PRELIMINARY ECONOMIC
ASSESSMENT
TECHNICAL REPORT

LAS MINAS PROJECT

VERACRUZ, MEXICO

EFFECTIVE DATE – SEPTEMBER 18, 2021
REPORT DATE – SEPTEMBER 18, 2021

PREPARED FOR

MEXICAN GOLD MINING CORP.

Suite 900, 999 West Hastings St.,
Vancouver, BC V6C 2W2 | Canada

PREPARED BY

JDS ENERGY & MINING INC.

Suite 900, 999 West Hastings St.,
Vancouver, BC V6C 2W2 | Canada

DATE AND SIGNATURE PAGE

This report entitled Preliminary Economic Assessment for the Las Minas, effective as of September 18, 2021 was prepared and signed by the following authors:

Original document signed and sealed by:

Gord Doerksen
Gord Doerksen, P. Eng. September 18, 2021

Original document signed and sealed by:

Michael Makarenko
Michael Makarenko, P. Eng. September 18, 2021

Original document signed and sealed by:

Michael Levy
Michael Levy, P. Eng. September 18, 2021

Original document signed and sealed by:

Garth Kirkham
Garth Kirkham, P. Geo. September 18, 2021

Original document signed and sealed by:

Tad Crowie
Tad Crowie, P. Eng. September 18, 2021

Original document signed and sealed by:

Ken Embree
Ken Embree, P. Eng. September 18, 2021

Original document signed and sealed by:

Tysen Hantelmann
Tysen Hantelmann, P. Eng. September 18, 2021

NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Mexican Gold 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.

Mexican Gold 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.

Table of Contents

List of Figures

List of Tables

1 EXECUTIVE SUMMARY

1.1 Introduction

This report summarizes the results of a Preliminary Economic Assessment (PEA) completed by JDS Energy & Mining Inc. (JDS) as commissioned by Mexican Gold Mining Corp. (Mexican Gold) for the Las Minas Project (the Project) and was prepared following the guidance of Canadian Securities Administrators’ National Instrument 43-101 and Form 43-101F1, collectively referred to as National Instrument (NI) 43-101.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the project presented in the PEA will be realized.

1.2 Project Description

Mexican Gold’s Las Minas property is located in central Veracruz State, Mexico, approximately 160 km (by road) northwest of the city of Veracruz and 250 km east of Mexico City. The project area is located within and surrounding the village of Las Minas in the municipalities of Las Minas and Tatatila. The project has an approximate geographic center at 19º 41’ 28” N latitude and 97º 08’ 46” W longitude.

The PEA plan presented in this report is to mine the deposit using underground mining methods and extract gold, silver, copper and magnetite from the mineralization using a 1,400 tonnes per day (t/d) flotation mineral processing facility and related infrastructure.

1.3 Location, Access and Ownership

The property is centered on the village of Las Minas which is partially connected by four-lane highways to the cities of Veracruz to the south and Mexico City to the west. From Veracruz, the village can be accessed by Highway 180 and then Highway 140 for a distance of 150 km, then turning north at the village of Cruz Blanca onto a 15 km gravel road that descends into the Rio Las Minas canyon. From Mexico City, access is via Highways 150D and 140D for a distance of 250 km to the turn-off at Cruz Blanca.

The Las Minas property consists of six mining concessions that cover approximately 1,616 ha. The mining concessions are titled according to Mexican mining law. The titles are valid for 50 years from the date titled and can be renewed for another 50 years once they expire. The current mineral resources underlie the Pepe, and Pepe Tres concessions. All of the concessions are owned by Mexican Gold subject to underlying royalty agreements on five of the concessions as discussed in Section 4.3. The San Valentin concession was staked by Source Exploration Corp. (Source) in 2012 and carries no royalty burden. Source changed their name to Mexican Gold Corp. in April of 2017.

1.4 History, Exploration and Drilling

The Las Minas mining region has been active for centuries. Malachite staining in the white marble cliffs would have been obvious to the earliest observers. Documentation and ruins of mining facilities and former town-sites remain from the early 1800's. Despite the long history, modern exploration only dates back a decade. Source Exploration initiated diamond drilling in the region in 2011, and the first geophysical surveys in 2012.

Since acquisition of the property in 2010, Mexican Gold has completed exploration activities including, diamond drilling; geological mapping; surface and underground sampling, and a ground magnetic survey.

Drilling has been conducted in 2011, 2012, 2014 through to 2020. The Las Minas drill database contains records for a total of 32,174 m of diamond-core (core) drilling in 229 drillholes within the Las Minas Project Area with 206 of those drillholes being within in the Las Minas resource area.

1.5 Geology and Mineralization

The Las Minas project is located in southeastern Mexico within the eastern portion of the Trans Mexico Volcanic Belt (TMVB), an east-west belt of Miocene to recent volcanic rocks that transects the country from the Pacific coast to the Gulf of Mexico. The pre-Miocene basement in the Las Minas region consists of a sequence of Jurassic and Cretaceous marine sedimentary rocks including sandstone, siltstone, limestone and shale. These have been intruded by Tertiary and Mesozoic plutonic rocks mapped as dominantly granodiorite and porphyritic dacite, with lesser amounts of granite, diorite and tonalite.

Copper and gold mineralization have been recognized in three settings within the Las Minas property: proximal skarn, distal skarn and quartz veins. Proximal-type skarn is the dominant skarn alteration observed within the Las Minas resource zones (El Dorado and Santa Cruz) while distal and gold-bearing quartz veins occur in the exploration targets to the east and north of the Las Minas resources.

Proximal skarn developed along marble-diorite contacts, both as exoskarn developed within the sedimentary rock, and as endoskarn developed within the intrusion. The skarn alteration has a typical zoning of marble-exoskarn-endoskarn-diorite. The distinction between exoskarn and endoskarn can be very difficult because the skarn alteration (especially garnet replacement) can be texturally destructive.

Proximal skarn alteration is dominantly garnet-rich with lesser amounts of pyroxene, and locally garnet appears to have replaced pyroxene. The skarn contains variable amounts of magnetite and lesser sulfide minerals.

Within the Las Minas resource zones, chalcopyrite is the dominant sulfide mineral with lesser amounts of bornite and pyrite. Sulfide grains usually are associated with magnetite and are present as relatively coarse-grained disseminations while sulfide blebs, bands, and veinlets cutting magnetite are also observed. Pyrite occurs as an accessory mineral in the main resource area.

Gold-silver-copper mineralization at El Dorado zone occurs as two horizons that are separated by a barren north-northwest trending diorite dike. The current modeling indicates that the El Dorado skarn zone on the west side of the diorite dike has an 800 m northwest strike length, extends up to 450 m to the southwest away from the diorite dike, is on average 15 to 20 m thick, and can reach over 50 m in thickness along the northwest-striking contact with the diorite dike. In contrast, the El Dorado zone on the east side of the dike has a strike length of 250 m northwest, extends up to 200 m to the northeast from the diorite dike, and is 5 to 10 m in thickness.

The Santa Cruz zone lies about 0.5 km south of the Las Minas pueblo and is well exposed on a west-facing canyon wall just above a tributary of the Rio Las Minas. Skarn within the Santa Cruz zone lies along the west side of the dike, immediately to the south of and stratigraphically higher than the El Dorado zone. The primarily east-dipping mineralization at Santa Cruz is more complex and discontinuous than observed at El Dorado due to the more variable intrusive-marble contact orientations (both near-vertical dike and east-dipping sills).

1.6 Metallurgical Testing and Mineral Processing

There have been two testwork programs run on material from the Las Minas deposit: a program in 2015 and a program in 2021. The gold and silver are generally associated with copper sulphide minerals allowing for a single metal concentrate to be produced. The deposit also has a significant quantity of magnetite which allows for a second concentrate to be produced.

The testwork programs were very similar to each other, although the results from the 2021 program indicated less copper and gold recovery. The differences are likely due to increased oxidation in the deposit. Since there were no oxide copper assays for the deposit, and because the difference in oxide copper between the two samples was not very high, it was determined best to use the results from the 2021 program to not be overly optimistic.

Both testwork programs included head assays, comminution testing, flotation testing, and magnetic separations testing. The comminution testing for the 2015 program included abrasion index testwork while the 2021 program was limited to Bond Ball Mill Work Index (Wi_{BM}). The flotation testing in both programs was very comprehensive including locked cycle tests with the conditions found to be optimal. The 2021 program included a more extensive magnetic separation program which demonstrated that a saleable concentrate could be produced with high recoveries.

The results from the testwork can be seen in Table 1-1.

Table 1-1: Estimated Metallurgical Recoveries, Concentrate Grades and Mineral Processing Factors

Headings	Units	Copper Concentrate	Magnetite Concentrate
Cu recovery	%	90	
Au recovery	%	80	
Ag recovery	%	70	
Fe Recovery	%		90

Headings		Units	Copper Concentrate	Magnetite Concentrate
Cu Concentrate Grade	Cu	%Cu	21.7	
	Au*	LOM g/t Au	25.5	
	Ag*	LOM g/t Ag	98	
Magnetite Concentrate Grade		%Fe		70
Bond Work Index (bWi)		**kWh/t**	**15.1**	

Notes:
*Variable with Cu concentrate pull factor.

1.7 Mineral Resource Estimate

The mineral resource estimates for Las Minas were prepared to industry standards and best practices and verified by Garth Kirkham, P.Geo., an Independent Qualified Person for the purposes of NI 43-101.

Within the Las Minas Project, 206 drill holes (32,058 meters) supports the mineral resource estimate. The deposit was segregated into multiple estimation domains based on geologic models for each of the mineralized units. The estimated mineral resources occur within the Las Minas gold-copper-silver-magnetite skarn deposit, which consists of the mineralized endo-skarn and exoskarn units within the El Dorado and Santa Cruz zones. The mineral domains were then used to code the block model, and assays within the modeled domains were evaluated geostatistically to establish estimation parameters. Assays were composited into 2-meter lengths. MineSight^{TM}, a commercially available geologic modeling and mine planning software package, was used to produce a three-dimensional block model while LeapFrog® Software was utilized to produce the solids models for the estimation domains.

The gold, copper, silver and iron grades were estimated into a three-dimensional, 12 m by 12 m by 3 m block model which was sub-blocked to 0.5 m in three dimensions. Gold (Au g/t), copper (Cu%), silver (Ag g/t) and total iron (Fe%) block grades were estimated from capped composited samples in a single pass. The mineral resources were estimated using ordinary kriging interpolation for the continuous mineralized domains. Search ellipse anisotropy and orientation were guided by the orientation of the domain solids models and omni-directional ellipsoids were employed in the individual zones.

Magnetite estimates were based on applying mathematical regression, as derived from SATMAGAN testing results, to the Total Fe% estimates. A total of 2,601 specific gravity readings were derived from measurements within individual rock types and estimated on a block-by-block basis using inverse distance.

Mineral resources are classified in accordance with the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves, and the 2019 CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.

Mineral Resources are classified under the categories of Indicated and Inferred according to CIM guidelines. Mineral Resource classification was based primarily on drill hole spacing and on continuity of mineralization. There are no measured resources at Las Minas. Indicated resources were defined as blocks with a distance to three drill holes of less than ~30 m to nearest composite and occurring within the estimation. Inferred resources were defined as those with a drill hole spacing of less than ~60 m.

Final resource classification shells were manually constructed on plan sections and all resources are constrained within lithological domains and by the continuous solids. Final Resource classification shells were manually constructed on sections. These interpreted boundaries were created for the indicated and inferred thresholds in order to exclude orphans and reduce potential “spotted dog” effect.

This estimate is also based upon the reasonable prospect of eventual economic extraction using estimates of reasonable operating costs and price assumptions. The mineral resources do not represent an attempt to estimate Mineral Reserves.

The Las Minas resources are reported in the following table at a base case cut-off of US$80 NSR.

Table 1-2: Las Minas Deposit Indicated and Inferred Mineral Resource Estimate at a US$80 NSR

Class	Tonnes	NSR (US$)	Au (g/t)	Au ('000 ounces)	Ag (g/t)	Ag ('000 ounces)	Cu (%)	Cu ('000 lbs)	Fe Magnetite (%)	Fe Magnetite ('000 tonnes)	AuEQ (g/t)	AuEq ('000 ounces)
Indicated	4,133	138.58	1.96	260	4.64	617	1.08	98,311	14.77	610	3.34	443
Inferred	5,200	112.83	1.44	241	5.97	997	0.95	108,802	17.54	912	2.16	361

Notes:

1. Mineral Resource Statement prepared by Garth Kirkham (Kirkham Geosystems Ltd.) in accordance with NI 43-101.
2. Effective date: September 18, 2021. All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (“CIM”) definitions, as required under NI 43-101.
3. Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under NI 43-101. Mineral resources are not Mineral Reserves and do not have demonstrated economic viability.
4. Underground Mineral Resources are reported at a cut-off grade of US$80 NSR. Cut-off grades are based on a price of US$1,700/oz gold, US$20/oz silver, US$3.50/lb copper and US$100/t magnetite concentrate and a number of operating cost and recovery assumptions, including a reasonable contingency factor.
5. Numbers are rounded.
6. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
7. The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

Source: Kirkham (2021)

The table below illustrates the sensitivity of the indicated and inferred mineral resource estimate to changes in cut-off grade. The base case at a cut-off grade of US$80 NSR is highlighted in bold. The table suggests that the mineral resource estimate is moderately sensitive to cut-off grade in terms of estimated contained metal.

Table 1-3: Sensitivity of Las Minas Indicated and Inferred Mineral Resource Estimate to Cut-Off Grade (base case is bolded)

Class	NSR COG (US$)	Tonnes	NSR (US$)	Au (g/t)	Au ('000 ounces)	Ag (g/t)	Ag ('000 ounces)	Cu (%)	Cu ('000 lbs)	Fe Magnetite (%)	Fe Magnetite ('000 tonnes)	AuEQ (g/t)	AuEq ('000 ounces)
Indicated	>=60	5,431	122.00	1.71	299	4.27	746	0.95	114,341	13.84	752	2.94	514
	>=70	4,750	130.25	1.83	280	4.44	678	1.02	106,373	14.35	682	3.14	479
	>=80	**4,133**	**138.58**	**1.96**	**260**	**4.64**	**617**	**1.08**	**98,311**	**14.77**	**610**	**3.34**	**443**
	>=90	3,549	147.47	2.09	239	4.87	555	1.14	89,467	15.31	543	3.55	405
	>=100	3,009	156.99	2.24	217	5.12	495	1.21	80,326	16.19	487	3.77	365
	>=110	2,572	165.96	2.38	197	5.36	444	1.27	72,146	16.86	434	3.98	329
Inferred	>=60	6,769	102.84	1.32	287	5.49	1,195	0.86	128,586	16.23	1,099	1.97	428
	>=70	6,012	107.69	1.38	266	5.73	1,108	0.91	119,959	16.95	1,019	2.06	398
	>=80	**5,200**	**112.83**	**1.44**	**241**	**5.97**	**997**	**0.95**	**108,802**	**17.54**	**912**	**2.16**	**361**
	>=90	4,228	119.33	1.54	209	6.19	842	1.00	93,057	18.00	761	**2.29**	311
	>=100	3,226	127.04	1.67	173	6.44	668	1.05	74,354	18.24	589	2.44	253
	>=110	2,106	138.88	1.84	125	7.07	479	1.14	52,930	18.42	388	2.66	180

Notes:

1. Mineral Resource Statement prepared by Garth Kirkham (Kirkham Geosystems Ltd.) in accordance with NI 43-101.
2. Effective date: September 18, 2021. All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under NI 43-101.
3. Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under NI 43-101. Mineral resources are not Mineral Reserves and do not have demonstrated economic viability.
4. Underground Mineral Resources are reported at a cut-off grade of US$80 NSR. Cut-off grades are based on a price of US$1,700/oz gold, US$20/oz silver, US$3.50/lb copper and US$100/t magnetite concentrate and a number of operating cost and recovery assumptions, including a reasonable contingency factor.
5. Numbers are rounded.
6. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
7. The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

Source: Kirkham (2021)

Additional drilling is recommended to increase drill density to potentially achieve a higher resource category in higher-grade areas. Additional drilling may increase resources, improve understanding and modelling of lithological units and better define the limits of the mineralization.

A review of QA/QC procedures is recommended to improve data quality and increase confidence in the dataset.

A comprehensive brownfields exploration program in the area is recommended to explore for additional targets.

Investigate and explore the historic mine workings and discoveries with the view of expanding resource base.

The most significant project risks are summarized below:

- Geological Complexity – The geological complexity of the Las Minas deposit could potentially lead to increased mining dilution and/or ore loss due to variability in mineralized domains. Grade control and proper mining execution will maintain minimal unplanned dilution, which would minimize potential impacts on grade, throughput, and operating costs. A comprehensive, tight grade control program and geological monitoring will help minimize unplanned dilution and negative impacts during mining;

- Data Quality – Data verification and data quality issues were encountered and addressed however, further issues could be discovered with ongoing data collection and exploration; and

- Drilling Uncertainty – There is no guarantee that further drilling will result in additional resources or increased classification. In addition, further work may disprove previous models and therefore result in condemnation of targets and potential negative economic outcomes. Refinement and continuous improvement of drilling planning and models will continue to advance understanding and increase confidence.

The main opportunities identified for the project are listed below:

- Data Validation – Ongoing data verification and ground truthing may result in being able to re-introduce data that has been excluded and may result in an improved understanding of the deposit and grade distribution;

- Mineral Resources – There is the potential for an increase in mineral resources with increased exploration drilling; and

- Exploration – There are many historic showings and discoveries that have been subject to limited exploration activities. These pose an excellent potential for expanding the project potential and resources.

1.8 Mineral Reserve Estimate

Mineral reserves can only be estimated as a result of an economic evaluation as part of a Preliminary Feasibility Study or a Feasibility Study of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the Project.

1.9 Mining

The deposit will be extracted using a combination of longhole stoping and room and pillar mining methods to meet a production rate of 1,400 t/d. Stopes are sequenced using a primary/secondary layout and are backfilled using cemented tailings (paste) and development waste rock. Development waste and lightly cemented tailings are used to backfill room and pillar areas to minimize surface storage.

The mine will be developed using conventional underground equipment consisting of development jumbos, longhole drills, bolters, LHDs, and haul trucks. Equipment and operations will be owner operated. Mineralized material will be hauled to an underground crusher located in the upper portion of the El Dorado Zone where the crushed material will be conveyed to the process facility. Stope optimizations were based on the Net Smelter Return (NSR) parameters contained in Table 1-4.

Table 1-4: NSR Input Parameters

Parameter	Unit	Value
COPPER CONCENTRATE		
Metal Prices		
Cu Price	US$/lb	3.25
Au Price	US$/oz	1,625
Ag Price	US$/oz	20.00
Exchange Rate	C$:US$	0.76
Royalties	% NSR	0.0
Recovery		
Copper Concentrate		
Cu Recovery	%	90.0
Au Recovery	%	80.0
Ag Recovery	%	70.0
Concentrate Grade		
Copper Concentrate		
Cu	%	21.7
Au	g/t	33.5
Ag	g/t	88
Moisture Content	%	8
Magnetite Concentrate		
Magnetite	% Magnetite	90.0
Smelter Payables		
Cu Payable	%	96.1
Min. Cu deduction	% Cu/tonne	1.1
Au Payable	%	97.2
Min. Au deduction	g/t concentrate	1
Ag Payable	%	60.9
Min. Ag deduction	g/t concentrate	30
Treatment & Refining Costs		
Cu TC	US$/dmt con	65

Parameter	Unit	Value
Cu RC	US$/payable lb	0.065
Au RC	US$/payable oz	5.00
Ag RC	US$/payable oz	0.40
Transport Costs		
Transport Costs	US$/wmt	56.20
Total Transport to Smelter	US$/dmt	61.09
MAGNETITE CONCENTRATE		
Magnetite Price (Iron Smelter Feed)	US$/tonne	100.00
Recovery		
Magnetite Feed Reporting to Tailings	%	94.5
Rougher Mag Separator Recovery	%	97.0
Estimated Magnetite Mineralized Material Recovery	%	98.5
Overall Magnetite Recovery	%	90.3
Concentrate Grade		
Magnetite	% Magnetite	90.0
Transport Costs		
Transportation Cost	US$/$t_{con}$	61.09

The total mineable resource is shown in Table 1-5. This does not constitute a mining reserve, as resources include inferred material which is not considered to be sufficiently proven geologically for reliance in an economic model.

Table 1-5: Summary of Mine Plan Resource Tonnes and Mill Head Grade

Class	Tonnes Mt	Cu %	Au g/t	Ag g/t	Magnetite %
Measured	-	-	-	-	-
Indicated	2.31	1.09%	1.95	4.86	14.6%
Inferred	1.74	1.02%	1.69	6.43	17.4%
Total Mine Plan	4.04	1.06%	1.84	5.53	15.7%

Note:

1. Mine Plan mineral resources are estimated at an NSR cut-off of $90/t;
2. Mine planning summary includes dilution and recovery; and
3. Some totals may not add due to rounding.

1.10 Recovery Methods

Mineral Processing will include crushing with a jaw and cone crusher, ball mill grinding at a nominal throughput of 1,400 t/d. The material would be ground to a particle size P_{80} of 150 µm and directed to a flotation circuit to recover copper, gold, and silver. The flotation tailings would then be processed in a magnetic separation circuit to produce a magnetite concentrate. The magnetic separation circuit would then be thickened and filtered and either placed underground as paste backfill or into the tailings storage facility.

A table of recoveries and concentrate grades can be found in Table 1-1 in Section 1.6.

1.11 Infrastructure

The project infrastructure is designed to support the operation of a 1,400 t/d mine and processing plant, operating on a 24 hour per day, seven day per week basis. The overall site layout will include a processing plant, filtered tailings and mine rock storage facilities, power plant and supporting infrastructure including an assay lab, warehouse, maintenance shop, fuel storage, mine dry, camp and administration offices.

Power for the site will be provided by an existing 15 MW capacity hydroelectric facility adjacent to the planned processing plant location and supplied by steel penstock tubes from a reservoir several hundred metres up the ridge. Stepdown transformers will be installed including switchgear throughout where required.

1.12 Environment and Permitting

Mexican Gold has conducted environmental studies in the project area in order to initiate development of a defensible baseline. Exploration work is conducted in a transparent manner with the local communities, supported with a strong community outreach and support program.

Current exploration activity is fully permitted and in good standing. Mine development will require the successful conclusion of an Environmental Impact Assessment and permitting. This is a recognized and regulated process in Mexico. There are no known environmental issues that could materially impact the ability of Mexican Gold to extract the mineral resources at the Las Minas Project.

1.13 Operating and Capital Cost Estimates

Total life of mine capital costs is estimated to be $145.1M Pre-production capital costs amount to $90.4M. Capital costs during production years total $44.7M. Closure costs have been estimated at $10M. These costs are summarized in Table 1-6. The project carried a blended contingency rate of 20%.

Table 1-6: Summary of Capital Cost Estimate

Capital Costs	Pre-Production (M$)	Sustaining / Closure (M$)	Total (M$)
Mining	12.3	34.7	47.0
On-site Development	4.2	-	4.2
Ore Crushing & Handling	3.2	-	3.2
Tailings Management	2.3	1.1	3.4
Mineral Processing Plant	22.6	1.4	24.0
Infrastructure	12.0	-	12.0
Project Indirects	7.3	-	7.3
Engineering & EPCM	5.4	-	5.4
Owner's Costs	6.2	-	6.2
Closure	-	10.0	10.0
Subtotal	**75.3**	**47.2**	**122.6**
Contingency	15.1	7.4	22.5
Total Capital Costs	**90.4**	**54.7**	**145.1**

Operating costs include mining, processing, tailings, and administration. Operating costs incurred during the construction phase (pre-production Years -2 and -1) are capitalized and form part of the capital cost estimate. Concentrate transportation, treatment and refining charges, and royalties are included separately as part of the economic assessment.

The average annual operating costs over the life of mine are expected to be approximately $28M and the LOM total is summarized in Table 1-7.

Table 1-7: Summary of Operating Cost Estimate

Operating Costs	$/t Milled	$/Payable AuEq Oz	LOM ($M)
Mining	35.83	378	145
Processing	14.55	154	59
G&A	7.37	78	30
Total	**57.76**	**609**	**234**

1.14 Economic Analysis

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

The Las Minas project is expected to derive its revenues from the sale of a copper concentrate containing gold and silver, as well as an iron concentrate into the international marketplace. Over an 8.5 year mine life, the project is expected to produce an approximate gold equivalent of 47k oz per year for a total of 383 k oz. Gold and Copper generate over 95% of the project net revenues.

The economic assessment used the commodity prices, treatment terms and tax rates are listed in Table 1-8. Base Case prices listed are between the current trailing 2-year and 3-year averages and were held constant over the LOM.

Table 1-8: Economic Assumptions

Assumptions	Unit	Value
Gold Price	US$/oz	1,625
Silver Price	US$/oz	20
Copper Price	$US/lb	3.25
Iron Concentrate Price	$US/dmt	100
Au Payable	%	94
Au Refining Charge	$/oz	5.00
Ag Payable	%	40
Ag Refining Charge	$/oz	0.40
Cu Payable	%	95
Cu Treatment Charge	$/dmt	65
Cu Refining Charge	$/lb	0.065
Cu Transportation Charge	$/wmt	56
Fe Concentrate Payable	%	100
Fe Transportation Charge	$/wmt	56
Mexican Corporate Tax	%	30
Mexican SMD Tax	% of EBITDA	7.5
Mexican EMD Tax	% of Gold & Silver Revenues	0.5

1.14.1 Results

A summary of the Las Minas project revenues, costs, taxes are shown in Table 1-9.

