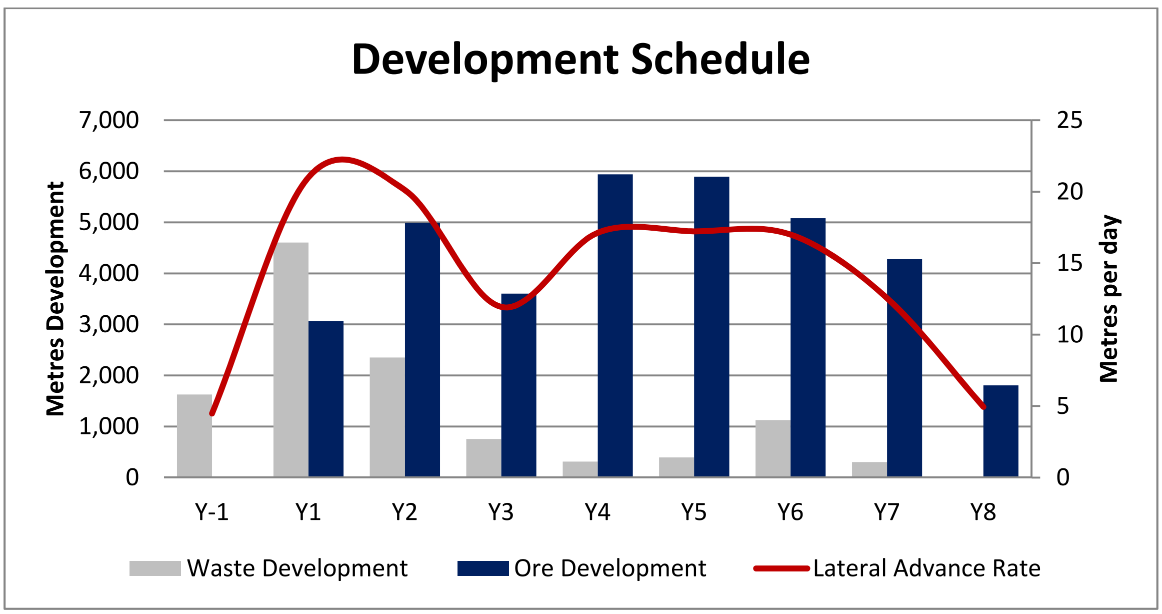
- Develop 1,100 m of service drifts for remucks, sumps, electrical bays, refuge stations, lay downs, magazines, and a shop;
- Develop 970 m of waste cross-cuts to intersect the ore body;
- Develop 720 m of raise bore ventilation raises;
- Develop 500 m of conventional raise for fresh air connections between mining levels; and
- Install ladders for secondary egress.

The development schedule was planned based on estimated cycle times for jumbo and raise development, and benchmarked against best practices of North American mining operations and contractors. The underground mine will be nearly fully accessible by ramp at Year 2 of mine production.

Total underground capital and sustaining lateral waste development is 11,480 m and averages 1,435 m/a or 3.9 m/d over the 8-year project life. Annual waste development is shown in Figure 16.21.

Total ore sub-level development is 33,607 m and averages 4,330 m/a or 12.0 m/d over the 8-year ore production period. Annual ore development is shown in Figure 16.22.

Figure 16.21: Annual Development



Source: JDS (2016)

16.12.2 Mine Production

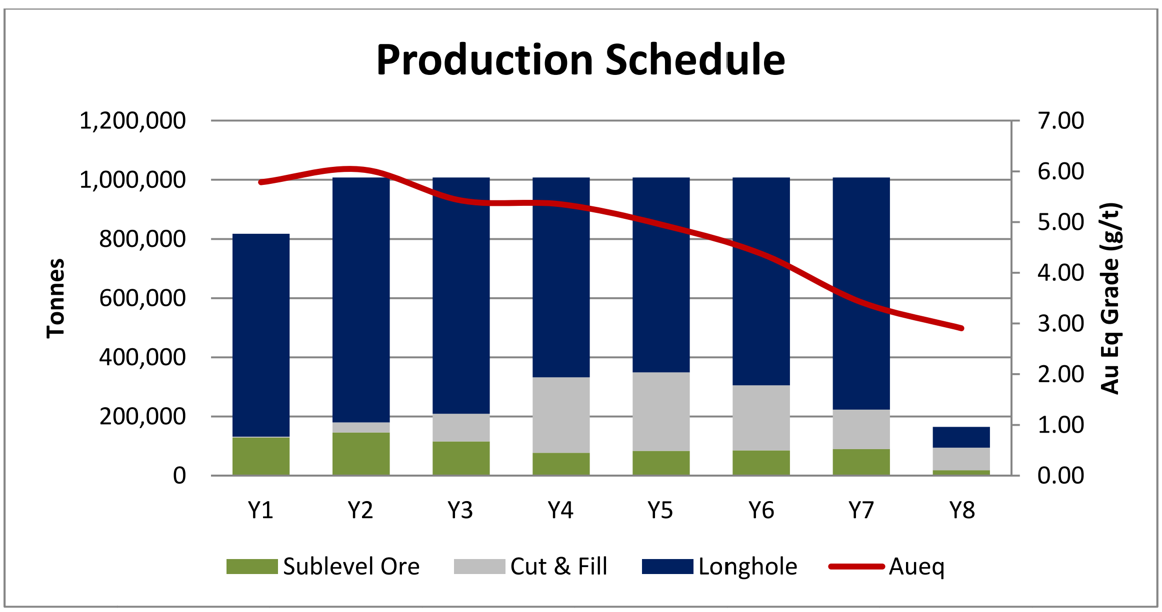
The criteria used for scheduling underground mine production at the Romero mine were as follows:

- Target the mining blocks with higher grade rock in the early stages of mine life to improve project economics;
- An average annual mill feed production rate of 1,008 kt/a was scheduled, including ore from development and stopes;
- The mine will operate two 12-hour shifts per day, 365 days per year;
- Provide enough production faces to support a daily mine production rate of 2,800 t/d; and
- Minimize mobile equipment requirements by smoothing ore and waste drifting.

The stope cycle times and productivities were estimated from the first principles. It will require four production stopes working at any time to meet daily production requirements of 2,800 t/d.

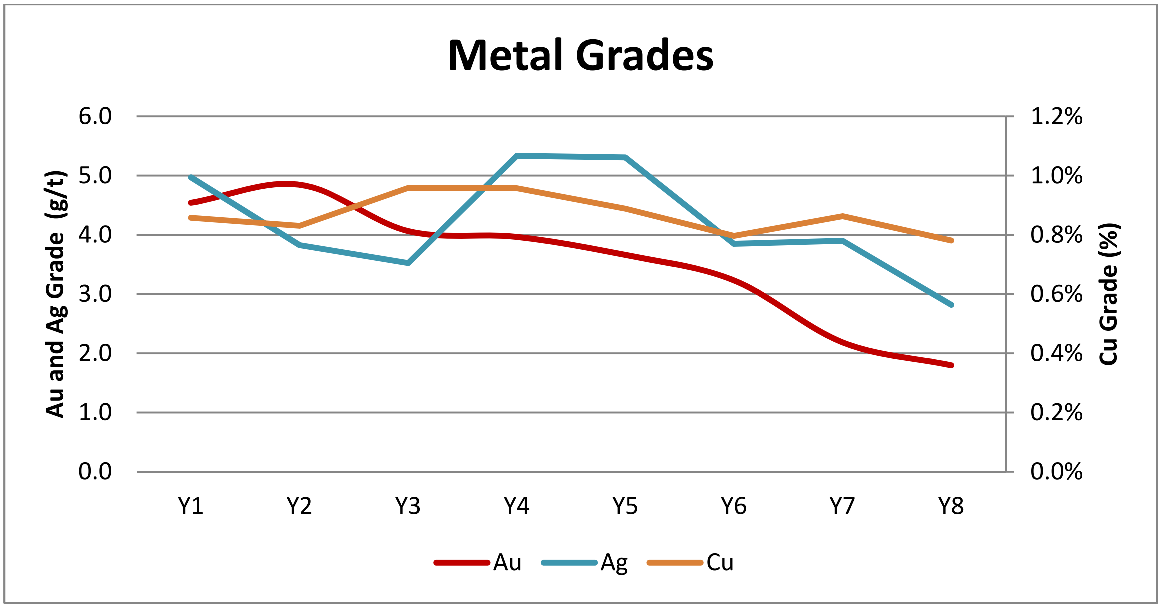
The average mined grades for the eleven-year mine life are 3.72 g/t gold, 4.33 g/t silver, and 0.88% copper. Annual production by ore source and metal grades are shown in Figure 16.22 and Figure 16.23.

Figure 16.22: Annual Ore Production by Source



Gold equivalent metal prices: Cu $2.50/lb Au $1,250/oz Ag $17.00/oz
Source: JDS (2016)

Figure 16.23 Annual Gold, Silver, and Copper Grades



Source: JDS (2016)

Detailed mine planning and scheduling has been done quarterly throughout the mine life but has been summarized annually in this report. The annual mine production schedule is provided in Table 16.28 and shows annual summaries of ore tonnage mined by deposit, ore grades and development quantities. Ore, waste and backfill tonnages have been rounded to the nearest thousand.

Table 16.28: Annual Production Schedule

Mine Production	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Mined Waste	kt	940	101	324	271	81	25	34	70	34	0
Mined Ore	kt	7,031	-	818	1,008	1,008	1,008	1,008	1,008	1,008	165
Gold Grade	g/t	3.72	-	4.54	4.85	4.06	3.96	3.66	3.23	2.18	1.80
Silver Grade	g/t	4.33	-	4.97	3.83	3.52	5.33	5.31	3.85	3.90	2.82
Copper Grade	%	0.88	-	0.86	0.83	0.96	0.96	0.89	0.80	0.86	0.78
Zinc Grade	%	0.26	-	0.18	0.36	0.36	0.31	0.20	0.24	0.19	0.12
NSR Value	$/t	121	-	140	146	132	130	120	106	84	72
Gold Equivalent	g/t	4.88	-	5.78	6.04	5.43	5.35	4.95	4.37	3.42	2.91
	koz	1,126	-	152	196	176	173	160	142	111	15

Gold equivalent metal prices: Cu $2.50/lb Au $1,250/oz Ag $17.00/oz
Source: JDS (2016)

Table 16.29: Annual Mine Production by Mine Method

Mine Production	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Sub-level Drifting	kt	744	-	129	146	116	77	84	86	90	18
Mechanized Cut and Fill	kt	1,080	-	3	34	94	255	265	219	134	76
LH Stoping	kt	5,206	-	686	828	799	676	659	703	785	70

Source: JDS (2016)

Table 16.30: Annual Mine Development Metres

Mine Development	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Ore Development	km	33.6	-	2.2	2.9	3.6	6.6	6.7	5.8	4.1	1.8
Waste Development	km	15.3	1.6	5.1	4.4	1.4	0.4	0.6	1.2	0.6	0.0
Total Metres Developed	km	48.9	1.6	7.3	7.3	5.0	6.9	7.3	7.0	4.7	1.8
Lateral Advance Rate	m/day	14.9	4.5	20.0	20.0	13.6	18.9	20.0	19.1	12.9	5.0
Raise Development	km	1.2	0.1	1.0	0.1	-	-	-	-	-	-

Source: JDS (2016)

Table 16.31: Annual Backfill Placement

Mine Backfill	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Paste Backfill	km^3	1,819	-	180	217	160	299	309	292	309	53
Waste Rock Backfill	km^3	453	-	84	109	166	27	16	34	16	1

Source: JDS (2016)

17 Process Description/Recovery Methods

The process design criteria and flowsheets have been developed based on the results from current and historical metallurgical test work programs summarized in Section 13 and industrial design factors. The flowsheet consists of primary crushing, SAG and ball mill grinding, gravity separation, flotation, dewatering and filtration. Tailings will either be dry stacked or combined with cement and pumped underground as paste backfill.

The mill is designed with a nominal capacity of 2,800 t/d at a planned average feed grade of 0.88% copper (Cu), 3.72 g/t gold (Au) and 4.33 g/t silver (Ag). The overall LOM recoveries based on test work are expected to be approximately 94.9% for copper, 78.2% for gold and 58.6% for silver. The grinding circuit product size is targeted at 80% passing (P_{80}) 75 µm and the rougher flotation concentrate will undergo further grinding to a P_{80} of 23 µm before the cleaning stage. The crushing circuit will operate at a utilization of 33%, while the process plant will operate 24-hours per day, 365 days per year at an availability of 92%.

17.1 Introduction

The plant will consist of the following unit operations:

- Primary Crushing – An apron feeder and jaw crusher in open circuit, producing a final product P_{80} of 120 mm;
- Crushed Ore Stockpile and Reclaim – A 2,500 t storage, crushed material stockpile with two reclaim belt feeders feeding the SAG mill feed conveyor;
- Grinding – A SAG mill in closed circuit with a pebble crusher followed by a ball mill and gravity circuit operating in closed circuit with hydrocyclones, producing a final product P_{80} of 75 µm;
- Flotation – Rougher flotation, regrind to a P_{80} of 23 µm and cleaner flotation;
- Concentrate Dewatering - Thickening, filtration and load-out;
- Rougher Tailings Dewatering – Thickening, filtration and dry stack or paste backfill; and
- First Cleaner Tailings Dewatering – Filtration and storage for paste backfill.

17.2 Process Design

17.2.1 Process Design Criteria

The process design criteria for the Romero Project is based on metallurgical test work programs undertaken by ALS Metallurgy in Kamloops, BC. The design criteria and mass balance are based on the test results outlined in Section 13 and the average ore head grades and tonnage from the mine plan. The results are summarized in Table 17.1.

Table 17.1: Process Design Criteria

Description	Units	Value	Source
Operating Data			
Daily ore throughput	t/d	2,800	Mine production schedule
Annual ore throughput	t/a	1,022,000	Mine production schedule
Ore Characteristics			
Ore Solids Density	SG	2.8	Average SG values from CWI and SMC results (KM4923)
JK Drop-Weight Parameters - A		66.4	Average from 8 SMC tests (KM3650, KM4923)
b		0.71	Average from 8 SMC tests (KM3650, KM4923)
ta		0.43	Average from 8 SMC tests (KM3650, KM4923)
Bond ball mill work index, Wi	kWh/t	15	Average from eight Bond tests (KM3650, KM4923)
Bond abrasion index, Ai	g	0.195	Average from eight abrasion tests (KM3650, KM4923)
Head Grade (Average LOM)	%Cu	0.88	Average LOM grade from the mine plan
	%Au	3.72	Average LOM grade from the mine plan
	%Ag	4.33	Average LOM grade from the mine plan
Production Rates			
Overall Crusher Availability	%	33	Designed to operate 8 h/d
Overall Plant Availability	%	92	Industrial design factor for SAG/Ball mill circuits
Final Copper Concentrate			
Concentrate mass pull	%	6.4	Mass balance calculations based on head grade and recovery projections
Concentrate production, daily	dry tpd	179 (Nominal)	Mass balance calculations based on head grade and recovery projections
		215 (Design)	
Concentrate grade	% Cu	13	Results from an economic analysis
Recovery	% Cu	94.6	Recovery projections (KM5085)
	% Au	78.1	Recovery projections (KM5085)
	% Ag	58.6	Recovery projections (KM5085)
Tailings			
Methodology		Dry Stack or Paste Backfill	Design selection

Source: JDS (2016)

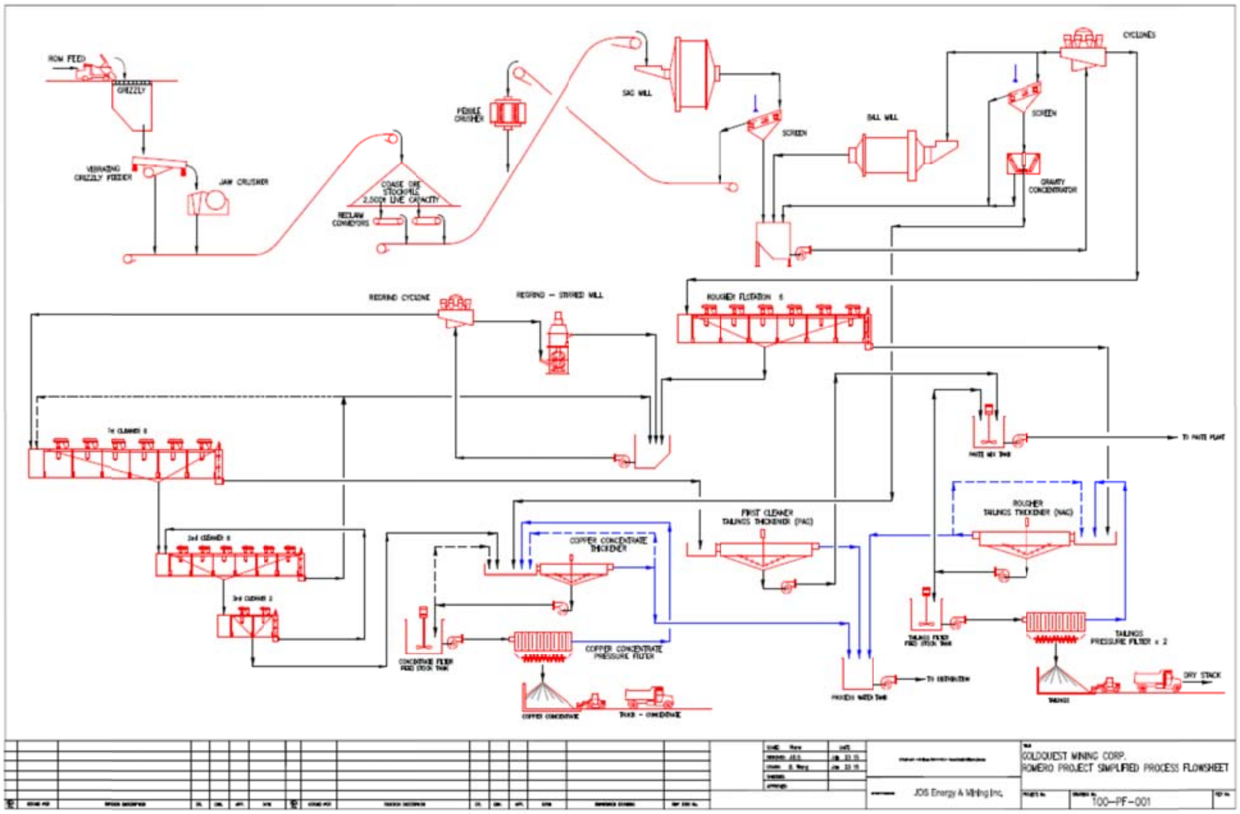
Further metallurgical test work will be completed in the next stage of engineering to provide confirmatory and/or additional information as discussed in Section 13.

The metallurgical plant is designed to process 127 dry tonnes per hour with a plant availability of 92%. Annual throughput is targeted at 1,022,000 dry tonnes.

17.3 Plant Design

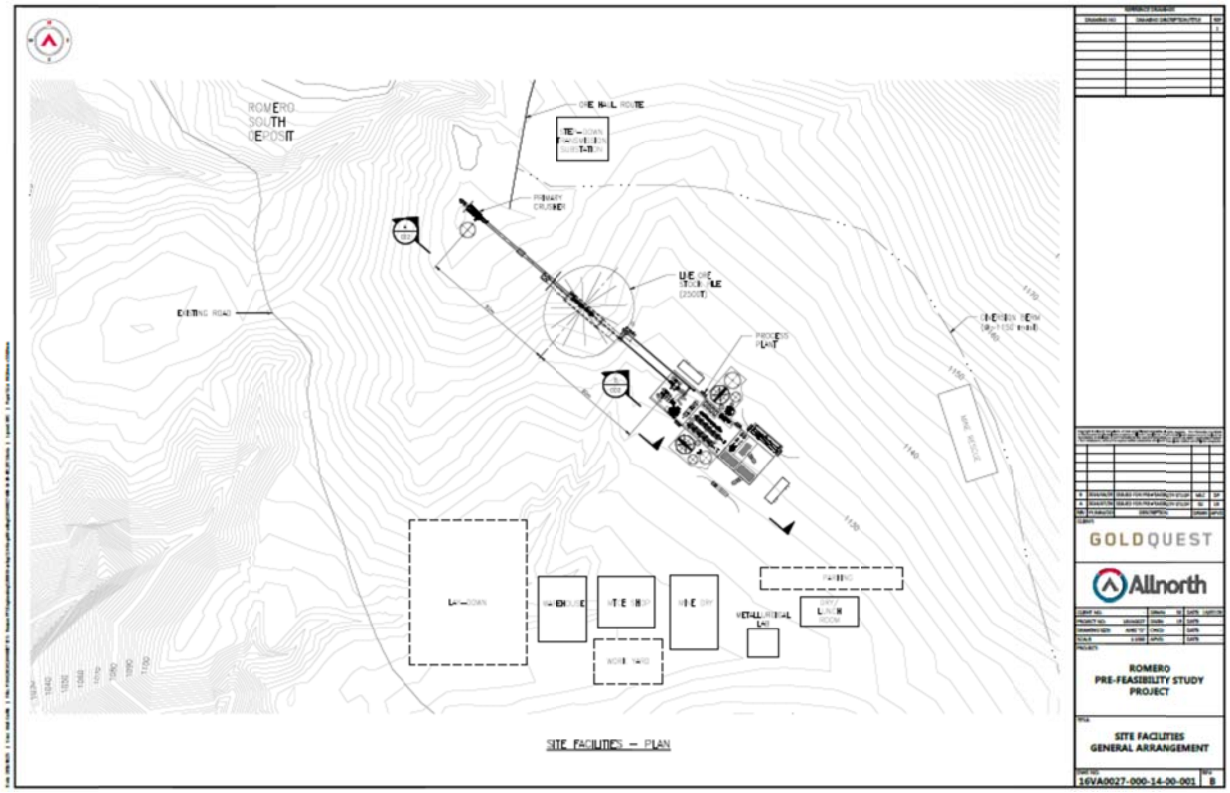
A summary of the process flowsheet and plant layout are shown in Figure 17.1 and Figure 17.2, respectively.

Figure 17.1: Plant Summary Process Flow Diagram



Source: JDS (2016)

Figure 17.2: Process Plant Layout



Source: Allnorth (2016)

17.4 Process Plant Description

17.4.1 Primary Crushing, Ore Storage and Reclaim

The crushing circuit consists of a mobile crushing unit that includes a truck dump pocket, vibrating grizzly feeder, jaw crusher, and discharge conveyor. A vibrating grizzly feeder will draw ore out of the dump pocket and provide a constant feed of material to the 1,000 mm x 760 mm jaw crusher installed with a 110 kW motor. Crushed ore will discharge from the stockpile feed conveyor to the 2,800 t crushed ore stockpile.

Two belt feeders will reclaim ore from the crushed ore stockpile and feed the SAG mill feed conveyor. Each 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 feeders and the resulting tonnage to the mill.

17.4.2 Grinding

Reclaimed ore will feed a 2.44 m diameter by 5.5 m long SAG mill driven by a 932 kW variable speed motor. This configuration will enable the SAG mill to vary power draw for circuit optimization under varying feed 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 pump box; while the screen oversize will be conveyed to the pebble crushing circuit. The critical sized pebbles will be crushed in a 90 kW cone crusher and recycled to the SAG mill feed conveyor via pebble recycle conveyors.

SAG discharge screen undersize material will be combined in the cyclone feed pump box with the ball mill discharge and gravity circuit tailings. The slurry will be pumped to a cluster of four 20” hydrocyclones for size classification. The underflow from the cyclones will be fed to a 4.3 m diameter by 7.0 m long ball mill installed with a 1,680 kW fixed speed induction motor, while the overflow will be piped to the copper rougher flotation circuit. The target P80 particle size of the cyclone overflow will be 75 µm.

A portion of the cyclone underflow will be diverted to a gravity circuit where a batch centrifugal gravity concentrator will recover any free gold. The gravity concentrate will report directly to the copper concentrate thickener, while the tailings will flow back to the cyclone feed pump box.

Process water will be added directly to the SAG mill feed chute to maintain a target slurry density of 72% 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.

17.4.3 Copper Processing

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

17.4.3.1 Rougher Flotation

Slurry from the cyclone overflow will gravitate to the rougher flotation circuit which consists of one bank of six 30 m3 cells. The cells will use a combination of reagents (PAX, MIBC and Lime), agitation, and air to recover the copper sulphides, and associated gold and silver, for further processing.

Rougher concentrate froth, approximately 30% of the feed, will be collected in a common launder which feeds a pump box. Slurry collected in the pump box will be pumped to the regrind circuit for further mineral liberation. Copper rougher tailings will be pumped to the tailings thickener.

The samples from the rougher feed, rougher concentrate, cleaner concentrate, and rougher tailings will be collected for metallurgical analysis.

17.4.3.2 Regrind Circuit

Rougher concentrate will be pumped to the regrind cyclone feed pump box. Cyclone underflow will feed a 699 kW vertical stirred mill. The mill product and the cyclone overflow, at a target P_{80} of 23 µm, will report to the first cleaner flotation circuit.

17.4.3.3 Cleaner Flotation

The cleaner flotation circuit will consist of six 20 m^3 first cleaner cells, six 5 m^3 second cleaner cells and two 5 m^3 third cleaner cells. Slurry from the regrind circuit will feed the first cleaner cells. The first cleaner concentrate will be collected in a common launder and flow by gravity to the first cleaner concentrate sump. This concentrate is pumped to the second cleaner cells and the resulting second cleaner concentrate reports toe the third cleaner cells. The third cleaner concentrate, or final 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 tailings, which will be directed to a thickener that feeds the paste mix tank.

17.4.3.4 Concentrate Dewatering and Storage

The concentrate dewatering circuits will remove water from the concentrate slurry, resulting in a damp filter cake for shipment. Test work carried out by Outotec in 2016 was used to confirm the equipment sizing.

The thickening operation concentrates suspended solids by gravity settling. Flocculant will be added as a dilute solution to the thickener, agglomerating fine solid particles and assisting with fine particle settling. Settled solids will be raked to the centre discharge cone, where the thickened slurry will be withdrawn using one of two centrifugal pumps for transfer to the concentrate stock tank. The thickener overflow will be pumped to the cleaner flotation circuit for process dilution water and launder spray water.

The concentrate stock tank will provide eight hours of surge capacity between the 6 m diameter concentrate thickener and concentrate pressure filter. The concentrate stock tank will be agitated to prevent sanding out of solids. Centrifugal slurry pumps feed thickened slurry from the concentrate stock tank to the concentrate filter.

A horizontal pressure filter is used for final concentrate dewatering to achieve a moisture content of approximately 8%. 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 then 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 thickener.

Copper concentrate will be transferred by front-end loader to trucks for transport to the port. The concentrate will be stored in a concentrate storage building at the port before being shipped to markets in Europe and Asia.

17.4.4 Tailings

Final tailings will be collected in a 12 m diameter thickener. Flocculant will be added to assist the settling of fine particles. Settled solids will be withdrawn using one of two hose pumps for transfer to the pressure filters. The resulting filter cake will then be dry stacked at the tailings facility. The thickener overflow reports to the process water tank and will be pumped to the plant as make-up or spray water in the grinding and rougher flotation circuits.

17.4.5 Paste Mixing

Thickened first cleaner tailings and a portion of thickened rougher tailings will be mixed in the agitated 10 m diameter by 12 m high paste mix tank, providing approximately 18 hours of storage. When the paste plant is not operational, the mix tank will provide approximately 30 hours of storage for first cleaner tailings. A total of 40% of the original feed to the plant will report to the paste plant.

17.4.6 Reagents Handling

Reagents consumed within the flotation circuits are prepared and distributed by the reagent handling circuits. This facility includes 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 metres and manual control valves. The storage tanks have been sized for a minimum of one day capacity. The reagents will be delivered in powder form, with the exception of MIBC and antiscalant, which will be delivered as solution.

17.4.6.1 Collector; PAX

PAX is used as a flotation reagent in the copper circuit. It promotes the flotation of selected sulphide particles contained within the ore. It will be delivered to the plant in the form of 900 kg bags of dry solid product. The bags will be lifted into a hopper using the flotation aisle crane. The solids will discharge into an agitated mixing tank, which will blend the solids with fresh water to a solution of 20% by weight of the dissolved product. From the mixing tank, the solution will be discharged by gravity into 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 flotation circuits.

17.4.6.2 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 850 kg totes and metred directly to the flotation circuits.

17.4.6.3 Flocculant

Flocculant will be received in 755 kg bags and prepared by a vendor supplied mixing system. Bags of solid product will be loaded into a hopper from which the particles will slowly be fed into the system via an educator, generating a concentration of 0.25% in the flocculant mix tank. From the mix tank, the flocculant will be transferred by gravity to a storage tank. In-line mixers will further dilute the flocculant to a concentration of 0.05% before delivery to the copper concentrate and final tailings thickeners.

17.4.6.4 Lime

Lime will be delivered in 1,000 kg sacks and mixed to a concentration of 20% solids for delivery to the flotation circuit as pH control.

17.4.6.5 Antiscalant

Antiscalant will be shipped to the plant in 50 kg drums. The antiscalant will be added at a rate of 4 g/t.

17.4.7 Plant Air Compressors

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

There are three compressors located in 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.