Table 1-9: Summary of Economic Results

Category	Unit	Value
Revenues	M $	623
Operating Costs	M $	234
Treatment/Refining/Transportation	M $	68
Cash Flow from Operations	M $	322
Initial Capital Costs	M $	90
Sustaining and Closure	M $	55
All-in Sustaining Cost# (net of by-product credits)	$/oz Au	145
All-in Sustaining Cost° (gold equivalent)	$/oz AuEq	928
Net Pre-Tax Cash Flow	M $	177
Pre-Tax NPV$_{5\%}$	**M $**	**114**
Pre-Tax NPV$_{8\%}$	**M $**	**86**
Total Taxes	M $	77
Net After-Tax Cash Flow	M $	99
Net After-Tax NPV$_{5\%}$	**M $**	**55**
Net After-Tax NPV$_{8\%}$	**M $**	**35**

Notes:

AISC formula: (Operating Costs + Refining Costs + Sustaining Capital + Closure – Net by-product credits) / Payable Au oz
° AISC formula: (Operating Costs + Refining Costs + Sustaining Capital + Closure) / Payable AuEq oz

1.14.2 Sensitivities

Sensitivity analyses were performed using metal prices, CAPEX, and OPEX as variables. The value of each variable was changed independently while all other variables were held constant. The results of the pricing sensitivity analysis are shown in Table 1-10. The other sensitivities are discussed in Section 23.

Table 1-10: Sensitivity of After-Tax Economic Results to Changes in Commodity Prices

	Base Case	Spot Price (July 29, 2021)	Upside	Downside
Gold (US$/oz)	1,625	1830	2000	1200
Silver (US$/oz)	20.0	25.5	28.0	14.0
Copper (US$/lb)	3.25	4.45	4.75	2.25
Iron Concentrate (US$/dmt)	100	213.5	220	65
Cumulative Cash Flow (US$M)	99	237	276	-22
After-Tax NPV5% (US$M)	55	157	187	-37
After-Tax NPV8% (US$M)	35	122	148	-43
After-Tax IRR (%)	16	31	35	-5
Capex Payback (Years)	4.4	2.8	2.6	n/a
EBITDA for First Year of Full Production (US$M)	43	70	77	19

Notes:

Upside and Downside commodity price scenarios represent the approximate high and low prices for each individual commodity in the last 3 years.

1.15 Conclusions

It is the conclusion of the Qualified Persons (QPs) that the PEA summarized in this technical report contains adequate detail and information to support the positive economic result shown by this study. The PEA proposes the use of industry standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the Las Minas Project.

1.16 Risks and Opportunities

1.16.1 Risks

The most significant potential risks associated with the project are uncontrolled dilution with waste rock and rock from different mineralized zones, geological complexity, data quality, drilling uncertainty, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing, metal price and US$ to Mexican peso exchange rate. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

1.16.2 Opportunities

The most significant opportunities are increasing the potential mine life by adding additional resources, optimizing the mine plan, increasing production and shipping concentrates to North American smelters.

1.17 Recommendations

It is recommended that the Las Minas Project proceed to the Pre-feasibility Study (PFS) stage in line with Mexican Gold's desire to advance the project. It is also recommended that environmental and permitting continue as needed to support Las Minas development plans.

This PFS will further detail:

- Mineral resources;
- Engineering design;
- Project scheduling;
- Process flowsheet parameters; and
- Capital and operating costs.

It is estimated that a PFS and supporting field work would cost approximately $2.2M.

Additional drilling is recommended to increase drill density to potentially achieve a higher resource category in higher-grade areas. Additional drilling may increase resources, improve understanding and modelling of lithological units and better define the limits of the mineralization.

A review of QA/QC procedures is recommended to improve data quality and increase confidence in the dataset.

A comprehensive brownfields exploration program in the area is recommended to explore for additional targets.

2 INTRODUCTION

JDS Energy & Mining Inc. (JDS) was commissioned by Mexican Gold to prepare a Preliminary Economic assessment (PEA) Technical Report following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, collectively referred to as National Instrument (NI) 43-101 for the las Minas Project located in Veracruz, Mexico.

One previous technical report, "*Technical Report and Estimated Resources for the Las Minas Project, Veracruz Mexico*" dated September 12, 2019 was completed on the project.

2.1 Scope of Work

This technical report summarizes the work of several consultants with the scope of work for each company listed below, which combined, comprises the total Project scope.

JDS Energy & Mining Inc. (JDS):

- Establishing an economic framework for the PEA;
- Mine engineering, design and scheduling;
- Geotechnical recommendations for underground mine design;
- Development of conceptual flowsheet, detailed flowsheets, specifications and selection of process equipment;
- Design oversight related to site infrastructure, access road, power line, plant facilities and other ancillary facilities;
- Estimating mining, process plant, G&A and site services OPEX and CAPEX;
- Establishing metal recovery values;
- Preparing a financial model and conducting an economic evaluation including sensitivity and Project risk analyses;
- Interpreting the results and making conclusions that lead to recommendations to improve Project value and reduce risks; and
- Developing and compiling the technical report and integrating sub-consultant report sections.

Kirkham Geosystems Ltd. (Kirkham):

- Deposit geology and mineralization;
- QA/QC, data verification; and

- Mineral Resource Estimation.

Knight Piésold (KP):

- Tailings management facility and mine rock management facility design;
- Overall project water balance;
- Water management, including design of ditches, channels and ponds for storm water controls; and
- Environment, Permitting, Social and Community Impacts.

2.2 Qualifications and Responsibilities

The Qualified Persons (QPs) preparing this report are specialists in the fields of geology, exploration, mineral resource estimation and metallurgy and mining.

None of the QPs or any associates employed in the preparation of this report has any beneficial interest in Mexican Gold and neither are any insiders, associates, or affiliates. The results of this report 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 Mexican Gold and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions / associations. The QPs are responsible for the specific report sections as listed in Table 2-1.

Table 2-1: QP Responsibilities

QP	Company	QP Responsibility / Role	Report Section(s)
Gord Doerksen, P. Eng.	JDS Energy & Mining Inc.	General items, Markets, Conclusions, Recommendations and Infrastructure	1.1,1.2,1.3,1.11,1.15,1.16, 1.17,2,3,4,5, 18 (except 18.6,18.7) 19,24,25,26,27,28,29
Michael Makarenko, P.Eng.	JDS Energy & Mining Inc.	Mining	1.4,1.9,12.4,15,16 (except 16.2)
Michael Levy, P. Eng.	JDS Energy & Mining Inc.	Mine Geotechnical	16.2
Tad Crowie, P. Eng.	JDS Energy & Mining Inc.	Metallurgy, Recovery Methods	1.6,1.10,12.3,13, 17

QP	Company	QP Responsibility / Role	Report Section(s)
Tysen Hantelmann, P. Eng.	JDS Energy & Mining Inc.	Capital & Operating Costs, Economic Analysis	1.13,1.14,21,22,23
Garth Kirkham, P. Geo.	Kirkham Geosystems Inc.	Geology, QA/QC, Data Verification, Drilling, Resource Estimate	1.4,1.5,1.7,6,7,8,9,10,11, 12.1,12.2,14
Ken Embree, P. Eng.	Knight Piésold	Tailings & Waste Management, Environment & Permitting	1.12,18.6,18.7, 20

2.3 Site Visit

Qualified Persons Garth Kirkham, P. Geo. and Gord Doerksen, P. Eng., visited the property January 16 through 19, 2021. The site visits included an inspection of the property, the adjacent hydro-power generation plant, offices, drill sites, outcrops, drill collars, core storage facilities, core receiving area, and tours of major centers and surrounding villages most likely to be affected by any potential mining operation.

The tour of the office and storage facilities showed a clean, well-organized, professional environment. On-site staff led the author through the chain of custody and methods used at each stage of the logging and sampling process. All methods and processes are up to industry standards and reflect best practices, and no issues were identified.

A visit to the collar locations showed that the collars were difficult to find in many cases as the area is subject to human and wildlife disturbance as well as experiencing significant weather and extensive vegetation. However, four collars were identified and marked however there appeared to be discrepancies between locations, particularly elevations, and recorded locations within the database. As a result of the uncertainties encountered, a plan to resolve was agreed with the company

A complete review of the drill core and sample chain of custody was performed and reviewed. All methods and procedures followed standard industry best practice and no issues were identified.

Based on the site visit and an inspection of all aspects of the project, The QPs are confident that the data and results are valid, including all methods and procedures. It is the opinion of the independent QPs that all work, procedures, and results have adhered to best practices and industry standards required by NI 43-101.

QP Gord Doerksen, P.Eng. provided site information to the other JDS QPs so they had a fully informed picture of the project. QPs Makarenko, Hantelmann, Crowie and Levy did not visit the site visit as there was little or no information available and things to see that would assist them in their work beyond what was provided by QP Gord Doerksen, P.Eng.

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 dollars (USD$ 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 sub-totals, 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

This report is based on information collected by JDS during a site visit performed between January 18-19, 2021 and on additional information provided by Mexican Gold throughout the course of JDS’s investigations. Other information was obtained from the public domain. The QPs conducted adequate verification of the information and take responsibility for the information provided by Mexican Gold.

3 RELIANCE ON OTHER EXPERTS

The QP's opinions contained herein are based on information provided by Mexican Gold 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.

Non-QP specialists relied upon for specific advice are listed below, along with the extent of their involvement and sections of the report to which their input applies:

- Raul Usla Lopez – Engineer, Comision Federal de Electricidad (CFE)
  - Electrical power distribution and capital costs required for the project to connect to the Mexican power grid summarized in CFE's memorandum dated June 18, 2021. The information contributed to Sections 18.4.5 and 21.1 of this report.

The QPs 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.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

Mexican Gold's Las Minas property is located in central Veracruz State, Mexico, approximately 160 km (by road) northwest of the city of Veracruz and 250 km east of Mexico City (Figure 4-1). The project area is located within and surrounding the village of Las Minas in the municipalities of Las Minas and Tatatila. The project has an approximate geographic center at 19º 41' 28" N latitude and 97º 08' 46" W longitude.

4.2 Mineral Tenure

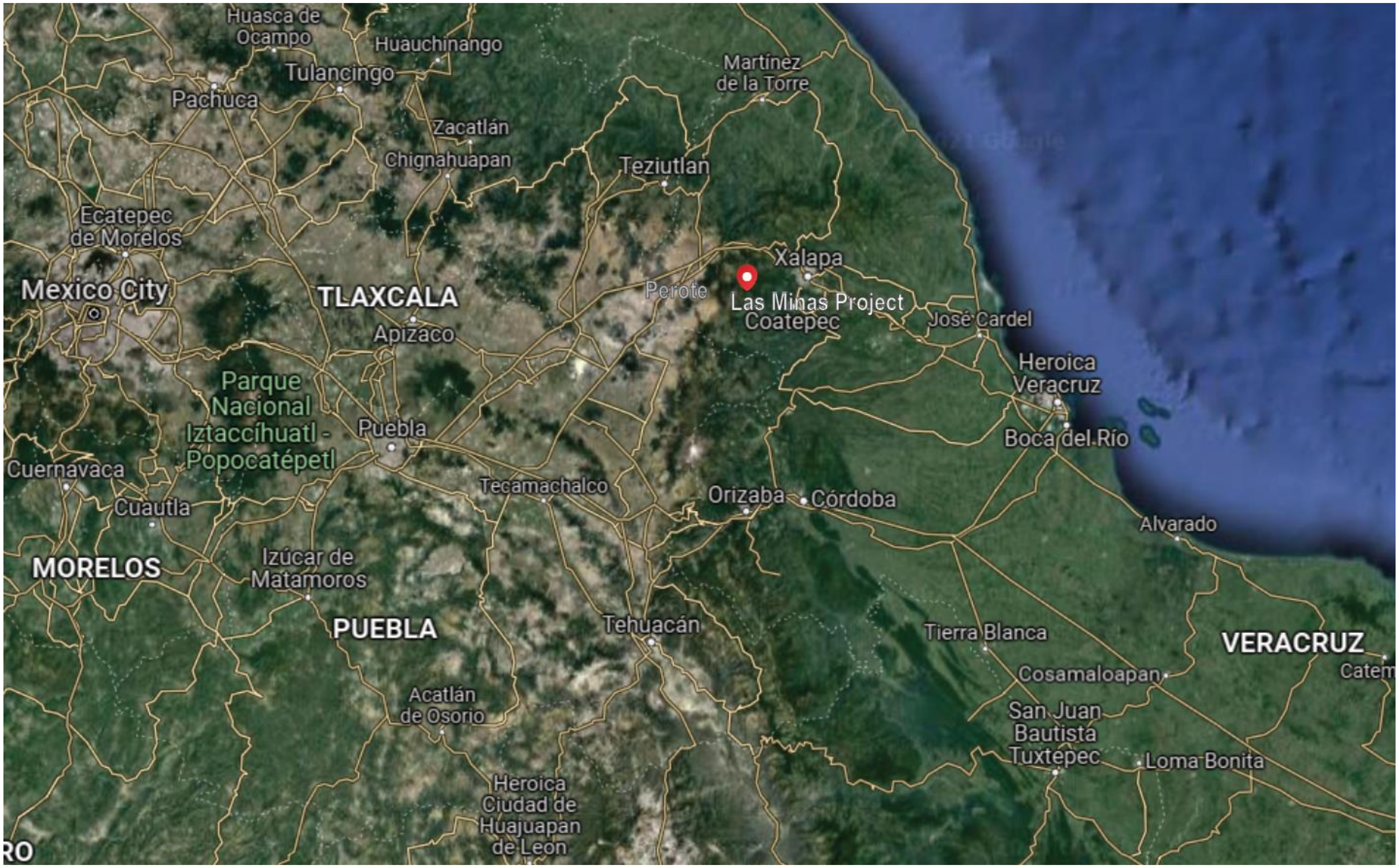
The Las Minas property consists of six mining concessions as listed in Table 4-1 that cover approximately 1,616 ha. The mining concessions are titled according to Mexican mining law. The titles are valid for 50 years from the date titled and can be renewed for another 50 years once they expire. The current mineral resources underlie the Pepe, and Pepe Tres concessions. All of the concessions are owned by Mexican Gold subject to underlying royalty agreements on five of the concessions as discussed in Section 4.3. The San Valentin concession was staked by Source Exploration Corp. (Source) in 2012 and carries no royalty burden.

Table 4-1: Las Minas Mineral Concessions

Concession ID	Title No.	Issuance Date	Expiration Date	Size (ha)
Pepe	195045	August 25, 1992	August 26, 2042	984
Pepe Tres	219668	March 27, 2003	March 27, 2053	121
San Jose	203831	October 8, 1996	October 9, 2046	12.7
Pueblo Nuevo	213450	May 10, 2001	May 10, 2051	97.5
La Luz I	215140	February 7, 2002	February 7, 2052	56
San Valentin	239890	February 29, 2012	February 28, 2062	345.2

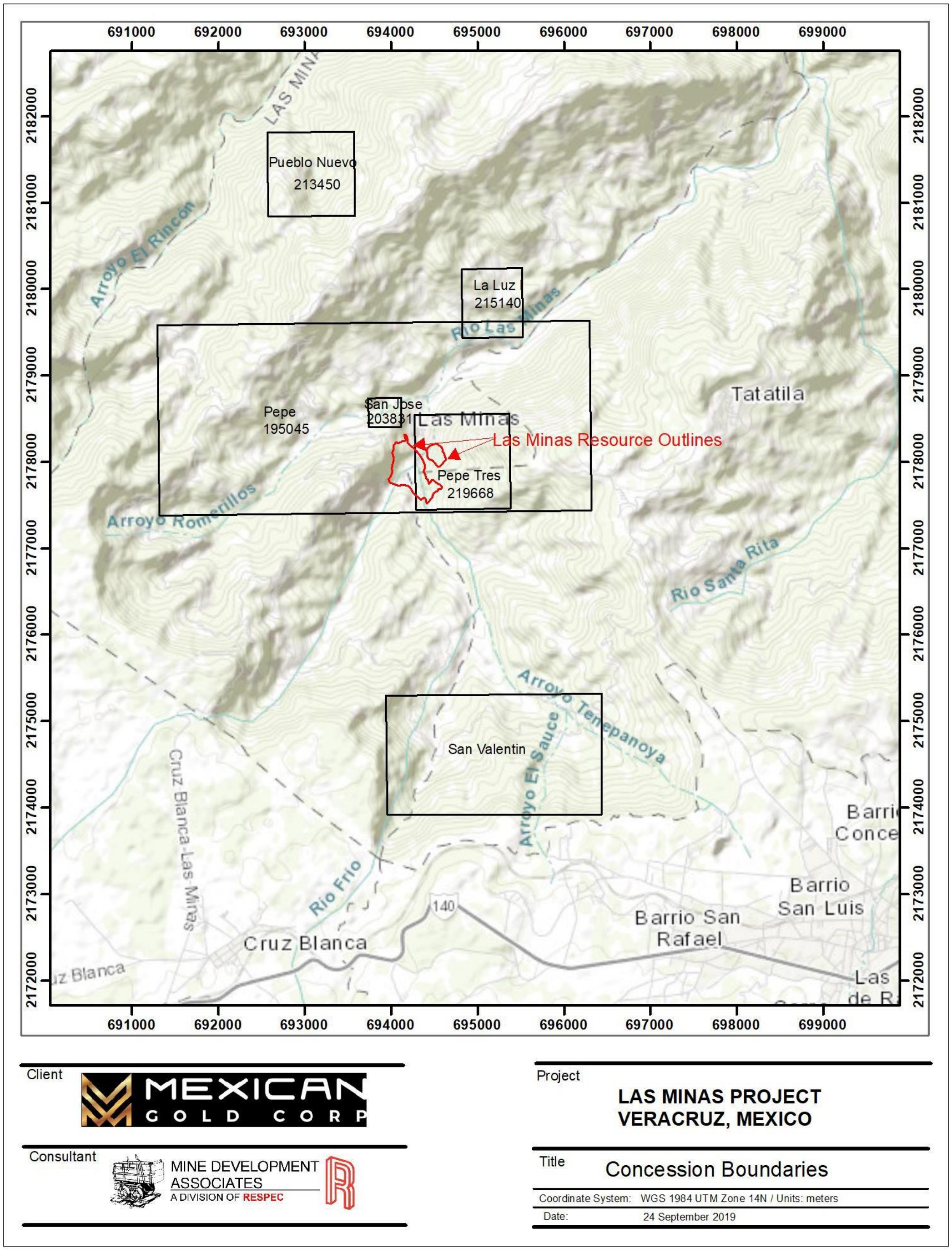
Source:

Figure 4-1: Location of the Las Minas Project



Source: Google (2021)

Figure 4-2: Las Minas Property Map



Source: MDA (2019)

4.3 Agreements and Encumbrances

The Pepe, Pepe Tres, and San Jose concessions were initially optioned in 2010 by Mexican Gold (formerly Source) through Roca Verde Exploración de Mexico (Roca Verde) a wholly-owned subsidiary of Mexican Gold. Mexican Gold purchased full ownership of the concessions via the following two underlying agreements, as per the title opinion provided by RGR, Mexican Gold's legal counsel. All of the concessions are owned by Mexican Gold with five of the concessions, including the two concessions containing the current mineral resources, subject to a 1.5% NSR royalty for production.

4.3.1 Farias Agreement

On May 9, 2017, Roca Verde and Mr. Ramon Farías Garcia executed an assignment of rights agreement regarding the Pueblo Nuevo and La Luz I concessions. The agreement was duly registered on July 12, 2017, at the Registro Público de Minería (RPM). With the execution of the agreement, Roca Verde granted Mr. Farías the right to a 1.5% NSR royalty for production from these two concessions.

4.3.2 Fernandez and Langenscheidt Agreement

On June 5, 2017, Roca Verde executed a purchase and sale agreement with Maria Elena Fernandez Anchondo (Fernandez) and Amalia Langenscheidt Salcedo (Langenscheidt), with reservation of domain regarding the Pepe, Pepe Tres, and San Jose concessions. The agreement on the Pepe Tres and San Jose concessions was duly registered on August 30, 2018, at the RPM, while the agreement regarding the Pepe concession was in progress according to the August 20, 2019, RGR title opinion.

On December 18, 2018, Fernandez and Langenscheidt executed an agreement to cancel the reservation of domain over the Pepe, Pepe Tres, and San Jose concessions. On February 13, 2019, said agreement was filed before the RPM and its recordation is still in progress as of the Effective Date of this report. With the execution of the agreement, Roca Verde granted Fernandez and Langenscheidt a 1.5% NSR for production from these three concessions.

4.3.3 Concession Status

The RGR title opinion states that all mining concession fees have been paid and legally required annual reports of proof of mining investments and work have been filed with the appropriate agency as of the Effective Date of this report.

4.3.4 Surface Ownership

Surface ownership across the property is mixed. Most of the surface is owned by various individuals with one very small parcel in the village owned by the local ejido, one parcel covering the hydroelectric plant owned by the Comision Federal de Electricidad (CFE), and Mexican Gold owns one parcel covering the Santa Cruz deposit, purchased in 2014. Several agreements are in place with individual landowners to grant surface access for exploration. The author has not conducted a review of these documents and is not aware if an independent legal review of surface ownership and Mexican Gold surface rights has been completed.

4.4 Environmental Liabilities and Considerations

Historic mining in the area has left numerous underground workings, mostly exposed near the canyon bottom, and there has been small-scale surface mining associated with the underground workings. Available information indicates that the majority of these are not extensive; most within the resource area have an underground extent on the order of 10 to 30 m.

Of the workings observed during the author’s site visit, none exhibited drainage. Only minor evidence of tailings or dumps are present, and most were likely washed away by the Rio Las Minas during storm events.

4.5 Permit Requirements

Mexican Gold has had an exploration permit issued by the federal agency Secretaria de Medio Ambiente y Recursos Naturales (SEMARNAT) in place since November 2018. The current permit, which allows for 25,000 m of drilling, expires in three years from the date of issuance, and covers all exploration-related activities at the project. The permit can be extended under certain circumstances. The author has not reviewed this permit.

4.6 Mining Rights

Mexican Gold does not have a mining permit as the project is still in the exploration stage.

4.7 Property Risks

The author is not aware of any significant factors or risks that may affect access, title or the right or ability to perform work on the property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The property is centered on the village of Las Minas which is partially connected by four-lane highways to the cities of Veracruz to the south and Mexico City to the west. From Veracruz, the village can be accessed by Highway 180 and then Highway 140 for a distance of 150 km, then turning north at the village of Cruz Blanca onto a 15 km gravel road that descends into the Rio Las Minas canyon. From Mexico City, access is via Highways 150D and 140D for a distance of 250 km to the turn-off at Cruz Blanca.

5.2 Climate

Las Minas has a temperate humid climate with an average annual temperature of 17.6º C. Temperatures vary from around 12º C in the winter to 20º C in the summer. Rainfall typically averages around 1,500 mm per year with rain common in the summer and into early autumn. Mining and exploration can be conducted year-round.

5.3 Physiography

The Las Minas property is characterized by very steep topography with moderately dense vegetation. The village of Las Minas, and the gold-copper skarn deposits which are the focus of the current resource estimate, lie at the bottom of the Las Minas canyon at an elevation of around 1,400 m. Project elevations at outlying exploration targets, such as Changarro [also spelled as Changaro on many maps and in reports; the names appear to be interchangeable. Changarro will be used in this current report] or Pueblo Nuevo, reach up to 1,800 m elevation with extreme (>500 m) local elevation differences. Access into these outlying targets is by footpath, many of which have been rehabilitated by Mexican Gold.

5.4 Local Resources and Infrastructure

The village of Las Minas has an estimated population of 2,900 people and semi-skilled labor can be sourced locally. Mexican Gold has a field office in the village and limited living and office accommodations are available. The town of Perote (population 38,000) is located 25 km southwest of Las Minas and is a primary source of supplies. The main economic activity in Las Minas is farming. Corn, beans and coffee are the main crops grown, and scattered pastures are maintained for the grazing of goats and sheep.

The Las Minas hydroelectric plant is located 0.5 km up-river from the village and has a capacity of 15 MW. The Rio Las Minas flows through the village and directly over the skarn deposit; this river is a potential source of water.

The process plant and infrastructure locations and the ability to store waste rock and tailings have been addressed with PEA-level designs and are found in the appropriate sections of this report. Other elements required for an operating mine such as water and power supply are available in adequate supply.

6 HISTORY

6.1 Management, Ownership and Exploration History

The Tatatila – Las Minas region is one of the oldest recognized mining regions in the state of Veracruz and there are many old mine workings distributed throughout the Las Minas Project that represent historic mining where high-grade values were exposed at surface mostly at stream-bed elevations within the very rugged terrain. Little modern exploration has been conducted within the Las Minas Project nor in the general region. Initial mining activities for gold date back to pre-Colonial times and were continued by Spaniards using arrastras. During the 1800s, several individuals had filed *denuncios* (mineral claims) in the region, but only minor extraction of mineralized material occurred. European interests employing stamp mills and concentrating tables conducted operations in the late 1800s to early 1900s, prior to the Mexican Revolution. Reportedly, high- grade mineralized material (20 g/t to 40 g/t Au and 15% to 30% Cu) was produced during the period from 1870 to 1910. It was during this time that most of the historical workings within the current Las Minas resource areas, along with workings within many of the current exploration targets, were constructed. Following the Mexican Revolution, some small-scale extraction of gold and copper occurred with high-grade mineralized material being shipped to San Luis Potosi, New York, and Europe.

More recent activity includes an evaluation in the 1970s by the federal Consejo de Recursos Minerales (CRM) which is now the Servicio Geológico Mexicano (SGM). This program included geologic mapping and sampling. In 1996, concessions were acquired by International Northair who also conducted a surface and underground sampling program. Results from 583 channel samples from surface and underground workings confirmed gold-copper mineralization over a wide area. The average metal grade from all samples was 1.7 g/t Au and 0.39% Cu. International Northair formed a joint venture in 1996 with Battle Mountain Gold (BMG) to explore the property. BMG subsequently terminated their participation in the project.

In 2006, Bell Resources acquired an option on properties in the Las Minas area and subsequently optioned them to Chesapeake Gold (Chesapeake) in 2007. Chesapeake undertook an extensive sampling program which showed several areas were anomalous in gold and copper, including the Las Minas area. Chesapeake consolidated additional ground in the area via staking and purchasing a small, mineralized area from the CRM.

Apparently neither Chesapeake nor Bell Resources ever controlled the core of Las Minas district, having never optioned the principal claims, although their sampling covered it. Chesapeake, however, still controls a large land position surrounding Mexican Gold's concessions.

In 2010, Mexican Gold (then named Source) leased the Pepe, Pepe Tres, and San Jose claims which make up the principal part of the Las Minas project. Several other claims located peripheral to the principal area were subsequently acquired. Mexican Gold's first sampling took place in 2010 and their first drilling campaign was carried out in 2011. Prior to this, no known drilling has occurred in the resource area. Source changed their name to Mexican Gold Corp. in April of 2017.