17.4.8 Flotation Air

Four blowers provide air at two different pressures to the flotation circuits. The higher pressure blowers will service the larger cells and the lower pressure blowers will service the smaller cells.

17.4.9 Assay Laboratory

The assay laboratory will consist of a sample preparation/metallurgical module and a wet laboratory module. 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 Overview

The Romero Project infrastructure and services are designed for the local conditions and rugged topography. They have been sized to support the operation of a 2,800 t/d underground mine and processing plant, operating on a 24-hour per day, 7-day per week basis.

The main infrastructure for the project consists of the following facilities:

- A 23.5 km access road between the existing municipal road network at Sabaneta Dam and leading to the site;
- A 2.8 km haul road connecting the underground workings with the processing facilities;
- Gold and copper processing plant with security, administration, and personnel facilities;
- Dry Stack Tailings Storage Facilities (DSTSFs);
- Paste backfill plant for providing cemented paste to the underground workings;
- Mine support facilities including mobile equipment maintenance, mine personnel facilities, and shotcrete mixing plant;
- Bulk emulsion storage area;
- Utility infrastructure for the site: water, sewer, fire protection and communications;
- 69 kV power transmission line connected to the national electricity grid at Sabaneta Dam;
- 5 kV distribution from on-site stepdown transmission substation to the underground mine;
- Water storage pond for process make-up water;
- Emergency water storage pond for the management excess water during the wet seasons;
- Runoff settling ponds; and
- Surface water diversion infrastructures to manage local streams and runoff from the facilities.

The overall site layout, showing location of the mining portals, processing plant, tailings storage facility (TSF) and other major facilities, is shown in Figure 18.1 below.

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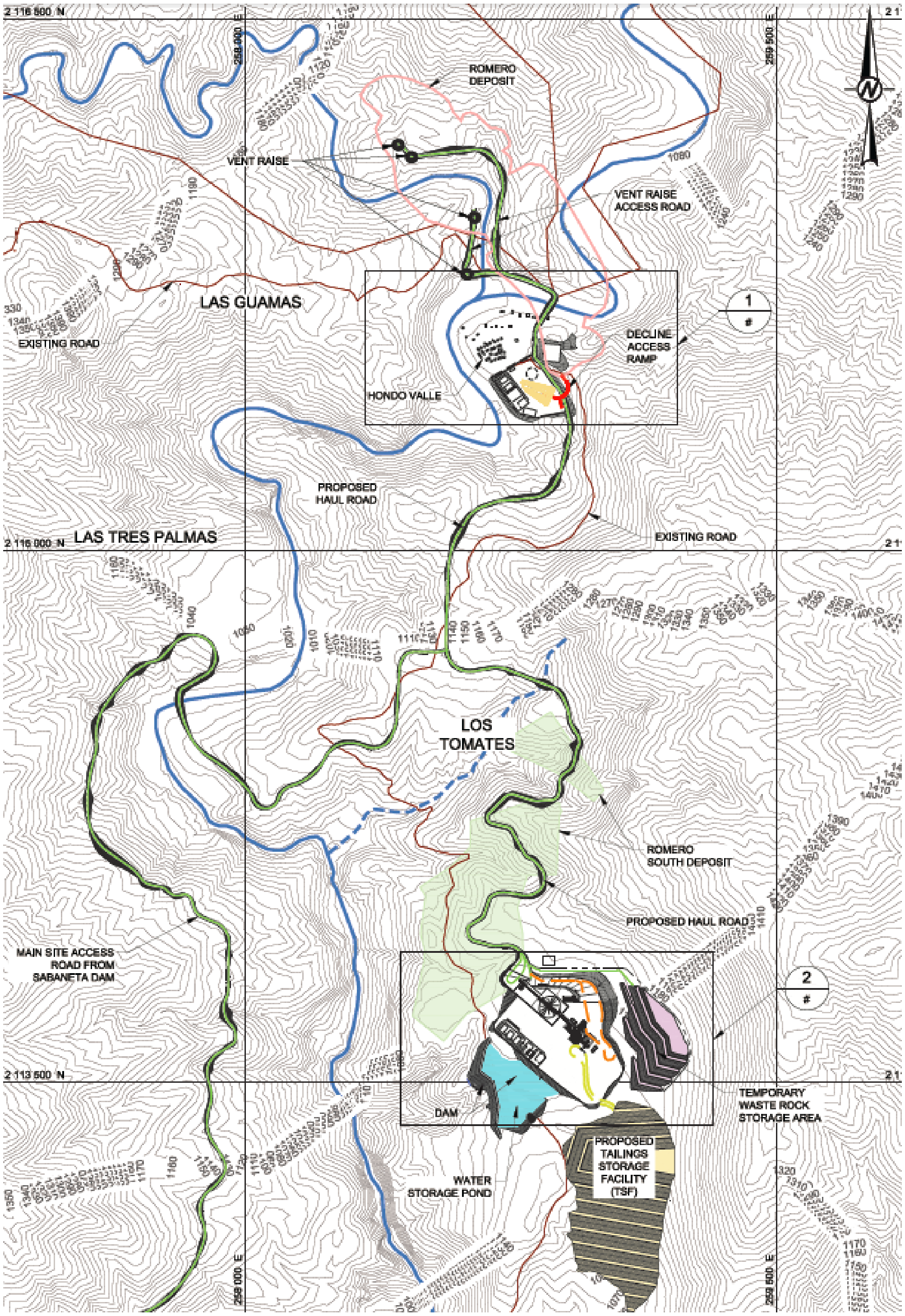
Figure 18.1: Overall Romero Site Layout



Source: JDS, Golder (2016)

18.2 Water Management

The foundation of the water management concept for the project is to provide the water supply without using water from the San Juan River or without using groundwater. The San Juan River feeds the Sabaneta irrigation reservoir some 10 km downstream of the Romero site. Therefore, it was a key design criteria for the Romero Project that the water system be independent from the San Juan River and be independent from the groundwater resource. The PFS shows that this design criterion is fulfilled. The complete mine operations' water demand will be satisfied based on recycling and reuse of the process effluents, and by the collection and storage of rainfall (surface runoff) from specific project site areas. The complete description of the Romero Project's water management system and water infrastructure is presented in Golder's report (2016b) titled "Romero Project Pre-Feasibility Water Management Report*"*. The key elements of the water management concept for the project are summarized as follows:

- Recycling and reuse of the process liquid streams will satisfy part of the water demand for the operation of the concentrator. Liquid streams that are generated by the process plant and by the paste plant will all be recycled and reused. As a result, the design of the project is such that no liquid effluents from the process will be discharged to the environment;
- The remainder of the water demand for the mine operations, namely process water make-up, dust suppression for the haul road, secondary site roads, crushing operation, washing of mobile equipment, showers and bathrooms at the site, will be satisfied by the collection and storage of natural surface runoff from specific areas of the project site. This anticipated water system will ensure a continuous water supply to the process plant during the prolonged dry periods, typical for the area;
- Under normal operating conditions, the water management system will also allow for the management of excess water collected during the rainy season. Instead of discharging excess water to the environment during the rainy season, the water management system was planned for excess water to be stored, transferred between facilities, and directed to water users at the site;
- As the project design does not include a tailings pond, there will be no discharge of tailings water. The tailings management strategy at the Romero site is based on returning the tailings to the underground mine as paste backfill and/or safely stored as inert dry tailings material in the dry stack tailings storage facilities;
- During operations, all runoff from the temporary (Years 1 to 5) waste rock storage area will be collected and reused in the process. At Year 5, the waste rock will be removed from the storage area and returned to the underground mine; and
- All water storage facilities of the project will be provided with adequate freeboard. Water retaining structures (dams) will be equipped with emergency spillways to protect the integrity of the structures under extreme flood events.

18.3 Tailings and Waste Rock Stockpile

18.3.1 Site Geotechnical Conditions

The complete description of the Romero Project dry stack tailings storage management system and infrastructures is presented in the Golder report entitled "GoldQuest Romero Project DSTSF and Temporary Waste Rock Stockpile Design" (Golder, 2016c).

The available geotechnical information at the site area is limited. To date, no geotechnical investigations have been carried out at the proposed processing plant facilities, DSTSFs; named as DSTSF 1 and DSTSF 2, and temporary Waste Rock Stockpile (WRS) areas, see Figure 18.1 for facilities location. Just a few mineral condemnation holes have been drilled in the vicinity of the proposed facilities and the information collected from these holes has been used as an indication of the site geotechnical conditions as summarized below.

The few mineral condemnation holes that were drilled in the vicinity of the DSTSF 1 footprint indicate up to about 4.5 m of overburden (LPT-23), generally saprolite materials, underlain by the bedrock. The ones drilled in the vicinity of the WRS / DSTSF 2 footprint indicate up to about 5.9 m of generally residual soil overburden (LPT-37), underlain by the bedrock.

A review of the LTP-80-A borehole log located within the WRS / DSTSF 2 footprint indicates the possibility of a pre-existing landslide within bedrock in this area. If present, an ancient landslide could have potential implications on the siting or stability of the proposed WRS / DSTSF 2.

A geotechnical investigation program will be carried out during the feasibility design to understand the geotechnical conditions at the project site including the investigation of the pre-existing landslide and its potential implications, if any. The geotechnical investigation program will include test pit excavation and borehole drilling including in-situ and laboratory testing to bring the facilities design to a feasibility level.

18.3.2 Waste Rock Stockpile – Temporary Surface Storage Area

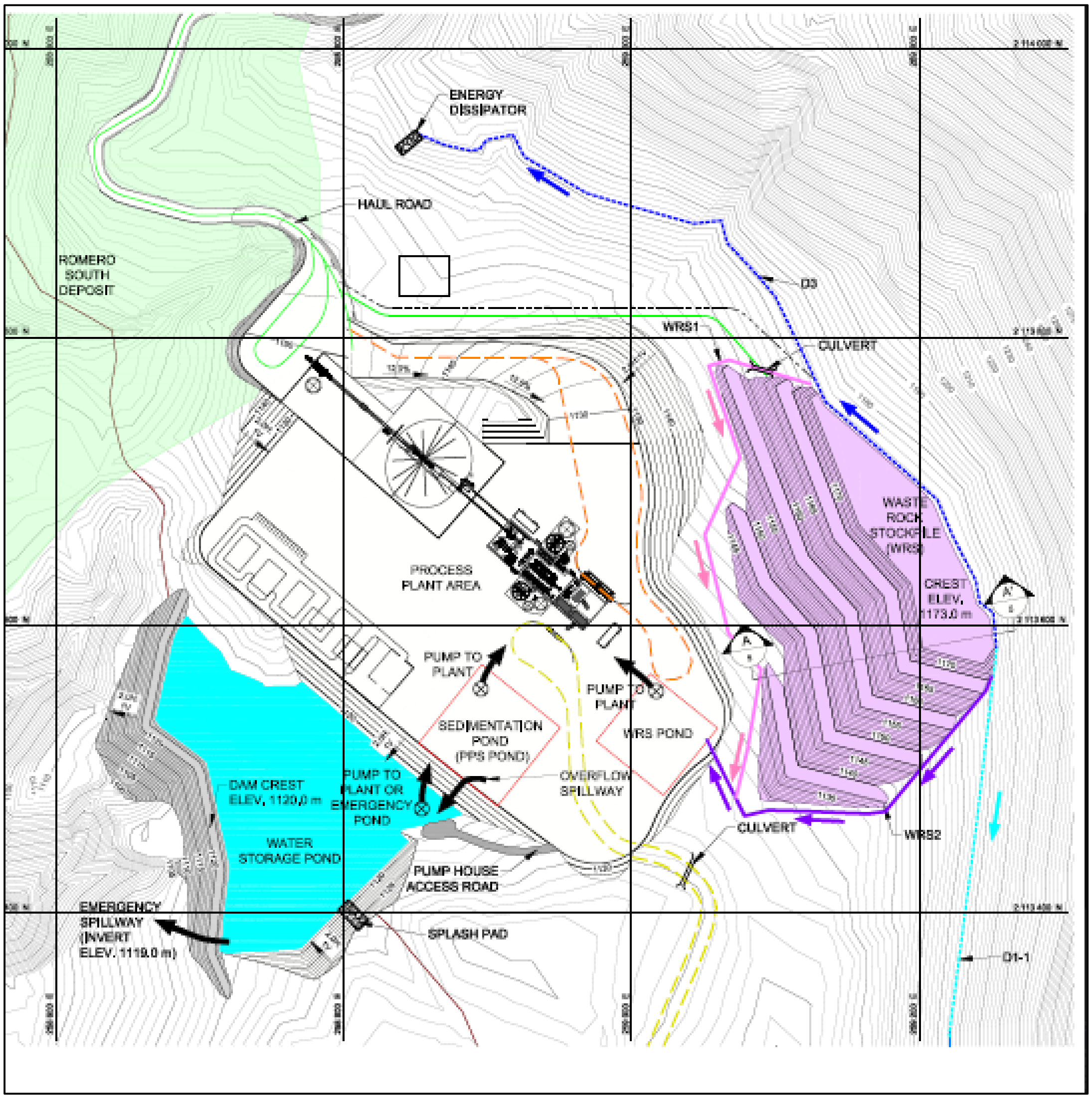
A temporary WRS will be required to manage the waste rock during operations in order to supply the underground mine backfilling requirements; this material could potentially produce ARD (Acid Rock Drainage), and therefore will not be left on surface at closure. The WRS will reach its maximum capacity in Year 2 and all waste rock will be returned to underground as backfill by Year 5 of operations. The WRS has been designed on the following basis:

- To provide sufficient storage capacity for 0.22 M-m^3 of waste rock;
- To contain the waste rock seepage using a clay liner. The water management system has been designed to collect the seepage and to return it to the process as water make-up.;
- To incorporate diversion channels to divert non-contact water around the facility footprint as needed; and
- To found the stockpile on a competent and stable foundation;

A plan view of the temporary WRS and the associated water management structures are shown in Figure 18.3, including:

- Diversion and collection channels;
- WRS pond located on the south side of the WRS in the processing plant facilities area;
- Pumping and pipeline systems for water recirculation; and
- Haul and access roads.

Figure 18.2: Process Plant Area, Water Storage Pond and WRS Plan View – Year 2 (WRS Ultimate Footprint)



The WRS will be located east of the processing plant facilities. It will provide storage for 0.22 Mm^3 of ARD waste rock with a maximum elevation of 1,173 m. The ultimate footprint of the stockpile will be approximately 3.7 hectares (ha), with a maximum height of 45 m (difference between the minimum toe elevation of 1,128 m at the southwest side and the top elevation of the facility). The stockpile will be developed in 10 m high benches, with 10 m wide berm between benches and 2H:1V inter-bench slopes. The overall slope of the stockpile will therefore be about 3H:1V. A cross section of the WRS is provided in Figure 18.5.

The entire footprint of the WRS will be cleared and grubbed. A 1 m thick clay liner will be constructed at the base of the WRS for seepage collection. Assuming that a subsequent geotechnical investigation shows that they are suitable, the in-situ residual soils will be reworked and compacted to produce the liner.

The WRS has been designed to be stable during the operations stage, under static and pseudo-static conditions.

Access ramps will be constructed to allow vehicular access to the WRS. A network of roads (access roads and haul roads) will be required on the proposed project site to connect up and to access the various project facilities.

The surface runoff from natural ground on the east side of the WRS will be diverted by channel D3 to the northwest of the facility. The diverted water will be released onto a natural stream towards the San Juan River; which is located downstream of the project facilities. Runoff from the surface of the WRS (contact water) will be collected and directed via collection channels to the WRS Pond, which will be located west of the WRS. Water collected in the WRS Pond will be pumped to the process plant. See Figure 18.2 for details.

The design of the water collection and diversion channels, and the WRS Pond is presented in the Golder report entitled “Pre-Feasibility Water Management, GoldQuest Romero Project.”

Figure 18.3: DSTSF 1 Plan View – End of LOM

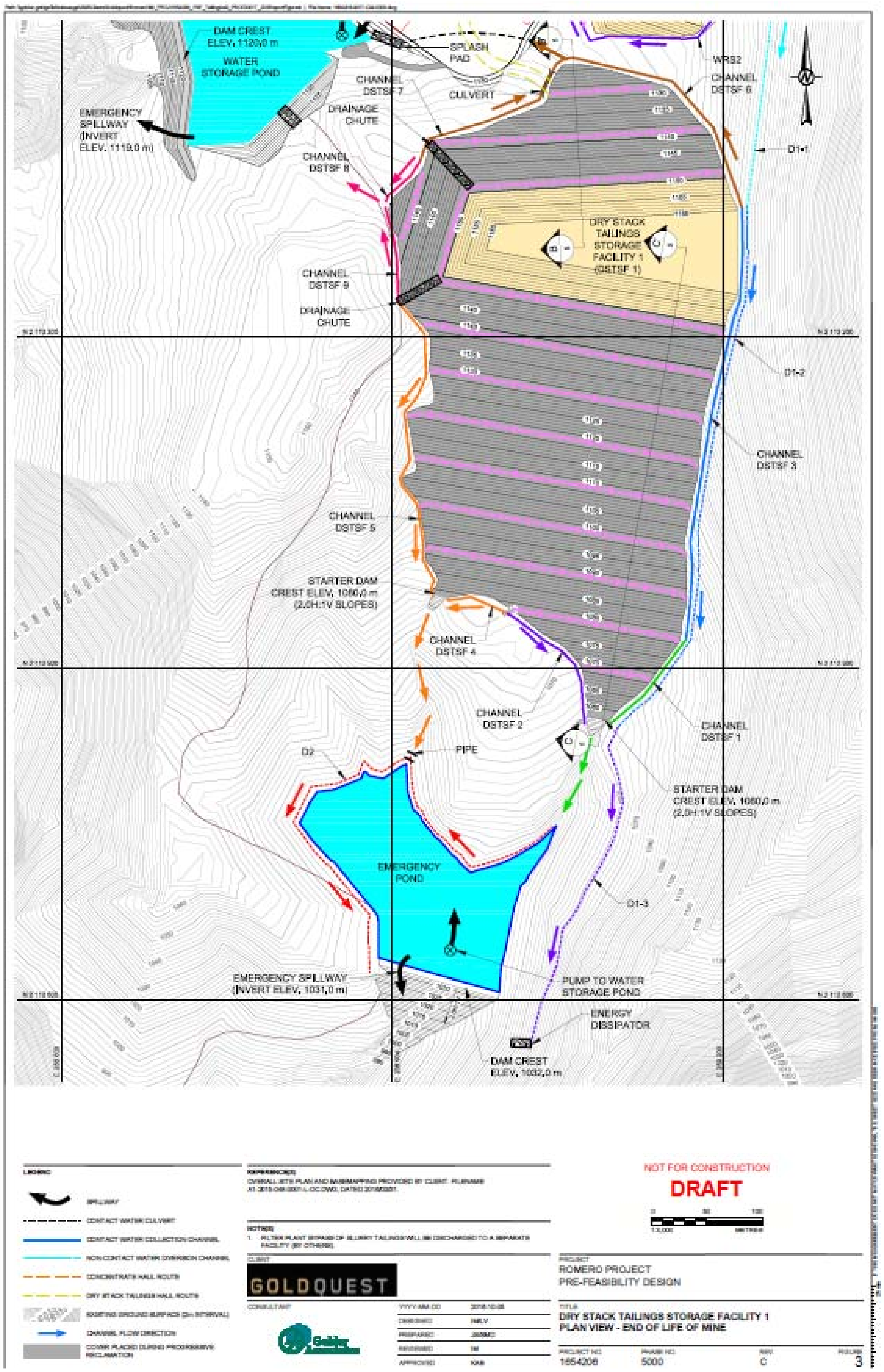


Figure 18.4: DSTSF 2 Plan View – Year 8

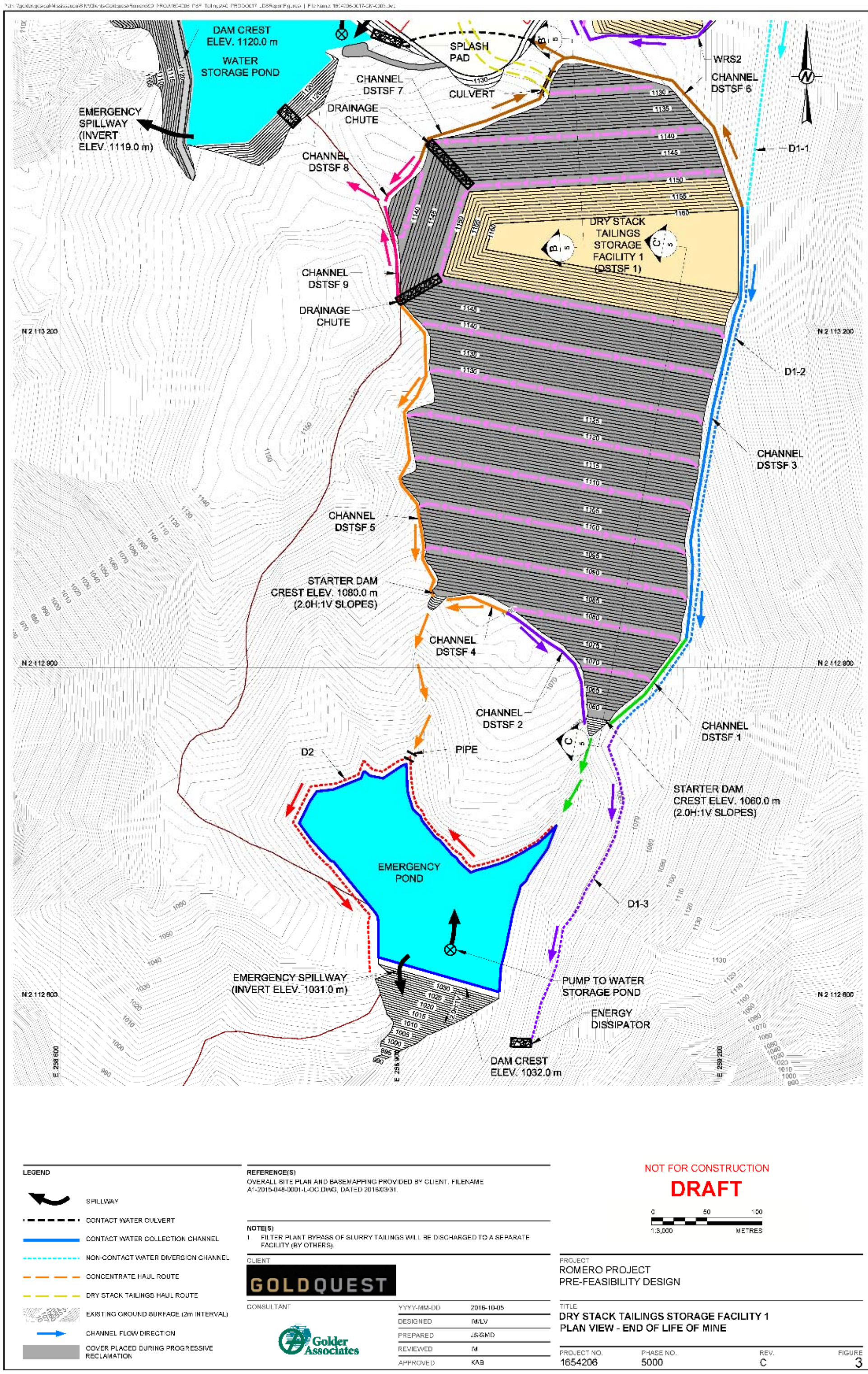
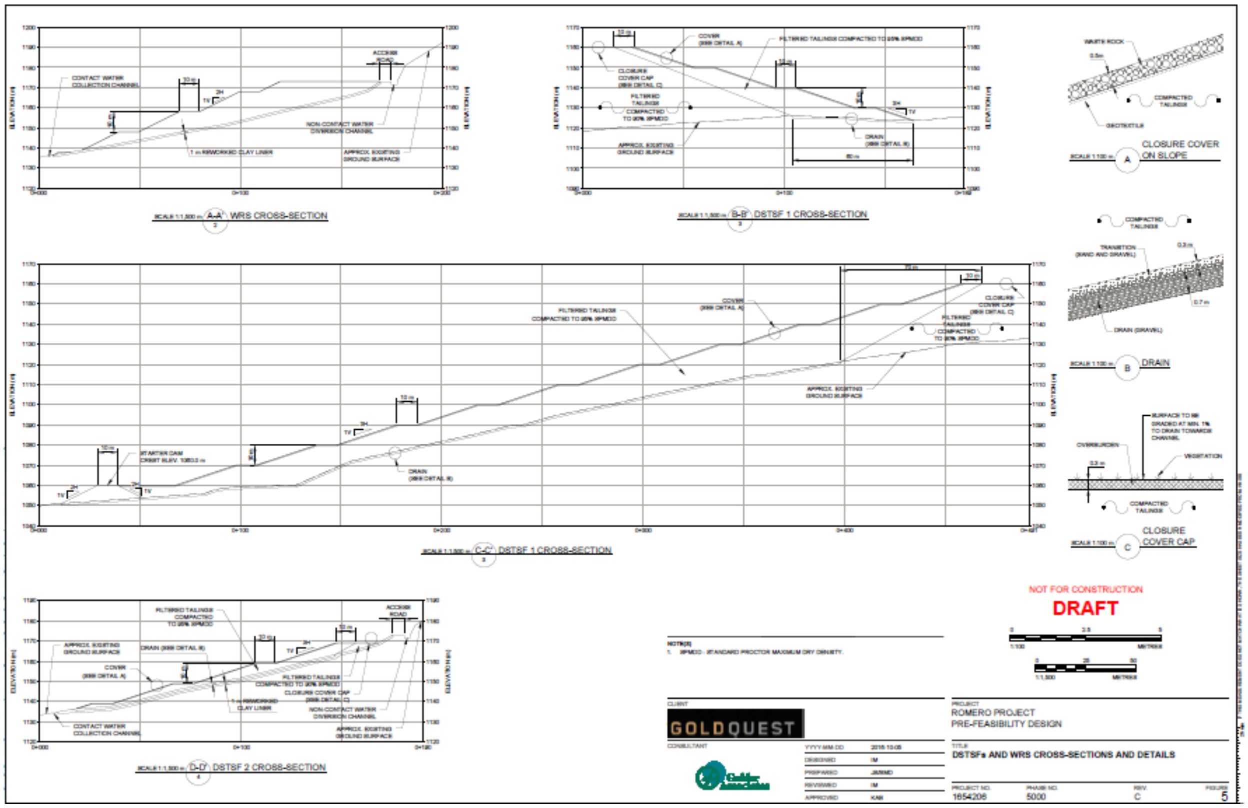


Figure 18.5: DSTSFs and WRS Cross-Sections and Details



18.3.3 Dry Stack Tailings Storage Facilities

The ARD generating fraction of the tailings to be generated by the project will be utilized in the paste backfill and the remaining non-ARD tailings will be thickened and filtered in order to be stacked in two dry stacks referred to as DSTSF 1 and DSTSF 2. The thickened and filtered tailings are referred to as "dry tailings" in this report.

The DSTSFs have been designed on the following basis:

- To provide sufficient storage capacity for 3.72 Mt (2.33 M-m^3, assuming that no filter plant bypasses occur - conservative approach) over a LOM of about eight years;
- To provide a small containment area for slurry tailings which bypasses the filter plant when it is not operational (design of the slurry impoundment will be carried out during the next design stage);
- To receive dry tailings with a geotechnical moisture content (defined as weight of water over weight of solids (Ww/Ws)), which is suitable for proper placement and compaction, as well to support necessary equipment traffic. This is currently estimated at 18% or lower; however, this will need to be verified by testing during the next design stage;
- In the case that some of the dry tailings reporting to the DSTSFs contains excessive moistures causing difficulties for workability and compaction, the "out-of-spec" tailings will be placed into a specified interior location and will receive special handling;
- To effectively drain the stacked dry tailings using an engineered underdrain system in order to reduce the risk of seismic liquefaction;
- To incorporate diversion channels to divert non-contact water around the facility footprint as needed;
- To collect dry stack tailings contact water for recycling to the process plant or for other usages such as dust suppression;
- To found the stack on a competent and stable foundation;
- To limit erosion with a cover made of geochemically inert rock material; and
- To place the cover progressively over the DSTSF 1 during operations to limit erosion during operations and to limit the work required after closure.

Plan views of the DSTSF 1 and DSTSF 2 and the associated water management components are shown in Figures 18.3 and 18.4, respectively. The associated components are:

- Diversion and collection channels;
- Collection pond on the south side of DSTSF 1 (i.e., the Emergency Pond) – including the dam and emergency spillway;
- Pumping and pipeline systems for water recirculation; and
- Haul and access roads.

The dry stack tailings will be loaded from the filter plant onto trucks, dumped in the DSTSFs, spread and compacted. DSTSF 1 will be located south of the processing plant facilities (which includes the filter plant) and will accommodate 2.18 M-m^3 of dry tailings, as shown in Figure 18.3. DSTSF 2 will be located at the former WRS area, which is located east of the processing plant facilities, and will accommodate 0.10 M-m^3 of dry tailings, as shown in Figure 18.4. If necessary, the remainder of dry stack tailings (i.e., 0.05 M-m^3) will be stored at the top area of DSTSF 1. The potential need for such additional storage will be confirmed during the operations stage.

- The configuration of DSTSF 1 shown on Figure 18.3 will provide storage for 3.49 Mt (2.18 Mm^3) of dry stack tailings with a maximum elevation of 1,160 m. The ultimate footprint of the facility will be approximately 13.0 ha, with a maximum height of 100 m (difference between the minimum toe elevation of 1,060 m at the south side from where it slopes up following the natural hill side and the top elevation of the facility);
- The configuration of DSTSF 2 shown on Figure 18.4 will provide storage for 0.16 Mt (0.10 Mm^3) of dry stack tailings with a maximum elevation of 1,169 m. The ultimate footprint of the facility will be approximately 3.2 ha, with a maximum height of 42 m (difference between the minimum toe elevation of 1,127 m at the southwest side and the top elevation of the facility);
- At both stacks, the dry stack tailings will be stacked to form benches at a maximum 10 m vertical spacing with 3H:1V inter-bench slopes. Each 10 m-high bench will be offset inward by a 10 m-wide horizontal platform. The overall slope of the stack will therefore be about 4H:1V. In order to reduce the amount of work such as slope flattening and re-contouring that will be required at closure, the exterior slope operational faces have been designed to match the closure configuration of the facility. Erosion protection measures will include placement of a closure cover progressively during operations. The cover will be constructed with waste rock, provided it has acceptable geochemical properties. The proposed closure cover will be 0.5 m thick and it will be placed over a geotextile filter. Cross-sections of the DSTSFs are provided in Figure 18.5;
- The exterior of the DSTSFs (i.e., the outer shell) will be compacted to a minimum 95% of Standard Proctor Maximum Dry Density (SPMDD). Excessively wet tailings produced by the filter plant (i.e., tailings having a geotechnical moisture content greater than the 18% used in the design criteria, and in practice tailings that do not provide adequate trafficability for equipment) must be placed within the interior of the DSTSFs (inner core). If required for trafficability, the large internal areas of the stacks may be constructed in a grid pattern, where trafficable dry tailings and waste rock (if available) will be used to construct access roads and to form the boundaries of wet cells. The size and number of these wet cells will be driven by the amount of wet tailings produced by the filter plant. The interior of the DSTSFs (inner core) will be compacted to at least 90% of SPMDD;
- An underdrain pad will be constructed at the base of each DSTSF to the extent shown in Figure 18.5 to facilitate the drainage of the placed dry tailings in order to reduce liquefaction risk. The underdrain pad will consist of a 0.7 m thick gravel drain layer overlain by a 0.3 m transition layer (sand and gravel). The underdrain pad for DSTSF 2 will be constructed over the clay liner placed at the former WRS area;

- The Canadian Dam Association (CDA) Dam Safety Guidelines (CDA 2013) were used to classify the DSTSFs with respect to the potential consequences of a presumed failure. Following the CDA (2013) classification methodology, the proposed DSTSFs have been designed for a Peak Ground Acceleration value representing the 1 in 5,000 year earthquake event, which corresponds to 0.79 g. The DSTSFs have been designed to be stable during the operations and closure stages, under static and pseudo-static conditions;
- The surface runoff from natural ground on the east side of the DSTSF 1 will be diverted by three diversion channels (D1-1 to D1-3). All ditches will convey runoff to the south side of the facility as shown in Figure 18.3. The surface runoff from the northeast catchment of the Emergency Pond will also be diverted by channels D1-3 and D2 to reduce inflow to the pond. The diverted water will be released onto natural streams towards the San Juan River; which is located downstream of the DSTSF 1;
- Runoff from the surface of the DSTSF 1 (contact water) will be collected and directed via collection channels to the Emergency Pond or to the Water Storage Pond, which will be located south and northwest of the DSTSF 1, respectively (Figure 18.3). Drainage measures at the DSTSFs will include the grading of all benches so that they drain towards one or other of the collection ditches or to the drainage chute (DSTSF 1) that will safely carry runoff down to the collection ditches;
- Water collected in the Emergency Pond will be pumped to the Water Storage Pond. Water from the Water Storage Pond will be pumped to the process plant;
- The water management system constructed for the WRS will stay in place and will be used to manage the water from the DSTSF 2. The water management for DSTSF 2 will be similar to that for the WRS, as described in the Section above;
- The design of the water collection and diversion channels, and the Emergency Pond is presented in the water management report;
- The entire footprint of the DSTSF 1 will be cleared and grubbed. The site preparation for DSTSF 2 will be carried out as preparation for the WRS. Access ramps will be constructed to allow vehicular access to both DSTSFs;
- The DSTSFs and associated components have been designed to meet closure requirements. At closure, the top surface of the DSTSFs will be regraded to prevent ponding. The final grading will provide positive drainage off the final top surface leading into the drainage channel. The closure cover will be placed progressively during the operation of the DSTSF 1 to the extent possible. The areas of DSTSF 1 and 2 that remain uncovered when mine operations cease will be covered as part of closure activities to prevent wind and runoff erosion of the tailings;
- Some of the haul and access roads will be decommissioned at the end of operations. Other access roads will be kept in service to allow equipment to access all surfaces of the DSTSFs for the purposes of closure and post-closure monitoring and maintenance requirements. Should erosion rills start to form after closure, it will be important that they be repaired before they become extensive;

- The diversion ditches will be permanent structures after closure and will have to be maintained. The contact water collection system, including the ponds, will be kept in service until the water quality is acceptable for direct release to the environment. At that time, the pond dams will be breached and the pumping systems will be removed;
- The DSTSFs closure and post-closure monitoring requirements should be prepared in conjunction with the overall project monitoring requirements; and
- Post-closure conditions for DSTSF 1 and DSTSF 2 are shown in Figures 18.6 and 18.7, respectively.

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Figure 18.6: DSTSF 1 Plan View – Post-Closure

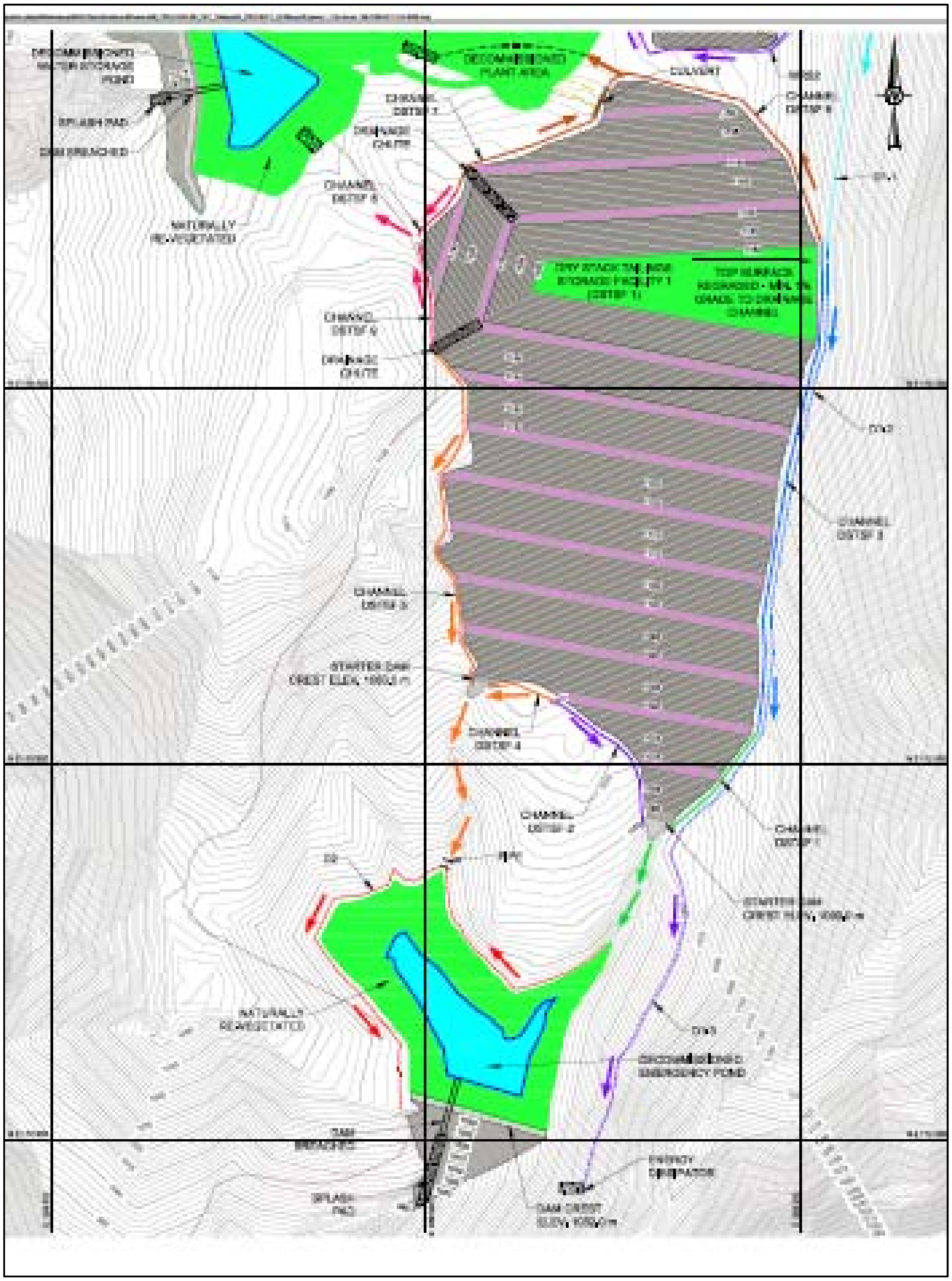
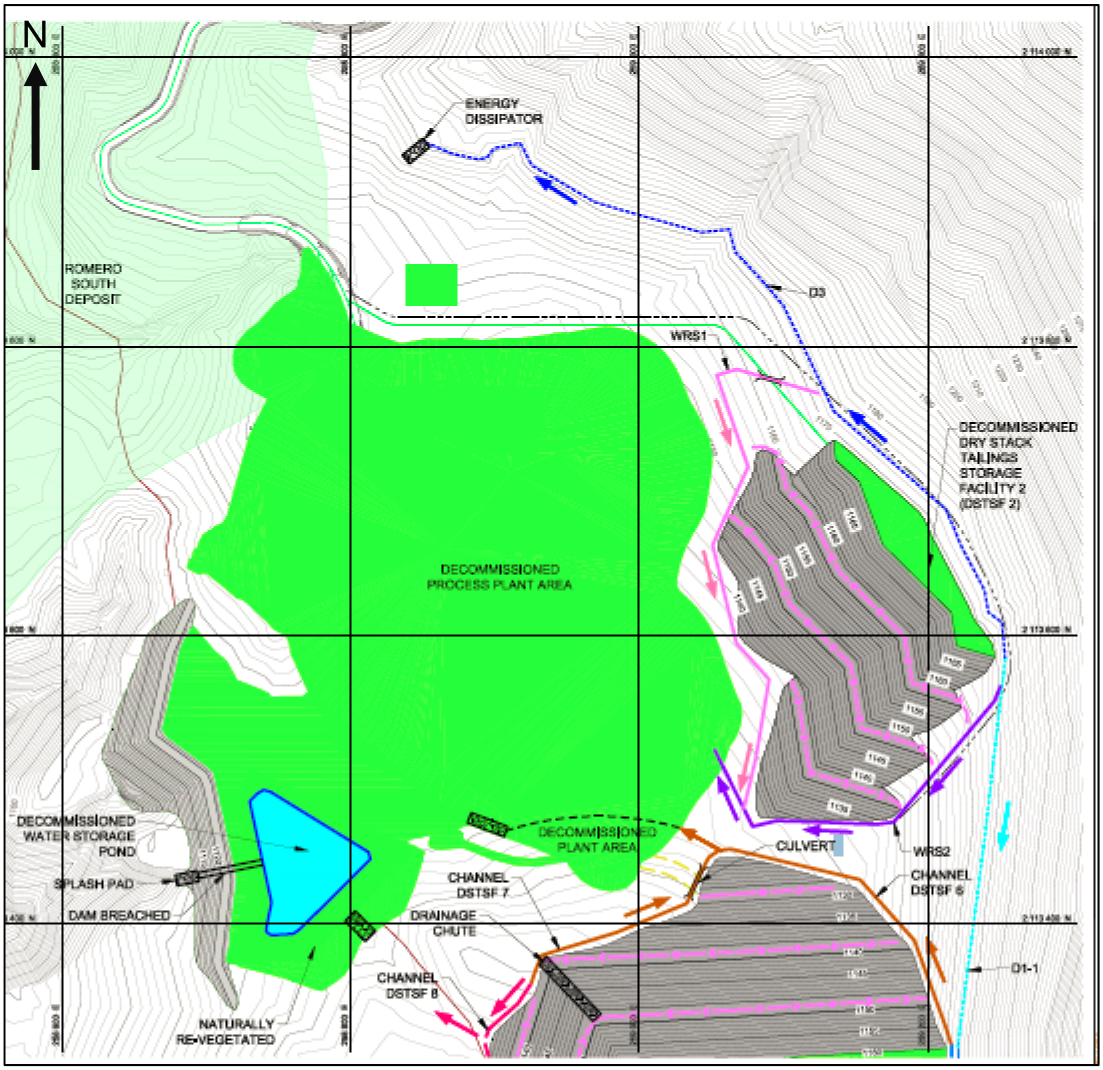


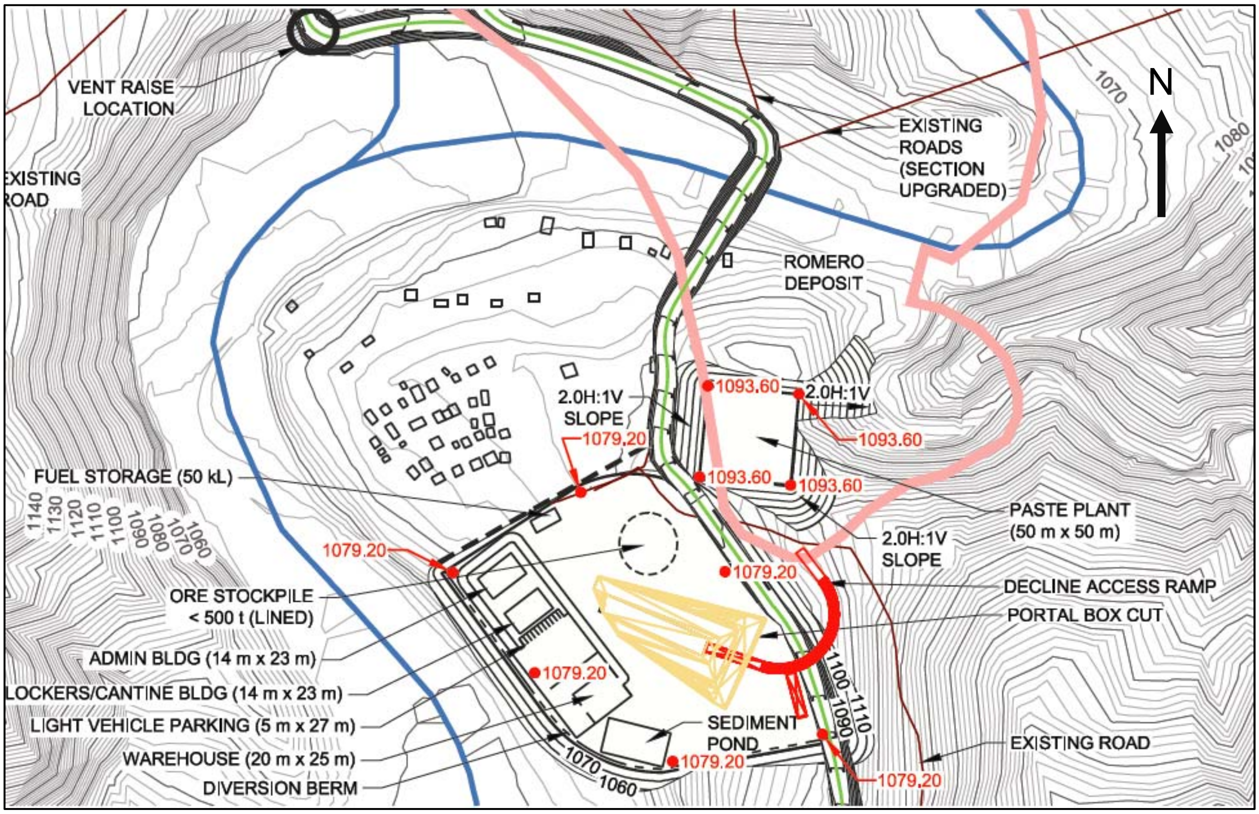
Figure 18.7: DSTSF 2, Process Plant and Water Storage Pond Plan View – Post-Closure



18.4 On-site Infrastructure

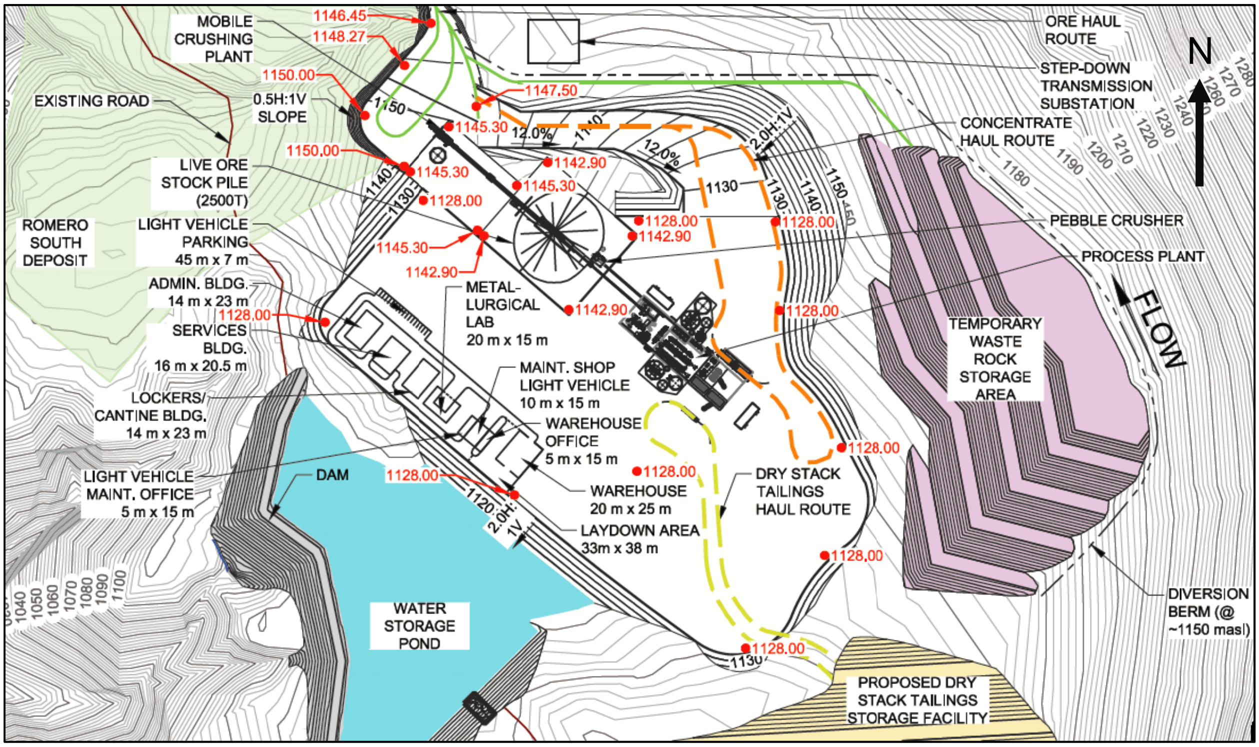
On-site infrastructure will be sited at either the portal location near Hondo Valle (Figure 18.8) or the process plant site (Figure 18.9), and will be located as close as possible to make efficient use of space.

Figure 18.8: General Layout at the Portal Site



Source: JDS, Golder (2016)

Figure 18.9: General Layout at the Process Plant Site



Source: JDS, Golder (2016)

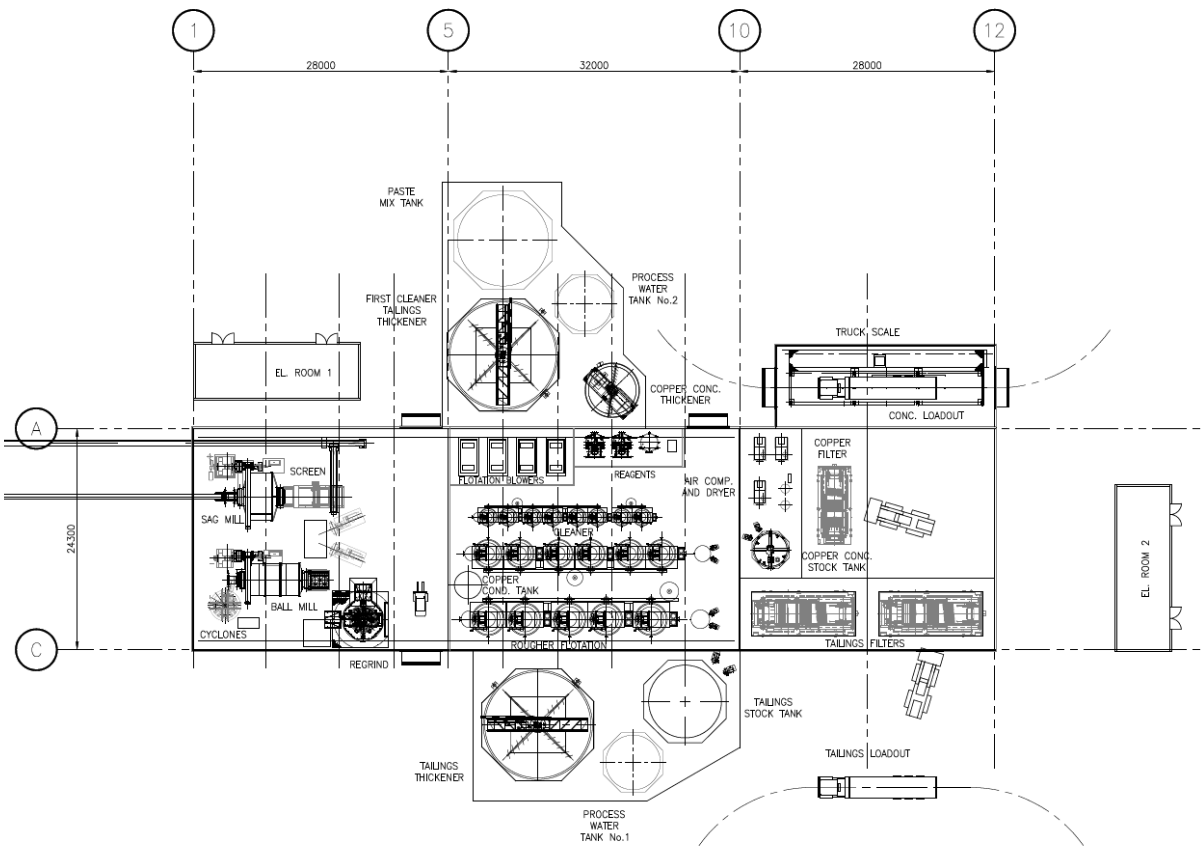
18.4.1 Process Plant

The primary process facilities include:

- ROM Stockpile pad;
- Primary crusher installation set on compacted fill, and a gabion basket retaining wall at ore loading bin;
- Live ore stockpile (2,800 t), reclaim and SAG mill feed system;
- Grinding and pebble crusher recirculation;
- Flotation circuit within structural building, partially cladded;
- Thickeners on concrete pad with concrete walled containment; and
- Pressure filtration for concentrate and tailings streams with covered load-out areas.

The process plant layout and facilities are shown in Figure 18.10.

Figure 18.10: Process Plant Layout



18.4.2 Maintenance Facility

The shop at the Romero site will consist of a 37 m long by 15 m wide concrete block wall structure designed to accommodate facilities for repair and maintenance of surface equipment and light vehicles. The building will also house warehouse storage space for spare parts, consumables and other materials and equipment. The shop area breakdown is provided in Table 18.1.

Table 18.1: Maintenance Shop/Warehouse Floor Areas

Description	Area (m^2)	Comments
Service Bays	220	2 truck bays, + 1 wash bay each 7.42 m wide x 9.8 m deep
Warehouse	370	9 sea-cans, 12.3 m long x 2.44 m wide

Source: JDS (2016)

The service bays are designated for the service and repair of the major surface hauling equipment which includes 35 t haul trucks and 3.0 m^3 front-end loaders. The facilities will include automatic hose reels in one bay for dispensing engine oil, transmission fluid, hydraulic oil, air, solvent, diluted coolant, and grease.

Tire repair will be done outside, weather permitting.

18.4.3 Laydown Area

Laydown areas for major process plant consumables are located to the south of the process plant and east of the ancillary facilities. Spare parts that require protection from the elements can be stored in the covered warehouse.

A separate construction laydown area has not been designated but the plant area pad was developed to allow for sufficient space around the infrastructure to store materials and equipment. Should additional storage area for construction materials be required, the north portion of the DSTSF may be utilized during the pre-production phase and first year of operations.

18.4.4 Mine Dry and Office Facilities

The 644 m^2 mine dry and office complex will be constructed with concrete block walls and concrete floors will be provided at the portal site, and a 322 m^2 administration building will be provided at the process plant site contractors. The facilities will comply with all building and fire code requirements.

The mine dry facility will service pre-production and operations staff during the life of the project and will contain the following:

- Male and female change room and locker areas; and
- Showers and washroom facilities with separate male and female sections.

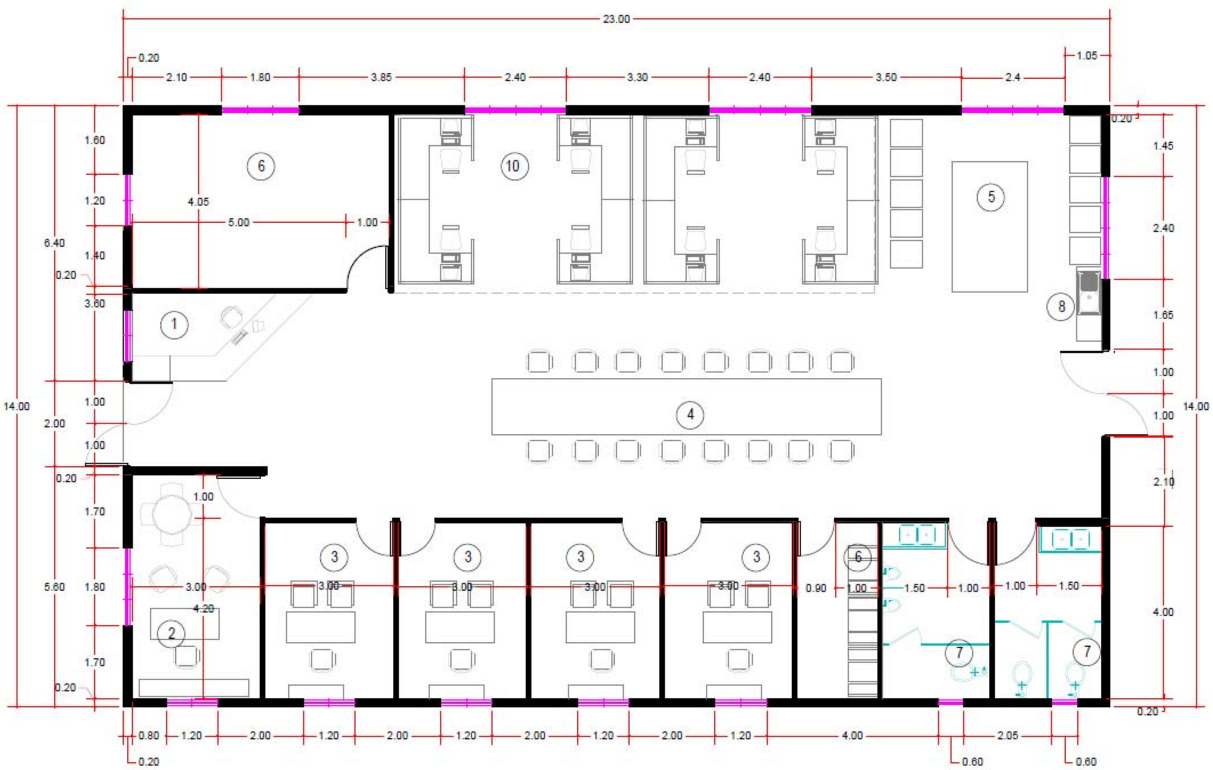
A male:female ratio of ~10:1 was assumed.

The site office facility will contain the following items:

- Private offices;
- Main boardroom; and
- Mine operations line-up area.