Exploration and drilling by Mexican Gold is discussed in Sections 9 and 10, respectively.

6.2 Historical Mineral Resource Estimate

One historical mineral resource estimate was completed on the Las Minas project. In 2017, Mexican Gold commissioned Boyd and Associates (Boyd) to prepare an independent technical report on the Las Minas project and to estimate the mineral resources. The report had an effective date of July 31, 2017 and was reportedly prepared to NI 43-101 standards (Read and Shoemaker, 2017). The 2017 estimate is relevant only for historical completeness, is not considered current, and has been superseded by the current mineral resources described in Section 14.

Boyd used inverse distance cubed (ID3) to estimate gold, copper, and silver resources within the El Dorado and Santa Cruz zones. Wireframes for the two zones, each based on an approximate gold equivalent (AuEq) cut-off grade of 0.5 g/t AuEq, were used to constrain mineralization and each metal was estimated independently within the wireframe. The block model used a variable block size of 10 m by 10 m by 10 m, sub-blocked to a minimum of 5 m by 5 m by 5 m, and model coding was block-in/block-out. A density of 3.28 g/cm^3 was used for mineralized rock and 2.7 g/cm^3 for waste. Both open pit and underground resources were estimated using cut-off grades of 0.46 g/t and 1.60 g/t AuEq for potential open pit and underground mining scenarios, respectively. The reported open pit resources were constrained within an economic pit shell calculated using the Whittle pit optimizer while the underground resources were those resources outside the pit shell. The metal prices used for the AuEq calculations are shown in Table 6-1 and the total estimated resources are shown in Table 6-2.

Table 6-1: Metal Prices and Recoveries Used by Boyd

	Metal Price	Metallurgical Recovery
Au	$1,250/oz	88%
Cu	$2.85/lb	95%
Ag	$16.75/oz	84%

Source: Read and Shoemaker (2017)

Table 6-2: Resource Table Summarizing 2017 Historical Open Pit and Underground Resources

Mining Method	Classification	Estimate Grade						Contained Gold Equivalent Troy Ounces
		Cut-off Grade (g/t Au)	Gold Equivalent (g/t Au)	Gold (g/t Au)	Copper (% Cu)	Silver (g/t Ag)	Tonnes	
Open Pit	Measured	0.46	2.32	1.24	0.65	4.66	62,000	5,000
	Indicated	0.46	1.86	0.90	0.58	4.33	4,685,000	281,000
Subtotal	**Measured + Indicated**	**0.46**	**1.87**	**0.91**	**0.58**	**4.34**	**4,747,000**	**286,000**

Mining Method	Classification	Estimate Grade						Contained Gold Equivalent Troy Ounces
		Cut-off Grade (g/t Au)	Gold Equivalent (g/t Au)	Gold (g/t Au)	Copper (% Cu)	Silver (g/t Ag)	Tonnes	
	Inferred	0.46	2.06	1.02	0.64	3.69	9,121,000	605,000
Underground	Measured	1.60	0.00	0.00	0.00	0.00	0	0
	Indicated	1.60	2.46	1.24	0.74	5.05	223,000	18,000
Subtotal	**Measured + Indicated**	**1.60**	**2.46**	**1.24**	**0.74**	**5.05**	**223,000**	**18,000**
	Inferred	1.60	2.99	1.57	0.86	6.25	1,183,000	114,000
Open Pit + Underground	Measured	-	2.32	1.24	0.65	4.66	62,000	5,000
	Indicated	-	1.89	0.92	0.58	4.36	4,908,000	299,000
Subtotal	**Measured + Indicated**	**-**	**1.90**	**0.92**	**0.59**	**4.37**	**4,970,000**	**304,000**
	Inferred	-	2.17	1.08	0.66	3.99	10,304,000	719,000

Notes:

1. The effective date for this mineral resource estimate is July 31, 2017 and are reported on a 100% ownership basis.
2. Mineral resources are calculated at a gold price of US$1,250 per troy ounce, a copper price of US$2.85 per pound and a silver price of US16.75 per troy ounce. Gold equivalent grade is calculated as AuEq (g/t) = Au (g/t) + Ag (g/t) * 0.013 + Cu (5) * 1.564. The factors for silver (0.013) and copper (1.564) will change depending on the metal price. The metal price numbers listed above were used to determine the conversion factors presented herein.
3. The mineral resources presented above are global and do not include a detailed pit or underground, only an economic pit shell was used to determine the in-pit mineral resources. The underground mineral resources are that material outside of the designated open pit mineral resources above the stated underground cut-off grade.

Source: Read and Shoemaker (2017)

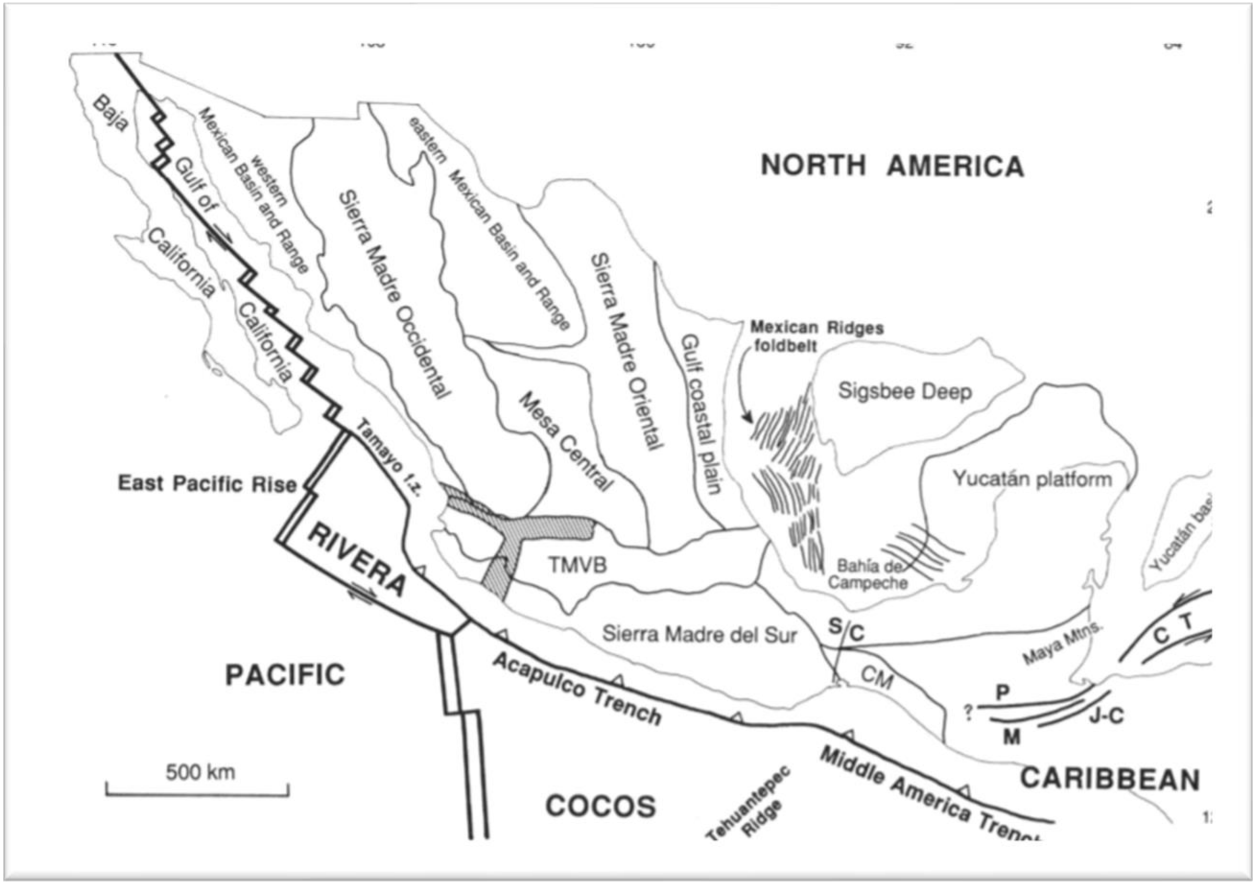
7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Las Minas project is located in southeastern Mexico within the eastern portion of the Trans Mexico Volcanic Belt (TMVB), an east-west belt of Miocene to recent volcanic rocks that transects the country from the Pacific coast to the Gulf of Mexico. The TMVB is a 1000 km long Neogene continental arc showing a large variation in composition and volcanic style, and an intra-arc extensional tectonics. It overlies the Rivera and Cocos slabs, which display marked changes in geometry (Ferrari et al, 2012, Campa and Comey, 1993).

These rocks are dominantly of calc-alkaline affinity and are related to subduction of the Rivera Plate to the west, and the Cocos Plate to the east, off the southern coast of Mexico as shown in Figure 7-1, (Ferrari et al, 2012; Gomez- Tuena et al, 2003, Sedlock et al, 1993).

Figure 7-1: Major Morphotectonic Features and Plate Tectonic Setting of Mexico



Source: Sedlock et al. (1993)

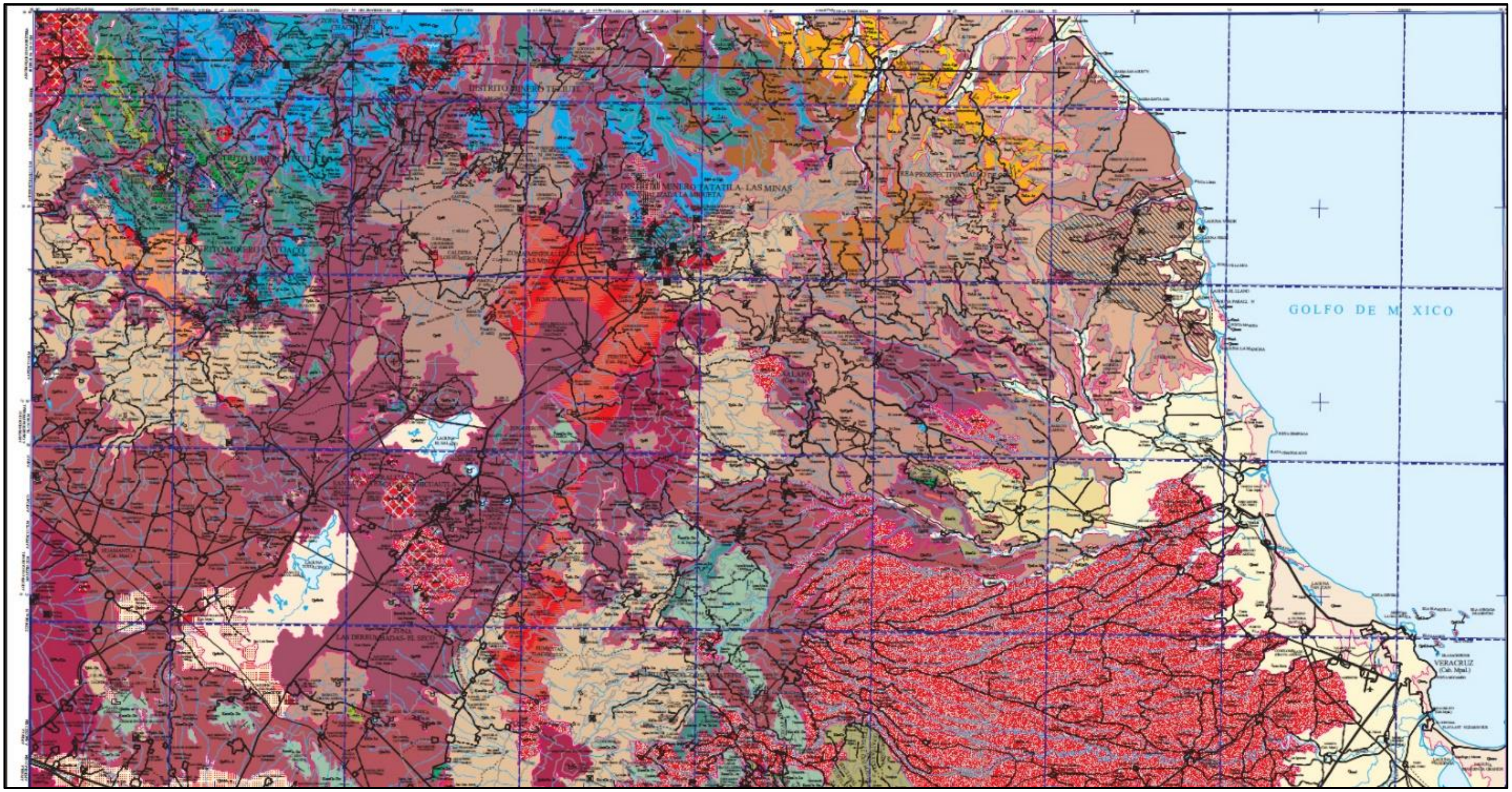
Portions of the TMVB are presently volcanically active. The high plains surrounding the Rio Las Minas canyon are comprised of pumaceous deposits, basalt flows and andesitic and rhyolitic tuffs. In this portion of the TMVB, the volcanic rocks are principally of Pleistocene to Holocene ages. Large stratovolcanos and shield volcanos occur in the region (e.g., Cofre de Perote, located just southeast of the town of Perote; and Pico de Orizaba, located approximately 60 km south of Perote on the Veracruz-Puebla border).

To the north of the volcanic plain, older rocks are exposed at lower elevations. These are principally Jurassic and Createaceous limestones and shales which constitute the southern portion of the Sierra Madre Oriental, an extensive north-trending mountain chain that extends northward to north-western Mexico. The Las Minas-Tatatila region is considered the southern terminus of the Sierra Madre Oriental because it is in this region that these rocks go under the volcanic cover of the TMVB.

The pre-Miocene basement in the Las Minas region consists of a sequence of Jurassic and Cretaceous marine sedimentary rocks including sandstone, siltstone, limestone and shale (Vergara Martinez et al., 2002). Tertiary and some Mesazoic plutonic rocks intrude the sedimentary base strata locally throughout the region. These have been mapped as dominantly granodiotite and porphyritic dacite, with lesser granite, diorite and tonalite (Vergara Martinez et al., 2002). In the project area, the intrusive rocks, through mapped as granodiorite, are principally dioritic in composition. Two potassium-argon (K-Ar) age determinations from the intrusive rocks in the Las Minas-Tatatila area gave dates in the range of 14-14.5 million of years from the present (Ma) (SGM, 2007).

Pre-tertiary rocks are exposed approximately 100 km west of the Las Minas-Tatatila area and are folded and thrust-faulted. These structures have a dominant northwest trend and are related to the Laramide (latest Createaceous-Paleocene) deformation of the Sierra Madre Oriental. In the Las Minas-Tatatila area, this same occurrence of rocks exhibits gentle dips, principally to the north-east-striking normal faults with lesser north-northwest faults also present. The regional geologic setting is shown in Figure 7-2 and Figure 7-3.

Figure 7-2: Regional Geologic Setting of the Las Minas Area



Source: Vergara Martinez et al. (2002)

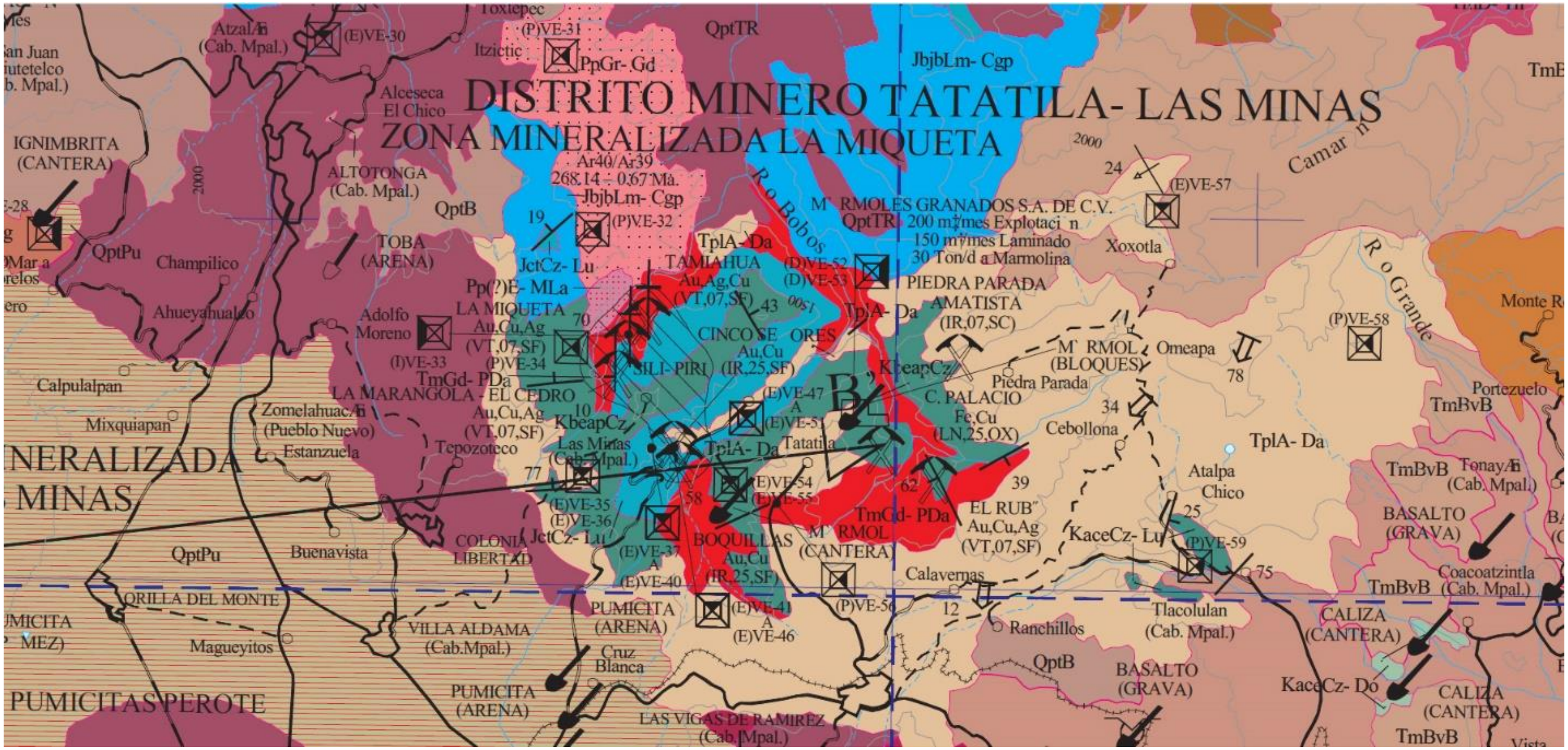
Figure 7-3: Regional Geology Setting

Source: Vergara Martinez et al. (2002)

7.2 Property Geology

The units exposed in and surrounding Mexican Gold's concessions are: (1) Cretaceous limestone which has been interpreted as Orizaba Formation (SGM, 2007a; 2007b); (2) Miocene-age dioritic intrusions, and (3) Quaternary dacitic to rhyolitic pyroclastic rocks. These units are cut by northeast-striking normal faults, one of which is where the Rio Las Minas canyon is located. The intrusion(s) exhibits a roughly circular outcrop pattern approximately 10 km in diameter surrounding Las Minas. The intrusions within the Las Minas property have been interpreted as the upper exposed portion of a batholith (Vergara Martinez et al., 2002), though it should be noted that the geometry at depth is largely unknown because most drillholes end in the intrusion. It has been suggested (M. Liard, pers. comm.) that the intrusion may represent a large sill as the upper contact is mostly planar and parallel with the overlying sedimentary sequence. Near the contact with the intrusion, the Cretaceous limestone has been contact-metamorphosed to marble, forming a thick metamorphic aureole. It is at the intrusive-marble contact that the mineralized skarn deposits were formed. Figure 7-4 depicts the geology in the project area.

Figure 7-4: Property Geology Map



Source: Vergara Martinez et al. (2002)

The principal intrusion and adjacent skarn are mainly exposed at lower elevations near the bottom of the Rio Las Minas canyon. These units are also exposed at somewhat higher elevations in the Molinillos and Macuerna areas. Tertiary volcanic rocks are exposed mainly at higher elevations surrounding the project area, although some dacitic rocks occur along the river northeast of the pueblo, presumably occupying a small graben or occurring as valley-fill.

The limestone of the Orizaba Formation is part of a platform carbonate sequence with thicknesses greater than 1 km has been described as unmetamorphosed limestone as pale gray in color and fossiliferous with local black chert (Geoconsulting Ingenieros, 2010). The marbleized part is pale gray to white and varies from fine- to coarse-grained with a granoblastic texture (anhedral, equigranular texture locally with sutured grain boundaries). The sedimentary sequence is horizontal to subhorizontal with regional dips up to 10°. Within the El Dorado resource area, the sedimentary sequence dips gently to the east.

Mapping within the lowest canyon elevations in the Pueblo Nuevo claim area shows clastic meta-sedimentary rocks, primarily phyllitic arkosic greywacke and quartzite, lying beneath the limestone just above the diorite intrusion.

Mapping of the principal intrusions in 2010 indicated them to be composed predominantly as granodiorite but ranging in composition from diorite to granite. In cases where they are weathered or altered, the unit is green and white with a fine- to medium-grained equigranular texture. Where the rock is fresh, it is light to dark grey, with colorless plagioclase and black horneblende. The rock is usually magnetic. The green color is presumably due to chloritization and sausseritization of the original mafic minerals. It is composed of calcic plagioclase with hornblende and may or may not include biotite. Accessory minerals include magnetite and ilmenite spatially associated with hornblende.

The diorite is spatially and genetically associated with skarn development. Drillhole logging indicated multiple phases of the diorite though this might reflect the type and degree of alteration. Locally there is evidence of a post-mineral phase as the diorite occurs as dike-like bodies that intrude through the skarn. In these cases, the intrusion appears nearly identical to the diorite evidenced elsewhere. Less well-defined is a post-mineral feldspar-porphyritic intrusive phase that has been intersected in a small number of drillholes. It displays equant, zoned plagioclase phenocrysts in a black fine-grained groundmass which is distinct from the diorite. The exact geometry and relationship to the main diorite body is currently unknown.

Several minor, un-mineralized dikes and sills composed of feldspar porphyry are known to cut the principal intrusion and sedimentary rocks in the project area. These are interpreted as post-mineral dikes and sills that are unrelated to the mineralization.

7.3 Alteration and Mineralization

Copper and gold mineralization have been recognized in three settings: proximal skarn, distal skarn and quartz veins.

7.3.1 Proximal Skarn

Proximal-type skarn is the dominant skarn alteration observed within the Las Minas deposits. Proximal skarn crops out in a narrow band along the canyon bottom to the west and south (Juan Bran historical mine area) and east (El Dorado mine area) of the Las Minas pueblo, and in a broader area in the Santa Cruz- Nopaltepec area which is located approximately 500 m south of village. At surface, the skarn occurs as a dark, weathered outcrops, commonly with copper staining, which are the focus of many small historical underground workings. The skarn developed along a somewhat irregular contact between marble and diorite. This is due to local irregularities in the contact geometry of the intrusion (sill, dike, and sub- horizontal basement contacts), and to the fact that some of the intrusion was altered to endoskarn.

Skarn alteration is dominantly garnet-rich with lesser amounts of pyroxene. Locally garnet appears to be replacing pyroxene. The garnet has been described as garnet as grossularite to andradite in composition, although it is not clear that this is based on actual mineral chemistry or just color variations (Geoconsulting Ingenieros, 2010). Pyroxene is a pale to medium green color. The skarn contains variable amounts of magnetite and sulphide minerals. At least some magnetite is usually present and can range from several percent of the rock to massive black metallic magnetite containing lesser calc-silicate minerals. In places, garnet has been observed cutting magnetite, indicating that magnetite was deposited early at high temperature. This is common in many proximal skarn occurrences.

Retrograde alteration of garnet and pyroxene to hydrous calc-silicate minerals is very minor at Las Minas. Epidote is occasionally abundant, with or without quartz, and is interpreted as being relatively late stage. Sulphide deposition likely occurred at lower temperatures during retrograde alteration. At Las Minas, chalcopyrite is the dominant sulphide mineral however, sulphide mineral content is variable. Sulphide grains usually are associated with magnetite and are present as relatively coarse-grained disseminations. Blebs, bands, and veinlets cutting magnetite were also observed. Pyrite occurs as an accessory mineral in the main resource area but is locally abundant in some of the peripheral skarn occurrences in the district such as Cinco Senores and Las Minillas. Lesser, though locally abundant, amounts of bornite also occur and chalcopyrite with quartz was also observed sporadically and copper carbonate minerals (malachite, azurite) occur at the ground surface. Within the Santa Cruz skarn zone, calc-silicate mineralogy locally includes olivine and serpentine. This magnesian skarn assemblage probably reflects local dolomitic protolith in the carbonate sequence.

Skarn within the Las Minas resource zones occurs both as exoskarn developed within the sedimentary rock and as endoskarn formed adjacent to the intrusion. The skarn alteration has a typical zoning of marble to exoskarn to endoskarn to diorite. Overall, this sequence occurs in a vertical sense from up (marble) to down (diorite) in the El Dorado zone. The alteration sequence can be inverted or horizontal in the more complex Santa Cruz zone where sill or dike contacts are more irregular than at El Dorado. Contacts of exoskarn with the weakly skarn-altered marble are commonly knife-edge with massive magnetite sometimes occurring at the contact.

The distinction between exoskarn and endoskarn can be very difficult because the skarn alteration (especially garnet replacement) can be texturally destructive. However, somewhat commonly, banded texture is observed in the skarn, reflecting relict bedding in the marble. This sometimes is exhibited as thin bands of magnetite-chalcopyrite in skarn.

Less commonly, igneous texture can be observed, indicating an intrusive protolith. The endoskarn transitions somewhat gradationally to chloritized, sausseritized diorite and finally into fresh diorite.

There is a zone of pervasive, post-mineral pyrite that affects, most notably, the Cinco Senores and Las Minillas areas. Here, the chalcopyrite-rich magnetite skarn has been strongly overprinted by pyrite, and the multiple, prominent dioritic dikes typical of these areas are also strongly pyritized and clay-altered. Pyrite flooded diorite may contain significant gold values of 300 to 1,300 ppb Au over 10's of metres, with or without associated skarn, as seen at the Mina Blanca occurrence. It is not known whether the pyritization is late- stage retrograde alteration, or a later Miocene event.

Gold within the proximal skarn occurs in magnetite- and sulphide-bearing skarn, especially with chalcopyrite. Copper occurs dominantly in chalcopyrite and in lesser, though locally abundant amounts in bornite. Silver is also present in the mineralized skarn, associated with gold and copper. It is not known in what phase(s) silver occurs.

Proximal skarn zones are often characterized by hydrothermal dissolution breccias in the overlying rocks, having random, anastomosing form extending up to 500 m into the limestone above the known skarns. The breccia matrix consists mainly of fine-grained calcite and iron-carbonate and is generally strongly hematitic red but may be dark green where unoxidized. Breccia clasts are very poorly sorted, angular to sub-rounded fragments of sub-mm to 30-cm sizes. Textures indicating clast rotation, milling and dissolution are common. In marble, this event is expressed more as multiple, irregular anastomosing breccia "veins" that are 1 cm to 20 cm wide. Hydrothermal breccia is also observed in intrusive rocks and skarn, with associated local areas of diorite altered to dark green chlorite and clay.

7.3.2 Distal Skarn

Distal skarn is exposed within the canyon walls at elevations from ~1,700 m to 2,100 m. The distal skarn is highly variable in character. Distal skarn consists of massive sulphides and/or massive magnetite in stockworks, pipes, and replacement pods within the variably marbleized limestone.

There are also swarms of discrete, narrow, co-planar magnetite-sulphide veins, all hosted in marble. When directions can be established, the co-planar veins generally trend north-south with near-vertical dips and with sub-horizontal conjugates, though east-west striking, near-vertical vein orientations are also observed.

Sulphide minerals include (in order of abundance) chalcopyrite, sphalerite and galena. As at Las Minas, pyrite, as a late alteration product, is rare and pyrrhotite or sulpharsenides have not been noted.

A low-temperature hematite-jasperoid-calcite alteration type is also present spatially associated with areas of distal skarn. All of the textures are hydrothermal and there is no evidence of a direct genetic relationship with any dikes which may be found in the mineralized zones.

Mineralization associated with various distal skarn mineral types is seen within the Changarro – La Perdida area at various locales such as Mina Changarro which appears to have exploited a near- vertical pipe of massive magnetite plus sulphides that has a width of 50 m. to 100 m.

Additionally, Guadalupe 2 presents a stockwork of massive chalcopyrite stringers, with high gold grades, in shattered marble adjacent to a significant north-south trending fault. There also two closely-spaced prospect pits in the canyon wall near Mina Escondida that expose texturally identical sheeted swarms of veinlets of gold-bearing sphalerite in one prospect pit, and veinlets of barren magnetite in the other pit.

The low-temperature jasperoid alteration type was worked at Mina La Perdida - Mina Escondida. The gold-bearing jasperoid contains no sulphide minerals, and gold values appear to be relatively proportional to intensity of jasperoid alteration.

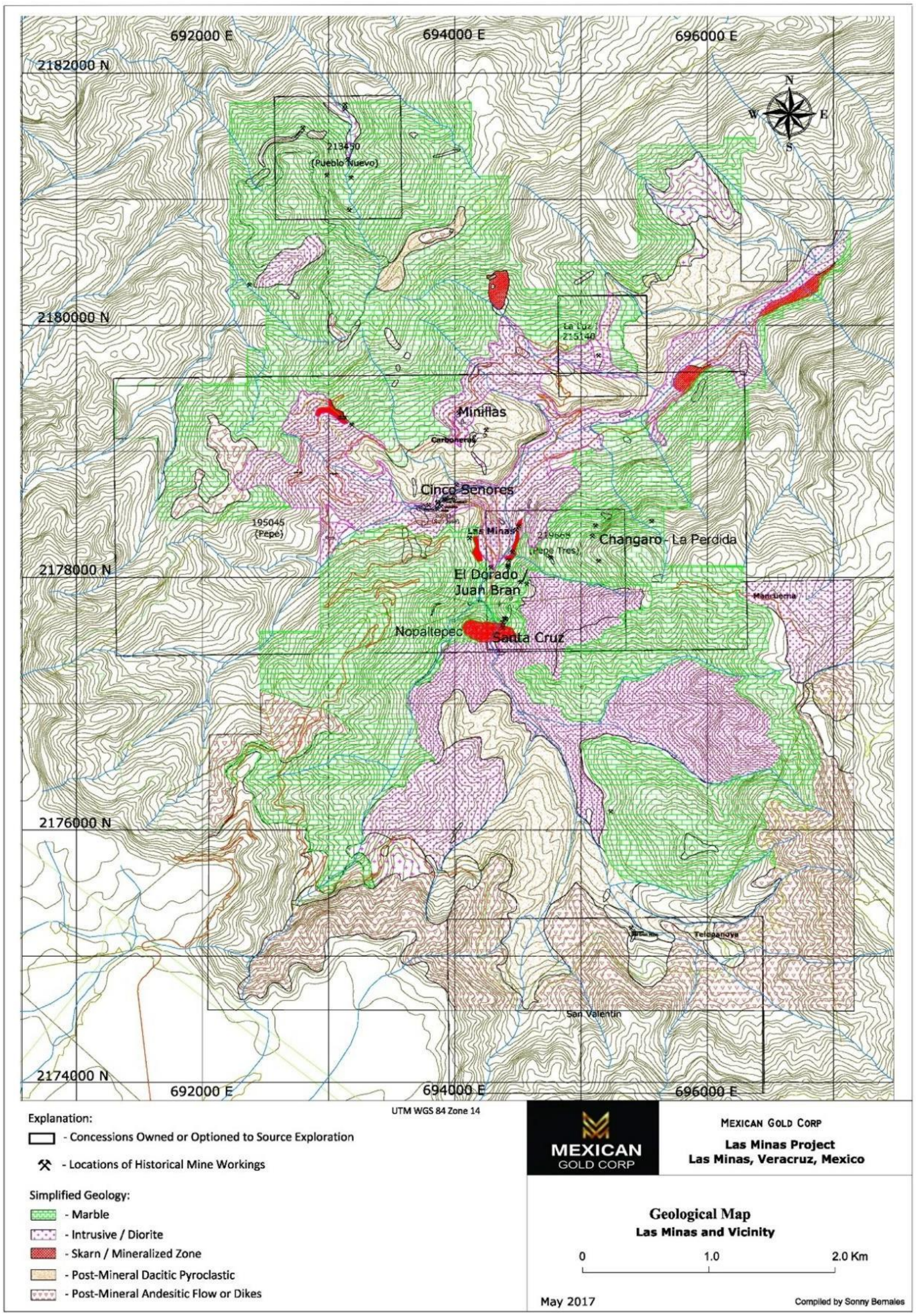
7.3.3 Gold-bearing Quartz Veins

Gold-bearing veins are found primarily at Pueblo Nuevo which is located about 4 km north of the Las Minas village. The veins are narrow, with widths ranging from 0.2 m to 0.8 m, strike generally within 10° of north-south, and have near-vertical dips. Veins with significant gold values persist over >1.5 km of strike and >650 m of elevation within a 600 m wide deformation zone, though the veining and deformation may be much more extensive. Vein gangue material is almost exclusively quartz, showing repeated or zoned open-space filling textures. Gold is associated with pyrite that forms < 5% of the vein fill, and with trace to locally 10% iron-rich sphalerite, galena and minor copper and silver minerals. These veins are prominent in both undeformed dioritic rocks which are believed to be the same intrusion that underlies the main skarn deposits at Las Minas, and in the overlying clastic meta-sedimentary rocks and sheared limestone, as observed at the La Miqueta mine. The lower levels of the veins within the meta-clastics tend to show higher gold grades than in the higher level, limestone-hosted portions of the veins. Wallrock alteration is generally minimal in all country rock types.

7.4 Deposit Geology and Mineralization

The following information describes the two primary mineral zones that contribute to the current resource estimate, with additional summary information on other exploration targets within the Las Minas property. Figure 7-5 shows the location of the Juan Bran, El Dorado and Santa Cruz zones along with the Changarro-La Perdida, Pueblo Nuevo, and Cinco Senores prospects.

Figure 7-5: Geology of Las Minas Project



Source: Bernales (2017)

7.4.1 El Dorado-Juan Bran Zone

Previous reports had used the name “El Dorado/Juan Bran” when referring to the sub-horizontal skarn mineralization extending south from the historical Juan Bran and El Dorado mine areas. Recent exploration has shown that the mineralization at both areas extends to the south along the same geologic horizon and has similar geologic characteristics. Due to the noted similarities, Mexican Gold has chosen to use just “El Dorado” as the zone name and this terminology will be used throughout this technical report. “Juan Bran” in this report designates only the historical mine area. The northern portion of the El Dorado mineral zone crops out along the canyon bottom to the southwest and southeast of the Las Minas pueblo.

El Dorado skarn occurs primarily at the contact between diorite and overlying marble. Being visually distinctive from the surrounding marble and intrusion, the skarn is seen in outcrop as dark-colored, with significant iron- and copper-staining. The occurrence of mineralized skarn was obvious to early explorers and was the locus of historical activity at the Juan Bran mine area on the west side of the canyon and El Dorado mine on the east side of the canyon. Both historical mine areas are marked by shallow adits and surface excavations worked along the canyon bottom, though greater production appears to have come from the El Dorado workings.

The El Dorado mineral zone had previously been considered a single contiguous, sub-horizontal skarn that extends to the south into the sub-surface under the rising canyon topography south of the Las Minas pueblo. Drilling by Mexican Gold indicates that the El Dorado zone is broken into two horizons that are separated by a barren, north-northwest trending, 100 m to 150 m wide diorite dike that extends up from the lower diorite intrusion.

Modeling indicates that the El Dorado skarn mineral zone on the west side of the diorite dike has an 800 m northwest strike length, extends up to 450 m to the southwest away from the diorite dike, is on average 15 to 20 m thick, and can reach over 50 m in thickness along the northwest-striking contact with the diorite dike. In contrast, the El Dorado zone on the east side of the dike has a strike length of 250 m northwest, extends up to 200 m to the northeast from the diorite dike, and is 5 to 10 m in thickness.

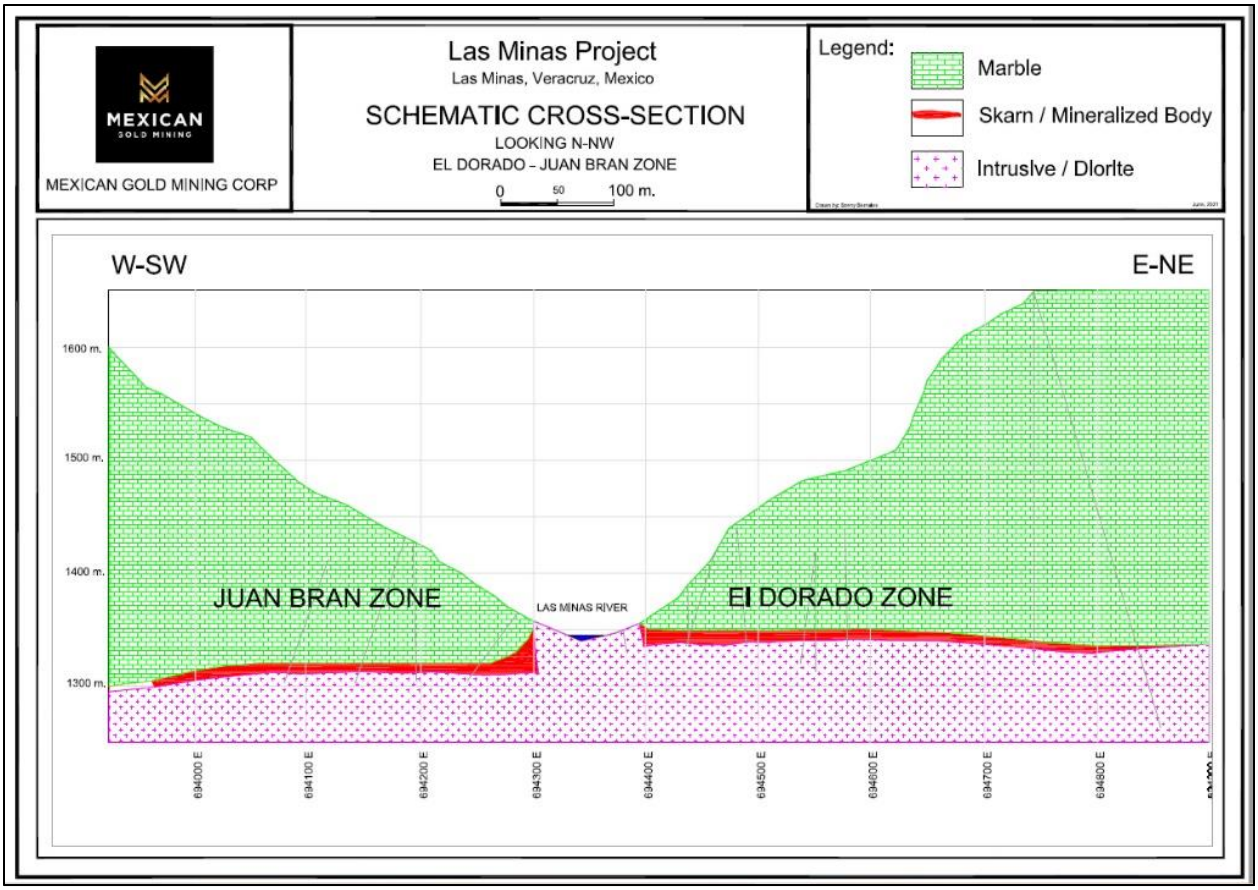
Highest metal grades within both zones are often within the skarn just below the marble-skarn contact. This contact can be very sharp with often a <0.1 m transition from weakly mineralized marble to high- grade skarn. The lower mineral contact is more gradational as the skarn alteration decreases into the weakly altered intrusive. The intrusive contact is sub-parallel to the observed remnant bedding in the overlying marble, which indicates that the intrusive contact might represent the upper contact of a thick sill.

The more significant drill intercepts within the Las Minas resource area occur where the Juan Bran skarn forms an east-dipping “keel” that extends into the diorite along the west side of the dike. Where the dike extends up into the carbonate section, the dike-skarn contacts can be highly variable due to local sill-like intrusions that extend out along gently east-dipping bedding planes. The result is an interlayering of marble, skarn and intrusions more than 100 m in thickness.

Extending away from the diorite dike to the west and east, the El Dorado mineralized skarn becomes generally thinner, but appears to remain open, especially to the west. However, the mineralization becomes progressively deeper in these directions under the steeply rising canyon topography making exploration by surface drilling increasingly difficult. The El Dorado mineralization is also open to the northwest towards Cinco Senores.

A cross-section through the El Dorado zone is shown in Figure 7-6 with the cross-section location shown on the drillhole map in Figure 10-1.

Figure 7-6: Geologic Section of El Dorado Zone



Source: Bernales (2021)

7.4.2 Santa Cruz Zone

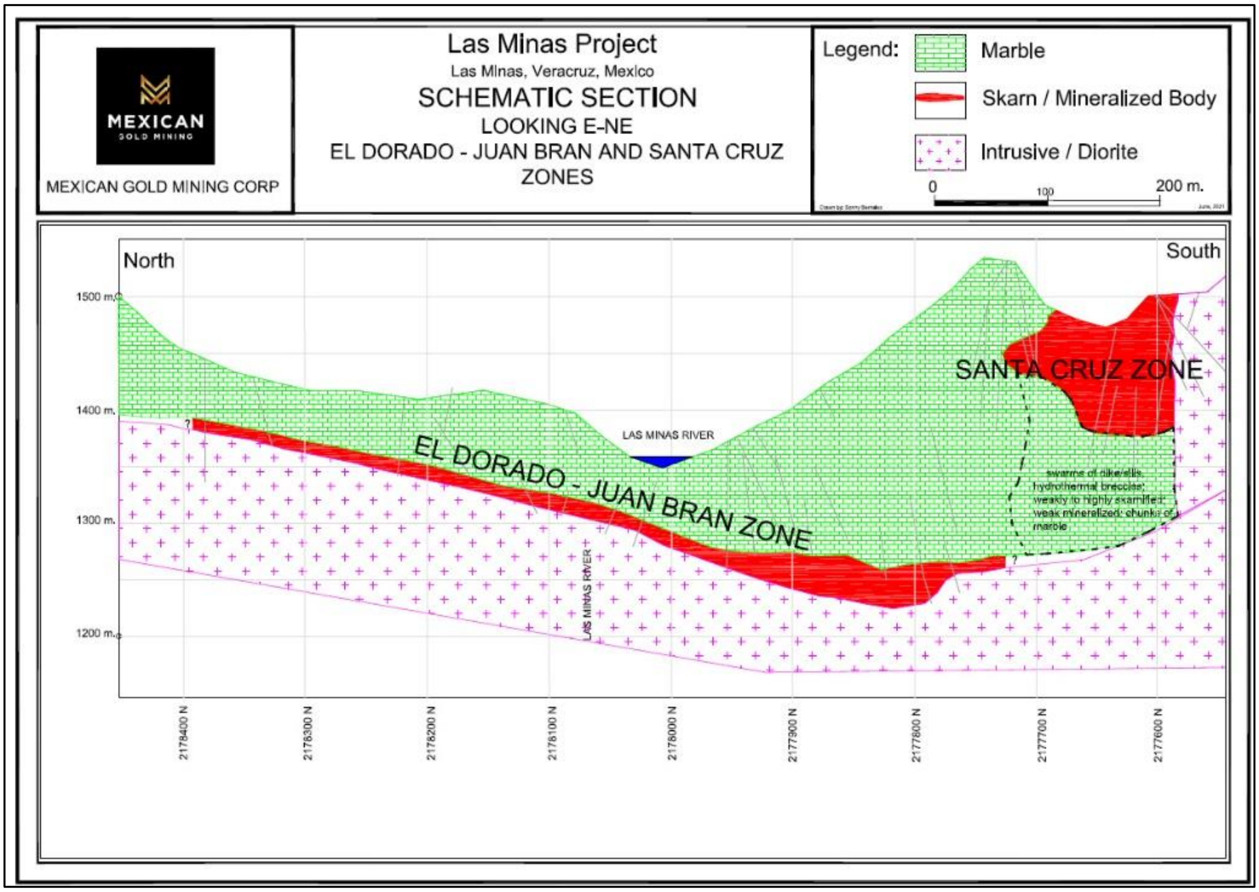
The Santa Cruz zone lies about 0.5 km south of the Las Minas pueblo and is well exposed on a west-facing canyon wall just above a tributary of the Rio Las Minas. Skarn within the Santa Cruz zone lies along the west side of the dike, immediately to the south of and stratigraphically higher than the El Dorado and Juan Bran zones. The primarily east-dipping mineralization at Santa Cruz is more complex and discontinuous than observed at El Dorado due to the more variable intrusive-marble contact orientations (both near- vertical dike and east-dipping sills). In contrast to the El Dorado zone, magnesian skarn is common as indicated by the presence of olivine and serpentinite along the intrusive contacts. The increased magnesian skarn indicates a more dolomitic protolith.

Due to the steep west-dipping topography, drilling has been primarily east-directed, which is sub-parallel to the bedding/sill/mineralization orientation, making geologic interpretations somewhat uncertain. The Santa Cruz zone has a 200 m northwest strike length, extends up to 200 m downdip to the northeast from the topographic surface, and can be up to 50 m thick, though thickness is highly variable and often consists of stacked sequences of mineralized skarn.

Occasional skarn intervals >100 m in thickness have been intersected, though this would include some intervals of dike and marble. In fact, quite commonly skarn (both exoskarn and endoskarn) are interlayered with thin (generally <5 m intervals of intrusion or marble.

A cross-section through the Santa Cruz zone is shown in Figure 7-7.

Figure 7-7: Geologic Section of El Dorado and Santa Cruz Zones



Source: Bernales (2021)

7.4.3 Other Mineralized Occurrences

There are several other skarn occurrences within the Mexican Gold property which have been subject to historical development and have been the focus of initial Mexican Gold exploration mapping, sampling, and limited drilling. Figure 7-5 shows the location of these historical prospects.

Changarro – La Perdida: The Changarro-La Perdida area is an historical skarn mining zone extending more than 400 m by 850 m on surface. The zone is exposed on the canyon walls between 280 m and 600 m above and to the east of the Las Minas resource skarn deposits. Gold occurs in both high-temperature, proximal skarn (garnet–magnetite-sulphides) related to dike contacts and structurally-controlled, low-temperature distal skarn (jasperoid-magnetite-tremolite-wollastonite +/- sulphides). A number of small historical mines are within this area including Changarro, La Perdida, Escondida, Guadelupe 2 and Guadelupe 3.

Pueblo Nuevo: Pueblo Nuevo is located about 4 km north of the Las Minas village. Mineralization within many small historical workings is hosted within both skarn and gold-bearing quartz veins. Most of the mines within the area are within the basement diorite below the contact with the overlying carbonate rocks, but the largest mine resides in altered carbonates above the contact. The La Miqueta mine, which has 21 known levels, lies within a 400 m wide zone of quartz-biotite schist developed within silicified limestone.

Cinco Senores: Cinco Senores is located approximately one-half kilometer northwest of the Las Minas village. The mineralized skarn at Cinco Senores is extensive and includes mining areas with numerous adits including Mayates, Narisillas, Muertos, San Antonio and Santa Elena, which are the names of the five senores (men). The mineralized skarn deposit extends for 150 m or more on strike along both sides of the Rio Las Minas streambed and was the first area developed by the Spanish in the 1700's. The vertical distance between the lower and the upper adits was estimated to be approximately 25 m and mineralization extends upwards from the streambed to a roadcut located approximately 75 m vertically above the valley floor.

San Jose del Oro: The San Jose del Oro deposit (Twomey (2013) is located one kilometer north of the Las Minas village in rugged terrain above the Rio Las Minas valley floor. Mineralization exposed at the Minillas open pit has a higher pyrite content than other skarn deposits within the project area. The Minillas mine was developed in a number of eras, lastly evaluated in 1914 and 1936.

8 DEPOSIT TYPES

The mineralized zones in the Las Minas district are Au-Cu-bearing skarn deposits. In general, skarn deposits can be classified by their principal contained metal such as Cu, Fe, W, Au, Pb-Zn, Sn (e.g., Meinert, 1992; Einaudi et al., 1981). In addition to primary metal content, these skarn classes can have distinct differences in calc-silicate minerals (e.g., garnet: pyroxene ratio), mineral composition, associated geochemistry and genetically associated plutonic rocks. The fact that the Las Minas skarns contain both abundant gold and copper and are low in the base metals Pb and Zn, indicates that they can be best classified as belonging to either the gold and/or copper skarn types. Copper skarns have long been described in the literature with many well documented examples worldwide (Einaudi et al., 1981). They are commonly, but not always, associated with porphyry copper deposits. Copper skarn deposits are generally typified by abundant andraditic garnet > diopsidic pyroxene and are characteristically associated with granodioritic plutonic rocks. Copper grades are on the order of 1% or greater and gold grades are generally <1 g/t (Meinert, 1987; Einaudi et al., 1981).

Gold grades can be significantly higher in some copper skarns, particularly in zones of high sulphide concentration or in those not related to porphyry copper deposits (Meinert, 1987). Skarn deposits that contain Au as the principal metal component have been recognized and studied for approximately 20 years and numerous examples exist such as Hedley, British Columbia; Fortitude/Phoenix, Nevada; McCoy, Nevada; Elkhorn, Montana; and the skarns of the Guerrero gold belt in western Mexico (Meinert, 2000; Meinert 1992; Everson and Read, 1992; Myers and Meinert, 1989; M3 Engineering, 2015). These gold skarns exhibit some distinct differences to copper skarns. They are generally typified by a more reduced mineral assemblage of iron-rich pyroxene, relatively iron-poor garnet, pyrrhotite and pyrite. Many examples are associated with equigranular diorite plutons (Everson and Read, 1992; Myers and Meinert, 1989).

Gold grades in gold skarns tend to have gold grades > 1 g/t and typically contain copper grades that are < 0.5%, however they may occasionally be greater (Meinert, 1987). Another distinctive feature of gold skarns is the gold-arsenic-bizmuth-tellurium geochemical association and the close association of gold deposition with bismuth and tellurium. Gold often occurs alongside or intergrown with minerals such as the bismuth telluride hedleyite, bismuthinite, and native bismuth, and may often occur in maldonite (Au2Bi) (Meinert, 2000; Everson and Read, 1992).

There also exists a sub-class of gold skarns that are more oxidized. These generally have a high garnet:pyroxene ratios, relatively iron-poor garnet and pyroxene, and pyrite > pyrrhotite, such as at McCoy, Nevada (Brooks et al, 1991).

The Las Minas skarns are characterized by a calc-silicate mineral assemblage consisting of abundant garnet with lesser pyroxene and a metallic mineral assemblage consisting principally of abundant magnetite, chalcopyrite, pyrite, and bornite. The dark brown color of the garnet suggests it may principally be andradite (Fe-rich) and the pale to medium green color of the pyroxene indicates it is relatively iron- poor diopside. These characteristics bear similarities to many copper skarns, particularly those that are not related to porphyry copper deposits (Meinert, 1992; 1987), but also to the oxidized sub-class of gold skarns (Meinert, 2000; Brooks et al, 1991).

Considering all characteristics, the Las Minas skarns have features which suggest the mineralization may be best classified as an example of the non-porphyry-related copper skarn class. However, many features of alteration and mineralization at Las Minas are also consistent with those of the oxidized gold skarn class. In addition, because the Las Minas skarn deposits contain copper and gold in association with iron oxide (magnetite), they could also be considered to have affinity to the iron oxide-copper-gold (IOCG) () deposit type or could be considered a hybrid or variation of an IOCG.