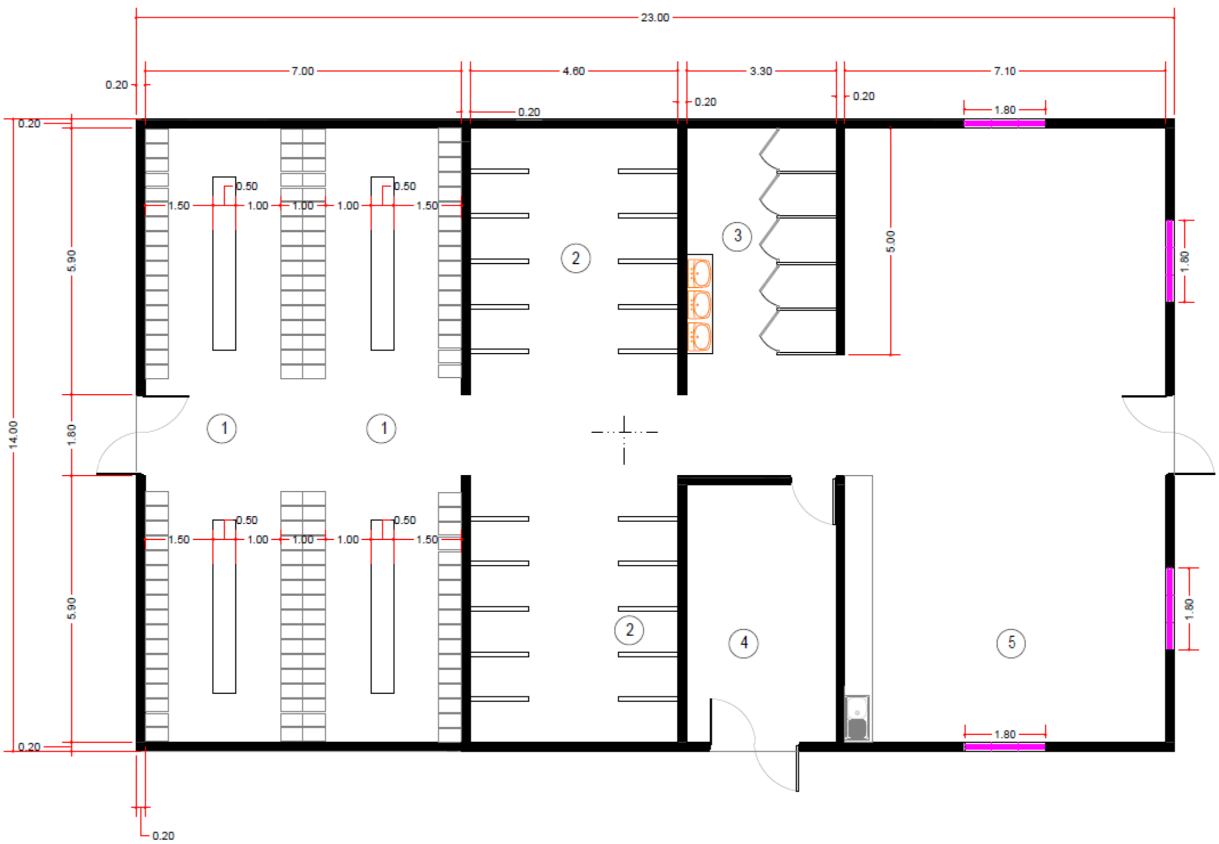
A layout of the mine dry/office complex is shown in the following figures.

Figure 18.11: Typical Administration Building Layout (Portal and Plant Site)



Source: GoldQuest (2016)

Figure 18.12: Mine Dry Layout



Source: GoldQuest (2016)

18.4.5 Fuel Storage

Diesel fuel storage capacity will consist of one 75,000 L double walled horizontal fuel tank inside containment structures. A fuel dispensing station will provide for vehicle fueling, and the entire installation will be protected with concrete bollards.

18.4.6 Bulk Emulsion Storage

Bulk emulsion required for the underground development and operation will be stored in a fenced area to the west of the main site areas, as per the national requirements. Explosives will be stored in a skid-mounted magazine at an appropriate distance from the bulk emulsion storage and the site facilities, until such time that the underground mine is sufficiently developed to move the explosives storage underground.

18.4.7 Site Security

Site security facilities will include guard posts at entrances to the property. The main entrance at the start of the main access road and the entrance onto the haul road will be controlled as checkpoints for vehicles and pedestrians entering the site. Potential access points from local community trails will have provisions for controlled access as necessary.

18.4.8 Assay Laboratory

The assay laboratory will be equipped with the necessary analytical equipment to perform all routine assays for the mine, the process facility, and the environmental departments, as well as metallurgical testing and sample preparation equipment for core and rock samples. The building will be a concrete block walled structure equipped with ventilation, dust collectors, temperature and climate controls.

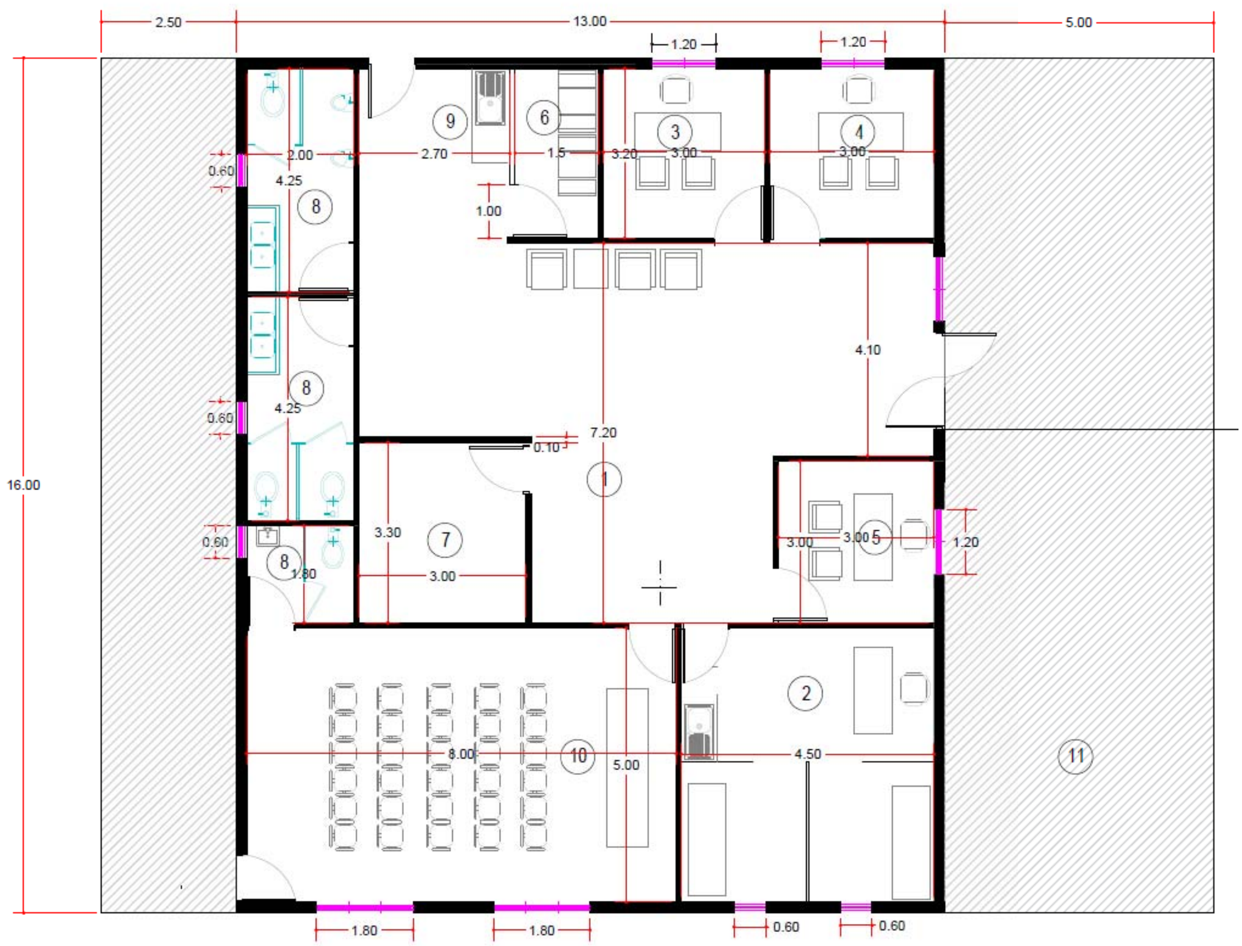
18.4.9 Paste Backfill Plant

A paste plant for controlled mixing and distribution of backfill to the underground will be located near the portal at Hondo Valle. The paste plant will include three main components: tailing feeder, paste preparation building, and binder silo. The paste preparation building will have two floors. In the main building, the lower floor will house the paste pumps and hydraulic power packs, while the second floor will house the paste mixer, the control room, and electrical room. The layout of these facilities will be arranged to make best use of the existing topography with minimal earthworks.

18.4.10 Medical Clinic and Mine Rescue Facility

The medical clinic and mine rescue facility will be housed in a single building with emergency vehicle parking outside. The medical clinic will include provisions for an emergency first aid station, consultation offices, and pharmaceutical storage. The mine rescue section will include provisions for mine rescue equipment storage, a mine rescue training facility, and offices for mine rescue staff and records. Figure 18.13 below provides the general layout of the facility.

Figure 18.13: Medical Clinic and Mine Rescue Layout



Source: GoldQuest (2016)

18.4.11 Utilities and Services

18.4.11.1 Waste Water Treatment Plant (WWTP)

Waste water and sewage will be treated by a membrane bioreactor (MBR) plant that will be constructed, assembled and tested prior to shipment to site. A sludge drying system will also be provided in a separate 40 ft container.

The treatment plant will include influent screening, an equalization/bioreactor tank (to handle the daily peaks in flow), a membrane system, a treated effluent storage tank and UV disinfection. The treated effluent will be regularly tested prior to being discharged to the surrounding environment.

18.4.11.2 Fire Protection

The Romero site facilities will be protected, at a minimum, from fire in accordance with applicable local codes and standards. The fire alarm system will consist of manual pull stations at building exits and audible and visual notification devices throughout the work areas.

All surface mobile equipment will be fitted with fire extinguishers at a minimum. The fleet of underground mining and surface haulage equipment will also contain fire suppression systems.

Fire suppression for the Romero site facilities will be provided by a firewater system fed by a firewater tank and modularized pump unit. The fire water pump system will include an electric main and jockey pump as well as a diesel-driven standby pump. The fire water pumping system will be housed in a modular building adjacent to the process plant. All buildings and conveyors will have fire extinguishers and some will have standpipe systems and hose connections. There are no sprinkler systems planned for inside the process plant. Instead, hydrants will be provided around the exterior of the building to provide access to water for fire response.

18.4.11.3 Communications

Site-wide communication design will incorporate reliable communications systems to ensure that personnel at the project site have adequate voice, data, and other communication channels available.

Communications will be facilitated by satellite internet connectivity initially until a communications line can be installed along the main access road to connect the site to the local communications network. A trunked radio system consisting of handheld, mobile and base digital radios will provide wide-area communications coverage.

18.5 Roads

The road network for the Romero site will consist of a main access road to connect the project with the municipal road networks as well as one primary haul road and several access roads.

In general, roads will be constructed with embankment fills sourced from cut sections along with the road alignments.

18.5.1 Main Access Road

To provide access to the project site from the municipal road networks, a 23.5 km access road will be constructed starting from the road overtop of the Sabaneta Dam. The road is broken down into three (sections):

- Section 1 largely follows the alignment of the existing road with new sections constructed around the villages along the route and sharp corners and steep hill sections improved;
- Section 2 will be constructed alongside the San Juan River and will include several small bridge crossings where the road moves from one side of the river to the other to follow better topography; and
- Section 3 will also be a new section of road constructed in proximity to an existing trail, but following a new alignment selected for optimal topography for most of the length.

The conceptual design of the main access road has been based on the following design criteria:

- Design vehicle: Medium truck (30T, tandem axle);
- Minimum width of travelling surface: 5 m;
- Design speed: 50 km/h;
- Side slopes: 0.5H:1V;
- Maximum grade: 12%;
- Safety berms (fills > 3 m in height): 0.5 m.

Locations in which the access road crosses small streams, natural storm water flow pathways or low points, corrugated steel culvert pipes will be installed to allow for water to pass underneath of the road.

Concrete culvert and spillway style crossings will be utilized for larger water courses, such as the San Juan River. These are typical of the region and effective during periods of high rainfall, when the river or streams are surcharged, due to the fact that water can flow over top of the crossing without damaging the structure.

The access road will be operated throughout the duration of the mine life with the exception of periods of high rainfall. During sustained rain events, the road may be closed for a period of several days so that traffic does not damage the saturated road. Graders will maintain the running surface, and excavators from the site can and will be used to maintain ditching and culverts.

Signage will be installed along the entire length of the road to provide controlled access at public intersections as well as to post speed limits, obstacles and cautionary signage for sharp curves and steep inclines/declines.

Construction of the access road will take approximately 15 months. Portions of the road may be constructed in advance of the full project implementation phase in order to advance the overall schedule.

18.5.2 Haul Road and Service Roads

The road network on-site at Romero will consist of one 2.8 km haul road and several, shorter service roads.

In general, the haul road and service roads will be designed and constructed in such a way as to balance the cut and fill volumes along each section of road.

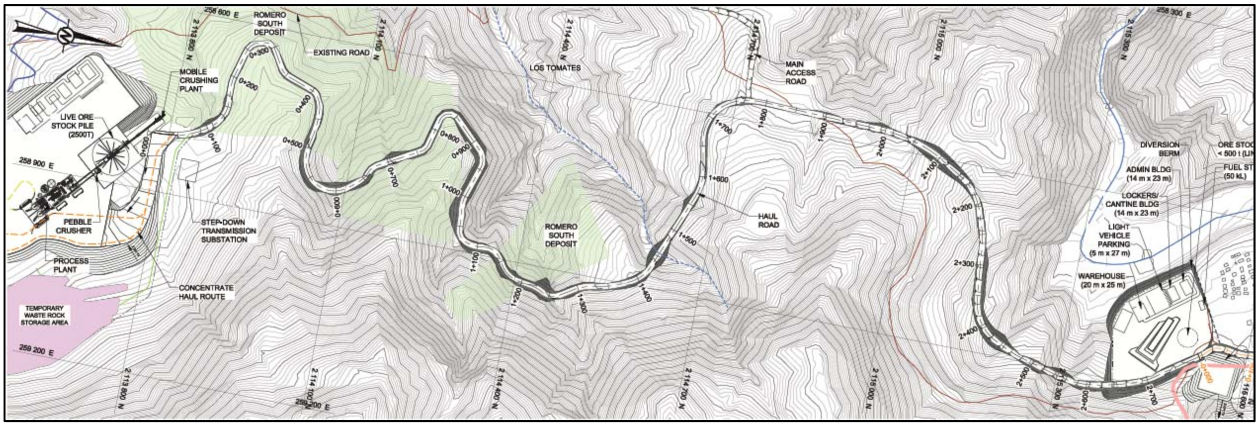
The haul road will connect the portal at Hondo Valle with the Main Plant Site and will be designed and constructed based on the following criteria:

- Design vehicle: Heavy truck (35T articulating haul truck);
- Minimum width of travelling surface: 6 m;
- Design speed: 40 km/h;
- Side slopes: 0.5H:1V;
- Maximum grade: 10%;
- Safety berms (fills > 3 m in height): 1.0 m.

A trench will be constructed along the uphill side of the haul road to accommodate the slurry tailings, water supply and water return lines running between the process plant and the paste plant. At 500 m intervals the road will be widened to provide pull-outs for vehicles to pass one another.

Figure 18.16 below provides the plan view of the haul road.

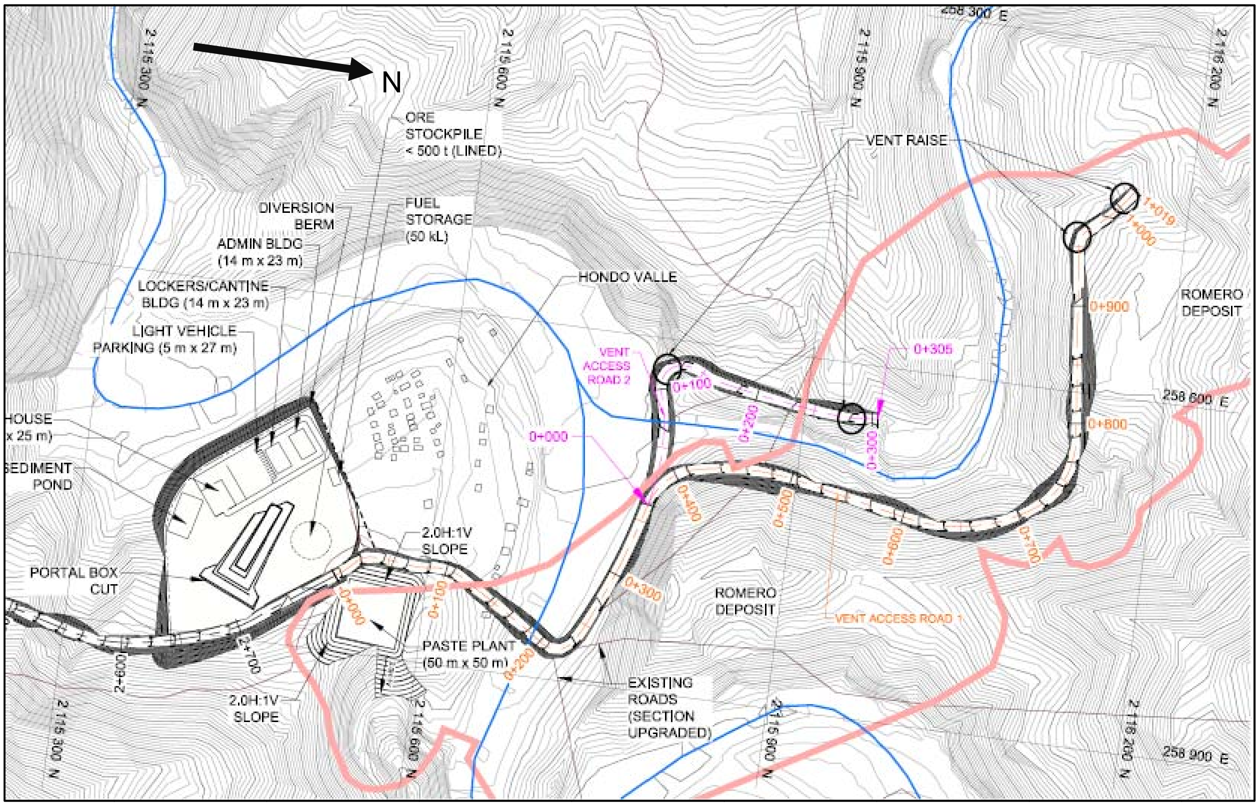
Figure 18.14: Haul Road Plan View



Source: JDS (2016)

Services roads will be constructed to access vent raise locations to the north of Hondo Valle as well as to access areas around the DSTSF. Figure 18.15 provides the plan view of the vent raise service road routes.

Figure 18.15: Vent Raise Service Roads



Source: JDS (2016)

18.6 Power Supply and Distribution

Power will be supplied through a 24.5 km 69 kV overhead transmission line connected to the Dominican Republic national grid. The line will begin Sabaneta Dam substation and end at the planned substation at the main process plant site.

18.6.1 Medium-voltage Transmission Line and Substation

Scoping and pricing for the 69 kV transmission line and the stepdown transmission substation on-site has been provided by Insel Ingeniería Y Servicioes Electrotechnicos, S.R.L. (“Insel”), a leading electrical contractor in Santo Domingo. A certificate of “no objection” for the connected load has been received from Empresa de Transmisión Electrica Dominicana for the connected load planned.

The Sabaneta Dam substation will require several additions and modifications in order to connect the transmission line to Romero, including:

- Voltage transformer;
- Disconnect switch with ground connection;
- Current transformer;
- Three-phase circuit breaker; and
- Control and protection cabinet.

The transmission line will generally follow the alignment of the main access road, but final routing will take into account environmental considerations, property taxes, technical and economic aspects. In general, final design work will be completed using the following criteria:

- Low environmental impact, to facilitate environmental authority license approval;
- Low environmental cost, to reduce execution time and cost;
- Avoid populated areas along route;
- Tower/pole site accessibility;
- Minimized number of vertices, to facilitate the construction and reduce costs;
- Constructability, to reduce execution time and cost; and
- Overall transmission line length.

18.6.2 Site Distribution

On-site power will be distributed from a 10 MW rated substation, connected to the 69 kV transmission line. The substation will have one 10 MVA transformer stepping down the 69 kV to 4160 V.

Power will be supplied to the mine portal, underground mine and paste plant via a 3 km, 4160 V overhead power line from the process facility substation. The power line will follow the haul road from the process facility and terminate at a 4160 V switchgear line-up in electrical room 3 (ER-3).

The total connected load for the project is calculated at 9.9 MW, with the total operating load calculated at 7.3 MW. The load breakout is anticipated as follows in Table 18.2.

Table 18.2: Total Connected and Operating Power Loads

Operational Area	Connected Load (MW)	Operating Load (MW)	Operating Load (MVA)
3000 - Process Facilities	5.7	4.2	4.9
4000 - Underground	2.2	1.7	2
5000 – Paste Plant	1.3	0.9	1
6000 – On-Site Infrastructure	0.7	0.5	0.8
Mine Site Totals	9.9	7.3	7.7

Source: JDS (2016)

18.6.3 Backup Power

The permanent standby power system will consist of two standby diesel generators.

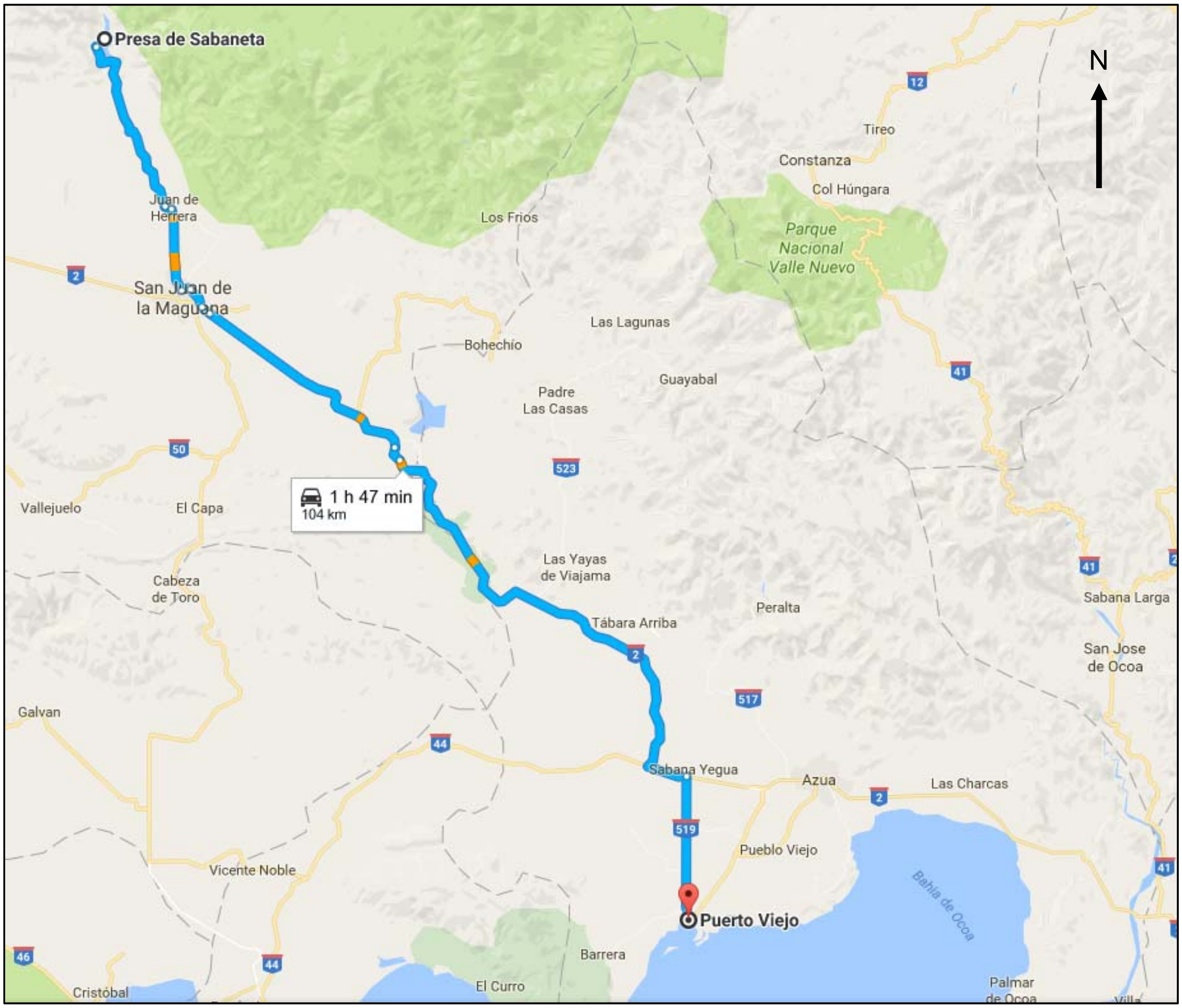
One generator will be located close to ER-2 at the process facility and will supply power to specific loads in the facility to enable purging of the process during utility supplied power outages and keep other essential systems (agitators, reagent ventilation fans, sump pumps, controls, communications etc.) operational. During the construction phase, this generator could be installed early in the schedule to provide temporary construction power.

One generator will be located close to ER-3 at the mine portal site and will supply power to the underground area to ensure ventilation, emergency lighting, sump pumps and refuge chambers remain operational in a power outage event, as well as other loads in the paste plant to enable purging of the process during utility supplied power outages. This generator will also keep other essential systems (controls, communications etc.) operational. During construction phase this generator could be installed early in the schedule to provide power to pit drills and temporary construction power.

18.7 Port Facilities and Concentrate Shipping

Concentrate will be loaded into 30t highway-rated trucks at the process plant and hauled approximately 127 km to Puerto Viejo near the town of Azua. The first 23.5 km will be along the main access road, with the balance being paved, municipal roads from Presa Sabaneta (Sabaneta Dam) to the port. Figure 18.16 provides an overview of the route travelling along the municipal roads and highways.

Figure 18.16: Concentrate Trucking Route, Presa Sabaneta to Puerto Viejo



Source: JDS and Google Maps (2016)

Concentrate will be stored in a covered shed at the port that will be constructed during the implementation phase of the project. It is envisaged that the storage shed will be sized to accommodate 15,000 t of concentrate, and built high enough that the trucks can dump underneath.

Concentrate will be shipped from the port to the smelter destination in lots of 10,000 t at a frequency of one shipment every 50 days. A ship loading system will be purchased and established at the port. When the vessel is ready to be loaded, the system will be setup and a loader and tandem axle dump truck will transport the concentrate from the storage shed to the dock and load the hold of the vessel. Concentrate will be loaded onto the vessel in bulk.

19 Market Studies and Contracts

19.1 Market Studies

A concentrate marketing firm has been consulted to provide guidance on concentrate terms and preliminary marketability. 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	%	90.0
Cu Minimum Deduction	%	1.0
Au Minimum Deduction	g/t	0.0
Ag Minimum Deduction	g/t	0.0
TC/RCs		
Treatment Charge	US$/dmt conc	85.00
Cu Refining Charge	US $/lb	0.085
Au Refining Charge	US $/oz	5.00
Ag Refining Charge	US $/oz	0.50
Transport Costs		
Moisture Content	%	8
Transport to Port	US$/dmt conc	88.93

Source: JDS (2016)

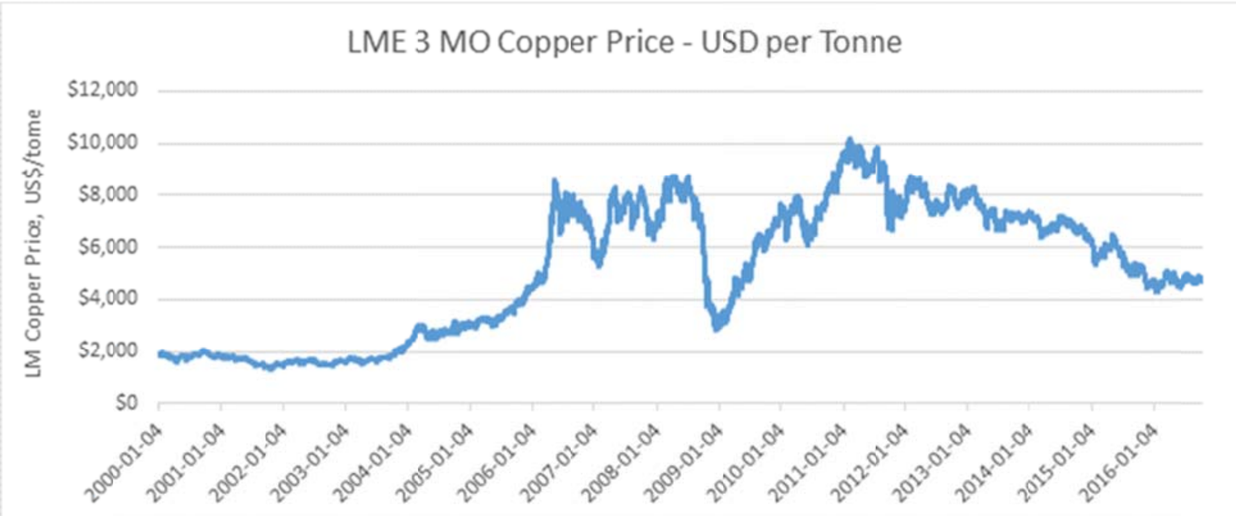
19.2 Royalties

The economic analysis has considered a 1.25% NSR royalty on all revenues. LOM royalties amount to $13.1M.

19.3 Metal Prices

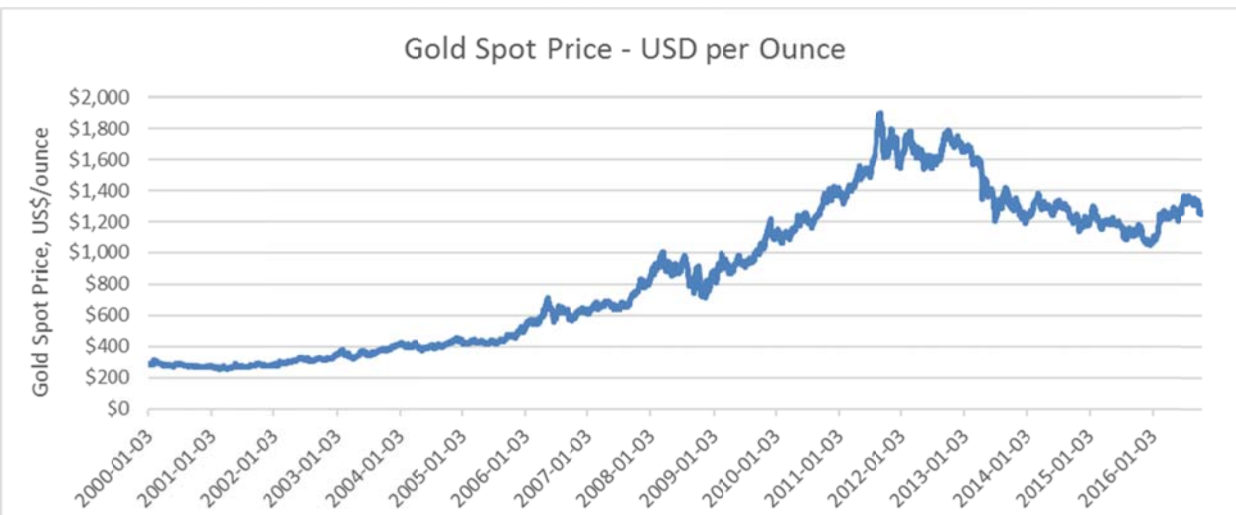
The base and precious metal markets benefit from terminal markets around the world (London, New York, Tokyo, Hong Kong, etc.) and fluctuate on an almost continuous basis. Historical metal price for copper are shown in Figure 19.1 through Figure 19.2 and demonstrate the change in metal price from 1998 through to 2016.

Figure 19.1: Historical Gold Price



Source: Bloomberg (2016)

Figure 19.2: Historical Copper Price



Source: Bloomberg (2016)

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

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

Metal Price and F/X Rate	Unit	Value
Cu Price	US$/lb	2.50
Au Price	US$/oz	1,300
Ag Price	US$/oz	20.00
F/X Rate	US$:C$	0.78

Source: JDS (2016)

20 Environmental Studies, Permitting and Social or Community Impact

The purpose of this Section is to discuss reasonably available information on environmental, permitting and social or community factors related to the project at its current Pre-Feasibility level. This Section covers the following:

- Environmental features of the Romero Project;
- Project permitting requirements and status of permit applications;
- Current status of the baseline studies and next steps;
- Social and community related requirements for the project and the status of the negotiations or agreements with local communities; and
- Mine closure requirements and conceptual closure plan.

20.1 GoldQuest Environmental Policy

GoldQuest has an environmental Policy by which the Company is committed to:

- Complying with the law and conduct all business in an ethical manner;
- Continuously review environmental achievements and technology to seek and implement methods for further improvement;
- Conduct regular environmental, health and safety preparedness and emergency response plans to verify compliance with the corporation’s policy and applicable regulations. Identify revisions or improvements to current practices in order to minimize environmental impacts. Report findings regularly to the Board of Directors;
- Educate employees in environmental matters and responsibilities relating to performance of their assigned tasks;
- Foster communication with shareholders, the public, employees, indigenous people and government to enhance understanding of environmental issues affecting the corporation’s activities;
- Work pro-actively with government and the public to define environmental priorities. Participate in the development of responsible laws for the protection of the environment;
- Allocate sufficient resources to meet the corporation’s environmental goals. Annually assess the projected costs of decommissioning and reclamation to ensure that there will be sufficient cash reserves to pay for these costs upon closure.

20.2 Environmental Features of the Romero Project

The PFS shows strong positive environmental features for the Romero Project, namely:

- The project technology is cyanide free;
- The project is not using the San Juan River for water supply;
- The water management system for the Romero site is developed to fulfill all of the water supply requirements based on water recycling, water reuse and by the collection and storage of rainfall over a small portion of the project site;
- The project is not using any groundwater, thus not impacting the natural equilibrium between the San Juan River and the regional aquifer;
- The design of the concentrator is such that all liquid streams are recycled and reused within the process, thus eliminating the discharge of any liquid effluents to the environment;
- The Romero Project has no tailings pond that would lead to the discharge of tailings water into the environment; it also has no tailings dam;
    - Tailings management for the Romero Project is based on the safe disposal of the tailings as “inert cement paste” into the underground mine and/or as a dry filtration cake for surface storage in a contained area to be revegetated at closure;
- The project will not leave any waste rock piles on the site after closure, since all waste rock is to be returned to the underground mine;
- The project has no air emissions from the process plant or the diesel power plant as the project is based on power supply by power line connected to the national grid. As a result, the carbon footprint of the project is non-significant; and
- The project will not change in any ways the natural landscape of the valley, as it is an underground mine.

Permitting of a new mine carries some risk due to the proximity of the project to a national park and the San Juan and La Guama Rivers. As the project plans will 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 with emphasis on local communities.

20.3 Permitting Requirements and Status of Permitting

At the time the PFS was prepared, the following permitting steps had been completed:

- The application for the exploitation of the Romero mine (“Solicitud De Concesion Para Exploitacion De Minerales Metalicos”) has been filed with the “Dirección General de Minería” of the Ministerio de Energia y Minas”. At the time the PFS was prepared, the application was being processed by the Ministry;

- In compliance with the requirement of the Mining Law ("Ley Minera No. 146 (1971)), a public project notice for the Romero Project has been published in local and National newspapers of the Dominican Republic;
- The application for authorizing the construction of the new access road to the project site has been filed with the "Dirección General de la Planification y Desarrollo or the Ministerio de Obras Publicas y Communicaciones". At the time the PFS was prepared, the application was being processed by the Ministry; and
- The application for authorizing the connection of the Romero"s project power line to the National Grid has been filed with the Corporacion Dominicana de Empresas Electicas Estatales (CDEEE).

In terms of environmental permitting, the permitting process is governed by the Dominican Republic Law No. 64-00 ("Ley General Sobre Medio Ambiante y Recursos Naturales, 64-00 – August 18, 2000"). The Law 64-00 is administered by the Dominican Republic State Secretariat of Environment and Natural Resources. Article 41 of the Law specifies that mining projects are subject to an environmental evaluation, and Article 38 specifies the evaluation process according to the following steps:

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

At the time the PFS was prepared, the environmental application under Law 64-00 has not been filed. The environmental application will be filed once the exploitation license is granted and terms of reference have been outlined.

In addition to the environmental evaluation process, the project will proceed, in a next step, with the preparation of the Socio-economic Impact Assessment (SIA) in compliance with the International Finance Corporation (IFC) Performance Standard and Equator Principles.

20.4 Baseline Studies

At the time the PFS was prepared, field work was completed to set up two field programs for meteorological and surface water monitoring (AMEC, 2013). The program includes flow measurements and water sampling at eight monitoring stations located in the vicinity of the project area. The program included the purchasing and field installation of flow measurement equipment and of a meteorological station (located next to the Exploration Camp).

The sampling program was fully developed, including but not limited to: methods of sampling, locations, frequency, list of parameters to be analyzed, shipping procedures to laboratories, reporting and methods of flow measurements.

A strategic pre-scoping environmental review was also completed (AMEC, 2014). The review included, without being limited to: a review of the Dominican Republic regulatory framework, the social and economic environment related to the project, and the preliminary framework for social engagement.

The project will pursue work for the baseline studies in preparation for the environmental impact assessment (EIA) and SIA.

20.5 Communities and Social Engagement

There is one small village located near the project site, Hondo Valle village, with a population of 80. Four other settlements are located along the 25 km road from Sabaneta town to the project site; Higuera (population of 170), La Cienega (population of 100), Higinito (population of 800), and Boca de Los Arrogas (population of 100). The site layout for the Romero Project has been developed so as not to interfere with the location of the Hondo Valle village. The town of Sabaneta (about 25 km from the project site) is the largest town in the vicinity of the project area.

As a next step, the project will proceed with the preparation of the SIA in compliance with the IFC Performance Standard and Equator Principles, together with its social engagement plan.

20.6 Mine Closure Concept

The Dominican Republic does not have any national mine closure guidelines. For the purpose of the PFS, a conceptual closure plan is presented based on industry standard best practices and to meet and or exceed standard Canadian mine closure practices (Kabir, 2015). These are considered to be among the highest standard globally.

The conceptual closure plan, is based on three major project components:

- Power line;
- Access road; and
- Mine and concentrator.
    - Removal or all surface facilities. Sloping and revegetation of mine waste stockpiles
    - Permanent sealing of all underground openings

20.6.1 Power Line

The power line and the electrical substation will not be demolished at closure. This infrastructure is anticipated to be an added value asset to be left in place, to benefit to the long term sustainable development of the area. Terms for leaving the power line infrastructure will be established in agreement with the Dominican Republic government.

20.7 Access Road

Similarly, the mine access road will not be removed, as this infrastructure is also anticipated to be an added value asset, to be left in place for the long term sustainable development of the area.

20.8 Underground Mine

The closure of the underground mine will take place progressively during operations as the mine will be backfilled with the tailings paste produced by the paste plant. Furthermore, at Year 5 of the operation, all waste rock will be moved from the temporary storage area at the process plant and returned to the underground mine. At the end of the mine life, ramps, remucks, sumps, and vent raises, will be paste backfilled as well using dry tailings from the storage area at the process plant. At final closure, all mine openings will be sealed according to the best safety practices that are described in Canada guidelines for mine openings.

20.8.1 Mine, Concentrator and Associate Site Infrastructures

The closure concept for the Concentrator and associated site infrastructures follows the industry standard best practices and Canadian closure guidelines:

- Heavy mining mobile equipment (haul trucks, shovels, drills) will be transported off-site, stored temporarily in San Juan, and sold;
- Mobile crusher will be transported off-site, temporarily stored in San Juan, and sold;
- Major process equipment (ball mills, flotation units, thickeners, filters, paste plant major equipment, tailings pumps, other large capacities water pumps, DCS) will be cleaned, dismantled, transported off-site with temporary storage in San Juan, and sold;
- All small mobile equipment, pick-up trucks, backhoes, loaders, graders, etc., will be transported off-site, with temporary storage in San Juan, and sold;
- All other mechanical equipment, such as piping, tanks, pumps, conveyors, silos, etc. will be cleaned, dismantled, transported off-site, with temporary storage in San Juan, and sold or sent to scrap;
- Fuel storage tanks and fuel distribution will be cleaned, dismantled, transported off-site, with temporary storage in San Juan, and then sold or sent to scrap;
- All office material, furniture, office equipment will be transported off-site and offered for free to the local people;
- Demolition of buildings: once all equipment will be dismantled and removed, the closure plan will require the demolition of all buildings at the Hondo Valle and process plant sites. The demolition approach will be based on the disposal of the demolition debris in the underground mine, together with the waste rocks and paste backfill. As much as possible, the demolition plan shall promote the segregation and recycling of demolition material, such as structural steel. However, no salvage value has been applied to the cost estimate of closure;
- Building foundations will be removed to the ground level;
- Progressive closure and revegetation will be implemented during the course of operation for the Dry Stacks Tailings Storage Facility (DSTSF);
- The water management infrastructure includes water collection ponds and water diversions channels. The closure concept for this infrastructure is presented in the Golder site water management report (Golder, 2016b); and

- The closure plan for the Romero Project was developed at conceptual level for the purpose of the PFS. As the project evolves towards the EIA and SIA, the closure plan will get the necessary basis to be further engineered and be expanded to social aspects.

20.8.2 Closure Guarantee

The Government of the Dominican Republic does not have any specific regulatory requirements regarding closure, such as surety bonds, credit application letter, financial assurances, etc. (World Bank, 2009). The closure cost estimate has however been developed and is provided as part of the CAPEX in Section 21.

.

21 Capital Cost Estimate

21.1 Summary and Estimate Results

LOM project capital costs total US$251M, consisting of the following distinct phases:

- Pre-production Capital Costs – includes all costs to develop the property to a 2,800 t/d production. Initial capital costs total $159M and are expended over a 36-month pre-production construction and commissioning period;
- Sustaining Capital Costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $92M and are expended in operating years 1 through 8;
- Closure Costs – includes all costs related to the closure, reclamation, and ongoing monitoring of the mine post operations. Closure costs total $11.0M (net of equipment salvage values), and are incurred in Years 9 through 13.

The capital cost estimate was compiled using a combination of quotations, database costs, and database factors. Once compiled, the overall cost estimate was top-down benchmarked against similar operations.

Table 21.1 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q3 2016 dollars with no escalation.

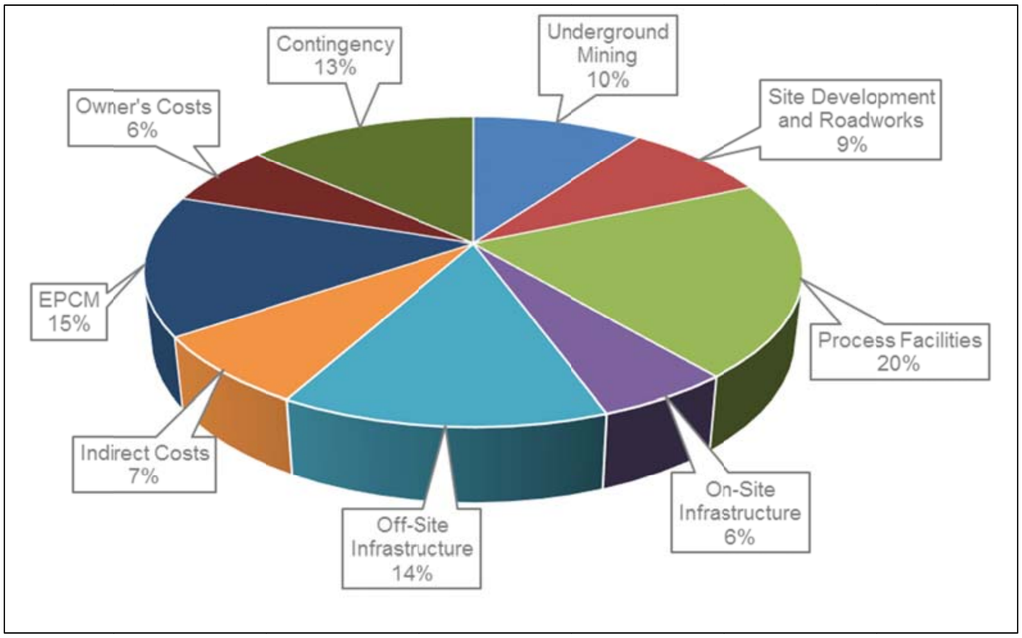
Table 21.1: Capital Cost Summary

WBS	Area	Pre-Production (M$)	Sustaining (M$)	Closure (M$)	Total (M$)
1000	Mining	15.7	57.4	-	73.1
2000	Site Development	13.5	4.0	-	17.5
3000	Process Facilities	32.4	5.2	-	37.6
4000	On-Site Infrastructure	8.8	4.1	-	13.0
5000	Off-Site Infrastructure	21.5	-	-	21.5
6000	Indirect Costs Incl. EPCM	11.8	-	-	11.8
7000	EPCM	23.2	-	-	23.2
8000	Owners Costs	10.2	-	-	10.2
	Closure Costs	-	-	15.5	15.5
	Salvage Value	-	-	(4.5)	(4.5)
	Subtotal Pre-Contingency	137.3	70.7	11.0	219.0
9000	Contingency	21.3	10.6	-	32.0
	Total Capital Costs	158.6	81.3	11.0	250.9

Source: JDS (2016)

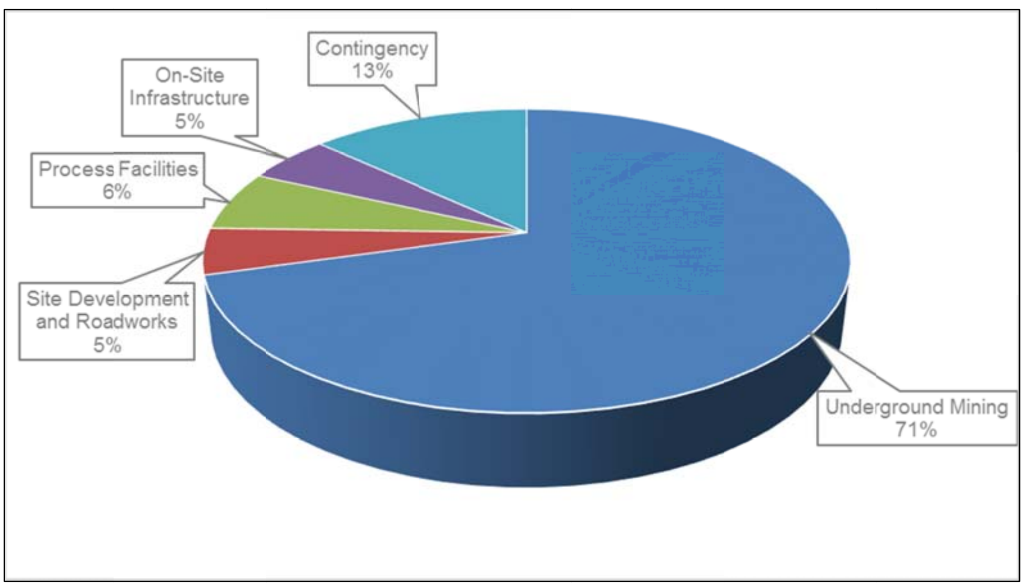
Figure 21.1 and Figure 21.2 present the capital cost distribution for the pre-production and sustaining phases. As typical with underground operations, the majority of sustaining capital costs relate to underground lateral and vertical development.

Figure 21.1: Initial Capital Cost Distribution



Source: JDS (2016)

Figure 21.2: Sustaining Capital Cost Distribution

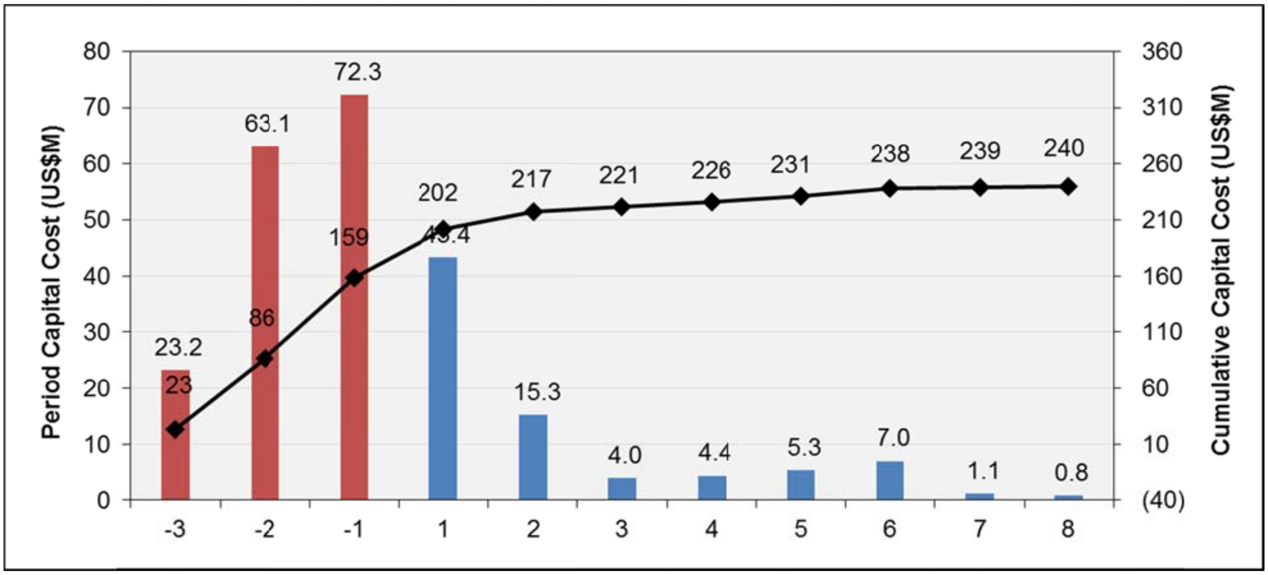


Source: JDS (2016)

21.2 Capital Cost Profile

All capital costs for the project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21.3 presents an annual LOM capital cost profile (excluding closure years).

Figure 21.3: Capital Cost Profile (Closure Years not Shown)



Source: JDS (2016)

21.3 Key Estimate Assumptions

The following key assumptions were made during development of the capital estimate:

- Underground mine development activities will initially be performed by a contractor, then phased to an owner team by operating Year 2; and
- All surface construction (including earthworks) will be performed by contractors.

21.4 Key Estimate Parameters

The following key parameters apply to the capital estimates:

- Estimate Class: The capital cost estimates are considered Class 4 estimates (-15%/+25%). The overall project definition is estimated to be 10%;
- Estimate Base Date: The base date of the estimate is September 1st, 2016. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- Units of Measure: The International System of Units (IS) is used throughout the capital estimate; and

- Currency: All capital costs are expressed in United States Dollars (US$). Portions of the estimate were estimated in other currencies and converted to US$ using the exchange rates shown in Table 21.2.

Table 21.2: Estimate Exchange Rates

Currency	Symbol	X : US$
United States Dollar	US$	1.00
Canadian Dollar	CA$	1.28
Dominican Peso	DOP	46.00
Australian Dollar	AU$	1.31

Source: JDS (2016)

21.5 Basis of Estimate

21.5.1 Labour Rates

21.5.1.1 Contract Labour Rates

Contractor labour rates were built up by applying appropriate burdens to base labour rates provided by Dominican contractors to determine all-in commodity unit labour rates.

Table 21.3: Contractor Labour Rates (US$)

Category	Civil	Conc	Struc	Arch	Mech	Pipe	Elec	Instr
Blended Direct Rate *(incl. small tools and protective personal equipment (PPE))*	12.50	15.00	20.00	20.00	22.54	20.59	15.20	17.97
Supervision and Management	2.50	3.00	4.00	4.00	4.51	4.12	3.04	3.59
Non-Productive Time	1.25	1.50	2.00	2.00	2.25	2.06	1.52	1.80
Overheads and Profit	1.63	1.95	2.60	2.60	2.93	2.68	1.98	2.34
Grand Total	**17.88**	**21.45**	**28.60**	**28.60**	**32.24**	**29.44**	**21.74**	**25.70**

Source: JDS (2016)

21.5.1.2 Operational (Owner) Labour Rates

Operational labour rates were built up from first principles, in consultation with GoldQuest. Base rates are based prevailing wages in the area, and legal premiums and benefits were built up to create all-in rates. Operational labour rates and staffing levels are described further within Section 22.

21.5.2 Fuel and Energy Supply

A delivered fuel price of $0.66/L has been used throughout the estimate, based on received budgetary quotations. An energy supply price of $0.12/kWh has been used throughout the estimate, based on preliminary discussions with the local power authority.

21.5.3 Mine Capital Costs

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers and contractor and in-house cost databases. Table 21.3 summarizes the underground mine capital cost estimate.

Table 21.4: Mine Capital Costs

WBS	Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
1100	Underground Mobile Equipment	-	33.9	33.9
1200	Underground Infrastructure	4.9	2.5	7.4
1300	Capital Development	4.0	21.0	25.0
1400	Capitalized Production Costs	0.6	-	0.6
1500	Paste Plant	6.2	-	6.2
	Total Mining (excl. Contingency)	**15.7**	**57.4**	**73.1**

Source: JDS (2016)

21.5.3.1 Underground Mobile Equipment

Underground mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities. Underground mobile equipment is supplied by the mining contractor during the pre-production phase. These equipment usage costs are included in the capital development cost area (WBS 1300). The Owner will purchase an underground fleet during the first year of operations to start production works, and supplement this fleet in Year 2 to take over capital development operations.

21.5.3.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling.

Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items, such as initial PPE, radios, water supply, refuge stations, and geotechnical investigations. Acquisition of underground infrastructure is timed to support the mine plan requirements.

21.5.3.3 Capital Development

Capital development includes the labour, fuel, equipment usage, power, and consumables costs for lateral and vertical development required for underground access to stopes and underground infrastructure. Capital development for the pre-production phase and the first two years of operations will be performed by a contractor. Budgetary quotations were received for development works from qualified contractors and applied to the units developed through the mine design process.

21.5.3.4 Capitalized Production Costs

Capitalized production costs are defined as mine operating expenses (operating development, mineralized material extraction, mine maintenance, and mine general costs) incurred by the owner prior to the introduction of feed to the processing facilities and the commencement of project revenues. They are included as a pre-production capital cost. Capitalized production costs are relatively low for this project, as many of the costs typically captured in this category are included within the contractor unit rates applied to the capital development costs in WBS 1300.

21.5.3.5 Paste Plant

A mechanical equipment list was developed for the paste plant, based on the design requirements. Budgetary quotations were received for major equipment, and database unit costs were applied to minor equipment quantities. Installation costs for mechanical, piping, electrical, and instrumentation were factored based on similar projects.

21.5.4 Site Development and Road Works

Material take-offs were developed from preliminary design drawings and 3D models for all on-site roads, pads, water management structures, and tails/waste storage facility foundations. The tailings storage facility is constructed in stages as capacity is required by the mine schedule.

Budgetary contractor unit rates for bulk earthworks, finish grading, ditching, lining, and retaining walls were obtained and applied to the material take-offs.

Table 21.5: Site Development Capital Costs

WBS	Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
2100	General Site Development and Pads	4.9	-	4.9
2200	On-Site Roads	2.5	-	2.5
2300	Surface Water Management	1.7	-	1.7
2400	Tailings Storage Facility	3.0	4.0	7.0
2500	Waste Rock Storage Facility	1.4	-	1.4
	Total Site Development and Road Works	**13.5**	**4.0**	**17.5**

Source: JDS (2016)

21.5.5 Process Plant

The process plant capital costs include all of the direct costs to construct the 2,800 t/d processing plant. A $650,000 annual allowance is applied during operations for miscellaneous sustaining projects, rebuilds, and modifications required to maintain the 2,800 t/d throughput.

Table 21.6: Process Plant Capital Costs

WBS	Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
3100	Primary Crushing	1.6	-	1.6
3200	Coarse Ore Stockpile and Reclaim	2.2	-	2.2
3300	Grinding and Gravity Concentration	8.5	-	8.5
3400	Flotation	5.2	-	5.2
3500	Regrind	2.3	-	2.3
3600	Concentrate Dewatering and Load-out	1.5	-	1.5
3700	Tailings Dewatering and Load-out	4.9	-	4.9
3800	Reagents	0.5	-	0.5
3900	Process Building and General	5.8	5.2	11.0
	Total Process Plant	**32.4**	**5.2**	**37.6**

Source: JDS (2016)

The process plant capital cost estimate was assembled form a combination of engineered take-offs, supplier quotations, contractor quotations, and database allowances. Table 21.7 presents a summary basis of estimate for the various commodity types within the process plant estimate.

Table 21.7: Process Plant Basis of Estimate

Commodity	Estimate Basis
Equipment	
Major Equipment	Budget quotations were solicited from qualified suppliers for the major equipment identified in the flow sheets and equipment register.
Minor Equipment	In-house data (firm and budgetary quotations from recent projects) was used for minor or low value equipment.
Installation (Labour and Materials)	
Concrete	Engineered take-off quantities were developed from preliminary design drawings. Budgetary quoted unit rates from local contractors were applied to design quantities.
Structural Steel including Process Plant Building	Engineered take-off quantities were developed from preliminary design drawings. Budgetary quoted unit rates from local contractors were applied to design quantities.
Mechanical Fixed Equipment	Database factor applied against mechanical equipment costs for installation.
Piping	Database cost factors applied against mechanical equipment costs.
Electrical	Engineered take-off quantities for major electrical equipment and materials were developed based on the site layouts, mechanical equipment lists, and single line diagrams. Database unit costs for supply were applied against the take-off quantities. Database factors were applied to equipment and material costs for installation.
Instrumentation and Controls	A bulk cost allowance was applied, based on similar sized process plants.

Source: JDS (2016)

21.5.6 On-Site Infrastructure

21.5.6.1 Summary

On-site infrastructure at the Romero Project includes power, water, and waste handling infrastructure, ancillary buildings (offices, mine dry, warehouses, and shops), the surface mobile support fleet, and information technology (IT) and communications systems.