Within Mexico, the closest significant analogues to the Las Minas skarns are in the prolific Guerrero Gold Belt in southwestern Mexico. The skarn deposits in this area have a long history with the involvement of many different mining companies (e.g., Teck Resources, Goldcorp, Newmont Mining, Torex Gold Resources, Leagold Mining Corp.). These skarns are closely associated with granodioritic and dioritic stocks, may contain locally abundant magnetite, and have Au grades on the order of 1-15 g/t Au developed in both exoskarn and endoskarn (M3 Engineering, 2015; Stantec, 2017). These characteristics are similar to Las Minas mineralization though in general are of higher gold grade than at Las Minas. Copper is present but usually not of economic grade, although the Media Luna deposit (Torex Gold) contains about 1% Cu. A number of these deposits are currently being mined and/or developed, such as El Limon, Media Luna and Los Filos.

9 EXPLORATION

Since acquisition of the property in 2010, Mexican Gold has completed exploration activities including, diamond drilling; geological mapping; surface and underground sampling, and a ground magnetic survey.

9.1 Geologic Mapping

Initial mapping in 2005 by previous operators provided some detail within the immediate vicinity of historical workings at Santa Cruz, El Dorado, and Juan Bran. Subsequent geologic mapping within the project area has been carried out by Mexican Gold's geologists and Geoconsulting Ingenieros S.C. (GISC) of Mexico City. This work included mapping the lithologic units in detail and refining the stratigraphy, particularly within the Tertiary volcanic section.

9.2 Surface and Underground Sampling

The exploration sampling database contains 1,814 chip, grab, and channel samples. As part of Mexican Gold's initial reconnaissance work in 2010 and 2011, over 1,400 samples were collected from exposures as part of an extensive program conducted throughout the district. Most of these samples are 2 m channel samples, representing more than 140 separate channels from the various adits and surface exposures associated with the numerous historical workings found throughout the property. In 2017 and 2018, the Pueblo Nuevo and Changarro areas were sampled where higher-grade gold assays, with associated silver and base-metal mineralization, was encountered. All of the surface and underground samples were assayed for Au, Ag, Cu, Pb, and Zn and analyzed by SGS Labs (SGS) in Durango, Mexico.

Channel samples which produced significant gold values within the resource area came from the Santa Cruz zone along with anomalous grades at the El Dorado and Juan Bran zones. Significant assays from this program included the following:

Santa Cruz

- 20 m @ 5.2 g/t Au, 1.19% Cu; La Garma adit
- 8 m @ 1.02 g/t Au, 0.63% Cu; La Republica 3 adit
- 8 m @ 3.5 g/t Au, 1.14% Cu; El Mayate adit

El Dorado-Juan Bran

- 12 m @ 0.76 g/t Au, 2.06% Cu; El Dorado 1 adit
- 16 m @ 1.171 g/t Au, 2.4% Cu; El Dorado 1 adit

- 42m @ 0.47 g/t Au, 0.77% Cu; El Dorado 3 adit
- 12 m @ 2.82 g/t Au, 1.5% Cu; Juan Bran 2 adit
- 22 m @ 0.52 g/t Au, 0.85% Cu; Juan Bran adit

Cinco Senores

- 18 m @ 1.88 g/t Au, 0.05% Cu; Los Muertos adit
- 24 m @ 1.54 g/t Au, 0.4% Cu; Narsillas

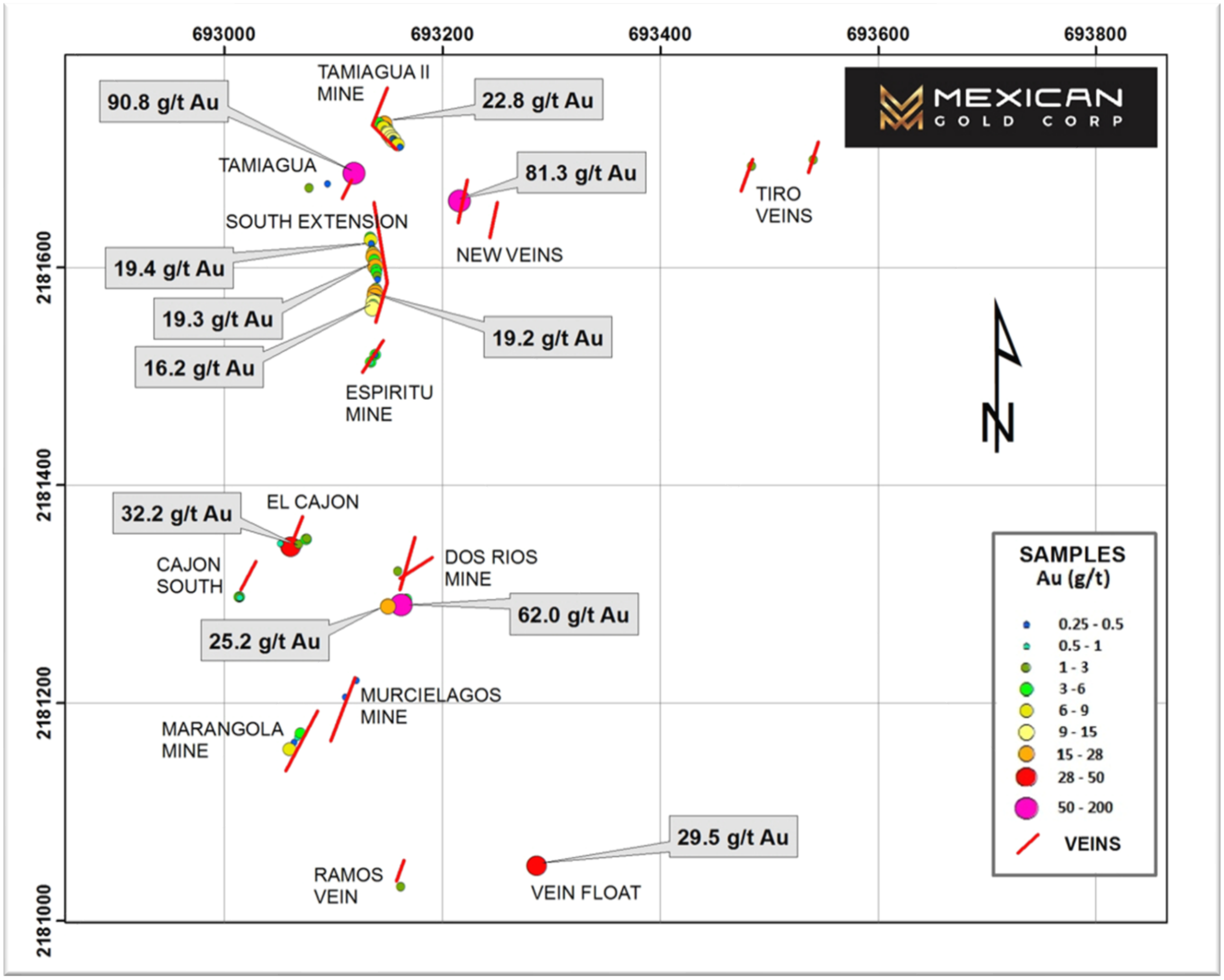
Las Boquillas

- 20 m @ 4.46 g/t Au, 0.33% Cu; LB-2 adit
- 20 m @ 3.43 g/t Au, 0.45% Cu; LB-2 adit
- 12 m @ 3.7 g/t Au, 0.4% Cu; surface

It should be noted that the channel lengths are not indication of true mineralized widths. The El Dorado zone is a horizontal plane which indicated that the horizontal adits and resulting channel samples are oriented along the trend of mineralization, not perpendicular or true width.

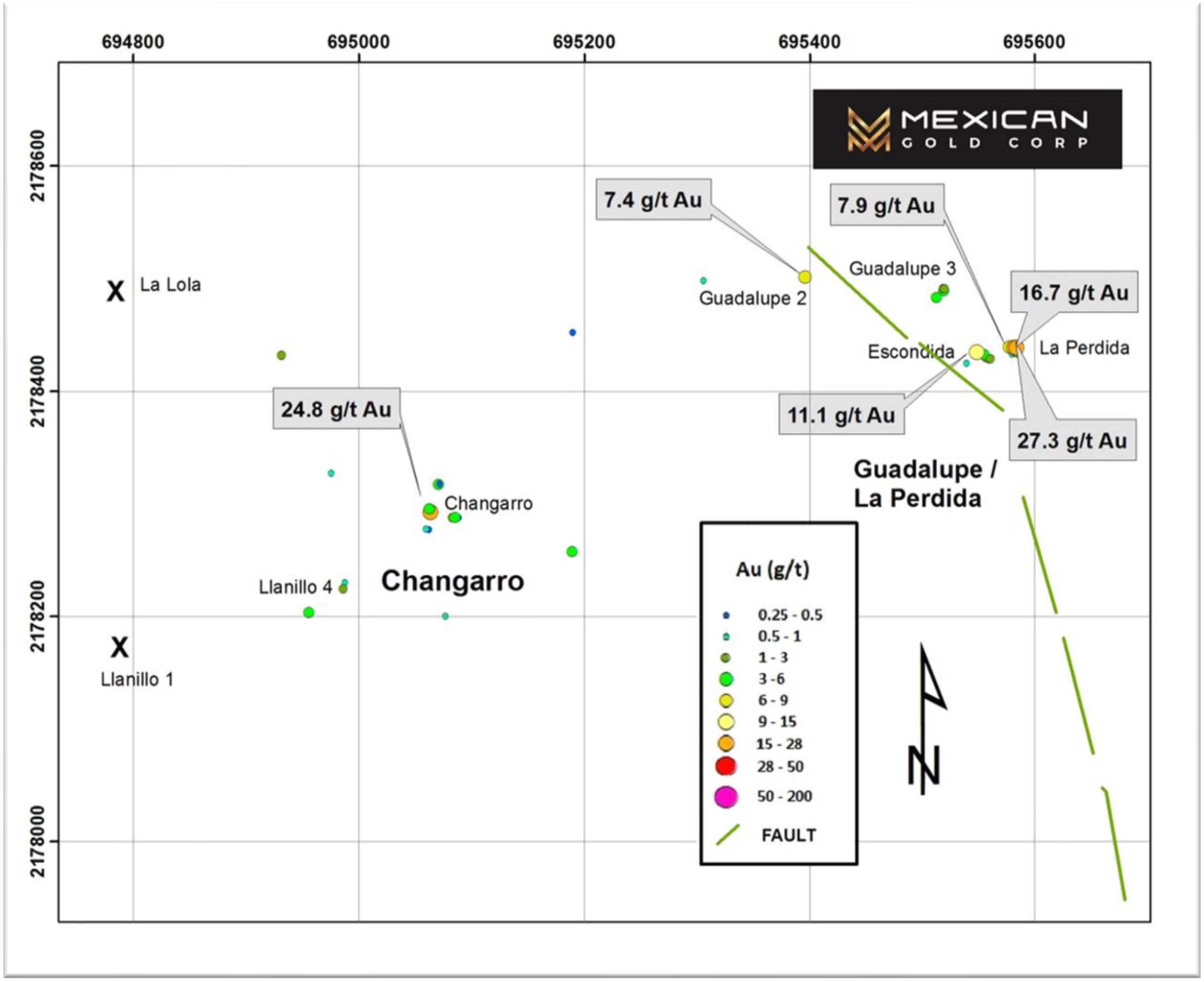
In addition, significant gold values (i.e., >1 g/t Au) were encountered in many of the historical workings outside of the current Las Minas resource, including Cinco Senores, San Jose del Oro, Las Boquillas, Changarro-La Perdida, and Pueblo Nuevo. The samples assaying over 10 g/t Au occur primarily within the Las Boquillas adit #2, located just north of the El Dorado resource zone, and within the Changarro-La Perdida and Pueblo Nuevo target areas. While the Las Boquillas and Changarro gold mineralization, and most of the other district mineralization, is skarn-hosted, the Pueblo Nuevo gold occurs both within skarn and also as banded quartz veins. The latter vein-type mineralization occurs within numerous adits within the Las Minas diorite and below the contact with the overlying silicified limestone. Figure 9-1 and Figure 9-2 show assay values for Pueblo Nuevo and Changarro, respectively.

Figure 9-1: Pueblo Nuevo Sample Location Map



Source: Tietz (2019)

Figure 9-2: Changarro – La Perdida Sample Location Map



Source: Tietz (2019)

9.3 Geophysical Surveys

9.3.1 Ground Magnetics

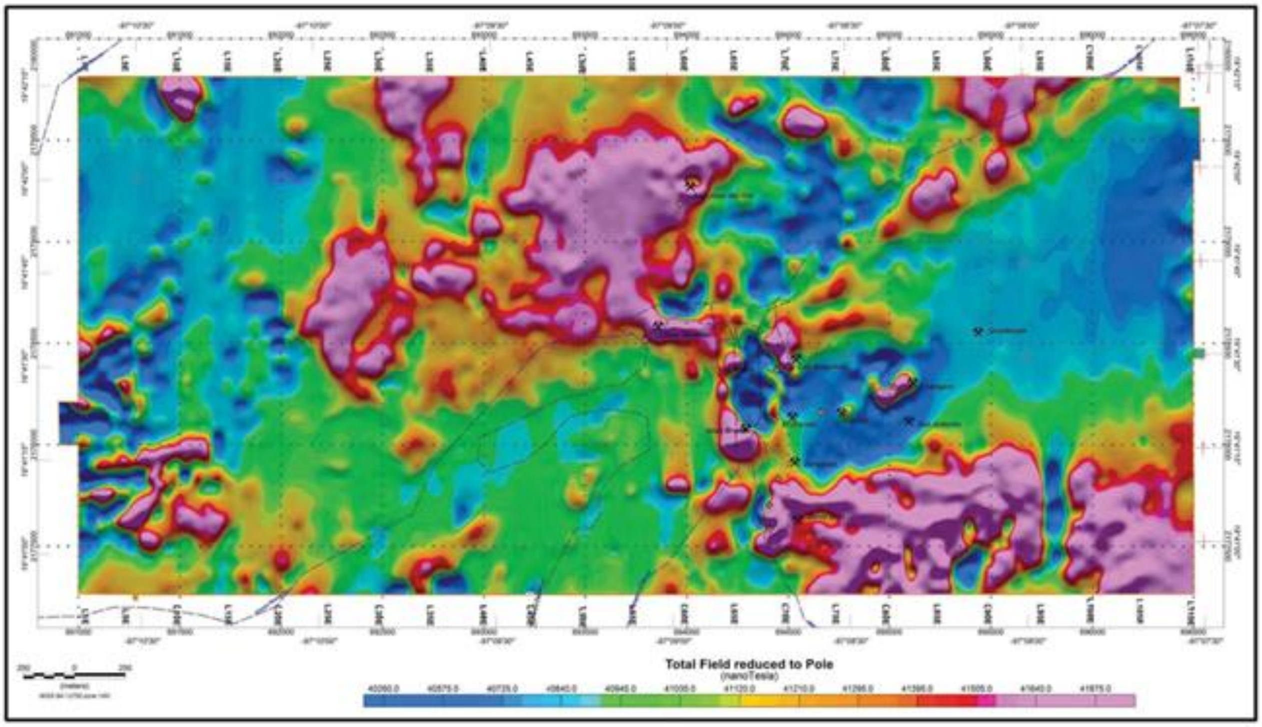
A ground magnetic survey was competed in 2013 by Geofisica TMC SA de CV, a subsidiary of Geophysique TMC of Quebec. The survey covered an area measuring 5.5 km (east-west) by 2.5 km (north-south), or 13 km², which was centered on the Las Minas pueblo, and covered the majority of the Pepe Tres claim. The survey was run on north-south lines spaced 100 m apart;

each line was 2.5 km in length. Readings of total magnetic field were collected along the lines every 2 seconds, with locations recorded by GPS (Geofisica TMC, 2013).

Figure 9-3 shows the total magnetic field reduced-to-pole image for the survey area. Numerous zones of magnetic highs exist throughout the survey area, including broad zones located northwest and southeast of Las Minas pueblo. It is likely that, for the most part, the magnetic high anomalies are reflecting underlying (or outcropping) intrusive. As has been noted, the dioritic and granodioritic rocks in the project area have high magnetite contents and thus should give rise to anomalously high total field zones. It is less clear if the magnetite-bearing skarn bodies are reflected in the survey. It is possible that due to the narrowness of the skarn zones compared to the adjacent diorite intrusive, a distinct magnetic anomaly attributable to just the skarn is not produced.

Strongly anomalous zones are present in several mineralized areas, notably Santa Cruz, Juan Bran, Cinco Senores, Las Boquillas, and Changarro, where magnetic highs occur coincident or adjacent to known skarn occurrences. The magnetic survey has served as a good exploration guide in the district.

Figure 9-3: Magnetic Survey – Total Field Reduced to Pole



Source: Geofisica TMC, S.A. de C.V. (2013)

9.3.2 Induced Polarization

In conjunction with the ground magnetic survey, Geofisica TMC completed an induced polarization/resistivity (IP/Res) orientation study of three test lines just east of Las Minas pueblo, within the magnetic survey grid. The IP/Res survey was carried out using a pole-dipole array with spacing at 100 m (Geofisica TMC, 2013). These three lines were oriented north-south, spaced at 200 m, and each was 2.4 km in length, covering the Las Boquillas, El Dorado, and Santa Cruz mineralized zones. These mine areas are characterized by low resistivity but are somewhat indistinct in chargeability response. Chargeability highs occur just north and south of Rio Las Minas, approximately 0.5 km northeast of the pueblo. The source of this anomalous chargeability is not known.

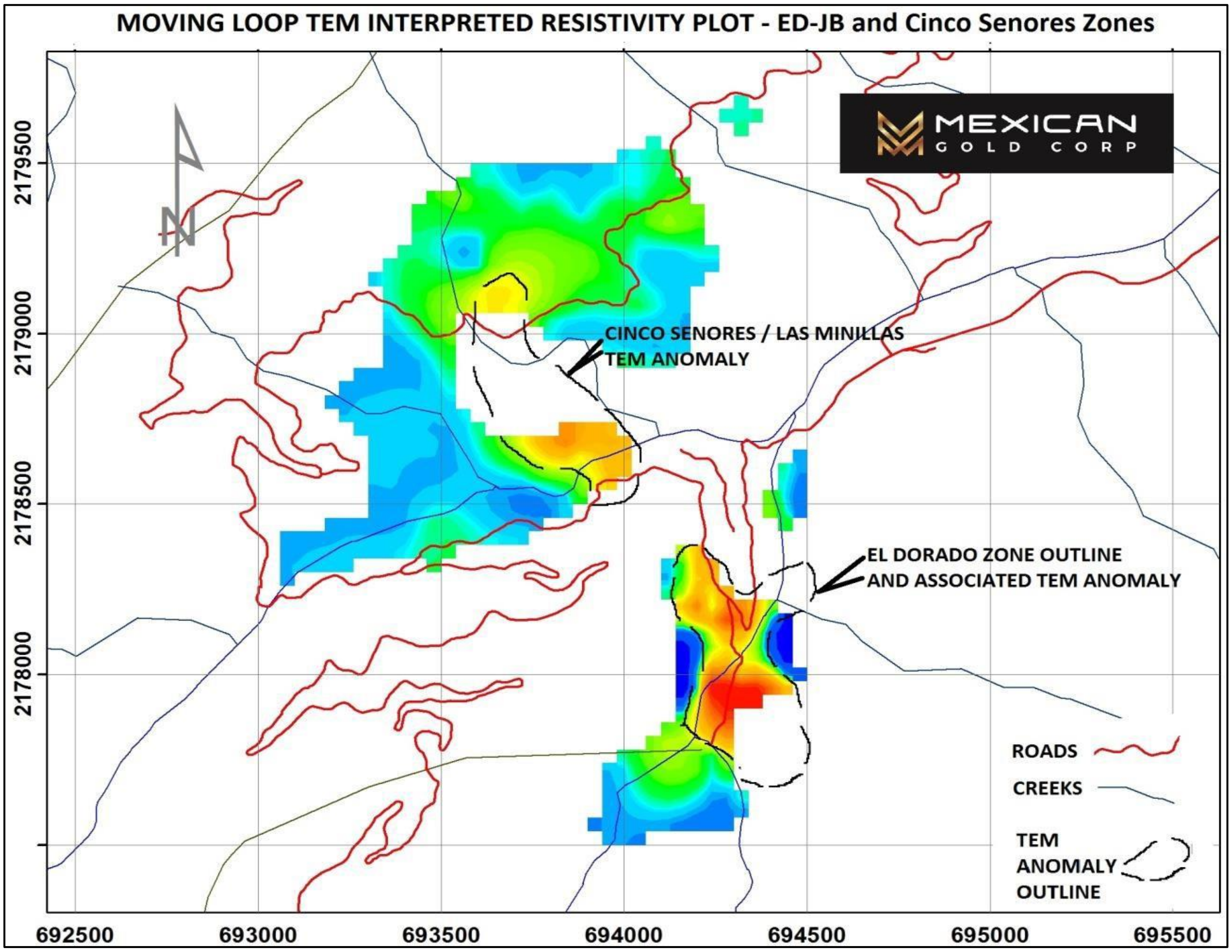
9.3.3 2017 Moving Loop TDEM and Ground Magnetics Surveys

Geotem Ingeniería of Mexico City performed ground-based, moving-loop, time-domain electromagnetic (TDEM) surveys and limited ground magnetic (MAG) surveys on the Pepe claim in 2017 (Geotem Ingeniería, 2017). The purpose of the surveys was to advance the understanding of potential drill targets at Cinco Senores and Las Minillas.

The TDEM survey was conducted over three grids; an orientation survey over the known Las Minas deposits, then the second at Cinco Senores and the third at Las Minillas. Analyses of the Las Minas TDEM sections and pseudo-sections successfully identified known mineralization over the El Dorado zone resulting in anomalies of indicative intensity and depth. Two additional anomalies were identified at Cinco Senores near the historical mine area, and a second target northeast of the historical area.

In 2018, the TDEM data was reprocessed (Lo, 2018) where two anomalies at Cinco Senores and north in the south-central Las Minillas grid area are seen in the resistivity image in addition to the El Dorado zone (Figure 9-4). These anomalies appear to be connected, though deep overburden between them masks portions.

Figure 9-4: Moving Loop Time Domain Electromagnetic Survey



Source: Geotem Ingeniería, 2017; Lo (2018)

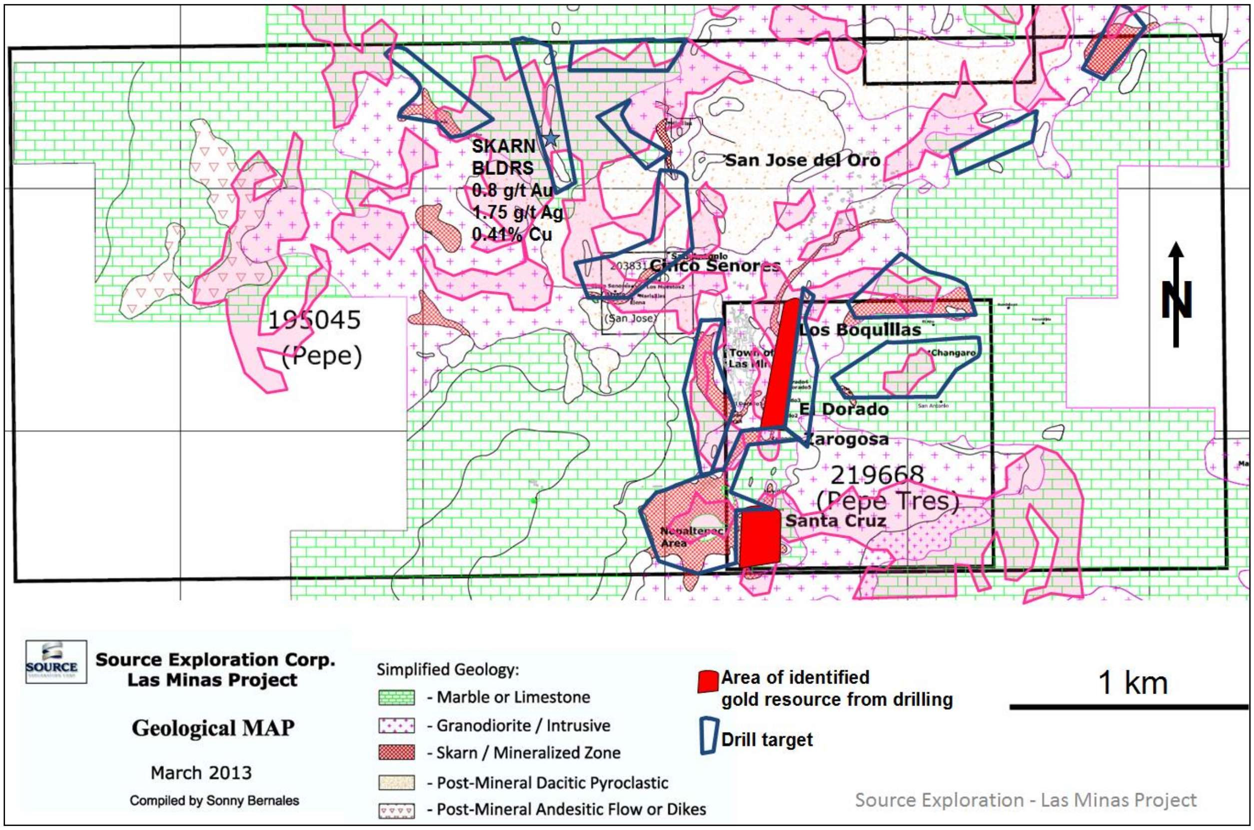
Ground magnetics were also performed on parts of the three grids, on east-west lines orthogonal to the 2012 ground magnetic survey to better discriminate north-south trending features. The main magnetic anomaly at Las Minillas was much better resolved than previously, and some magnetite skarn may be visible on the margins of a thick dike. Some of the dikes at Cinco Senores are also evident.

9.4 Interpretation and Exploration Potential

There are many old mine workings distributed throughout the Las Minas Project that represent historic mining where high-grade values were exposed at surface mostly at stream-bed elevations within the very rugged terrain. Little modern exploration has been conducted within the Las Minas Project nor in the general region. Mexican Gold conducted geological mapping within the property and has identified several untested skarns that continue under younger cover rock. In addition, the ground magnetic survey that was conducted over the entire Pepe Property which covers approximately 8 km^2 with line spacing was 100 m for approximately 275 line-km. The ground magnetics first vertical derivative map showed that most skarn zones are spatially associated with the flanks of high magnetic areas. There are a number of untested magnetic extensions where skarn rocks that are likely to be found underneath post-mineral dacitic pyroclastic rocks as well as under limestone or marble.