Table 21.8: On-Site Infrastructure Capital Costs

WBS	Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
4100	Site Utilities	2.5	-	2.5
4200	Mine Site Ancillary Facilities	2.1	-	2.1
4300	Portal Site Ancillary Facilities	0.3	-	0.3
4400	Explosives Storage Facilities	0.2	-	0.2
4500	Surface Mobile Equipment	3.3	4.1	7.4
4600	Bulk Fuel Storage and Distribution	0.1	-	0.1
4700	IT and Communications	0.3	-	0.3
	Total On-Site Infrastructure	**8.8**	**4.1**	**13.0**

Source: JDS (2016)

21.5.6.2 Site Utilities

Site utilities include the on-site electrical substation, on-site power distribution, a chlorinator water treatment plant, and waste water treatment plant. Database unit pricing was used for these facilities.

21.5.6.3 Ancillary Facilities

Ancillary buildings are located at both the plant site and mine portal site areas. A total of 11 ancillary buildings are included in the capital estimate. These buildings are described within Section 18.

Local contractor quotations were used for the building supply/erection costs. A quotation was received for a modular assay lab, including equipment. Cost allowances were made for water supply, laydown pads, and fencing.

21.5.6.4 Surface Equipment Fleet

Surface equipment fleet requirements are determined based on material movement requirements and experience at similar operations, and considering site conditions specific to the project. Waste rock/tailing handling equipment requirements are based on equipment utilization requirements for the haulage operations. No equipment replacements are anticipated for the surface equipment fleet due to the short mine life and relatively low utilization of equipment.

Approximately half of the surface support fleet purchase is deferred until the start of operations.

A combination of quoted and database unit pricing has been applied to the surface equipment fleet quantities.

21.5.7 Off-Site Infrastructure

21.5.7.1 Summary

Off-site infrastructure is required for the project for reliable road connection for access and concentrate shipments, power connection, and port infrastructure to support concentrate handling for sea shipment to the refinery.

Table 21.9: Off-Site Infrastructure Capital Costs

WBS	Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
5100	Main Access Road	16.5	-	16.5
5200	69kV Power Transmission Line and Substation	4.2	-	4.2
5300	Pueblo Viejo Port	0.8	-	0.8
	Total Off-Site Infrastructure	**21.5**	**-**	**21.5**

Source: JDS (2016)

21.5.7.2 Main Access Road

A budgetary contractor quotation was used, based on engineered take-off quantities from preliminary design drawings.

21.5.7.3 Power Transmission Line

A budgetary estimate provided by experienced line contractor operating in the area of the project was used for the estimate.

21.5.7.4 Pueblo Viejo Port

Database unit pricing was used for a concentrate storage building. Quotations were received for the mechanical equipment (conveyor and belt feeder) required at the port.

21.5.8 Indirect and Owners Costs

21.5.8.1 Summary

Indirect costs are those that are not directly accountable to a specific cost object. Table 21.10 presents the detail of the indirect and owners costs categories.

Table 21.10: Indirect Capital Costs

WBS	Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
6100	General Construction Services	0.8	-	0.8
6200	Temporary Facilities and Utilities	1.3	-	1.3
6300	Contractor Indirects	2.0	-	2.0
6400	Logistics	5.2	-	5.2
6500	Commissioning and Start-up	2.5	-	2.5
7100	Engineering and Procurement	9.7	-	9.7
7200	Construction and Project Management	13.5	-	13.5
8200	Owners Costs - Processing Labour and Power	1.0	-	1.0
8300	Owners Costs - General and Administration	9.2	-	9.2
	Total Indirect and Owners Costs	**45.2**	**-**	**45.2**

Source: JDS (2016)

21.5.8.2 Indirect and EPCM Costs

Table 21.11 presents the basis of estimate for each of the indirect cost categories. The majority of indirect costs in the estimate are factors or allowances based on recently completed definitive estimates for similar projects.

Table 21.11: Indirect Cost Basis of Estimate

Commodity	Basis
Construction Support Services	Time based cost allowance for general construction site services (temporary power, heating and hoarding, contractor support, etc.) applied against the surface construction schedule
Temporary Facilities and Utilities	Allowance for construction offices and ablution facilities Allowance for a combination of diesel and transmission line construction power
Contractor Mobilization	Lump sum cost allowance for contractor mobilization and miscellaneous expenses; equivalent to 3.2% of the total direct contractor costs. Note that contractor profit on labour and materials are included in the direct cost unit rates
Logistics and Freight	Lump sum cost allowance for all freight and logistics; equivalent to 10.7% of the total direct material and equipment costs
Start-up and Commissioning	Lump sum allowance of $600,000 for pre-operational contractor commissioning labour. Factored allowance (2.5%) for spare parts Lump sum allowance of $500,000 for first fills and mill charges. Factored allowance (2.5%) for the provision of vendor services for commissioning support
Detailed Engineering and Procurement	Factored (15%) allowance of total direct construction costs (excluding mining)
Project and Construction Management	Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, and contract administration Database unit (hourly) rates

Source: JDS (2016)

21.5.8.3 Owners Costs

Owner's costs are items that are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements described below are described in more detail within Section 22.

- Pre-production milling: Costs of the Owner's processing labour, power, and consumables incurred before declaration of commercial production;
- Pre-production general and administration: Costs of the Owner's labour and expenses (safety, finance, security, purchasing, support labour, maintenance, equipment usage, management, etc.) incurred prior to commercial production.

21.5.9 Closure Costs and Salvage Value

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an underground mine. Typical activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMF;
- Closure of the underground mine portals;
- Access road closure;
- Power transmission line and substation removal;
- Revegetation and seeding; and
- Ongoing site monitoring.

A total lump sum closure cost of $15.5M has been used for the estimate, based on factored costs from similar underground projects. Closure costs are incurred over a five year period following the completion of operations.

Due to the relatively short mine life, a salvage value was estimated at $4.5M, which is used to offset the closure costs.

21.5.10 Cost Contingency

An overall contingency of 15% was applied to the LOM capital costs of the project. LOM project contingency amounts to $32.0M.

21.5.11 Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any project sunk costs (studies, exploration programs, etc.);
- Closure bonding; and
- Escalation cost.

22 Operating Cost Estimate

22.1 Introduction and Estimate Results

LOM operating costs for the project average $45.97/t processed. This includes the following sectors:

- Underground mining;
- Ore re-handling;
- Mineral processing; and
- General and administration.

The operating costs described in this section exclude off-site costs (such as shipping and refining costs), taxes, and government royalties. These cost elements are used to determine the NSR in the economic model, and are discussed in Section 23.

Table 22.1 presents a summary of the LOM operating costs, expressed in US$ with no escalation. Figure 22.1 illustrates the distribution of operating costs among the cost sectors.

Table 22.1: Operating Cost Summary

Sector	Average US$ M/year	LOM US$ M	$/t processed
Underground Mining	27	195	27.67
Ore Re-handling	1	9	1.28
Mineral Processing	11	81	11.58
General and Administration	5	38	5.44
Total Mine Operating Costs	**45**	**323**	**45.97**

Source: JDS (2016)

Figure 22.1: Operating Cost Distribution, by Sector

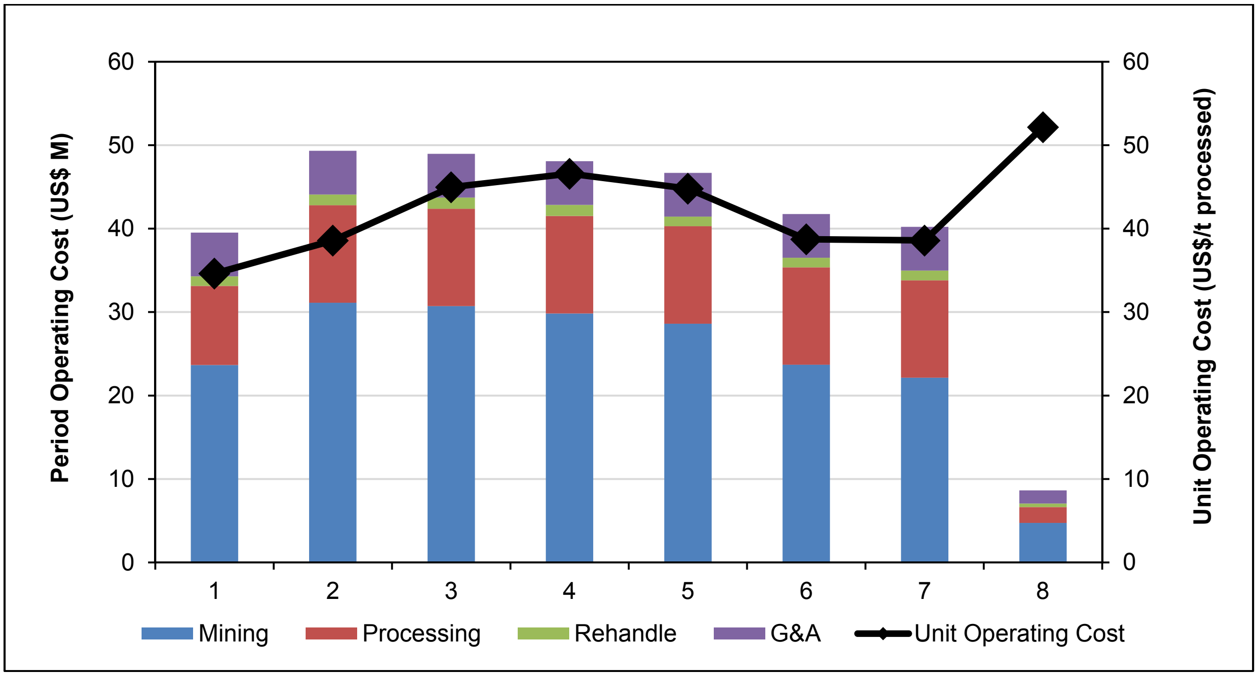
Source: JDS (2016)

The operating cost estimate was compiled utilizing input from engineers, contractors, and suppliers with experience operating projects in the area. Wherever possible, bottom up first principle estimates were developed and benchmarked against other projects of similar size with similar site conditions.

22.2 Operating Cost Profile

All operating costs have been included in the economic cash flow model according to the development schedule. Figure 22.2 presents an annual LOM operating cost profile.

Figure 22.2: Operating Cost Profile, by Year



Source: JDS (2016)

22.3 Operational Labour Rate Buildup

Operational staff labour rates have been built up by applying legal and discretionary burdens against base labour rates. Eight wage scales were defined for each sector (mining, milling, and G & A), and applied to the various operational positions based on skill level and expected salary. GoldQuest operational personnel were involved in the buildup and verification of the operational labour rates.

22.4 Mine Operating Cost Estimate

The mine operating costs are broken down into the following functional areas:

- Waste Drifting – Costs include labour, equipment parts, fuel, oil and lube, explosives and ground support and other consumables for non-capitalized lateral waste development, such as attack ramps and sub-level drifting;
- Production – Costs include labour, equipment parts, fuel, oil and lube, explosives and ground support and other consumables for lateral ore development and LH and MCF stoping;
- Backfill – Costs include labour, equipment parts, fuel, oil and lube, cement, piping and past plant labour and consumables for the production, distribution, and placement of backfill;
- Mine General – Costs include support equipment costs (parts, fuel, oil and lube), site power, technical services, definition drilling, and miscellaneous supplies; and

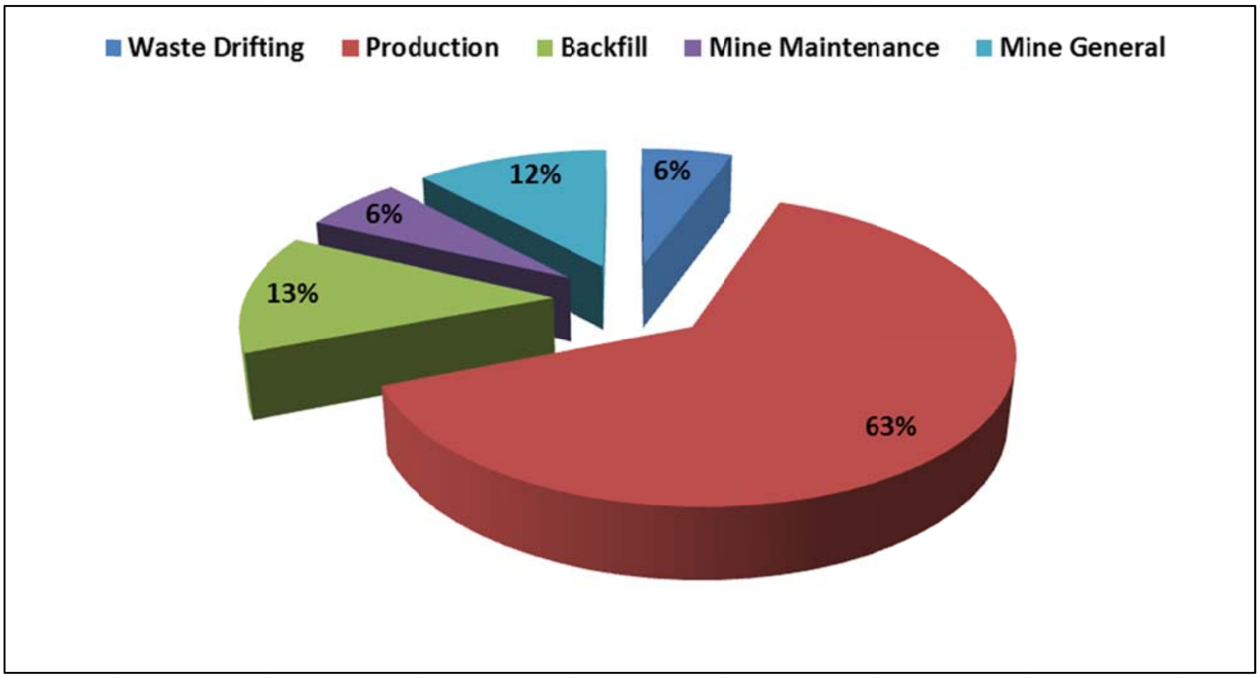
- Mine Maintenance – Costs include labour and shop consumables to maintain and repair the underground mining mobile equipment.

Table 22.2: Mine Operating Costs by Area

Total Operating Cost - By Area	LOM $M	$/tonne milled
Waste Drifting	11	1.55
Production	123	17.54
Backfill	25	3.62
Mine General	23	3.27
Mine Maintenance	12	1.68
Operating Cost - Total	**194**	**27.67**

Source: JDS (2016)

Figure 22.3: Mine Operating Cost Distribution



Source: JDS (2016)

22.4.1 Mining Labour

Mining labour was calculated using the personnel numbers summarized in Section 16.11 of this report. Labour costs are based on fully burdened staffing wage bandings, as described in Section 22.3.

Table 22.3: Mining Labour

Area	Max. Staff	LOM US$ M	US$/t processed
Mining Management	5	3.7	0.53
Mining Operations (Production)	116	18.8	2.68
Contractor Services (Expats)	21	33.9	4.83
Mining Operations (Services)	32	5.8	0.82
Mine Maintenance	24	5.7	0.82
Mine Technical Services	17	8.8	1.25
Total Mining Labour	**213**	**76.8**	**10.93**

Source: JDS (2016)

22.4.2 Equipment and Consumables

Drilling, mucking and hauling operating costs were developed from first principles from the mine plan and required equipment operating hours. Haulage profiles were developed for ore and waste rock to determine required haulage hours.

Equipment fuel and factored oil and lube consumption cost are based on Original Equipment Manufacturers (OEM) recommendations for the expected operating conditions. Parts costs were provided by OEMs based on the life expectancy of the equipment. These include the following:

- Major components (engine, torque converter, transmission, final drives, etc.);
- Major hydraulic/suspension cylinders (suspension, hoist/steering cylinders, etc.);
- Minor components (hydraulic pumps, motors, turbo chargers);
- All tools to remove and install components;
- Preventative maintenance (including filters, seals, screens, midlives);
- System parts (hydraulic, steering, transmission, cooling, cab, rear axle, suspension, brake, front axle, enclosures);
- Hoses and fittings; and
- Electrical wiring, sensors.

Life expectancy for major underground mine equipment is summarized in Table 22.4.

.

Table 22.4: Major Equipment Life Expectancy

Equipment Type	Expected Life (Hours)
Two Boom Jumbo	18,000
LH Drill	15,000
6 m^3 LHD with Remote	17,500
40 Tonne Truck	20,000
Mechanized Bolter	15,000
ANFO Loader	20,000

Source: JDS (2016)

Tire replacement costs are included within the equipment unit rates and are based on expected tire life hours. Management of tires is considered to be of critical importance for the operation of the mine. Allowances for cleanup of drift floors and roadways, plus a grader, are included in mining costs. Table 22.5 summarizes the major underground equipment tire life expectancy, while major underground equipment operating costs per hour, excluding labour and drill tooling, are shown in Table 22.6.

Table 22.5: Major Equipment Tire Life Expectancy

Equipment Type	Expected Life (Hours)
6 m^3 LHD with Remote	1,500
40 Tonne Truck	3,500

Source: JDS (2016)

Table 22.6: Major Underground Equipment Hourly Operating Cost

Equipment Type	Fuel $/hr	Oil/Lube $/hr	Parts $/hr	Tires $/hr	Total $/hr
Two Boom Jumbo	2.90	1.88	75.09	1.51	81.38
LH Drill	3.13	2.52	100.82	1.73	108.20
6 m^3 LHD with Remote	30.33	2.11	42.21	16.43	114.29
40 Tonne Truck	47.08	2.58	51.63	6.31	111.45
Mechanized Bolter	2.66	1.07	42.87	1.14	47.74
ANFO Loader	3.38	0.47	9.36	0.24	12.30

Source: JDS (2016)

Consumables usage was based on required drift and stope services, explosives quantities, ground support patterns and drilling equipment tooling. Consumables usage by major drift and stope types are summarized in Table 22.7.

Table 22.7: Underground Mining Consumables Unit Costs

Excavation Type	Units	Drilling	Blasting	Support	Shotcrete	Services	Total
Ramp	$/m	39	186	303	7	182	717
Footwall	$/m	39	183	303	7	182	714
Level Access	$/m	39	189	313	20	182	744
Large Service Drift	$/m	39	183	345	58	174	798
Small Service Drift	$/m	30	138	253	12	-	432
Large Cross-Cut	$/m	41	195	307	15	84	643
Small Cross-Cut	$/m	31	146	241	-	84	502
Cut and Fill Ramp	$/m	40	194	303	35	177	749
Stope Sub-level Waste	$/m	34	170	288	2	42	535
Stope Sub-level Ore	$/tonne	0.73	3.47	7.63	2.28	0.67	14.78
Cut and Fill Underhand (P)*	$/tonne	0.76	3.80	6.62	5.98	0.75	17.91
Cut and Fill Underhand (S)*	$/tonne	0.76	3.82	6.17	-	0.75	11.51
Cut and Fill Overhand (P)	$/tonne	0.76	3.80	5.50	9.29	0.84	20.19
Cut and Fill Overhand (S)	$/tonne	0.76	3.82	5.50	9.05	0.85	19.98
LH Large (P)	$/tonne	0.52	0.29	0.01	-	0.60	1.42
LH Large (S)	$/tonne	0.50	0.28	0.03	-	0.60	1.42
LH Small (P)	$/tonne	0.52	0.32	0.00	-	0.60	1.45
LH Small (S)	$/tonne	0.50	0.31	0.03	-	0.60	1.45
LH Drop Raise	$/m	44.18	50.47	30.15	-	-	125

*(P) Primary, (S) Secondary
Source: JDS (2016)

22.4.3 Backfill

Backfill costs were based on an average cement content of 5.8% by weight. Other consumables used include pipe and barricades for paste fill distribution, and the parts, fuel and lubricants required for mobile equipment to place rock fill.

22.5 Re-Handle Operating Cost Estimate

The re-handle operating cost estimate includes costs to perform the following activities:

- Load ore from the portal area, haul, and dump it at the run of mine stockpile located near the ore crushing area; and
- Load waste from the waste rock storage facility, haul, and dump it at the mine portal area for re-handling by underground equipment and eventual use in cut/fill operations.

Costs have been assembled from first principles, based on the requirements of the production schedule and the calculated equipment operating hours.

Table 22.8: Re-Handle Operating Costs

Capital Costs	Total LOM Staff	Average US$ M/year	LOM US$ M	US$/t processed
Labour Costs	16	0.4	2.6	0.36
Equipment Maintenance Costs	-	0.5	3.6	0.51
Equipment Fuel Costs	-	0.4	2.8	0.41
Total Re-Handle Operating Cost	**16**	**1.3**	**9.0**	**1.28**

Source: JDS (2016)

22.6 Processing Operating Cost Estimate

22.6.1 Mineral Processing Labour

Milling operations and maintenance staffing levels have been built up based on experience at similar operations. Labour costs are based on fully burdened staffing wage bandings, as described in Section 22.3.

Table 22.9: Processing Labour

Area	Total LOM Staff	Average US$ M/year	LOM US$ M	US$/t processed
Mill Management	7	0.4	2.7	0.38
Primary Crusher and Reclaim	4	0.1	0.6	0.08
Process Plant	20	0.6	4.0	0.57
Assay Laboratory	13	0.4	2.5	0.35
Process Maintenance	21	0.8	5.4	0.77
Dry Stack Tailing Facility Operations	9	0.2	1.7	0.24
Total Processing Labour	**74**	**2.5**	**16.9**	**2.40**

Source: JDS (2016)

22.6.2 Mineral Processing Power

Electrical power consumption has been based on the equipment connected loads, discounted for operating time and the anticipated operating load level.

Estimated total annual electricity cost within the processing facilities is $4.3M, or $4.22/tonne processed at a unit rate of $0.12/kWh.

22.6.3 Mineral Processing Consumables

Grinding media and liners have been estimated on a kilogram/tonne basis, based on experience at similar operations.

Milling reagent consumption rates have been determined from the metallurgical test data or experience from other operations (when test data was not available). Unit pricing is based on budgetary quotations.

Maintenance parts costs have been factored based on the direct capital costs of the equipment within each area.

Table 22.10: Processing Consumables

Item	Average US$ M/year	LOM US$ M	US$/t processed
Grinding Media	1.4	9.8	1.39
Liners and Wear Parts	1.0	6.6	0.94
Reagents	0.7	5.1	0.73
Maintenance Parts	0.7	4.6	0.65
Assay Lab Consumables	0.3	1.9	0.26
Total Processing Consumables	**4.1**	**27.9**	**3.97**

Source: JDS (2016)

22.6.4 Tailing Facility Equipment Operations

Tailing facility equipment operating costs include the costs to load, transport, place, and compact dried tailing material from the tailing stockpile building to the Tailing Storage Facility ("TSF") using articulated surface haul trucks.

Estimated average annual equipment operating costs are $1.0M, or $0.99/t processed.

22.7 General and Administration Operating Cost Estimate

22.7.1 General and Administration Labour

General and administration staffing levels have been built up based on experience at similar operations. Labour costs are based on fully burdened staffing wage bandings, as described in Section 22.3.

Table 22.11: General and Administration Labour

Area	Total LOM Staff	Average US$ M/year	LOM US$ M	US$/t processed
Management and Administration	2	0.2	1.7	0.24
Accounting	3	0.1	0.8	0.11
Human Resources and Training	6	0.3	2.0	0.28
Community Relations	1	0.1	0.2	0.03
IT and Communications	1	0.1	0.3	0.04
Procurement and Logistics	7	0.2	1.7	0.24
Environment	8	0.3	2.3	0.32
Security	8	0.1	1.0	0.14
Surface Infrastructure and Maintenance	20	0.6	4.0	0.57
Total G & A Labour	**56**	**1.9**	**13.9**	**1.97**

Source: JDS (2016)

22.7.2 General and Administration Services and Expenses

G & A services and expenses have been estimated in consultation with GoldQuest area managers, and considering other similar operations. Major items (logistics, mobile equipment, and insurance) are built up from first principles. Minor items are factored, based on other estimate parameters (such as number of staff) or are general allowances.

Table 22.12: G & A Services

Item	Average US$ M/year	LOM US$ M	US$/t processed
Health Safety, Medicals and First Aid	0.3	1.8	0.26
Surface Support Equipment	0.6	4.4	0.62
Surface Infrastructure Power	0.3	2.3	0.33
Facilities Maintenance	0.1	0.9	0.13
3rd Party Support Services	0.1	0.9	0.13
Environmental	0.3	1.8	0.26
Human Resources	0.2	1.3	0.18
Operations Insurance	0.7	5.3	0.75
Community Relations	0.1	0.7	0.10
Legal and Insurance	0.2	1.4	0.21
External Consulting	0.2	1.4	0.21
IT and Communications	0.2	1.4	0.20
Site Office	0.1	0.7	0.10
Total G & A Services	**3.4**	**24.4**	**3.47**

Source: JDS 2016

22.8 Contingency

No operating cost contingency provision has been included in the estimate.

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 2016 dollars). The economic analysis has been run with no inflation (constant dollar basis).

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.

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Table 23.1: LOM Plan Summary

Summary of Results	Unit	Value
Probable Reserves	kt	7,031
Cu	%	0.88
Au	g/t	3.72
Ag	g/t	4.33

Source: JDS (2016)

23.3 Assumptions

The following economic assumptions were used in the economic analysis:

- Discount rate of 5% (sensitivities using other discount rates have been calculated) - refer to Section 23.6;
- Closure cost of $11.0 M was considered (net of salvage value of $4.5 M);
- Nominal 2016 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, rehandle, processing, and G & A) in Year 1 (assumed to be required in Year -1). The working capital is recuperated during the last year of production (Year 8);
- 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 23.3 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 is 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 and F/X Rate	Unit	Value
Cu Price	US$/lb	2.50
Au Price	US$/oz	1,300
Ag Price	US$/oz	20.00
F/X Rate	US$:C$	0.78

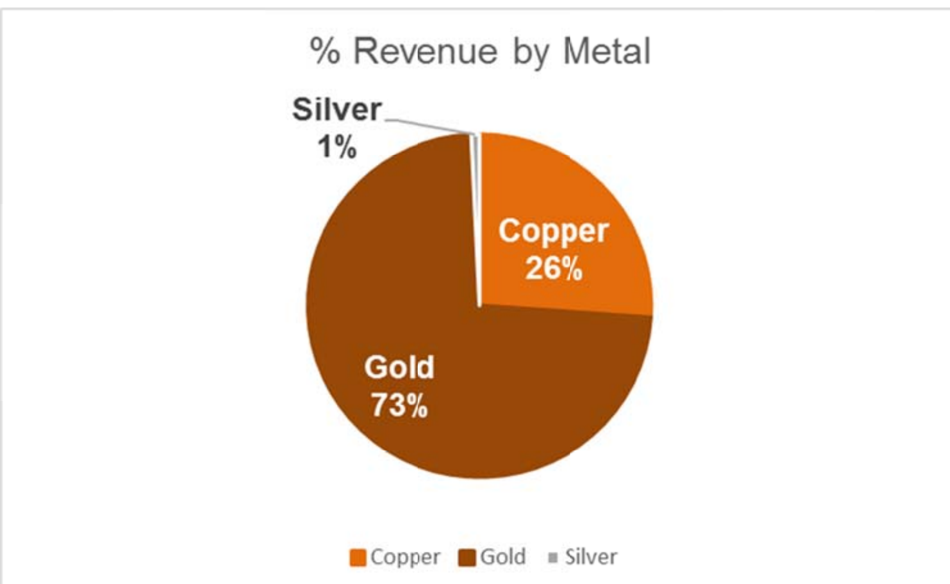
Source: JDS 2016

23.4 Revenues

Mine revenue is derived from the sale of copper concentrate into the international marketplace. No contractual arrangements 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) $1,032M over the approximately 7-year mine life.

Figure 23.1: Payable Metal by Value



Source: JDS (2016)

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 $149M.

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 units of production basis was considered on all capital expenditures beginning in Year 2;
- A net asset tax of 0.5% is not considered at the project level;
- A corporate income tax of 27%;
- A maximum of 20% loss carryforward per year;
- An export withholding tax of 5%, with the same amount credited against corporate taxes payable; and
- A local community tax of 5%.

23.6 Results

The project is economically viable with an after-tax IRR of 28.2% and a net present value using a 5% discount rate ($NPV_{5\%)}$ of $203M using the Base Case metal prices. Table 23.3 summarizes the economic results of the project.

The break-even gold price for the project (using the Base Case metal prices for the after-tax NPV) is approximately $724/oz, based on LOM presented herein and a copper price of US$2.50/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$M LOM	1,137
Total Operating Cost	US$/t milled	45.97
	US$M LOM	323
Net Operating Income	US$M LOM	709
Pre-Production Capital (Incl. Contingency)	US$M	159
Sustaining Capital (Incl. Contingency	US$M	92
Total Capital (Incl. Contingency)	US$M	251
LOM Pre-Tax Free Cash Flow	US$M	458
Average Annual Pre-Tax Free Cash Flow	US$M/a	64
Pre-Tax $NPV_{5\%}$	US$M	317
Pre-Tax IRR	%	38.7
Pre-Tax Payback	Years	1.9
NPV to Pre-Production CAPEX	times	2.0
Taxes	US$M	149
LOM After-Tax Free Cash Flow	US$M	309
Average Annual After-Tax Free Cash Flow	US$M/a	43
After-Tax $NPV_{5\%}$	US$M	203
After-Tax IRR	%	28.2
After-Tax Payback	Years	2.5
Break-Even Au Price‡ (for after-tax NPV)	US$/Au oz	724
Cash Cost*	US$/Au oz	669
Cash Cost Net of By-Products**	US$/Au oz	191
All-In Sustaining Cost per Ounce Au, Net of By-products (AISC)	US$/Au oz	595

(‡) Based on constant Cu price of US$2.50/lb

(‡) Based on constant Cu price of US$2.50/lb, Ag price of US$20.00/oz

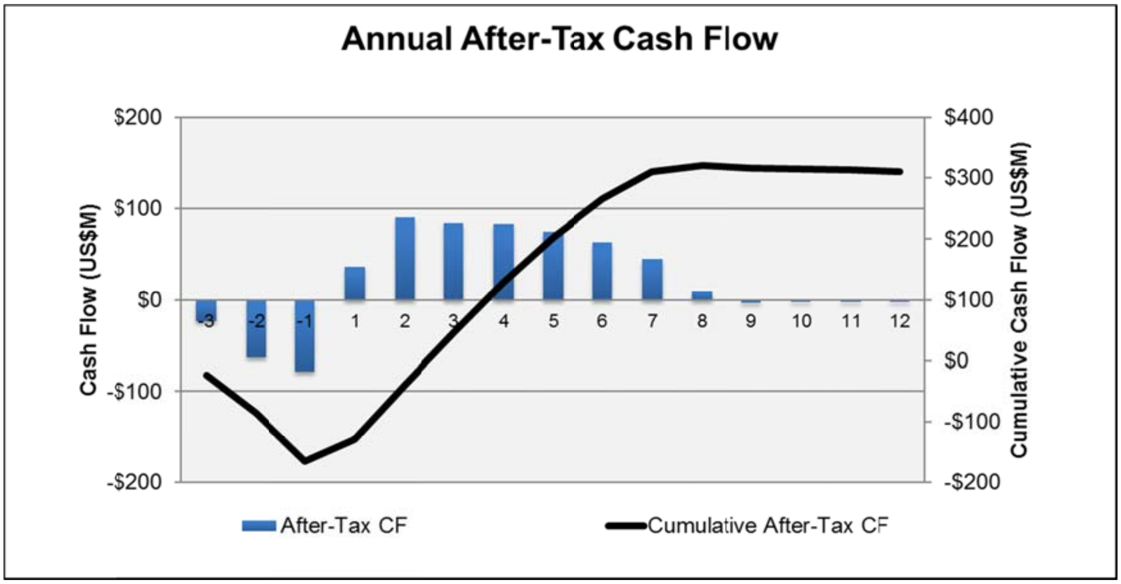
(*) Cash Cost = (Treatment Charges + Refining Charges + Concentrate Handling and Shipping + Royalties + Operating Costs)/Payable Au oz

(**) Cash Cost Net of By-Products = ((Treatment Charges + Refining Charges + Concentrate Handling and Shipping + Royalties + Operating Costs) – (Payable Cu lbs * 2.50/lb) – (Payable Ag oz * $20/oz)) / Payable Au oz

(***) Au oz equivalent payable is calculated by the following: Au oz payable + ((Cu lbs payable * $2.50/lb)+(Ag oz payable * $20/oz))/$1,300oz)

Source: JDS (2016)

Figure 23.2: Annual After-Tax Cash Flows



Source: JDS (2016)

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 $NPV_{5\%}$ for the Base Case evaluated.

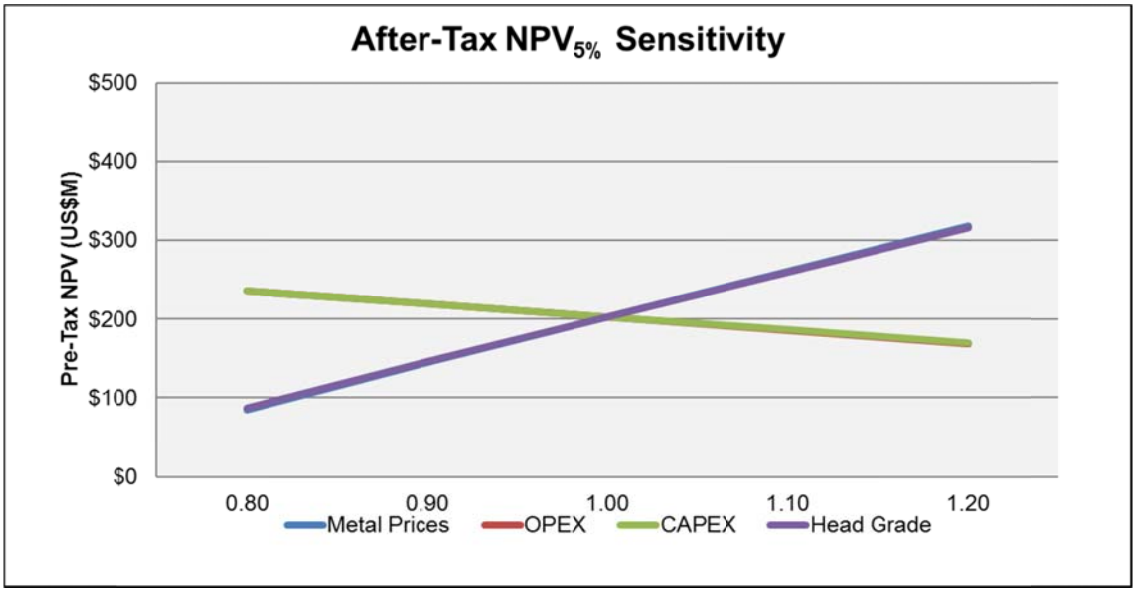
The project was also tested under various discount rates. The results of this sensitivity test are demonstrated in Table 23.4.

Table 23.4: After-Tax Sensitivity Test Results

After-Tax $NPV_{5\%}$ (US$M)					
Variable	80%	90%	100%	110%	120%
Metal Prices	84	145	203	260	318
Head Grade	87	146	203	259	316
OPEX	236	219	203	186	169
CAPEX	235	219	203	186	170

Source: JDS (2016)

Figure 23.3: After-Tax Sensitivity Test Results



Source: JDS (2016)

Table 23.5: Discount Rate Sensitivity Test Results

Discount Rate %	Pre-Tax NPV (US$M)	After-Tax NPV (US$M)
0	458	309
5	**317**	**203**
8	254	155
10	219	129
12	188	106

Source: JDS (2016)

Table 23.6: Economic Model

Please see next page for Economic Model.

Parameters	Source	Unit	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
KEY INPUTS																			
Metal Prices																			
Cu	link	US$/lb	**2.50**	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50
Au	link	US$/oz	**1,300**	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Ag	link	US$/oz	**20.00**	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
PRODUCTION SCHEDULE																			
Mined Material																			
Ore Mined	link	ktonnes	**7,031**			-	818	1,008	1,008	1,008	1,008	1,008	1,008	165	-	-	-	-	-
Waste Mined	link	ktonnes	**940**			101	324	271	81	24	34	70	34	1					
Total Mined	**calc**	**ktonnes**	**7,971**	**-**	**-**	**101**	**1,142**	**1,279**	**1,089**	**1,032**	**1,042**	**1,078**	**1,042**	**166**	**-**	**-**	**-**	**-**	**-**
Mined Grades																			
Cu	link	%	**0.88%**			-	0.86%	0.83%	0.96%	0.96%	0.89%	0.80%	0.86%	0.78%	-	-	-	-	-
Au	link	g/t	**3.72**			-	4.54	4.85	4.06	3.96	3.66	3.23	2.18	1.80	-	-	-	-	-
Ag	link	g/t	**4.33**			-	4.97	3.83	3.52	5.33	5.31	3.85	3.90	2.82	-	-	-	-	-
PROCESSING SCHEDULE																			
Ore Throughput																			
Total Mill Feed	link	ktonnes	**7,031**	-	-	-	818	1,008	1,008	1,008	1,008	1,008	1,008	165	-	-	-	-	-
Operating Days	input	days	**2,614**	-	-	-	365	365	365	365	365	365	365	59	-	-	-	-	-
Plant Throughput	calc	tpd	**2,702**	-	-	-	2,241	2,762	2,762	2,762	2,762	2,762	2,762	2,800	-	-	-	-	-
Head Grades																			
Cu	calc	%	**0.88%**	-	-	-	0.86%	0.83%	0.96%	0.96%	0.89%	0.80%	0.86%	0.78%	-	-	-	-	-
Au	calc	g/t	**3.72**	-	-	-	4.54	4.85	4.06	3.96	3.66	3.23	2.18	1.80	-	-	-	-	-
Ag	calc	g/t	**4.33**	-	-	-	4.97	3.83	3.52	5.33	5.31	3.85	3.90	2.82	-	-	-	-	-
Au Equiv	calc	g/t	**4.94**	-	-	-	5.75	6.00	5.38	5.31	4.91	4.34	3.38	2.87	-	-	-	-	-
Contained Metal																			
Cu	calc	ktonnes	**62**	-	-	-	7	8	10	10	9	8	9	1	-	-	-	-	-
	calc	Mlbs	**135.9**	-	-	-	15.5	18.5	21.3	21.3	19.7	17.7	19.2	2.8	-	-	-	-	-
Au	calc	kg	**26,135**	-	-	-	3,714	4,886	4,096	3,997	3,689	3,256	2,201	296	-	-	-	-	-
	calc	koz	**840**	-	-	-	119	157	132	128	119	105	71	10	-	-	-	-	-
Ag	calc	kg	**30,470**	-	-	-	4,066	3,856	3,551	5,375	5,350	3,877	3,931	464	-	-	-	-	-
	calc	koz	**980**	-	-	-	131	124	114	173	172	125	126	15	-	-	-	-	-
Au Equiv	calc	koz	**1116.75**	-	-	-	151.15	194.49	174.41	172.06	159.21	140.64	109.59	15.20	-	-	-	-	-
Recovery to Bulk Concentrate																			
Overall Recovery	calc	Cu %	**94.6%**				94.6%	94.5%	94.8%	94.8%	94.6%	94.4%	94.6%	94.3%					
	calc	Au %	**78.1%**				79.3%	79.6%	78.7%	78.5%	78.0%	77.2%	73.8%	71.5%					
	link	Ag %	**58.6%**	-	-	-	58.6%	58.6%	58.6%	58.6%	58.6%	58.6%	58.6%	58.6%	-	-	-	-	-
Metal in Concentrate	calc	Cu ktonnes	**58.3**	-	-	-	6.6	7.9	9.2	9.1	8.5	7.6	8.2	1.2	-	-	-	-	-
	calc	Cu Mlbs	**128.6**	-	-	-	15	17	20	20	19	17	18	3	-	-	-	-	-
	calc	Au kg	**20,423**	-	-	-	2,944	3,889	3,222	3,139	2,879	2,514	1,624	212	-	-	-	-	-
	calc	Au koz	**657**	-	-	-	95	125	104	101	93	81	52	7	-	-	-	-	-
	calc	Ag kg	**17,855**	-	-	-	2,383	2,260	2,081	3,150	3,135	2,272	2,304	272	-	-	-	-	-
	calc	Ag koz	**574**	-	-	-	77	73	67	101	101	73	74	9	-	-	-	-	-
Bulk Concentrate Grade	link	Cu %	**13%**	-	-	-	13%	13%	13%	13%	13%	13%	13%	13%	-	-	-	-	-
	calc	Au g/t	**45.5**	-	-	-	57.7	63.9	45.7	44.6	44.2	43.1	25.7	22.7	-	-	-	-	-
	calc	Ag g/t	**39.8**	-	-	-	46.7	37.1	29.5	44.8	48.1	39.0	36.4	29.1	-	-	-	-	-
Bulk Concentrate Produced	**calc**	**dmt**	**448,705**	**-**	**-**	**-**	**51,016**	**60,832**	**70,437**	**70,340**	**65,169**	**58,285**	**63,291**	**9,335**	**-**	**-**	**-**	**-**	**-**
	calc	**wmt**	**487,723**	**-**	**-**	**-**	**55,452**	**66,122**	**76,562**	**76,456**	**70,835**	**63,354**	**68,795**	**10,146**	**-**	**-**	**-**	**-**	**-**
Mass Factor	calc		**16**	-	-	-	16	17	14	14	15	17	16	18	-	-	-	-	-
Moisture Content	link	%	**8%**	-	-	-	8%	8%	8%	8%	8%	8%	8%	8%	-	-	-	-	-
NET SMELTER RETURN																			
Payable Metals																			
Cu Payable	link	%	**96.5%**	-	-	-	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	-	-	-	-	-
Cu Min. Deduction	link	%	**1%**	-	-	-	1%	1%	1%	1%	1%	1%	1%	1%	-	-	-	-	-
Payable Based on Cu Payable	calc	%	**13%**	-	-	-	13%	13%	13%	13%	13%	13%	13%	13%	-	-	-	-	-
Payable Based on Min. Deduc	calc	%	**12%**	-	-	-	12%	12%	12%	12%	12%	12%	12%	12%	-	-	-	-	-
Payable Cu in Bulk Concentrate	calc	Mlbs	**118.7**	-	-	-	13.5	16.1	18.6	18.6	17.2	15.4	16.7	2.5	-	-	-	-	-
	calc	US$M	**296.8**	-	-	-	33.7	40.2	46.6	46.5	43.1	38.5	41.9	6.2	-	-	-	-	-
Au Payable	link	%	**97.5%**	-	-	-	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	96.5%	96.5%	-	-	-	-	-
Au Min. Deduction	link	g/t	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Au Payable in Bulk Conc	calc	koz	**639.6**	-	-	-	92.3	121.9	101.0	98.4	90.3	78.8	50.4	6.6	-	-	-	-	-
	calc	US$M	**831.5**	-	-	-	120.0	158.5	131.3	127.9	117.3	102.5	65.5	8.5	-	-	-	-	-
Ag Payable	link	%	**90.0%**	-	-	-	90%	90%	0%	90%	90%	90%	90%	0%	-	-	-	-	-
Ag Min. Deduction	link	g/t	**0.0**	-	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-	-	-	-	-
Ag Payable in Bulk Conc	calc	koz	**434.2**	-	-	-	68.9	51.0	-	91.1	90.7	65.7	66.7	-	-	-	-	-	-
	calc	US$M	**8.7**	-	-	-	1.4	1.0	-	1.8	1.8	1.3	1.3	-	-	-	-	-	-
Total Payable Metals	**calc**	**US$M**	**1,136.9**	**-**	**-**	**-**	**155.1**	**199.7**	**177.9**	**176.2**	**162.2**	**142.3**	**108.7**	**14.7**	**-**	**-**	**-**	**-**	**-**
AuEq oz payable	**calc**	**koz**	**874.6**				**119.3**	**153.6**	**136.8**	**135.6**	**124.8**	**109.5**	**83.6**	**11.3**					
Refining and Transportation Costs																			
Cu TC	link	US$/dmt	**85.00**	-	-	-	85.00	85.00	85.00	85.00	85.00	85.00	85.00	85.00	-	-	-	-	-
	calc	US$M	**38.1**	-	-	-	4.3	5.2	6.0	6.0	5.5	5.0	5.4	0.8	-	-	-	-	-
Cu RC	link	US$/pay lb	**0.085**	-	-	-	0.085	0.085	0.085	0.085	0.085	0.085	0.085	0.085	-	-	-	-	-
	calc	US$M	**10.1**	-	-	-	1.1	1.4	1.6	1.6	1.5	1.3	1.4	0.2	-	-	-	-	-
Au RC	link	US$/pay oz	**5.00**	-	-	-	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	-	-	-	-	-
	calc	US$M	**3.2**	-	-	-	0.5	0.6	0.5	0.5	0.5	0.4	0.3	0.0	-	-	-	-	-
Ag RC	link	US$/pay oz	**0.50**	-	-	-	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	-	-	-	-	-
	calc	US$M	**0.2**	-	-	-	0.0	0.0	-	0.0	0.0	0.0	0.0	-	-	-	-	-	-
Concentrate Handling and Shipping	link	US$/dmt	**88.93**	-	-	-	88.93	88.93	88.93	88.93	88.93	88.93	88.93	88.93	-	-	-	-	-
	calc	US$M	**39.9**	-	-	-	4.5	5.4	6.3	6.3	5.8	5.2	5.6	0.8	-	-	-	-	-
Cu Conc NSR	**calc**	**US$M**	**1,045.4**	**-**	**-**	**-**	**144.6**	**187.1**	**163.6**	**161.9**	**149.0**	**130.4**	**96.0**	**12.8**	**-**	**-**	**-**	**-**	**-**
Royalties																			
NSR Royalty	link	% NSR	**1.25%**	-	-	-	1.25%	1.25%	1.25%	1.25%	1.25%	1.25%	1.25%	1.25%	-	-	-	-	-
	calc	US$M	**13.1**	-	-	-	1.8	2.3	2.0	2.0	1.9	1.6	1.2	0.2	-	-	-	-	-
NSR After-Royalties	**calc**	**US$M**	**1,032.3**	**-**	**-**	**-**	**142.8**	**184.8**	**161.5**	**159.9**	**147.1**	**128.8**	**94.8**	**12.7**	**-**	**-**	**-**	**-**	**-**
	calc	**US$/t milled**	**146.83**	**-**	**-**	**-**	**174.57**	**183.34**	**160.23**	**158.60**	**145.92**	**127.80**	**94.02**	**77.00**	**-**	**-**	**-**	**-**	**-**

Parameters	Source	Unit	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
OPERATING COSTS																			
Mining	calc	US$/t milled	**27.67**				28.94	30.88	30.47	29.60	28.38	23.50	21.97	28.71	-	-	-	-	-
	link	US$M	**194.5**				23.7	31.1	30.7	29.8	28.6	23.7	22.1	4.7					
Processing	link	US$/t milled	**11.58**	-	-	-	11.58	11.58	11.58	11.58	11.58	11.58	11.58	11.58	-	-	-	-	-
	calc	US$M	**81.4**	-	-	-	9.5	11.7	11.7	11.7	11.7	11.7	11.7	1.9	-	-	-	-	-
Rehandle	link	US$/t milled	**1.28**	-	-	-	1.28	1.28	1.28	1.28	1.28	1.28	1.28	1.28	-	-	-	-	-
	calc	US$M	**9.0**				1.1	1.3	1.3	1.3	1.2	1.2	1.2	0.4	-	-	-	-	-
G&A	calc	US$/t milled	**5.44**	-	-	-	6.41	5.20	5.20	5.20	5.20	5.20	5.20	9.62	-	-	-	-	-
	link	US$M	**38.3**	-	-	-	5.2	5.2	5.2	5.2	5.2	5.2	5.2	1.6	-	-	-		
Total OPEX	**calc**	**US$/t milled**	**45.97**	**-**	**-**	**-**	**48.33**	**48.93**	**48.58**	**47.71**	**46.31**	**41.43**	**39.89**	**52.43**	**-**	**-**	**-**	**-**	**-**
	calc	**US$M**	**323.2**	**-**	**-**	**-**	**39.5**	**49.3**	**49.0**	**48.1**	**46.7**	**41.8**	**40.2**	**8.6**	**-**	**-**	**-**	**-**	**-**
Au Cash Cost	*calc*	*US$/oz*	*669*	-	-	-	*562*	*527*	*647*	*655*	*685*	*701*	*1,074*	*1,624*	-	-	-	-	-
Au Cash Cost (Net of BP)	*calc*	*US$/oz*	*191*	-	-	-	*181*	*189*	*186*	*164*	*187*	*195*	*217*	*684*	-	-	-	-	-
AuEq Cash Cost	*calc*	*US$/oz*	*489*	-	-	-	*435*	*418*	*478*	*476*	*495*	*505*	*647*	*942*	-	-	-	-	-
INCOME																			
Net Operating Income	**calc**	**US$M**	**709.13**	**-**	**-**	**-**	**103.2**	**135.5**	**112.5**	**111.8**	**100.4**	**87.1**	**54.6**	**4.0**	**-**	**-**	**-**	**-**	**-**
	calc	**US$/t milled**	**747.87**	**-**	**-**	**-**	**126.2**	**134.4**	**111.7**	**110.9**	**99.6**	**86.4**	**54.1**	**24.6**	**-**	**-**	**-**	**-**	**-**
CAPTIAL COSTS																			
Underground Mining	link	US$M	**73.1**	-	1.6	14.2	30.5	12.7	2.8	1.6	4.0	5.4	0.3	0.0	-	-	-	-	-
Site Development and Roadworks	link	US$M	**17.5**	0.3	11.0	2.2	2.4	-	-	1.6	-	-	-	-	-	-	-	-	-
Process Facilities	link	US$M	**37.6**	-	13.0	19.4	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	-	-	-	-	-
On-Site Infrastructure	link	US$M	**13.0**	-	3.5	5.4	4.1	-	-	-	-	-	-	-	-	-	-	-	-
Off-Site Infrastructure	link	US$M	**21.5**	12.4	8.3	0.8	-	-	-	-	-	-	-	-	-	-	-	-	-
Indirect Costs	link	US$M	**11.8**	-	4.7	7.2	-	-	-	-	-	-	-	-	-	-	-	-	-
EPCM	link	US$M	**23.2**	7.3	10.1	5.8	-	-	-	-	-	-	-	-	-	-	-	-	-
Owner's Costs	link	US$M	**10.2**	0.3	2.8	7.2	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure	input	US$M	**15.5**	-	-	-									7.5	2.0	2.0	2.0	2.0
Salvage	link	US$M	**-4.5**												(4.5)				
Subtotal	calc	US$M	219.0	20.2	54.8	62.2	37.7	13.3	3.5	3.8	4.7	6.1	1.0	0.7	3.0	2.0	2.0	2.0	2.0
Contingency	link	US$M	**32.0**	3.0	8.2	10.1	5.7	2.0	0.5	0.6	0.7	0.9	0.1	0.1	-	-	-	-	-
Total CAPEX	**calc**	**US$M**	**250.9**	**23.2**	**63.1**	**72.3**	**43.4**	**15.3**	**4.0**	**4.4**	**5.3**	**7.0**	**1.1**	**0.8**	**3.0**	**2.0**	**2.0**	**2.0**	**2.0**
Pre-Production	*link*	*US$M*	*158.6*	*23.2*	*63.1*	*72.3*	-	-	-	-	-	-	-	-	-	-	-	-	-
Production	*link*	*US$M*	*92.3*				*43.4*	*15.3*	*4.0*	*4.4*	*5.3*	*7.0*	*1.1*	*0.8*	*3.0*	*2.0*	*2.0*	*2.0*	*2.0*
WORKING CAPITAL																			
Working Capital	calc	US$M	0.0			6.6								-6.6					
ROYALTY BUYOUT OPTION																			
Royalty Buyout Option	input	US$M	0.0			0.0													
TAXES																			
Taxes	link	US$M	149.4		-	-	23.6	30.3	24.2	24.2	20.8	16.7	8.9	0.6	-	-	-	-	-
CASH FLOWS																			
Pre-Tax																			
Net Pre-Tax Cash Flow	calc	US$M	458.2	(23.2)	(63.1)	(78.9)	59.9	120.2	108.5	107.4	95.1	80.1	53.4	9.9	(3.0)	(2.0)	(2.0)	(2.0)	(2.0)
Cumulative Pre-Tax Cash Flow	calc	US$M		(23.2)	(86.3)	(165.2)	(105.4)	14.8	123.4	230.7	325.8	405.9	459.3	469.2	466.2	464.2	462.2	460.2	458.2
After-Tax																			
Net After-Tax Cash Flow	calc	US$M	**308.8**	(23.2)	(63.1)	(78.9)	36.3	89.9	84.3	83.2	74.3	63.3	44.5	9.2	(3.0)	(2.0)	(2.0)	(2.0)	(2.0)
Cumulative After-Tax Cash Flow	calc	US$M		(23.23)	(86.3)	(165.2)	(129.0)	(39.0)	45.3	128.5	202.7	266.1	310.6	319.8	316.8	314.8	312.8	310.8	308.8
ECONOMIC RESULTS																			
Pre-Tax																			
Pre-Tax IRR	**calc**	**%**	**38.7%**																
Pre-Tax Payback	**calc**	**Years**	**1.9**																
Pre-Tax NPV @ 5%	**calc**	**US$M**	**317.2**																
Pre-Tax NPV @ 0%	**calc**	**US$M**	**458.2**																
After-Tax																			
After-Tax IRR	**calc**	**%**	**28.2%**																
After-Tax Payback	**calc**	**Years**	**2.5**																
After-Tax NPV @ 5%	**calc**	**US$M**	**202.7**																
After-Tax NPV @ 0%	**calc**	**US$M**	**308.8**																

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

This Pre-Feasibility Study indicates that the Romero Project can support a 2,800 t/d underground mine and processing plant. In the opinion of JDS, the project should proceed to the Feasibility Study stage where further technical evaluations will lend support to the concepts developed here. Support is based on the total Probable Mineral Reserve of 7.031 Mt grading 3.72 g/t Au, 4.33 g/t Ag, and 0.88% Cu, containing 840,000 oz gold, 980,000 oz silver and 136 M lbs of copper. A bulk 13% copper concentrate with gold credits will be exported to international smelters.

At this stage of study, there are a number of risks and opportunities associated with the project. These are described in the following sections.

26.1 Risks

26.1.1 Backfill

- Cleaner tails will have very high SG and there is risk of solids settling in the distribution pipes over time. Other operations have had issues with segregation of the paste and sanding of paste lines due to the settling of solids during paste transport

26.1.2 Mining

- Mining advance rates are based on good operating conditions. It may be required to reduce drill steel length from 16ft to 6ft in areas of very poor rock quality, which would hinder advance rates.
- Capital development was designed to minimize distances excavated, and as such is close enough to the Mineral Resource that some long-term infrastructure crosses through zones of poor rock quality. Although appropriate ground support controls have been planned and budgeted for these areas a trade-off may be warranted to investigate keeping all capital development outside of the zones of argillic alteration to improve advance rates and drift stability at the cost of longer drives into the production levels.
- Assumptions were made on the type and amount of ground support and advance rates based on the rock mass quality to be encountered when developing the ramp, access drifts and production drifts. Since there is an incremental cost change from one type or class of support to the next, there is a geotechnical threat related to the impact on the cost and schedule if the ground conditions encountered during excavation are worse than that assumed in the PFS study. Two remediation controls are proposed:
    1. The development of more simultaneous headings/workplaces where possible to minimize impact to schedule.
    2. The application of a high standard systematic ground support throughout the underground mine.

- Cold joints, formed from the contact of backfill with different curing times, could cause back instabilities in underhand MCF. The causes for this threat are related to the contact between adjacent drifts and/or when there are delays in backfilling a given drift (i.e., backfilling is done in different shifts or days). The impact is potential injury to mine workers, requirement to add shotcrete of additional ground support in the back, and loss of production from that heading. Potential remediation controls are:
    - i) Alternate the alignments of the underhand drifts so that they do not overlap or continuously follow a backfill contact in the back.
    - ii) Ensure that the paste fill plant has enough capacity to produce the required volume of backfill to meet mining schedule.
    - iii) Plan maximum drift lengths that could be backfilled within one to maximum two shifts.

26.1.3 Hydrogeology

- Groundwater inflow was assumed by JDS based on limited hydrogeologic data made available. Additional test work is recommended to validate inflow and pumping requirements estimated in this study.

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.	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, FS field work should begin as soon as possible.
Overall Mine Stability	Mining with backfill may increase dilution and overall mine recovery. The current design calls for all mined voids to be filled with paste backfill.	Overall geotechnical stability of the mine needs to be assessed in more detail at the feasibility level.

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Ability to Attract Experienced Professionals	The ability to attract and retain competent, experienced professionals 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 (2016)

26.2 Opportunities

Mining

- Potential increase to mining recovery could be seen with smaller stope dimensions and increased resolution of the mineable resource. The reduction of stope dimensions would increase the number of stopes and thus unit mine costs, so a trade-off study would be required to confirm any economic gains.

Table 26.2: Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Expansion of the Mine	Approximately half of the estimated resources have been contemplated for mining in the PFS. The remaining resources offer opportunity for expansion; the mineral resource has not been fully delineated and there is an opportunity to expand the mineable resource.	Increased mine life.
Increased Production	Increased production may be possible in high TVPM levels. There is an opportunity for the mine to produce more tonnes for short durations on the high tonnage levels of the mine.	Reduced unit operating costs and increased revenue.
Optimize Mine Plan	Optimize the mine plan and stope sequence.	Decrease ramp-up duration and potentially higher grades earlier in the mine life.
Contract Mining	Contract mining instead of owner mining.	Reduce CAPEX (but likely increase OPEX).
Backfill Cement Content	Paste backfill testing may reduce the cement content assumption.	Reduce mining costs.
Concentrate Smelting	Copper and bulk concentrates are currently assumed to be shipped overseas. There may be potential to source North American smelter capacity to reduce concentrate transport costs. It may be possible to obtain better treatment and/or refining terms from smelters through formal negotiations in the future.	Reduced transportation and concentrate shipping costs. Reduced concentrate treatment and refining costs.
Satellite Deposits	Potential additional resources at Romero South could provide additional feed for the mill.	Additional mill feed (especially at higher grade) could improve the project economics by speeding up project payback and/or extending the mine life.