The areas drilled thus far represent only a small proportion of areas with good mineral potential. This drilling has focused on the east side of Las Minas valley within the Pepe Tres claim. Pepe Tres is located within the larger Pepe claim that is 5 km long by 2 km high. The potential is good in these areas for exploration and expansion. Therefore, drilling to expand areas of known mineralization as well as testing of new zones has excellent exploration target potential as shown in Figure 9-5.

Figure 9-5: Plan Map of Geology with Magnetically Anomalous Areas



Source: Twomey (2019)

10 DRILLING

10.1 Drilling Summary

The first two drill campaigns (2011 and 2012) were contracted to Drilcor, S.A. de C.V. based in Durango, Mexico. Drilling in 2014 through 2020 was done by KDL Mexico, S.A. de C.V. of Durango, Mexico, a direct subsidiary of Kluane Drilling Ltd. of Whitehorse, Yukon Territory, Canada.

The Las Minas drill database contains records for a total of 34,703 m of diamond-core (core) drilling in 229 drillholes within the Las Minas Project Area with 206 of those drillholes being within in the Las Minas resource area.

The project drilling compiled by target area is summarized in Table 10-1 and Figure 10-1. This includes the Santa Cruz and El Dorado resource zones along with the more significant exploration targets with a complete listing of drillhole locations presented in Table 10-2. Figure 10-2 shows the locations of the Las Minas drillhole collars within the Santa Cruz and El Dorado resource zones.

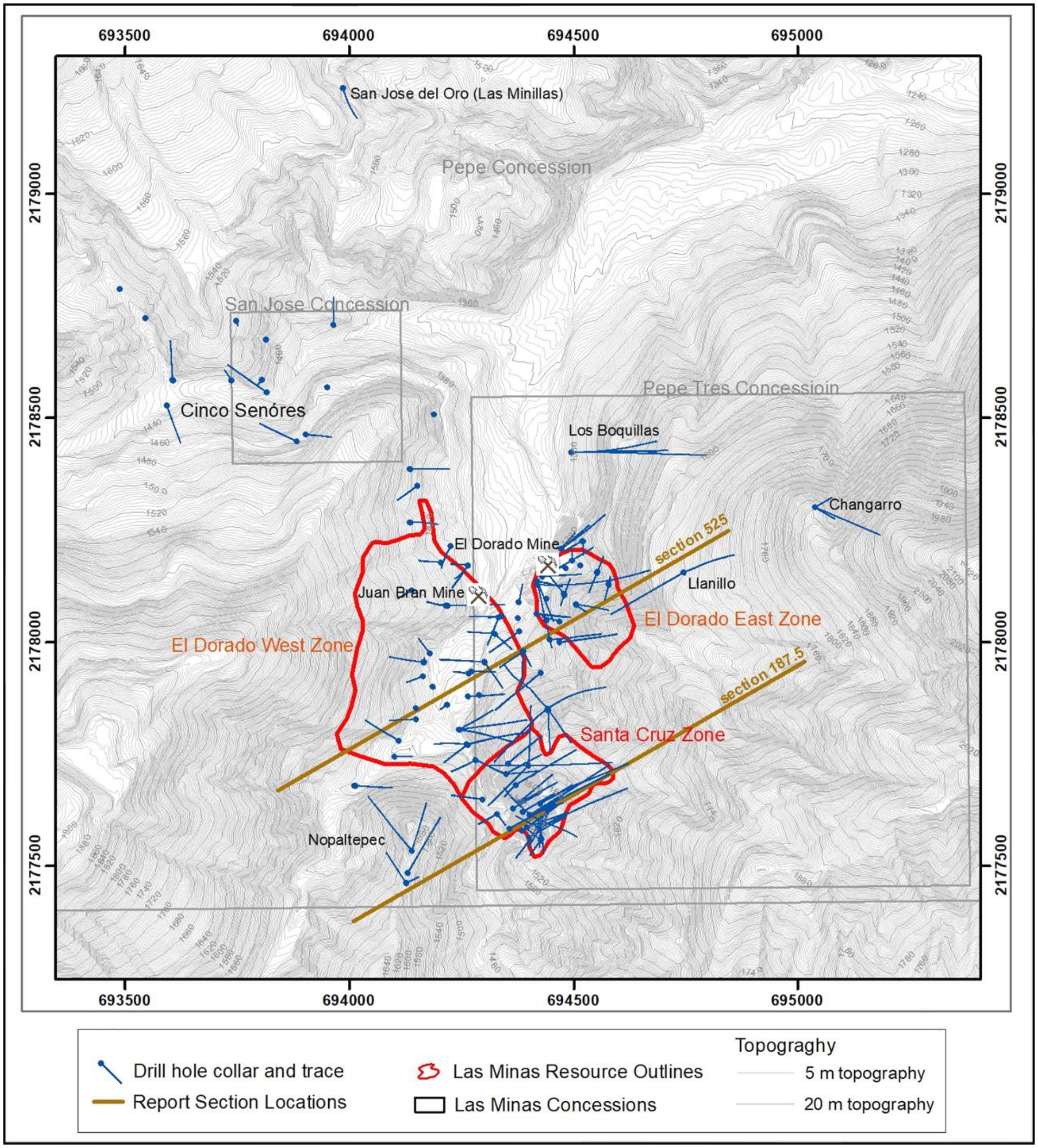
Table 10-1: Las Minas Drill Database - Yearly Summary by Target Area

Year	Prospect	DH Prefix	# Drillholes	Metres Drilled
2011	El Dorado	ED	14	1,193
	Las Boquillas	LB	5	1,096
	Santa Cruz	SC	10	4,281
	Santa Cruz	ZZ	6	
	Llanillo	LL	4	1,138
	Las Minillas	LM	2	255
	Cinco Senores	CS	5	332
	El Changarro	EC	4	647
	Total		**50**	**8,942**
2012	**Santa Cruz**	**SC**	**10**	**1,081**
2014	Cinco Senores	CS	3	300
	El Dorado	ED	19	2,917
	Juan Bran	JB	8	
	Santa Cruz	SC	12	1,537
	Total		**42**	**4,754**

Year	Prospect	DH Prefix	# Drillholes	Metres Drilled
2015	El Dorado	ED	5	765
	Juan Bran	JB	1	
	Santa Cruz	SC	6	759
	Total		**12**	**1,524**
2017	Cinco Senores	CS	8	917
	El Dorado	ED	24	5,348
	Juan Bran	JB	15	
	Total		**47**	**6,265**
2018	Cinco Senores	CS	1	79
	El Dorado	ED	6	1,372
	Juan Bran	JB	2	
	Santa Cruz	SC	6	1,480
	Total		**15**	**2,931**
2019	**Juan Bran**	**JB**	**3**	**271**
2020	El Dorado	ED	28	8,298
	Juan Bran	JB	18	
	Las Boquillas	LB	4	637
	Total		**50**	**8,935**
Total	**El Dorado**		**143**	**20,164**
	Santa Cruz		**50**	**9,138**
	Cinco Senores		**17**	**1,628**
	Las Boquillas		**9**	**1,733**
	El Changarro		**4**	**647**
	Llanillo		**4**	**1,138**
	Las Minilas		**2**	**255**
	TOTAL		**229**	**34,703**

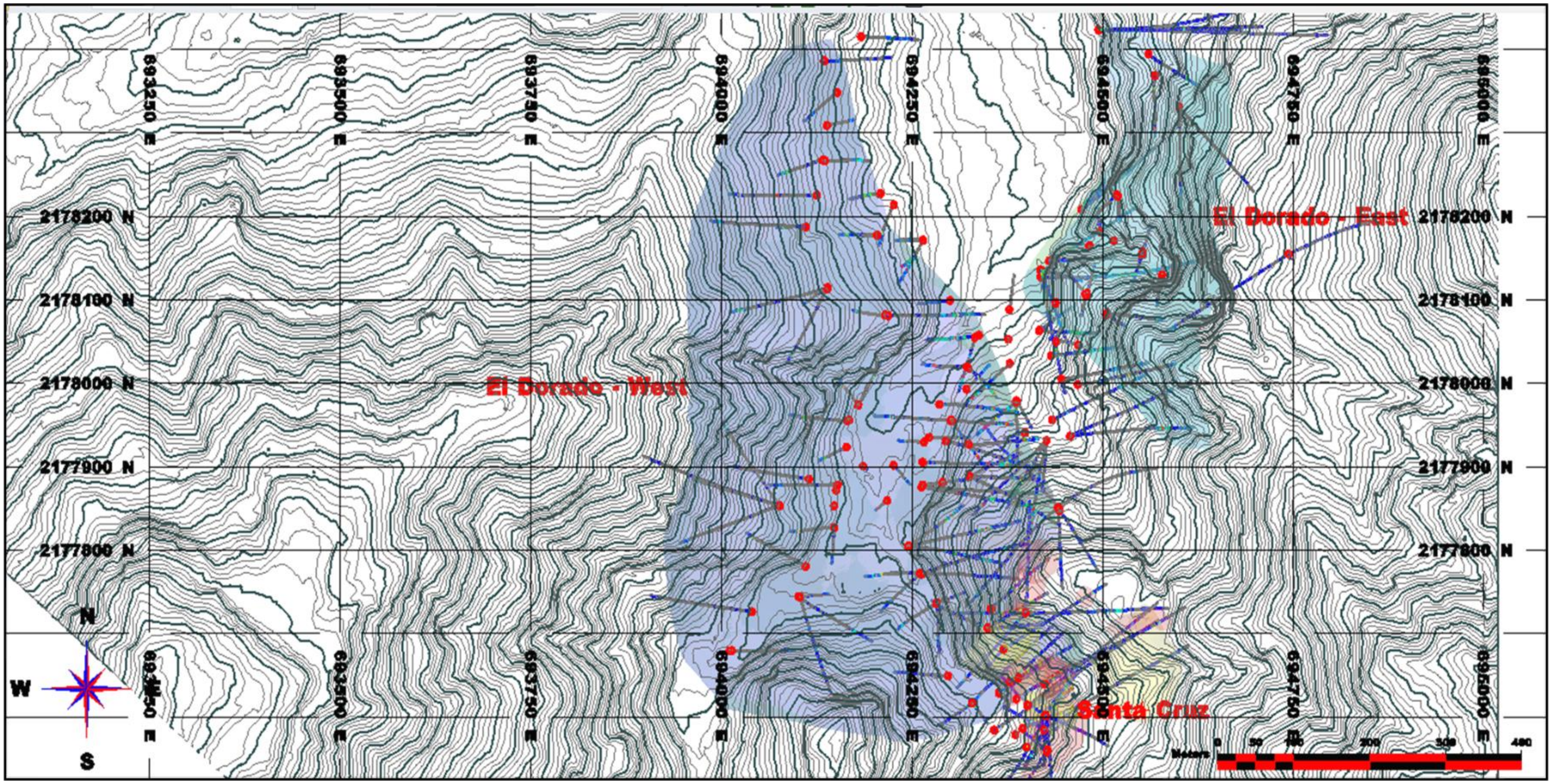
Source: Kirkham (2021)

Figure 10-1: Location of Las Minas Project Drillholes



Source: Tietz (2019)

Figure 10-2: Location of Las Minas Project Drillholes and Mineral Zones



Source: Kirkham (2021)

Mexican Gold, and its predecessor company Source, has completed all of the known drilling on the Las Minas property. The drilling was performed in 2011, 2012, 2014, 2015, 2017, 2018, 2019 and 2020. The sub-horizontal El Dorado zone was drilled by primarily vertical or steeply inclined drillholes so sample length are an accurate representation of true thickness.

Table 10-2: Drillhole Collars

Hole #	East (m)	North (m)	Elevation (m)	Azimuth (degrees)	Dip (degrees)	Depth (m)	Total Depth (m)
LM11ED10	694418.71	2178135.21	1346.25	159.2	-28.3	107.5	1193.2
LM11ED11	694418.37	2178136.08	1347.7	158.76	10	14.8	
LM11ED12	694418.22	2178135.03	1347.36	96	-3	11.7	
LM11ED13	694418.45	2178134.96	1346.54	101	-27.5	62.7	
LM11ED14	694417.68	2178135.13	1348.21	99.5	15.5	29.8	
LM11ED15	694418.34	2178127.54	1347.74	92.5	-5.5	254.7	
LM11ED16	694471.64	2178207.98	1352.66	38.5	22.2	16.9	
LM11ED17	694471.64	2178207.98	1352.66	50	-15	50.1	
LM11ED18	694471.72	2178208.19	1351.68	56.2	0	125	
LM11ED26	694471.66	2178208.09	1352.2	50	12	83.8	
LM11ED6	694429.59	2178146.78	1345.64	88.9	-28.8	101.6	
LM11ED7	694429.35	2178146.82	1347.19	86.4	9.8	50.6	
LM11ED8	694429.49	2178146.81	1346.43	64.3	-7.3	131.5	
LM11ED9	694418.57	2178135.46	1347.14	170	-7.2	152.5	
LM11LB1	694494	2178423.2	1357.56	90	10.9	192.5	1095.7
LM11LB2	694494.2	2178423.25	1357.11	90	-0.9	150.1	
LM11LB3	694494.62	2178423.27	1356.49	90	-12.3	302.5	
LM11LB4	694494.37	2178423.2	1356.09	90	-25.6	241.1	
LM11LB5	694494.07	2178423.21	1357.92	90	25	209.5	
LM11LL40	694744	2178155	1643.77	240	-60	372.4	1137.5
LM11LL43	694744	2178155	1643.77	0	-90	339.7	
LM11LL46	694744	2178155	1643.77	67	-74.21	412.25	
LM11LL48	694744	2178155	1643.77	150	-60	13.15	
LM11SC34	694398.76	2177725.29	1521.05	90	-59.6	368.5	4280.55
LM11SC36	694398.76	2177725.29	1521.05	270	-76	359.85	
LM11SC37	694425.21	2177601.46	1479.29	300	-65	275.9	
LM11SC41	694425.21	2177601.46	1479.29	0	-88.7	273.15	
LM11SC42	694425.21	2177601.46	1479.29	110	-61.7	191.7	
LM11SC44	694425.21	2177601.46	1479.29	30	-61.5	230.25	
LM11SC45	694425.21	2177601.46	1479.29	210	-60.2	206.6	

Hole #	East (m)	North (m)	Elevation (m)	Azimuth (degrees)	Dip (degrees)	Depth (m)	Total Depth (m)
LM11SC47	694398.76	2177725.29	1521.05	4	-59	261.35	
LM11SC49	694398.76	2177725.29	1521.05	225	-65.45	309	
LM11SC50	694398.76	2177725.29	1521.05	0	-90	190.5	
LM11ZZ27	694443.12	2177847.98	1514.8	140	-59	170.4	
LM11ZZ28	694442.15	2177847.71	1514.89	180	-59.9	189.15	
LM11ZZ29	694441.23	2177851.12	1514.3	236	-61.5	341.15	
LM11ZZ30	694441.23	2177851.12	1514.3	320	-68.2	383.7	
LM11ZZ31	694441.23	2177851.12	1514.3	56	-58.8	276.75	
LM11ZZ32	694442.19	2177849.55	1514.38	0	-90	252.6	
LM12SC51	694369.97	2177681.94	1468.71	225	-70	108.55	1081.4
LM12SC52	694423.08	2177584.68	1467.08	210	-60	87	
LM12SC53	694423.08	2177584.68	1467.08	0	-90	105	
LM12SC54	694423.08	2177584.68	1467.08	30	-60	150.7	
LM12SC55	694423.08	2177584.68	1467.08	300	-60	125.4	
LM12SC56	694423.08	2177584.68	1467.08	210	-45	70.5	
LM12SC57	694424.05	2177601.92	1479.41	180	-60	121.1	
LM12SC58	694425.21	2177601.46	1479.29	175	-45	104.2	
LM12SC59	694427.08	2177560.5	1488.32	215	-60	93.05	
LM12SC60	694426.46	2177558.31	1488.3	310	-70	115.9	
LM14ED01	694476.94	2178106.96	1445.47	320	-65	131.15	2916.97
LM14ED02	694479.01	2178107.94	1445.6	20	-75	125.05	
LM14ED03	694478.03	2178105.33	1445.53	200	-80	140.3	
LM14ED04	694504.13	2178082.64	1457.87	120	-80	161.65	
LM14ED05	694504.74	2178083.8	1457.91	100	-63	155.55	
LM14ED06	694467.44	2178046.16	1437.87	285	-60	76.25	
LM14ED07	694467.73	2178046.28	1438.03	0	-90	149.45	
LM14ED08	694437.91	2178096.46	1399.03	0	-90	83	
LM14ED09	694482.33	2178164.68	1371.85	280	-75	48.8	
LM14ED10	694514.13	2178170.71	1379.28	0	-90	48.8	
LM14ED11	694551.3	2178155.03	1399.98	0	-90	109.8	
LM14ED12	694551.14	2178154.82	1399.89	200	-65	115.9	
LM14ED13	694552.31	2178157.63	1399.84	20	-65	85.4	
LM14ED14	694576.91	2178127.08	1415	200	-70	100.65	
LM14ED15	694577.6	2178129.89	1415.07	7	-80	94.55	
LM14ED16	694518.48	2178225.42	1436.63	240	-62	93.02	
LM14ED17	694519.88	2178224.49	1436.55	170	-75	112.85	
LM14ED18	694495.6	2178181.63	1373.7	355	-55	48.8	

Hole #	East (m)	North (m)	Elevation (m)	Azimuth (degrees)	Dip (degrees)	Depth (m)	Total Depth (m)
LM14ED19	694438.86	2178049.63	1403.74	340	-80	39.65	
LM14JB01	694218.42	2178081.14	1429.36	90	-50	199.8	
LM14JB02	694217.75	2178081.18	1429.78	0	-90	131.75	
LM14JB03	694214.28	2178082.03	1429.79	270	-70	132.15	
LM14JB04	694133.73	2178266.82	1431.6	90	-50	97.6	
LM14JB05	694135.27	2178266.65	1431.6	0	-90	112.9	
LM14JB06	694134.86	2178386.25	1439.8	90	-50	137.25	
LM14JB07	694134.28	2178386.28	1440	0	-90	102.5	
LM14JB08	694188.12	2178507.55	1390.96	0	-90	82.4	
LM14SC01	694394.57	2177586.59	1442.69	70	-45	131.15	1536.8
LM14SC02	694377.65	2177642.32	1432.69	50	-45	134.6	
LM14SC03	694386.33	2177621.87	1435.74	60	-45	115.9	
LM14SC04	694369.97	2177681.94	1468.71	60	-45	213.5	
LM14SC05	694424.01	2177639.4	1457.66	60	-45	115.9	
LM14SC06	694423.08	2177584.68	1467.08	60	-45	100.65	
LM14SC07	694349.5	2177706.6	1473.24	60	-46	128.1	
LM14SC08	694349.2	2177706.6	1473.2	60	-65	112.85	
LM14SC09	694369.72	2177681.94	1468.71	60	-65	109.8	
LM14SC10	694388.97	2177646.96	1448.33	60	-45	115.9	
LM14SC11	694357.12	2177584.51	1412.97	60	-45	159.95	
LM14SC12	694363.5	2177629	1398.25	60	-45	98.5	
LM15ED20	694300.84	2177956.59	1372.52	270	-45	153.75	765.55
LM15ED21	694301.05	2177956.57	1372.4	0	-90	128.1	
LM15ED22	694321.5	2178020	1364	270	-55	102.6	
LM15ED23	694244.3	2177805	1381.5	0	-90	118.85	
LM15ED24	694289.5	2177881.5	1381.05	0	-90	149.5	
LM15JB09	694147.3	2177853.3	1388.4	0	-90	112.75	
LM15SC13	694353.05	2177729.64	1476.39	50	-52	157.85	759.55
LM15SC14	694352.88	2177729.53	1476.48	50	-77	122	
LM15SC15	694399.89	2177564.05	1445.64	60	-45	131.2	
LM15SC16	694384.92	2177578.93	1429.59	52	-45	133.25	
LM15SC17	694400.8	2177613.82	1447.87	65	-45	112.75	
LM15SC18	694349.5	2177705.56	1473.37	85	-48	102.5	
LM17ED25	694323.49	2178018.45	1364.15	130	-67	135.3	5348.37
LM17ED26	694245.16	2177805.57	1381.4	62	-60	180.4	
LM17ED27	694260.05	2177772.24	1376.23	275	-60	139.4	

Hole #	East (m)	North (m)	Elevation (m)	Azimuth (degrees)	Dip (degrees)	Depth (m)	Total Depth (m)
LM17ED28	694262.97	2177771.62	1376.36	75	-75	133.25	
LM17ED29	694263.42	2177771.69	1376.45	70	-45	202.95	
LM17ED30	694260.05	2177772.24	1376.23	228	-60	135.3	
LM17ED31	694300.8	2177955.05	1372.51	140	-60	162.95	
LM17ED32	694387.03	2177979.11	1388.28	150	-75	148.6	
LM17ED33	694375.69	2178053.07	1361.16	0	-90	92.25	
LM17ED34	694265.28	2177929.64	1365.68	270	-75	139.4	
LM17ED35	694216.85	2177859.86	1372.59	230	-80	112.75	
LM17ED36	694296.84	2177649.57	1396.11	270	-65	159.9	
LM17ED37	694281.05	2177736.79	1379.77	240	-63	131.2	
LM17ED38	694332.17	2178054.92	1355.79	195	-50	132.55	
LM17ED39	694377.18	2178088.61	1352.48	15	-45	59.45	
LM17ED40	694386.03	2177978.3	1388.28	235	-70	194.25	
LM17ED41	694337.4	2178057.73	1355.15	260	-75	115.25	
LM17ED42	694337.4	2178057.73	1355.15	260	-45	97	
LM17ED43	694416.17	2178063.06	1376	95	-45	108.95	
LM17ED44	694426.13	2177931.75	1422.42	232	-58	256.2	
LM17ED45	694445.77	2178005.15	1425.31	85	-52	191.2	
LM17ED46	694467	2177998.86	1446.39	85	-50	207.4	
LM17ED47	694386.24	2177979.39	1388.41	180	-67	201.3	
LM17ED48	694244.81	2177804.84	1381.47	78	-55	269.4	
LM17JB10	694010.52	2177679.4	1403.48	0	-90	142.5	
LM17JB11	694013.15	2177679.69	1403.52	90	-60	166.1	
LM17JB12	694166.91	2177956.19	1368.71	0	-85	93.25	
LM17JB13	694185.36	2177901.06	1364.49	135	-85	80	
LM17JB14	694263.97	2178171.68	1372.04	270	-70	75.85	
LM17JB15	694163.82	2177923.76	1371.06	250	-70	112.75	
LM17JB16	694147.65	2177827.5	1379.86	270	-65	145.55	
LM17JB17	694100.64	2177744.53	1392.11	90	-72	139.4	
LM17JB18	694179.16	2177974.64	1363.71	324	-55	72.25	
LM17JB19	694164.7	2177954.98	1368.64	275	-50	120.95	
LM17JB20	694110.07	2177780.71	1384.65	300	-57	159.6	
LM17JB21	694203.66	2178177.46	1407.04	270	-58	97.6	
LM17JB22	694264.17	2178171.06	1372.26	215	-45	60.6	
LM17JB22A	694264.17	2178171.06	1372.26	215	-47	89.97	
LM17JB23	694225.81	2178213.99	1390.67	210	-52	85.4	

Hole #	East (m)	North (m)	Elevation (m)	Azimuth (degrees)	Dip (degrees)	Depth (m)	Total Depth (m)
LM18ED49	694263.33	2177878.58	1372.01	90	-65	199.75	1371.8
LM18ED50	694328.39	2177616.91	1403.03	240	-65	189.1	
LM18ED51	694328.39	2177616.91	1403.03	140	-80	198.25	
LM18ED52	694377.76	2178024.06	1374.82	225	-60	160.6	
LM18ED53	694272.08	2177935.16	1365.47	89	-69	147	
LM18ED54	694245.3	2177805.13	1381.45	100	-55	226.5	
LM18JB24	694151.66	2178348.06	1417.23	240	-55	99.15	
LM18JB25	694138.96	2178115.05	1453.14	245	-75	151.45	
LM18SC05A	694424.01	2177639.4	1457.66	60	-45	286.05	1479.85
LM18SC06A	694423.08	2177584.68	1467.08	60	-45	295.85	
LM18SC17A	694400.8	2177613.82	1447.87	65	-45	281.65	
LM18SC19	694348.2	2177707.3	1473.2	295	-70	237.9	
LM18SC20	694349.2	2177706.6	1473.24	0	-90	83.25	
LM18SC21	694401.54	2177614.3	1447.74	60	-45	295.15	
LM19JB26	694179	2177974	1363.71	30	-45	86.5	271.00
LM19JB27	694150	2177872	1382.42	0	-90	111.3	
LM19JB28	694299.58	2178099.21	1358.51	265	-55	73.2	
LM20ED55	694320.7	2177993.21	1373.41	45	-60	84.45	8297.95
LM20ED56	694438.86	2178049.63	1403.74	16	-70	47.3	
LM20ED57	694431.92	2178033.27	1398.39	85	-50	151.05	
LM20ED58	694416.6	2178063.84	1373.41	58	-55	88.45	
LM20ED59	694456.82	2177937	1423.6	68	-50	230.25	
LM20ED60	694281.05	2177736.79	1379.77	68	-75	152.5	
LM20ED61	694282.84	2177737.57	1379.77	180	-65	143.75	
LM20ED62	694102.31	2177743.89	1392.11	125	-50	166.2	
LM20ED63	694102.31	2177743.89	1392.11	160	-45	172.5	
LM20ED64	694302.19	2177955.79	1372.77	93	-45	112.85	
LM20ED65	694294.15	2177931.11	1373.74	106	-48	135.7	
LM20ED66	694263.46	2177905.8	1368.7	93	-55	280.65	
LM20ED67	694328.39	2177616.91	1403.03	45	-50	222.65	
LM20ED68	694225.39	2177902.35	1365.97	135	-75	128.1	
LM20ED69	694426.13	2177931.75	1422.42	190	-63	259.25	
LM20ED70	694433.52	2177956.45	1414.27	68	-50	204.35	
LM20ED71	694324.13	2177927.1	1402.73	90	-48	137.25	
LM20ED72	694426.13	2177931.75	1422.42	185	-75	262.3	
LM20ED73	694456.82	2177937	1423.6	80	-55	283.65	
LM20ED74	694324.58	2177889.02	1415.76	70	-60	140.3	

Hole #	East (m)	North (m)	Elevation (m)	Azimuth (degrees)	Dip (degrees)	Depth (m)	Total Depth (m)
LM20ED75	694324.58	2177889.02	1415.76	115	-55	225.7	
LM20ED76	694262.16	2177876.06	1372.01	99	-50	228.75	
LM20ED76A	694262.16	2177876.06	1372.01	99	-50	228.75	
LM20ED77	694285.26	2177974.97	1364.14	91	-53	170.8	
LM20ED78	694263.42	2177771.69	1376.45	88	-55	218.35	
LM20ED79	694324.58	2177889.02	1415.76	95	-50	138.8	
LM20ED80	694397.5	2177940.72	1418.24	252	-82	207.7	
LM20ED81	694426.13	2177931.75	1422.42	212	-70	288.2	
LM20JB29	694114.42	2177885.48	1402.9	280	-50	178.45	
LM20JB30	694147.65	2177827.5	1379.86	180	-70	164.7	
LM20JB31	694138.84	2178113.59	1453.53	260	-45	192.15	
LM20JB32	694138.84	2178113.59	1453.53	220	-55	183	
LM20JB33	694132.58	2178267.52	1431.6	260	-60	128.1	
LM20JB34	694076.05	2177853.6	1432.33	285	-45	271.45	
LM20JB35	694076.05	2177853.6	1432.33	287	-55	218.05	
LM20JB36	694076.05	2177853.6	1432.33	322	-52	208.95	
LM20JB37	694076.05	2177853.6	1432.33	252	-52	228.95	
LM20JB38	694039.83	2177726.45	1397.08	280	-65	161.65	
LM20JB39	694039.83	2177726.45	1397.08	278	-45	195.2	
LM20JB40	694152.25	2177878.46	1382.42	278	-53	155.55	
LM20JB41	694110.55	2178187.62	1446.2	275	-45	180.95	
LM20JB42	694110.55	2178187.62	1446.2	270	-75	157.55	
LM20JB43	694182.68	2178414.99	1405.94	90	-48	114.35	
LM20JB44	694208.33	2178227.3	1387.03	265	-55	73.2	
LM20JB45	694138.07	2178308.7	1425.66	80	-65	88.45	
LM20JB46	694124.87	2178225.38	1439.81	270	-45	170.8	
LM20LB01	694559.69	2178394.16	1430.67	130	-45	115.9	637.45
LM20LB02	694568.24	2178368.4	1428.78	190	-48	109.8	
LM20LB03	694599.7	2178331.85	1471.82	180	-50	210.45	
LM20LB04	694599.7	2178331.85	1471.82	140	-45	201.3	

Source: Kirkham (2021)

10.2 Significant Assay Results

Table 10-3 and Table 10-4 provides a selection of significant drillhole intervals within lower grade intervals from the Las Minas drillhole database. Drillhole intervals are reported as actual core lengths and approximate the true thickness.