Source: JDS (2016)

27 Recommendations

It is recommended that Romero proceed to the Feasibility Study stage in line with GoldQuest's desire to advance the project towards a production decision. Several technical programs, including baseline environmental studies, are required to de-risk the project and provide the level of detail necessary to complete a feasibility level evaluation. It is also recommended that the company continue with its efforts with respect to community engagement and project permitting.

It is estimated that a Feasibility Study, technical studies and supporting field work would cost approximately $4.8 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 FS Stage

Component	Estimated Cost (M$)	Comment
Resource Drilling and Updated Resource	1.0	Conversion of Inferred resources to Indicated within and immediately adjacent to the proposed mine. Drilling will include holes for combined resource, geotechnical and metallurgical purposes
Metallurgical Testing	0.3	Variability test work including expanded comminution, grinding, flotation and filtration test work as well as multi-element ICP tailings and concentrate analysis for smelter interest and pricing
Access Road	0.3	Reconnaissance, test pitting, borrow source identification, geotechnical investigations and road design
Backfill Testing	0.2	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.), and process plant area/piezometers/flow monitoring/geochemical test work
Engineering and Design	2.0	FS-level mine, infrastructure, tailings storage, paste backfill and process design, cost estimation, scheduling and economic analysis
Environment	0.5	Baseline environmental investigations including, water quality, fisheries, wildlife, weather, traditional land use and archaeology
Total	**4.8**	Excludes corporate overheads and future permitting activities

Source: JDS (2016)

Further details on recommendations not mentioned in Table 27.1 are found in the next sections.

27.1 Geology

Drilling outside of the Romero and Romero South deposits is relatively limited and there are large areas of untested ground near the deposits which provide brownfields resource growth potential. A project site exploration program of up to 5,000 m is recommended to test existing targets.

Additional drilling at the Romero and Romero South deposits should be completed to achieve multiple objectives for a feasibility study, including potentially improving classification of resources, collecting geotechnical data, performing packer tests and gathering material for metallurgical test work. Up to 3,000 m of oriented core drilling is recommended for various technical studies for a feasibility study and to potentially improve resource classification.

The estimated cost for the drilling is $1 million for the project site exploration and $1 million for the feasibility study drilling.

27.2 Metallurgy

The flowsheet developed from recent test work is based on a primary grind of 75 µm with a regrind P_{80} of 23 µm to produce a 13% copper concentrate with a recovery of 94.9%. With gravity and flotation, the final concentrate will include 78.2% gold and 58.6% 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 three years of operation by rock type and variability samples of varying grades. From this test work, GoldQuest can proceed with some confidence towards a full-scale feasibility level study.

Engineering work should include:

- Updated design criteria based on test work to confirm flowsheet with more sample variability;
- Updated mass and process water balance calculations;
- Confirmation of equipment sizing and specifications;
- Detailed flowsheets for each unit operation;
- Piping and instrumentation drawings for each unit operation; and
- Detailed operational and capital cost estimates.

Further studies should include:

- Looking at regrind energy requirements with further test work to confirm the results for variability samples and composites representing the first three years of operation;
- Investigating opportunities to recover more gold in the later stages of flotation, including a gravity concentrator in the regrind circuit;
- Evaluating methods to reduce gold loss in the first cleaner flotation circuit, including installing a gravity concentrator or leaching the tailings; and
- Conducting additional test work on Romero South to better define a flowsheet with improved grade and recoveries while producing a saleable concentrate. Alternative recovery methods should also be considered.

The following metallurgical testing programs are recommended:

- Lock-cycle flotation test work on samples representing the first three years of operation and varying copper and gold head grades;
- Flotation optimization;
- Gravity concentration and leaching test work on first cleaner tailings samples; and
- Regrind studies and specific energy testing;

27.3 Geotechnical

Geotechnical Drilling

- Minimum two geotechnical holes with packer testing, where the decline will cross under the San Juan River, to investigate the geotechnical and hydrogeological conditions of the rock mass beneath the river.

Further geotechnical analysis

- Improve geotechnical (RQD) and alteration block models for use in FS mine design.
- Effects of hydrogeological data on stability analysis and support recommendations.
- Strength tests to validate intact rock strength envelope and increased understanding of degradation of argillic rocks.
- 3D numerical modelling with the planned PFS mining sequence.
- Geotechnical logging of new exploration holes, including measurements of RQD, fracture frequency, rock hardness, joint surface conditions, etc.

27.4 Paste Backfill

- Detailed backfill schedule as to better understand and budget the cleaner and rougher tails consumption in paste production;
- Extend the yield stress measurements in paste rheology test work from 150 Pa to 350 Pa, with accompanying Bingham plastic viscosity measurements, for the proportional mixes of cleaner and rougher tailings to be mixed in the plant;
- Incorporate storage of cleaner tailings as filter cake to allow for up to five continuous days of shutdown in the paste plant, as to avoid mine shut downs for lack of access to filtered tails.
- Detailed barricade cost estimation;
- UCS of paste cylinders to determine the paste strength at various binder contents.
- Optimize the proportion of cleaner tailings used in the paste to provide the best properties needed for backfill reporting to LH stopes and MCF stopes; and
- Degradation study to examine high pyrite pastes over 120-day period.

27.5 Mining

- Perform trade-off studies for stope dimensions and cut-off to optimize FS mine design criteria.
- Advanced negotiations with mining contractors and visit sites in operation.
- Detailed design of mine infrastructure, including ventilation, pump, and shop equipment and installations.

Reliable, safe infrastructure is key to the success of the proposed Romero mine. There are three primary areas that will need further study if the project proceeds in to the feasibility study stage – access road, power and port facilities.

Access road

- Continue technical evaluation and detailed design of primary access road.
- Conduct a geotechnical investigation program to evaluate sub-surface conditions along the proposed road alignment
- Conduct a geotechnical investigation of the foundation conditions for bridge abutments and approaches
- Detailed design of culverts and bridge crossings
- Detailed design of signage and traffic control measures

Power

- Continue discussions with domestic power utility to receive firm power rate pricing
- Feasibility level design of power transmission line and substations

Port Facilities

- Continue discussions with port owners relating to access and concentrate handling terms
- Complete technical investigations of pier and concentrate storage areas to aid in the detailed design
- Complete detailed design of concentrate storage areas, security, administration and ship loading equipment

27.6 Environment and Permitting

Continue with environmental baseline studies for the project, including;

- Terrestrial Habitat and Wildlife Studies
- Vegetation Community Studies
- Waste Rock, Ore, and Soil Geochemistry
- Acid Rock Drainage and Metal Leaching Prediction Studies
- Hydrogeology and Hydrology
- Surface Water and Groundwater Quality
- Air Quality, Noise and Vibration Studies
- Species at Risk Screening Studies
- Archaeological and Traditional Land Use Studies

Continue providing regular communication of project information with local residents and government agencies.

Continue collecting seasonal data on water quality and flows from both surface and groundwater sources.

Preparation of the SIA in compliance with the IFC Performance Standard and Equator Principles, together with a social engagement plan.

28 References

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29 Units of Measure, Abbreviations and Acronyms

Symbol/Abbreviation	Description
'	Minute (Plane Angle)
"	Second (Plane Angle) or Inches
°	Degree
°C	Degrees Celsius
3D	Three-Dimensions
A	Ampere
a	Annum (Year)
AA	Atomic Absorption
AAS	Atomic absorption spectrometry
ac	Acre
ADR	Adsorption-Desorption-Recovery
AES	Atomic Emission Spectroscopy
AIM	Alternative Investment Market
ALS	ALS Chemex Ltd
amsl	Above Mean Sea Level
ANFO	Ammonium nitrate fuel oil
ARD	Acid Rock Drainage
Au	Gold
BD	Bulk Density
BFA	Bench Face Angles
BTU	British Thermal Unit
BV/h	Bed Volumes Per Hour
BVI	British Virgin Islands
C$	Dollar (Canadian)
Ca	Calcium
CDA	Canadian Dam Association
CDE	Canadian Development Expense
CDEEE	Corporacion Dominicana de Empresas Electicas Estatales
CDP	Cyanide Detoxification Plant
CF	Cumulative Frequency
cfm	Cubic Feet Per Minute
CHP	Combined Heat And Power Plant
CIC	Carbon-In-Column
CIM	Canadian Institute Of Mining And Metallurgy
cm	Centimetre
CM	Construction Management
cm^2	Square Centimetre
cm^3	Cubic Centimetre
COG	Cut-Off Grades
Cr	Chromium
CSA	Canadian Securities Administrators
CSRM	Certified standard reference materials
Cu	Copper
CV	Coefficient of Variation
d	Day
d/a	Days per Year (Annum)
d/wk	Days per Week

Symbol/Abbreviation	Description
dB	Decibel
dBa	Decibel Adjusted
DCIP	Direct current induced polarization
DCS	Distributed Control System
DGPS	Differential Global Positioning System
dmt	Dry Metric Ton
DSTSF	Dry Stack Tailings Storage Facility
DTM	Digital terrain model
EA	Environmental Assessment
EDA	Exploratory Data Analysis
ELOS	Equivalent linear over-break/slough
EMR	Energy, Mines and Resources
EP	Engineering and Procurement
EPCM	Engineering, Procurement and Construction Management
FEL	Front-End Loader
FS	Feasibility Study
ft	Foot
ft^2	Square Foot
ft^3	Cubic Foot
ft^3/s	Cubic Feet Per Second
g	Gram
G & A	General And Administrative
g/cm^3	Grams Per Cubic Metre
g/L	Grams Per Litre
g/t	Grams Per Tonne
gal	Gallon (Us)
GJ	Gigajoule
GPa	Gigapascal
gpm	Gallons Per Minute (US)
GRG	Gravity recoverable gold
GSC	Geological Survey of Canada
GW	Gigawatt
h	Hour
h/a	Hours Per Year
h/d	Hours Per Day
h/wk	Hours Per Week
ha	Hectare (10,000 M2)
HG	High Grade
HLP	Heap Leaching Pads
HMI	Human Machine Interface
hp	Horsepower
HPGR	High-Pressure Grinding Rolls
HQ	Drill Core Diameter Of 63.5 Mm
HSE	Health, Safety and Environmental
HVAC	Heating, Ventilation, and Air Conditioning
HW	Hanging Wall
Hz	Hertz
IFC	International Finance Corporation
in	Inch
in^2	Square Inch
in^3	Cubic Inch

Symbol/Abbreviation	Description
IP	Internet Protocol
IRR	Internal Rate Of Return
IT	Information technology
JDS	JDS Energy and Mining Inc.
K	Hydraulic Conductivity
k	Kilo (Thousand)
KE	Kriging Efficiency
kg	Kilogram
kg	Kilogram
kg/h	Kilograms Per Hour
kg/m^2	Kilograms Per Square Metre
kg/m^3	Kilograms Per Cubic Metre
km	Kilometre
km/h	Kilometres Per Hour
km^2	Square Kilometre
KNA	Kriging Neighbourhood Analysis
kPa	Kilopascal
kt	Kilotonne
kV	Kilovolt
KV	Kriging Variance
kVA	Kilovolt-Ampere
kW	Kilowatt
kWh	Kilowatt Hour
kWh/a	Kilowatt Hours Per Year
kWh/t	Kilowatt Hours Per Tonne
L	Litre
L/min	Litres Per Minute
L/s	Litres Per Second
LAN	Local Area Network
LDD	Large-Diameter Drill
LDRS	Leak Detection And Recovery System
LG	Low Grade
LG	Lerchs- Grossman
LH	Long hole
LHD	Load-haul-dump
LOI	Letter of Intent
LOM	Life Of Mine
m	Metre
M	Million
m/min	Metres Per Minute
m/s	Metres Per Second
m^2	Square Metre
m^3	Cubic Metre
m^3/h	Cubic Metres Per Hour
m^3/s	Cubic Metres Per Second
Ma	Million Years
mamsl	Metres Above Mean Sea Level
MAP	Mean Annual Precipitation
masl	Metres Above Mean Sea Level
Mb/s	Megabytes Per Second
mbgs	Metres Below Ground Surface

Symbol/Abbreviation	Description
mbs	Metres Below Surface
mbsl	Metres Below Sea Level
MCC	Motor Control Centres
MCF	Mechanized cut and fill
mg	Milligram
mg/L	Milligrams Per Litre
min	Minute (Time)
mL	Millilitre
Mm^3	Million Cubic Metres
MMER	Metal Mining Effluent Regulations
mo	Month
MPa	Megapascal
MRE	Mineral Resource Estimate
Mt	Million Metric Tonnes
MVA	Megavolt-Ampere
MW	Megawatt
MWMT	Meteoric Water Mobility Tests
MWTP	Mine Water Treatment Plant
NAD	North American Datum
NG	Normal Grade
Ni	Nickel
NI 43-101	National Instrument 43-101
Nm^3/h	Normal Cubic Metres Per Hour
NPI	Net profits interest
NPV	Net present value
NPVS	NPV Scheduler
NQ	Drill Core Diameter of 47.6 Mm
NRC	Natural Resources Canada
NSR	Net smelter return
OEM	Original Equipment Manufacturers
OIS	Operator Interface Stations
OP	Open Pit
OSA	Overall Slope Angles
oz	Troy Ounce
P.Geo.	Professional Geoscientist
Pa	Pascal
PAG	Potential acid generating
PAG	Potentially Acid Generating
PAX	Potassium Amyl Xanthate
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PFS	Preliminary Feasibility Study
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	Probable Maximum Flood
ppb	Parts Per Billion
PPE	Protective personal equipment
ppm	Parts Per Million
psi	Pounds Per Square Inch
QA/QC	Quality Assurance/Quality Control
QKNA	Qualitative Kriging Neighbourhood Analysis

Symbol/Abbreviation	Description
QMA	Quartz Mining Act
QML	Quartz Mining License
QMS	Quality Management System
QP	Qualified Person
QQ	Quartile-Quartile
RC	Reverse Circulation
RDI	Resource Development Inc
RMR	Rock Mass Rating
ROM	Run-Of-Mine
rpm	Revolutions Per Minute
RQD	Rock quality designation
RQD	Rock Quality Designation
s	Second (Time)
S.G.	Specific Gravity
SARA	Species At Risk Act
Scfm	Standard Cubic Feet Per Minute
SD	Standard deviations
SEDEX	Sedimentary Exhalative
SG	Specific Gravity
SIA	Socio-economic Impact Assessment
SMR	South Mcquesten Road
SPMDD	Standard Proctor Maximum Dry Density
SVOL	Search Volume
t	Tonne (1,000 Kg) (Metric Ton)
t/a	Tonnes Per Year
t/d	Tonnes Per Day
t/h	Tonnes Per Hour
TCR	Total Core Recovery
tph	Tonnes Per Hour
ts/hm^3	Tonnes Seconds Per Hour Metre Cubed
TSF	Tailings storage facility
TSS	Total Suspended Solids
UCS	Uniaxial compression
US	United States
US	United States
US$	Dollar (American)
UTM	Universal Transverse Mercator
V	Volt
VEC	Valued Ecosystem Components
VoIP	Voice Over Internet Protocol
VSAT	Very Small Aperture Terminal
VSEC	Valued Socio-Economic Components
w/w	Weight/Weight
WAD	Weak-Acid-Dissociable
WBS	Work Breakdown Structure
wk	Week
wmt	Wet Metric Ton
WRS	Waste Rock Stockpile
WRSA	Waste Rock Storage Area
WUL	Water Use License
XRF	X-ray fluorescence

Symbol/Abbreviation	Description
µm	Microns

Scientific Notation	Number Equivalent
1.0E+00	1
1.0E+01	10
1.0E+02	100
1.0E+03	1,000
1.0E+04	10,000
1.0E+05	100,000
1.0E+06	1,000,000
1.0E+07	10,000,000
1.0E+09	1,000,000,000
1.0E+10	10,000,000,000

APPENDIX A
QP CERTIFICATES

PARTNERS IN
ACHIEVING
MAXIMUM
RESOURCE
DEVELOPMENT
VALUE

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
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jdsmining.ca

CERTIFICATE OF AUTHOR

I, Garett Macdonald, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Prefeasibility Study Technical Report on the Romero Project, Dominican Republic”, with an effective date of September 27, 2016, (the “Technical Report”) prepared for GoldQuest Mining Corp.;

2. I am currently employed as Vice President Project Development 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 Laurentian University with a B.Eng. in Mining Engineering, 1996. I have practiced my profession continuously since 1996;

4. I have worked in technical, operations and management positions at mines in Canada. I have been an independent consultant for over one year and have managed preliminary economic assessments, pre-feasibility studies, feasibility studies and technical due diligence reviews.

5. I am a Registered Professional Mining Engineer in Ontario (#90475344)

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 r equirements 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 on May 20-21 , 2016;

8. I am responsible for Sections 1, 2, 3, 18, 19, 20, 21, 22, 24, 25, 26, 27, 28, 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: September 27, 2016
Signing Date: November 10, 2016

(original signed and sealed) “Garett Macdonald, P.Eng.”

Garett Macdonald, P. Eng.

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CERTIFICATE OF AUTHOR

I, Mathangi (Indi) Gopinathan, P. Eng., C.P.A., C.M.A., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Romero Project, Dominican Republic", with an effective date of September 27, 2016, (the "Technical Report") prepared for GoldQuest Mining Corp.;

2. I am currently employed as a Project Manager with JDS Energy & Mining Inc. with an office at Suite 3670 – 130 King Street West, Toronto, Ontario, M5X 1E2;

3. I am a graduate of the University of Toronto with a B.A.Sc. in Civil Engineering, 1996 (P.Eng., 2001) and Chartered Professional Accountant (C.P.A., C.M.A., 2008);

4. I have worked in, operations, financial and management positions at mining companies and financial institutions in Canada over the past 18 years. I have been an independent consultant for one year, and have performed G&A cost analysis, tax and economics analysis and report writing for mining projects worldwide;

5. I am a Registered Professional Engineer in Ontario (#90483173) and Registered Chartered Professional Accountant in Ontario (#31026349);

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 May 20-21, 2016;

8. I am responsible for Section 23 (Economic Analysis) 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: September 27, 2016
Signing Date: November 10, 2016

(original signed and sealed) "Mathangi (Indi) Gopinathan, P.Eng."

Mathangi (Indi) Gopinathan, P. Eng., C.P.A., C.M.A.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Kelly S. McLeod, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Romero Project, Dominican Republic", with an effective date of September 27, 2016, (the "Technical Report") prepared for GoldQuest Mining Corp.;

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 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;

4. I am a Professional Metallurgical Engineer (P.Eng. #15868) registered with the Association of Professional Engineers, Geologists of British Columbia;

5. 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;

6. I did not visited the Romero Project site;

7. I am responsible for Section 13 and 17 of this Technical Report;

8. I have had no prior involvement with the property that is the subject of this Technical Report;

9. 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;

10. 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: September 27, 2016
Signing Date: November 10, 2016

(original signed and sealed) "Kelly McLeod, P.Eng."

Kelly S. McLeod, P. Eng.

JDS Energy & Mining Inc.
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Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Michael E. Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Romero Project, Dominican Republic", with an effective date of September 27, 2016, (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 nine 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 15 and 16 (except 16.5 and 16.9.6) of this Technical Report;

9. I have had prior involvement with the property that is the subject of this Technical Report and was QP for the "Preliminary Economic Assessment Technical Report for the Romero Project, Dominican Republic", with an effective date of April 29, 2015;

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: September 27, 2016
Signing Date: November 10, 2016

(original signed and sealed) "Michael E. Makarenko, P.Eng."

Michael E. Makarenko, P. Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Marcel Pineau, Ph.D.,M.Sc.f.P.Eng. do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Romero Project, Dominican Republic", with an effective date of September 27, 2016, (the "Technical Report") prepared for GoldQuest Mining Corp.;

2. I am currently employed as a Senior Technical Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I have an engineering degree from Laval University and my professional engineer registration (OIFQ, #79007) is under the Law of the Province of Quebec; I have a Master and Ph.D. degrees in mathematical modeling applied to complex water systems from the *National Scientific Research Institute of the University of Quebec,* and completed post-doctoral studies at the Water Resources Engineering Dept. of the University of Arizona. I have 25 years of experience in the development and operation of mines sites water infrastructures and in managing mine closure, permitting, EIA, IBA for mining projects and mining operations in Northern Canada, Greenland, Chile and New Caledonia.

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 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;

5. I visited the Romero project on May 20-21, 2016;

6. I am responsible for Sections 16.8.2 and 20 of this 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: September 27, 2016
Signing Date: November 10, 2016

(original signed and sealed) by "Marcel Pineau, Ph.D.,M.Sc.,f.P.Eng."

Marcel Pineau, Ph.D.,M.Sc.,f.P.Eng.

CERTIFICATE OF AUTHOR

As an author of this report entitled “Prefeasibility Study Technical Report on the Romero Project, Dominican Republic”, prepared for GoldQuest Mining Corp. and with an effective date of September 27, 2016, (the “Technical Report”), I, B. Terrence Hennessey, P.Geo., do hereby certify that:

1. I am employed as 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 over 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 20 years as a consulting geologist working in precious, ferrous and base metals as well as industrial minerals.

6. I visited the Romero project form January 9 to 12, 2013.

7. I am responsible for the preparation of Sections 4 to 12, 14 and any summaries therefrom in Sections 1, 26 and 27 of the Technical Report.

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 have had no prior involvement with the mineral properties in question.

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 this Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make this report not misleading.

Effective date: Romero Mineral Resource: January 14, 2016
Romero South Mineral Resource: October 29, 2013

Dated this 10th day of November, 2016

"B. Terrence Hennessey" {signed, sealed and dated}

B. Terrence Hennessey, P.Geo.

CERTIFICATE OF AUTHOR

I, Alan J. San Martin MAusIMM(CP), do hereby certify that:

1. This certificate applies to the Technical Report entitled *"NI 43-101 Pre-Feasibility Study Technical Report for the Romero Gold Project, Dominican Republic"*, with an effective date of September 27, 2016, (the "Technical Report") prepared for GoldQuest Mining Corp.;

2. I am currently employed as a Mineral Resource Specialist with Micon International Limited. with an office at Suite 900 – 390 Bat Street, Toronto, Ontario, M5H 2Y2 | +1 416 362 5135;

3. I am a gr aduate of the Universidad Nacional de Piura, Peru with a B.Sc. in Mining Engineering, 1998. I have practiced my profession continuously since 1999;

4. I have worked in mineral exploration projects in technical management positions in Peru and Ecuador. I have been an independent consultant with Micon for over seven years and have performed mineral resource estimates for a v ariety of mineral deposits, technical due diligence reviews and report writing for exploration and mining projects worldwide;

5. I am a R egistered Chartered Professional in Geology with The Australasian Institute of Mining and Metallurgy – AusIMM (#301778), a Ing. CIP with Colegio de Ingenieros del Perú (#79184) and a m ember of the Canadian Institute of Mining, Metallurgy and Petroleum, (#151724);

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 an accepted foreign professional association (AusIMM), 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 have not visited the Romero project;

8. I am a co-author responsible for Sections 14 and any related summaries in sections 1, 26 and 27 of this Technical Report;

9. I have had prior involvement with the property that is the subject of this Technical Report in "Preliminary Economic Assessment (PEA) for The Romero Project, Tireo Property, Province of San Juan, Dominican Republic", effective date May 27, 2014 and " A Mineral Resource Estimate for The Romero Project, Tireo Property, Province of San Juan, Dominican Republic", effective date October 29, 2013;

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 t echnical 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: January 14, 2016 (Romero), October 29, 2013 (Romero South)
Signing Date: November 10, 2016

(original signed and sealed) "Alan J. San Martin MAusIMM(CP)."

Alan J. San Martin MAusIMM(CP).

MineFill Services, Inc.
International Specialists in Rock Mechanics and Mine Backfill

CERTIFICATE OF AUTHOR

I, David Stone, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Romero Project, Dominican Republic", with an effective date of September 27, 2016, (the "Technical Report") prepared for GoldQuest Mining Corp.
2. I am currently employed as President of MineFill Services, Inc., that is a Washington, USA, domiciled Corporation.
3. I am a graduate of the University of British Columbia with a B.Ap.Sc in Geological Engineering, a Ph.D. in Civil Engineering from Queen's University at Kingston, Ontario, Canada, and an MBA from Queen's University at Kingston, Ontario, Canada.
4. I have practiced my profession for over 30 years and have considerable experience in the preparation of engineering and financial studies for base metal and precious metal projects, including Preliminary Economic Assessments, Preliminary Feasibility Studies and Feasibility Studies.
5. I am a licensed Professional Engineer in Ontario (PEO #90549718) and I am licensed as a Professional Engineer in a number of other Canadian and US jurisdictions.
6. I have read the definition of 'Qualified Person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and 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.
7. I have not visited the property that is the subject of this report.
8. I am responsible for the report content related to the paste backfill plant (Section 16.9.6)
9. I am independent of the Issuer applying all the tests in Section 1.5 of NI 43-101.
10. I have had no prior involvement with the property.
11. I have read NI 43-101 and NI 43-101F1 and this Technical Report has been prepared in compliance with that instrument and form.
12. As of the Effective Date of the Technical Report (September 27, 2016), to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Effective Date: September 27, 2016
Signing Date: November 10, 2016

(original signed and sealed) "David Stone, P.Eng."

David M Stone, P.Eng.

CERTIFICATE OF AUTHOR

I, Luiz Castro, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Romero Project, Dominican Republic", with an effective date of September 27, 2016, (the "Technical Report") prepared for GoldQuest Mining Corp.;
2. I am currently employed as a Principal and Senior Rock Mechanics Engineer with Golder Associates Ltd. with an office at Suite 100 – 6925 Century Avenue, Mississauga, Ontario, L5N 7K2;
3. I am a graduate of the Catholic University of Rio de Janeiro, Brazil with a B.Sc. in Civil Engineering, 1980; a M.Sc. in Soil Mechanics from the New University of Lisbon, Portugal, 1987; and a Ph.D. in Rock Mechanics from the University of Toronto, 1996. I have practiced my profession continuously since 1988;
4. I have been the lead rock mechanics engineer and managed several underground and open pit projects from Scoping Level to Feasibility Level to Operations, located in Africa, Asia, and Americas. I have been working at Golder Associates for more than 20 years and have performed geotechnical and hydrogeological field investigations, elaboration of geotechnical model, complex numerical modelling, open pit slope design, slope performance audit, ground control audit, crown pillar design, bulkhead design, geotechnical hazard assessment and underground mine geomechanics, including ground support and mining sequence evaluations for excavations under rock burst prone conditions.
5. I am a Registered Professional Mining Engineer in Ontario (#90517921);
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 site from January 20 to 22, 2016;
8. I am responsible for Section 16.5 of this Technical Report;
9. I have not had 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: September 27, 2016
Signing Date: November 10, 2016

(Original signed and sealed) "Luiz Castro, P.Eng."

Luiz Castro, P. Eng.

Golder Associates Ltd.
6925 Century Avenue, Suite #100, Mississauga, Ontario, Canada L5N 7K2
Tel: +1 (905) 567 4444 Fax: +1 (905) 567 6561 www.golder.com

CERTIFICATE OF AUTHOR

I, Kenneth A. Bocking, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Romero Project, Dominican Republic", with an effective date of September 27, 2016, (the "Technical Report") prepared for GoldQuest Mining Corp.;

2. I am currently employed as a Principal with Golder Associates Ltd. with an office at Suite 100 – 6925 Century Avenue, Mississauga, Ontario, L5N 7K2;

3. I am a graduate of the University of Saskatchewan with a B.Sc. in Civil Engineering, 1974 and a M.Sc. in Geotechnical Engineering, 1978. I have practiced my profession continuously since 1974;

4. I have worked as a consulting geotechnical engineer since graduation. Since 1988, my consulting work has been almost exclusively for mining sector clients;

5. I am a Registered Professional Engineer in Saskatchewan (#772059) and Ontario (#4253654) and as a Licenced Professional Engineer in the Northwest Territories (#L400);

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 have not visited the Romero project site;

8. I am responsible for Section 18.3 of this Technical Report;

9. I have not had 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: September 27, 2016
Signing Date: November 10, 2016

(original signed and sealed) "Kenneth A. Bocking, P.Eng."

Kenneth A. Bocking, P. Eng.

CERTIFICATE OF AUTHOR

I, Luis F. Vasquez, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Romero Project, Dominican Republic", with an effective date of September 27, 2016, (the "Technical Report") prepared for GoldQuest Mining Corp.;

2. I am currently employed as a Senior Water Resources Engineer with Golder Associates Ltd. with an office at Suite #100 – 6925 Century Avenue, Mississauga, Ontario, L5N 7K2;

3. I am a graduate of Universidad de Los Andes (Bogotá, Colombia) with a B.Sc. in Civil Engineering, 1998 and a master's degree in Water Resources Engineering, 1999. I have practiced my profession continuously since 1999;

4. I have continuously worked as a consultant with Golder Associates Ltd. since 2004 carrying out hydrological assessments, water management planning, design of water management facilities and infrastructure, cost estimation, report writing and supervision of preparation of design drawings for mining projects worldwide;

5. I am a Registered Professional Water Resources Engineer in Ontario (#100210789);

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 did not visit the Romero project site;

8. I am responsible for Section 18.2 of this Technical Report;

9. I have not had 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: September 27, 2016
Signing Date: November 10, 2016

(Original signed and sealed) "Luis F. Vasquez, P.Eng."

Luis F. Vasquez, P. Eng.