Table 10-3: Significant Drillhole Intersections – Gold Grades Greater Than 5 g/t Au

Drillhole Name	From (m)	To (m)	Assay Interval (m)	Au Grade (g/t)	Ag Grade (g/t)	Cu Grade (%)	Total Fe Grade (%)
LM-11-LL-40	335.8	337.8	2	27.05	6.8	0.911	20.7
LM-11-SC-36	91.7	93.7	2	6.452	19.1	3.07	4.87
LM-11-SC-36	122.3	124.3	2	5.902	63.1	5.35	16.4
LM-11-SC-36	226.6	228.6	2	10.15	4	0.582	5.12
LM-11-SC-45	55.6	57.6	2	35.22	17.4	2.28	12.2
LM-11-SC-45	57.6	59.6	2	39.3	33.4	5.73	26.4
LM-11-SC-49	263.6	265.6	2	6.964	9	0.911	36.5
LM-12-SC-52	50.5	52.5	2	7.901	14.9	1.68	23.1
LM-12-SC-57	6.2	8.2	2	5.38	0.6	0.0657	12.3
LM-12-SC-57	66.2	68.2	2	5.214	5.1	3.33	13.5
LM-12-SC-57	68.2	70.2	2	5.226	6.5	2.49	33.4
LM-14-JB-02	13.5	15.5	2	12.65	0.7	0.0068	5.29
LM-14-SC-03	75.2	77.2	2	12.64	16.3	1.92	20
LM-14-SC-03	83.1	85.1	2	9.214	126	9.74	33.4
LM-14-SC-04	173	175	2	5.608	4.7	0.316	8.06
LM-14-SC-07	16	18	2	7.08	15	0.971	2.55
LM-14-SC-07	63	65	2	7.16	8	0.811	9.2
LM-14-SC-08	45.4	47.4	2	5.791	24.1	4.75	8.45
LM-14-SC-08	47.4	49.4	2	12.38	46.1	7.5	10
LM-14-SC-08	49.4	51.4	2	15.94	72.6	11.4	12.5
LM-14-SC-08	51.4	53	1.6	16.23	77.8	10.7	14
LM-14-SC-08	95	96.2	1.2	11.3	13.8	3.84	9.04
LM-14-SC-09	21	23	2	5.749	9	0.958	2.52
LM-14-SC-09	23	25	2	9.61	16.5	1.71	2.08
LM-14-SC-10	16	17.5	1.5	9.53	31.1	7.49	26.4
LM-15-ED-21	89.6	91.6	2	5.004	13.3	1.9	39.5
LM-15-SC-15	22	24	2	5.729	17.6	3.36	10.3
LM-15-SC-15	31.9	32.15	0.25	21.3	3.8	11.7	11.1

Drillhole Name	From (m)	To (m)	Assay Interval (m)	Au Grade (g/t)	Ag Grade (g/t)	Cu Grade (%)	Total Fe Grade (%)
LM-15-SC-15	100	102	2	5.111	12.5	1.2	>15
LM-17-ED-28	94.36	96.36	2	5.615	5.7	1.84	>15
LM-17-ED-28	104.36	106.36	2	6.997	1.4	1.16	21.5
LM-17-ED-37	2	4	2	5.454	30.1	2.06	10.9
LM-17-ED-40	45	46	1	11.15	7.2	1.05	11.2
LM-17-ED-40	123.6	125.6	2	28.01	8.1	4.09	30.4
LM-17-ED-40	125.6	127.6	2	6.633	5.7	2.51	29.4
LM-17-ED-40	131.6	133.6	2	5.32	5.5	1.09	20
LM-17-ED-40	141.6	143.6	2	6.124	38.5	2.49	38.9
LM-17-ED-40	143.6	145.6	2	5.041	31	2.46	53.68
LM-17-ED-40	147.6	149.6	2	5.697	36.7	2.84	59.08
LM-17-ED-40	149.6	151.6	2	5.107	30.5	1.9	49
LM-17-ED-42	53	55	2	19.76	2.2	0.203	23.7
LM-17-ED-44	214.8	216.8	2	12.01	17.7	7.71	12.8
LM-17-ED-44	218.8	220.8	2	13.4	11.5	5.2	21.2
LM-17-ED-45	159.8	161.8	2	10.05	3.3	0.811	22.2
LM-17-ED-46	41	43	2	5.711	1.1	0.0064	8.9
LM-17-ED-47	22	24	2	5.243	0.6	0.0602	4.36
LM-17-ED-48	181.5	183.5	2	7.067	6	2.1	54.96
LM-17-ED-48	183.5	185.5	2	13.52	6.2	2.09	54.02
LM-17-JB-12	65.35	67.35	2	5.78	32.7	1.53	23.1
LM-17-JB-22A	56.3	58.3	2	5.224	26.7	5.6	32.4
LM-18-ED-49	144.1	146.1	2	6.118	30.6	8.73	>15
LM-18-ED-49	148.1	150.1	2	5.177	14.8	5.67	>15
LM-18-ED-49	164.1	166.1	2	7.273	16.4	2.1	>15
LM-18-ED-49	172.1	174.1	2	6.381	22.1	1.99	>15
LM-18-ED-49	174.1	176.1	2	7.004	27.5	2.98	>15
LM-18-ED-49	176.1	178.1	2	14.72	29	2.48	>15
LM-20-ED-60	119.3	121.3	2	5.31	1.6	1.5	>15
LM-20-ED-60	131.3	133.3	2	20.13	1.8	0.616	13.3
LM-20-ED-61	5	6	1	6.24	6.5	1.47	12.1
LM-20-ED-61	13	14	1	8	0.6	0.103	12.8
LM-20-ED-63	168.4	170.4	2	5.07	0.5	0.002	3.22
LM-20-ED-64	100	101.5	1.5	5.13	6.4	3.05	11.9
LM-20-ED-64	101.5	102.5	1	6.35	32.8	7.9	11.1
LM-20-ED-66	152	154	2	5.14	17.1	1.53	49.53

Drillhole Name	From (m)	To (m)	Assay Interval (m)	Au Grade (g/t)	Ag Grade (g/t)	Cu Grade (%)	Total Fe Grade (%)
LM-20-ED-66	154	156	2	14.85	14.9	1.27	48.7
LM-20-ED-69	180	182	2	17.89	24.2	6.15	12.5
LM-20-ED-69	223.6	225.6	2	6.01	6.5	1.76	44.3
LM-20-ED-69	229.6	231.6	2	6.71	23.6	1.89	9.53
LM-20-ED-69	233.6	235.6	2	6.55	27.6	2	30.8
LM-20-ED-71	106	108	2	9.74	33.9	17.8	22.5
LM-20-ED-72	60.5	62.5	2	9.3	14.4	2.5	7.44
LM-20-ED-72	68.5	70.5	2	6.38	7.6	0.771	7.91
LM-20-ED-75	160.7	161.2	0.5	6.98	10.6	2.02	10.9
LM-20-ED-75	161.2	163	1.8	9.51	0.7	0.0179	11.7
LM-20-ED-76	77.3	77.8	0.5	5.07	9.5	4.52	11.6
LM-20-ED-76	189	191	2	11.64	1.9	0.453	10.6
LM-20-ED-76	191	193	2	11.34	0.8	0.0185	4.94
LM-20-ED-76	193	195	2	9.49	2.6	0.249	7.45
LM-20-ED-76	195	197	2	47.4	3	0.0235	9.13
LM-20-ED-81	148	150	2	8.81	29.7	7.03	12.2
LM-20-ED-81	197	199	2	7.69	5.9	1.8	45.8
LM-20-LB-02	90.9	92.9	2	11.07	2.2	0.225	14.8
LM-20-LB-02	94.9	95.9	1	5.2	2.1	0.261	27.3

Source: Kirkham (2021)

Table 10-4: Significant Drillhole Intersections – Copper Grades Greater Than 3 % Cu

Drillhole Name	From (m)	To (m)	Assay Interval (m)	Au Grade (g/t)	Ag Grade (g/t)	Cu Grade (%)	Total Fe Grade (%)
LM-11-ED-9	98.3	100.2	1.9	0.983	9.2	3.05	>10
LM-11-LB-3	4.9	6.7	1.8	1.756	5	7.79	5.91
LM-11-SC-36	91.7	93.7	2	6.452	19.1	3.07	4.87
LM-11-SC-36	122.3	124.3	2	5.902	63.1	5.35	16.4
LM-11-SC-36	270.6	272.6	2	4.151	18.4	3.53	9.33
LM-11-SC-45	57.6	59.6	2	39.3	33.4	5.73	26.4
LM-12-SC-53	52	54	2	2.419	9.5	3.22	42
LM-12-SC-54	93	95	2	1.451	49.6	3.21	42.5

Drillhole Name	From (m)	To (m)	Assay Interval (m)	Au Grade (g/t)	Ag Grade (g/t)	Cu Grade (%)	Total Fe Grade (%)
LM-12-SC-54	95	97	2	2.168	80.8	4.57	47.9
LM-12-SC-57	66.2	68.2	2	5.214	5.1	3.33	13.5
LM-12-SC-57	72.2	74.2	2	4.255	11.5	3.84	>40
LM-14-ED-03	124.13	126.13	2	0.771	13.4	5.37	21.5
LM-14-SC-03	83.1	85.1	2	9.214	126	9.74	33.4
LM-14-SC-03	87.2	89.2	2	2.602	38.8	3.07	53.92
LM-14-SC-03	89.2	91.2	2	2.827	47	3.98	39.9
LM-14-SC-04	7.7	9.7	2	0.985	22.5	6.05	2.5
LM-14-SC-08	43.4	45.4	2	3.802	27.6	3.52	9.41
LM-14-SC-08	45.4	47.4	2	5.791	24.1	4.75	8.45
LM-14-SC-08	47.4	49.4	2	12.38	46.1	7.5	10
LM-14-SC-08	49.4	51.4	2	15.94	72.6	11.4	12.5
LM-14-SC-08	51.4	53	1.6	16.23	77.8	10.7	14
LM-14-SC-08	95	96.2	1.2	11.3	13.8	3.84	9.04
LM-14-SC-10	16	17.5	1.5	9.53	31.1	7.49	26.4
LM-14-SC-10	19.5	21.5	2	0.02	22.2	3.43	37.4
LM-15-ED-20	116.7	118.7	2	3.123	20.1	3.13	39.8
LM-15-ED-21	85.6	86.1	0.5	0.798	8.8	3.6	27.8
LM-15-ED-24	113.2	115.2	2	2.866	13.8	4.52	29.8
LM-15-SC-15	18	20	2	2.844	7.7	4.15	6.76
LM-15-SC-15	22	24	2	5.729	17.6	3.36	10.3
LM-15-SC-15	31.9	32.15	0.25	21.3	3.8	11.7	11.1
LM-15-SC-16	101	103	2	2.994	29	3.17	>15
LM-15-SC-16	105	107	2	4.224	35.2	3.08	>15
LM-15-SC-17	32.2	34.2	2	2.464	45.8	4.28	4.21
LM-15-SC-17	86.2	88.2	2	2.263	59.1	3.59	40.5
LM-17-ED-25	120.62	122.62	2	4.036	24	5.35	42.4
LM-17-ED-25	122.62	124.62	2	1.244	13.2	3.16	>50
LM-17-ED-38	57	59	2	4.272	6.3	5.1	27.2
LM-17-ED-40	123.6	125.6	2	28.01	8.1	4.09	30.4
LM-17-ED-40	135.6	137.6	2	4.146	7.5	4.17	46.6
LM-17-ED-41	70.6	72.6	2	4.249	33.6	7.59	49.4
LM-17-ED-41	72.6	74.6	2	2.106	17.9	3.98	56.07
LM-17-ED-43	68.9	70.9	2	1.609	5.8	3.64	46.2
LM-17-ED-44	186	188	2	3.097	9.6	3.74	14.6
LM-17-ED-44	214.8	216.8	2	12.01	17.7	7.71	12.8

Drillhole Name	From (m)	To (m)	Assay Interval (m)	Au Grade (g/t)	Ag Grade (g/t)	Cu Grade (%)	Total Fe Grade (%)
LM-17-ED-44	216.8	218.8	2	2.2	7.7	3.11	11.5
LM-17-ED-44	218.8	220.8	2	13.4	11.5	5.2	21.2
LM-17-JB-22	58.3	60.6	2.3	3.958	29.8	5.36	27.2
LM-17-JB-22A	56.3	58.3	2	5.224	26.7	5.6	32.4
LM-17-JB-22A	58.3	60.3	2	3.166	20.4	3.93	32.8
LM-18-ED-49	142.1	144.1	2	4.994	50.2	9.28	>15
LM-18-ED-49	144.1	146.1	2	6.118	30.6	8.73	>15
LM-18-ED-49	146.1	148.1	2	4	13.9	5.12	25.1
LM-18-ED-49	148.1	150.1	2	5.177	14.8	5.67	>15
LM-18-ED-49	150.1	152.1	2	1.669	13.6	6.45	>15
LM-18-ED-49	152.1	154.1	2	2.632	10.3	3.98	>15
LM-18-ED-52	120	122	2	2.91	6.4	3.32	>15
LM-20-ED-57	96.3	98.3	2	1.03	17.5	3.07	22.1
LM-20-ED-64	100	101.5	1.5	5.13	6.4	3.05	11.9
LM-20-ED-64	101.5	102.5	1	6.35	32.8	7.9	11.1
LM-20-ED-66	199	200.2	1.2	1.5	10.4	3.15	33.3
LM-20-ED-69	180	182	2	17.89	24.2	6.15	12.5
LM-20-ED-71	104	106	2	2.07	19.5	4.21	11.7
LM-20-ED-71	106	108	2	9.74	33.9	17.8	22.5
LM-20-ED-76	77.3	77.8	0.5	5.07	9.5	4.52	11.6
LM-20-ED-81	132	134	2	3.32	9	4.16	17.9
LM-20-ED-81	148	150	2	8.81	29.7	7.03	12.2
LM-20-JB-46	73.2	75	1.8	3.62	28.3	4.63	8.6

Source: Kirkham (2021)

10.3 Drilling Methods

All drilling on the property has been diamond core drilling. The 2011 and 2012 drilling used HQ and NQ- sized core. The 2014 through 2020 drilling utilized NTW (NQ Thin-Wall; 56 mm) core size. Core drilling has been carried out using man-portable drills. This has allowed for constructing drill sites (by hand methods) in places that would not be amenable to road construction due to the steep topography in the project.

When drilling, a single drill rig was operated by drill crews that worked two 12-hour shifts per day. Core runs were approximately 1.5 m in length although the occasional shorter run was used within broken ground. The rock is generally weakly fractured and core recovery has been consistently very good.

Upon drillhole completion, the collar was marked with a wooden stake, though some holes located in the valley just south of the Las Minas village have a concrete monument marking the drillhole location. As discussed in Section 12, Data Verification, numerous issues and potential risks were identified. Also, due to the topography where there are significantly steep hillsides and rapid natural re-vegetation, many of the hole locations which were marked using wooden stakes that can no longer be identified, which makes finding the actual collar locations difficult.

10.4 Drill Collar Surveys

Drill collars were surveyed during each yearly drill campaign by Topografia Digital Prissmax (Prissmax), an independent surveyor based in Jalapa, Veracruz. The survey was done using optical total station methods because the steep canyon walls make GPS surveying difficult. Collar coordinates are in UTM metres, WGS84 datum.

It was noticed by Mexican Gold in 2017 that database collar elevations for some drillholes did not coincide with available topographic mapping. Mexican Gold re-surveyed a number of drill collars which found that the topography was in error. The issue was described in the MDA Report (Tietz 2019) as follows:

> *“The project topography is based on a survey completed in 2011 by PhotoSat of Vancouver, Canada. It was recognized early in the project that there was an elevation discrepancy between the topography base map and the surveyed drillhole collars. The drill collars were all above the topographic surface by about 10 to 12 m. To resolve this issue, Mexican Gold completed survey collar checks in 2017 which confirmed that the collar coordinates and elevations were accurate and that a problem existed with the elevations of the topographic base map. The author’s review of email between Source Exploration and PhotoSat indicates that the topography was based on orthometric elevations while the collar surveys were based (correctly) on ellipsoid elevations. To correct this problem for purposes of the 2017 mineral resource estimate, new topographic surfaces was generated over the majority of the Santa Cruz zone and portions of the El Dorado zone. These areas of revised topography were then stitched into the larger topographic base which had been raised 12 m to better match the drill surveys away from the immediate resource zones. This corrected the most significant elevation issues, and this revised topography is in use for the current resource model and estimate. It is highly recommended that an accurate LiDAR survey be completed over the project area before any significant future development is begun.”*

However, during the January 18-19, 2021 site visit and subsequent review of the database, several issues were identified related to collar locations and in particular with the collar elevations. In addition, a subsequent review and inspection of the drillhole database also identified several significant issues. The company was informed of these issues and corrective plan was established which entailed performing a detailed ground survey to place known monuments and to tie in identifiable collars and landmarks. This was then followed by a LIDAR (Light Detection and Ranging) survey in order to give accurate location and elevation tie-ins to survey and to assist in determining precise coordinates for the drillhole collars and therefore data locations in 3-dimensions.

10.5 Downhole Surveys

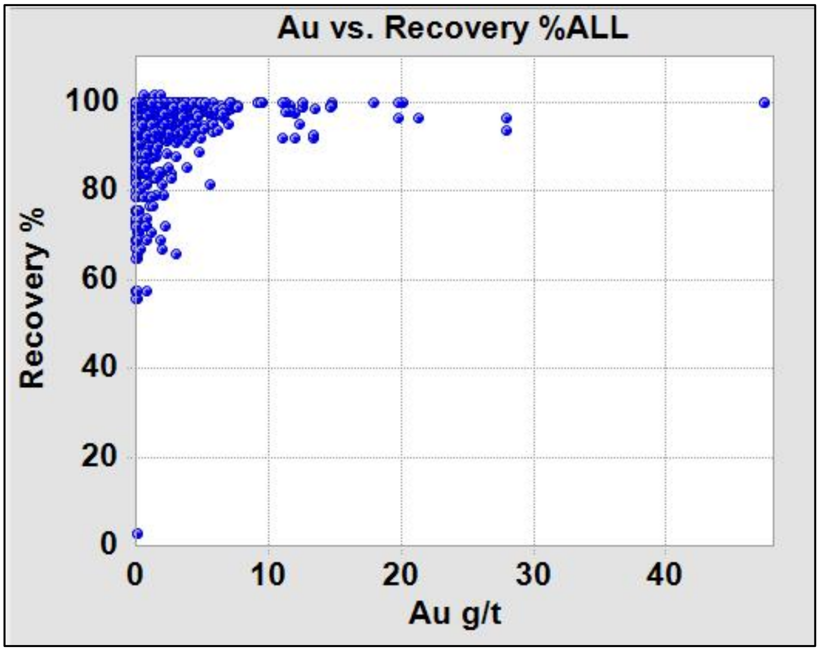
Prior to the 2019 drill campaign, downhole directional surveys were conducted in 157 of the 181 drillholes in the Las Minas drill database. Within the resource area, 143 of the 149 drillholes have downhole surveys. Azimuth and inclination readings were taken at 3 m intervals in the 2011 and 2014 drillholes, and at variable intervals from 30 m to 100 m in the 2012, 2015, 2017, and 2018 drill campaigns. A reading was also taken approximately 1 m above total depth in all drillholes. Data were directly downloaded from the apparatus.

During the 2011 and 2012 drilling campaigns, the majority of drillholes were surveyed using a Maxibor optical instrument with a few holes surveyed with a gyro. Both methods are specified not to be susceptible to variations in the local magnetic field due to the existence of magnetite. The 2015 through 2020 campaigns have used Reflex “EZ-Shot” or “Devi-Shot” methods which are susceptible to variations in the magnetic field and could lead to spurious results depending on proximity of the instrument to magnetite zones in the skarn. During the 2014 drilling, both gyro and “EZ shot” methods were used, and some double surveys were done to verify the accuracy of the EZ shot method. Inspection of the downhole surveys during drilling by Mexican Gold staff discovered sharp deviations in the azimuth or inclination readings and/or anomalous values for the instrument’s magnetic field reading which resulted in approximately 5% to 10% of the data being discarded. Most holes were relatively shallow which show only minor deviation however longer holes, particularly those that have been drilling at higher dips, may have issues with accuracy. A review of the down hole survey records against models and adjacent data along with visual inspection of sections and plans showed issues with the 2011 drillholes that were drilled through the Santa Cruz zone. It was decided by Kirkham to exclude 16 drillholes due to uncertainties related to downhole survey accuracy. Should these drillholes be adequately validated and verified, they may be utilized in the future.

10.6 Core Recovery and RQD

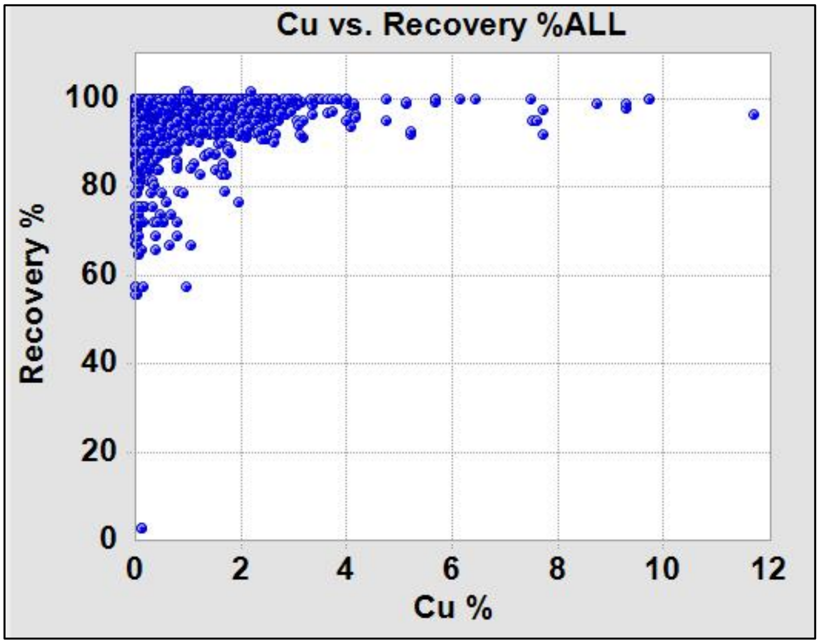
Average core recovery for all Las Minas core holes is 92.4%, while average core recovery for those intervals contributing to the current mineral estimate is 96.1%. Intervals of poor core recovery are unlikely to have an impact on grade assessment as shown in Figure 10-3 and Figure 10-4. It is clear that gold and copper grades are not significant within intercepts with low core recoveries.

Figure 10-3: Core Recovery versus Gold Grade – All Modeled Mineralization



Source: Kirkham (2021)

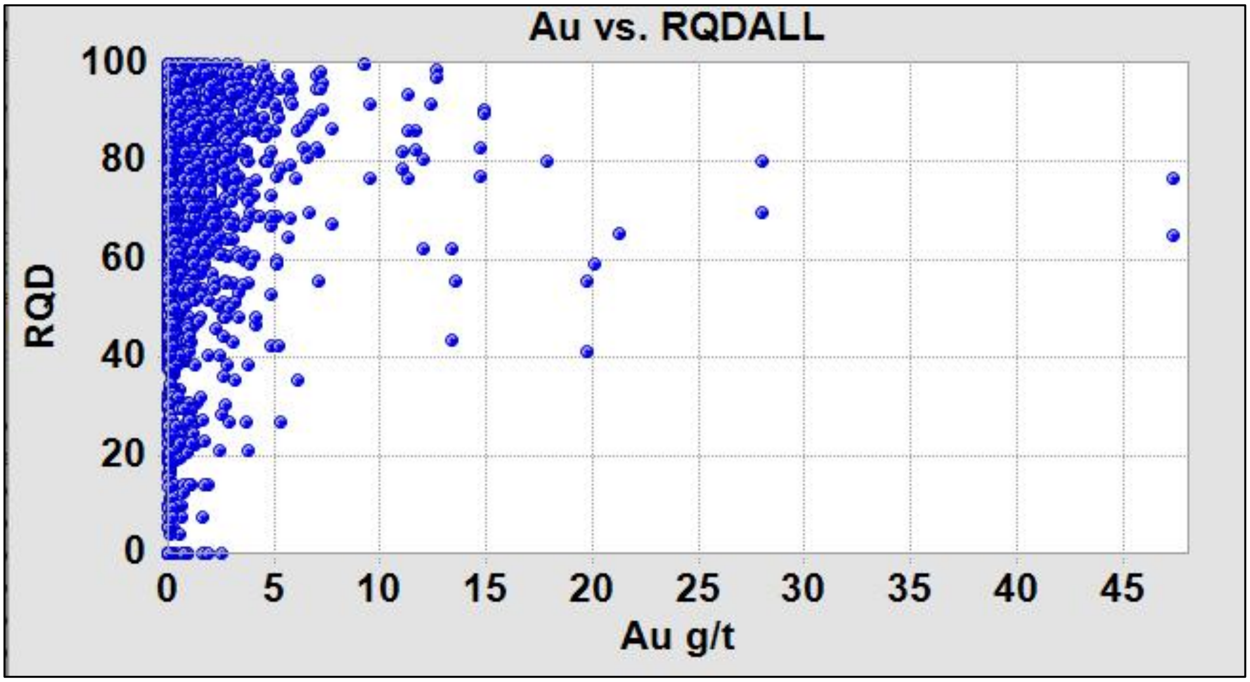
Figure 10-4: Core Recovery versus Copper Grade – All Modeled Mineralization



Source: Kirkham (2021)

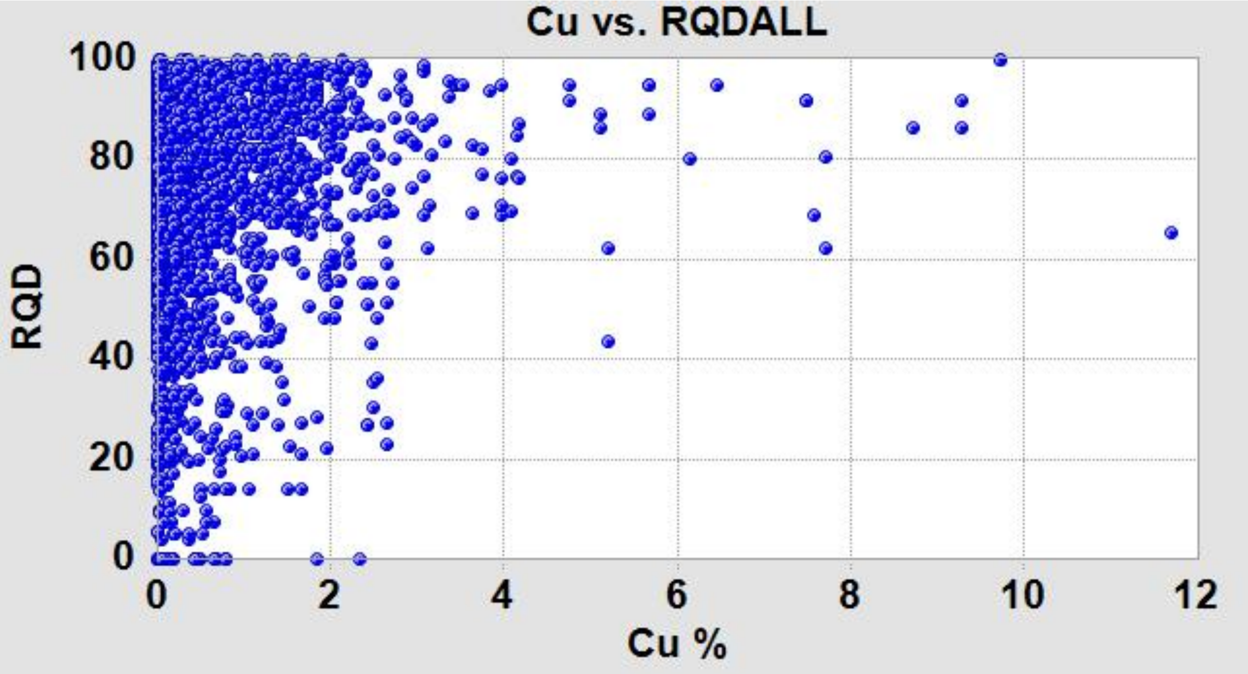
The core is generally weakly fractured both within and adjacent to the mineralized intervals, and rock quality designation (RQD) measurements for those intervals within the mineralized zones average 72.4%.

Figure 10-5: RQD versus Gold Grade – All Modeled Mineralization



Source: Kirkham (2021)

Figure 10-6: RQD versus Copper Grade – All Modeled Mineralization



Source: Kirkham (2021)

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling Method & Approach

11.1.1 Chip and Channel Sampling Preparation, Analyses & Security

The chip and channel sampling programs were supervised by professionals representing Mexican Gold. Sample preparation and analysis of samples was done by SGS in Durango, Mexico, which is an ISO/IEC 17025 accredited laboratory.

Sampling was marked by a geological staff with sample lengths being based on lithologic units which are generally at 2.0 m. The orientation, GPS location and direction of each sample were recorded. For QA/QC, a duplicate was inserted for every 10 samples submitted. Technicians sampled with hammer and moil into a plastic bag and the bag secured with a tamper-proof tag. Assay intervals are stated as metric lengths in two orientations, horizontal and vertical.

All of the samples were kept within Mexican Gold's core shack, which was locked when vacated until shipment to the laboratory. Samples were sealed in labelled plastic samples bags and securely packed for shipping. Bags of samples were then shipped by commercial transport or by SGS, to the SGS preparation facility in Durango.

Samples were prepared (crushed and pulverized) at Durango and the pulps were then analyzed there. The pulps and rejects are stored at the SGS facility in Durango. The samples were logged in their tracking system, then weighed and the entire sample was crushed to approximately 8 mesh (2.38 mm). A sub-sample split of 250 to 500 grams was then pulverized to > 90% passing 150 mesh (0.104 mm).

The sample was fire assayed and the resultant bead analyzed for gold using Atomic Absorption (AA) spectroscopy. Lower detection limits were 5 ppb Au. For samples that had an initial assay value reported of greater than 10 g/t Au, a coarse reject split of 1,000 g was taken and pulverized to 90% passing 150 mesh. This sub-sample was screened with two 30.2 g aliquots analyzed from the pulp fraction and assayed gravimetrically.

Some of the samples were selected for metallic screen analysis where a coarse reject split of 1,000 g was taken and pulverized to 90% passing 150 mesh. This sub-sample was screened with two 30.2 g aliquots analyzed from the pulp fraction and the entire metallic fraction (>+150 mesh) assayed. The contributions from the fractions were then combined using the weighted average for the sub-samples. Samples with visible gold were not flagged for special treatment.

11.1.2 Drillhole Preparation, Analyses & Security

The drill core was retrieved daily at the drill site by Mexican Gold staff, or drill contractor staff, and was brought to the Mexican Gold's secure core logging and sample preparation facility located at the north end of the Las Minas townsite. The core was laid out on tables and reviewed to make sure the core run blocks were inserted and marked, and then arranged so that the individual core pieces "fit" together. Trained Mexican Gold technicians then measured and recorded the core recovery, RQD (rock quality determination) and number of fractures. The core was then photographed.

The core was logged by Mexican Gold project geologists. Logging was done electronically on a laptop computer utilizing GEMS Logger^{TM} software. Recording of geologic parameters included lithology, alteration, sulphide minerals, and magnetite content. Lithology was detailed by basic rock type, e.g., intrusion, marble, exoskarn, endoskarn. More detailed coding was applied to capture attributes for unique mineralogy or alteration-style of the various major lithologies. A verbal geologic description of the core was also entered into GEMS Logger.

After logging was completed, the geologist marked the “from-to” sample intervals in the core box with a stapled sample tag and also marked a cut line on the core. The sample ID was a “blind” sample number from a project worksheet which correlated the sample ID with the actual from-to sample meterage. Maximum sample lengths were generally 2 m with the occasional shorter interval at geological breaks. The geologist also added sample tags for those intervals where quality assurance / quality control (QA/QC) samples were to be inserted.

The core was then cut using a diamond bladed core saw by Mexican Gold technicians following the cut line marked on the core by the geologist, with half-core samples collected and bagged while the remaining core was placed back in the core box. The bags were marked with the sample ID and laid out in numerical sequence on the floor of the facility. Where the sample interval was marked to insert a field duplicate, the half core to be sampled was cut into quarter core. One quarter was placed back in the core box while the other quarter was inserted as the field duplicate. The entire sample stream, including the QA/QC samples, were then put into large sacks, sealed with a secure band, and stored until transport to the SGS laboratory in Durango, Mexico. Transport to the SGS lab was usually by trucks belonging to SGS, though occasionally, if the sample shipment was small, the samples would be transported by Mexican Gold personnel in company trucks.

The core logging and core storage facility within the Las Minas townsite which also serves as the storage of returned sample pulps is within a fenced enclosure with a locked gate.

Samples were transported in security-sealed bags to Inspectorate Laboratories in Guatemala City for sample preparation up until March 2020, and thereafter to Inspectorate Laboratories in Managua due to closure of the Guatemalan facility in 2020. Samples were prepared at Inspectorate by crushing and pulverizing the drill core down to 85% passing -75 µm. Pulps were weighed and individually packaged into 100 g envelopes and shipped for analysis.

All Las Minas project drill and surface samples have been sent to SGS for processing, except for a limited number of second-lab QA/QC check samples that were sent to ALS Metallurgy Kamloops. SGS is presently accredited by the International Organization for Standardization (ISO) and has ISO 9001 certification and fulfills ISO/IEC 17025 testing requirements. SGS and ALS are independent of Mexican Gold and its subsidiaries.

Samples were prepared following protocol for mineral sample preparation including weighing, drying, crushing, sieving, splitting, and pulverization. Samples were analyzed for gold and silver using fire assay techniques, and for copper and 33 other elements using ICP-AES (“inductively coupled plasma - atomic emission spectroscopy”) techniques. SGS has employed the same analytical methods and procedures for all of Mexican Gold's drill samples commencing in 2011. The analytical methods used included:

Gold	30 g fire assay with atomic absorption spectroscopy (AAS) finish (SGS code FAA313); analytical limits of 0.005 ppm to 10.0 ppm. For samples that assayed greater than 10.0 ppm, a 30 g fire assay with a gravimetric finish (SGS code FAG303) was performed.
Silver	2 g aqua regia digest with AAS determination; (SGS code AAS12E); 0.3 - 300 ppm analytical limits. For samples that assayed greater than 300 ppm, a 30 g fire assay with gravimetric finish was performed (SGS code FAG313).
34 Elements	Major, minor and trace elements including Cu were assayed using ICP-AES following an aqua-regia digestion (SGS code ICP14B); analytical limits including 0.5 ppm to 10,000 ppm limits for Cu. For samples that assayed greater than 10,000 ppm Cu, another ICP-AES determination was performed following a sodium peroxide fusion (SGS code GO-ICP90Q).

Pulps are shipped for regular and QA/QC analysis to Inspectorate Laboratories (a division of Bureau Veritas) in Reno, Nevada, USA and ALS Chemex in Vancouver, BC, Canada, respectively. Both are ISO 17025 accredited laboratories. Gold and silver were analyzed by a 30 g charge with atomic absorption with gravimetric finish for values exceeding 5 g Au/t and 100 g Ag/t.

11.2 Quality Assurance & Quality Control

11.2.1 QA/QC Performance & Discussion of Results for Drillhole Data 2011-2020

Since 2011, Mexican Gold has implemented a comprehensive QA/QC program employing industry standards and best practices for all drill core. QA/QC samples were inserted into the sample stream sent to SGS on a regular basis for all Mexican Gold drill campaigns. The QA/QC samples consisted of pulp blanks, certified reference materials (CRM or standards), duplicate samples and repeat (umpire) samples. The duplicates samples consisted of field duplicates (quarter-core splits), preparation pulp duplicates, coarse rejects, and second-lab pulp and coarse reject re-assays (repeat samples). The QA/QC samples have made up about 10% of the total samples analyzed.

A total of 862 control samples (Table 11-1) were assigned for QA/QC purposes, accounting for approximately 10% of the total samples taken during the program.

Table 11-1: Quantity of Control Samples by Type

Control Type	Number
Standards	560
Pulp Blanks	260
Pulp and Coarse Reject Duplicates	142
Total	**862**

Source: Kirkham (2021)

QA/QC assay results were checked by project geologist on a batch-by-batch basis for analytical or batch errors.

Analytical "pulp" blanks are non-mineralized material sourced from CDN Resource Laboratories Ltd. of Langley, BC, Canada (CDN Laboratories) (Table 11-2) that are inserted into the sample series every 40 samples. Field blanks are inserted to test for any potential carry-over contamination that might be occurring at the crushing phase of sample preparation due to poor laboratory cleaning practices and to monitor for calibrations problems.

Table 11-2: Analytical Blank Used by Year

CDN Blank	#	Year
BL-7	45	2011
BL-8	32	2011
BL-9	17	2012-2020
BL-10	166	2011-2013

Source: Kirkham (2021)

Duplicate analysis of pulps and quarter-core are used to evaluate the analytical precision and to determine if any biases exist between laboratories. Duplicate analysis of coarse rejects is used to analyze preparation error.

Standards are used to test the accuracy of the assays and to monitor the consistency of the laboratory over time. A variety of certified standards of various gold grades were purchased from CDN Laboratories (see Table 11-3) and inserted by the logging geologists.

Table 11-3: Summary of Standards

Certified Standard	Gold Grade (g/t)	Standard Deviation (+/-)	Copper Grade (%)	Standard Deviation (+/-)	Years Used
CGC-22	0.64	0.03	0.725	0.014	2011, 2012, 2014
CGC-26	1.64	0.055	1.58	0.035	2011, 2014, 2015, 2017, 2018, 2020
CGC-28	0.727	0.038	2.033	0.054	2011, 2012
CGC-30	0.338	0.048	0.154	0.011	2020
CM-13	0.74	0.047	0.786	0.018	2011, 2012
CM-15	1.253	0.059	1.28	0.045	2017, 2018, 2020
CM-24	0.521	0.028	0.371	0.009	2014, 2015
CM-42	0.576	0.05	0.526	0.022	2020
GS-4B	3.77	0.175	n/a	n/a	2012, 2014
GS-5L	4.74	0.11	n/a	n/a	2017, 2018
GS-5T	4.86	0.13	n/a	n/a	2017, 2019
ME-11	1.38	0.05	2.44	0.055	2011
ME-2	2.1	0.11	0.48	0.018	2011

Source: Kirkham (2021)

11.2.1.1 QA/QC Performance & Discussion of Results for Drillhole Data 2011-2020

The performance of the quality control programs for the drillhole data is relatively good for all elements with a few specific exceptions. The programs between 2011 through 2020 included the insertion of blanks, standards and the analysis of duplicates.

Blanks

Blanks are barren samples with an expected very low grade. These are submitted to ensure that there is no contamination between samples during the sample preparation or assaying. If the blank samples following high grade samples have elevated grades, then there have been problems.

Blank performance in 2011 which utilized CDN BL-7 and BL-8 resulted in 3 and 1 failures, respectively for a total of 4 failures in 77 or 5.2%. Although not a high rate of failure, this may be an indication of issues at the laboratory such as insufficient cleaning practices or potential calibration issues. In subsequent years, 2012 through 2020, in which CDN BLK-9 and BL-10 were utilized there was only 1 failure in 183 or 0.5% failure rate. Analyses of both pulps consistently yielded gold values near or below the detection limit of the primary laboratory. No sample contamination was detected.

Standards

Standards are samples of known (usually certified) grade that are submitted to monitor the accuracy of a laboratory, i.e., the ability of the laboratory to get the correct or known result. A laboratory showing a systematic difference from the expected result is said to exhibit a bias. Standard samples ensure that the laboratory quality control procedures are effective, and that significant bias is not evident within or between assay batches. These standard samples may be variously referred to as CRMs.

The control standards for gold showed excellent performance with only 2 failures in 560 or 0.4% (3 standard deviations) and 26 warnings or 4.6% (3 standard deviations) for the entirety of the database as shown in Table 11-3. However, the results for the copper analyses were relatively poor with an overall failure rate of 6.0% and warning rate of 13.6%. CGS-22, CGS-28, CM-13, ME-2 and ME-11 have high levels of failure and particularly warnings. These standards were used in the 2011 QA/QC sample stream and it is clear that the issues were predominant in 2011 with one of the two gold failures, 33% of the gold warnings, 64% of the copper failures and 73% of the copper earnings occurring during this early campaign. In the QP’s opinion, the issues are not related to the control samples used however this cannot be completely discounted as subsequent standards are not experiencing the same issues however, it appears that there were systemic QA/QC issues in 2011 that were not detected but appear to be addressed in subsequent years.

Table 11-4: Summary of Results for Control Standards

STD-ID	#N	Grade Au	SD	Grade Cu	SD	Failure-Au	Failure-Au	Warning - Au	Warning - Au	Failure-Cu	Failure-Cu	Warning - Cu	Warning - Cu
		(g/t)	(g/t)	(%)	(%)	(#)	(%)	(#)	(%)	(#)	(%)	(#)	(%)
CGC-22	68	0.64	0.06	0.725	0.028	0	0.0%	5	7.4%	**5**	**7.4%**	**17**	**25.0%**
CGC-26	121	1.64	0.11	1.58	0.07	1	0.8%	0	0.0%	4	3.3%	8	6.6%
CGC-28	19	0.727	0.076	2.033	0.108	1	5.3%	**4**	**21.1%**	1	5.3%	**4**	**21.1%**
CGC-30	14	0.338	0.048	0.154	0.011	0	0.0%	0	0.0%	1	7.1%	1	7.1%
CM-13	19	0.74	0.094	0.786	0.036	0	0.0%	0	0.0%	**3**	**15.8%**	**5**	**26.3%**
CM-15	85	1.253	0.118	1.28	0.09	0	0.0%	7	8.2%	3	3.5%	6	7.1%
CM-24	29	0.521	0.056	0.371	0.018	0	0.0%	0	0.0%	0	0.0%	0	0.0%
CM-42	21	0.576	0.05	0.526	0.022	0	0.0%	**4**	**19.0%**	0	0.0%	0	0.0%
GS-4B	38	3.77	0.35	n/a	n/a	0	0.0%	1	2.6%	n/a	n/a	n/a	n/a
GS-5L	29	4.74	0.22	n/a	n/a	0	0.0%	0	0.0%	n/a	n/a	n/a	n/a
GS-5T	30	4.86	0.26	n/a	n/a	0	0.0%	0	0.0%	n/a	n/a	n/a	n/a
ME-11	44	1.38	0.1	2.44	0.11	0	0.0%	3	6.8%	**6**	**13.6%**	**7**	**15.9%**
ME-2	43	2.1	0.11	0.48	0.018	0	0.0%	2	4.7%	**5**	**11.6%**	**15**	**34.9%**
	560					2	0.4%	26	4.6%	28	6.0%	63	13.6%

Source: Kirkham (2021)

Table 11-5: Results for Control Standards by Year

YEAR	Failure-Au #	Failure-Au %	Warning-Au #	Warning-Au %	Failure-Cu #	Failure-Cu %	Warning-Cu #	Warning-Cu %
2020	0	0%	6	25%	1	4%	4	6%
2019	0	0%	2	8%	1	4%	1	2%
2018	0	0%	4	17%	1	4%	1	2%
2017	0	0%	0	0%	4	14%	6	10%
2015	0	0%	0	0%	1	4%	1	2%
2014	1	50%	3	13%	1	4%	3	5%
2012	0	0%	1	4%	1	4%	1	2%
2011	1	50%	8	33%	18	64%	46	73%
TOTAL	**2**		**24**		**28**		**63**	

Source: Kirkham (2021)

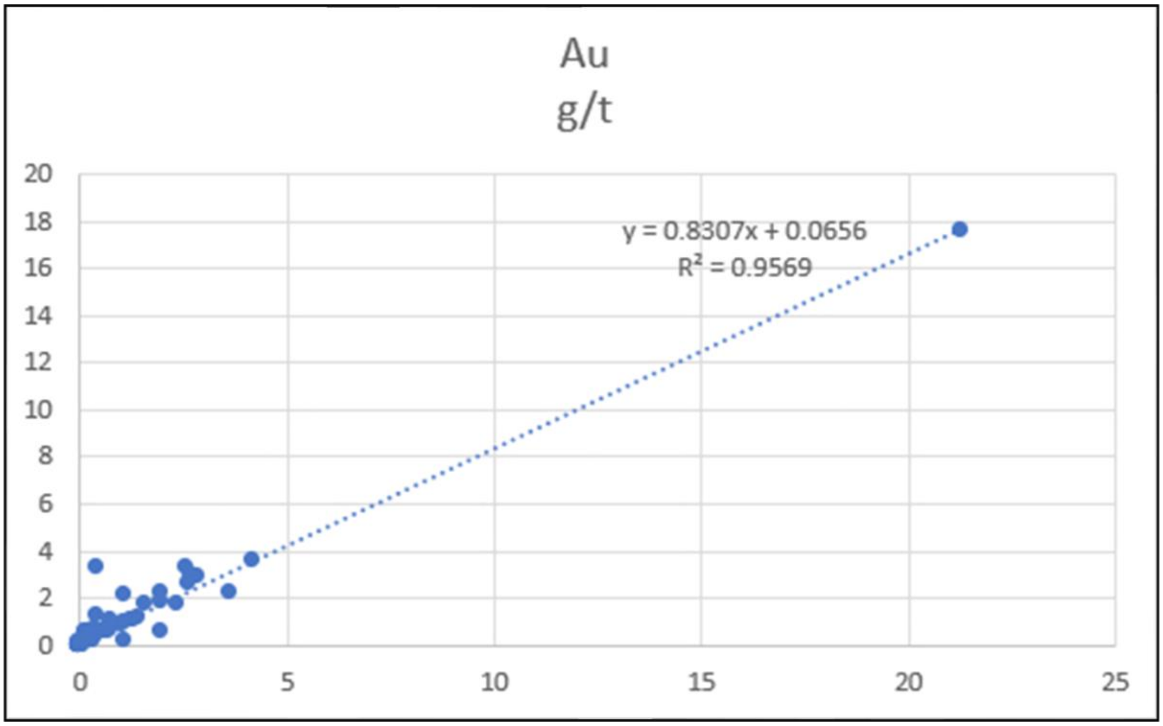
Duplicates

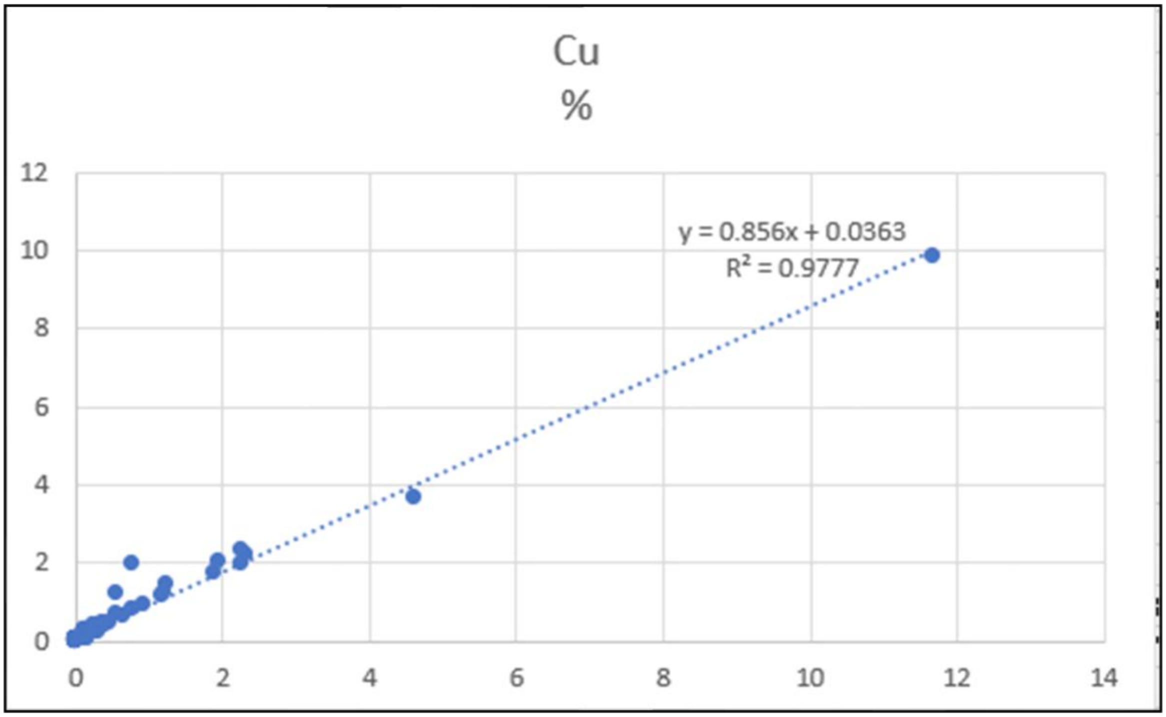
Duplicates are samples collected, prepared and assayed in an identical manner as an original sample, to provide a measure of the total error of sampling. When this error is derived in relative terms, the total error is the sum of the errors due to splitting the initial duplicate, preparing the sample and assaying the sample. Residues of samples may be collected at all stages of the sampling protocol.

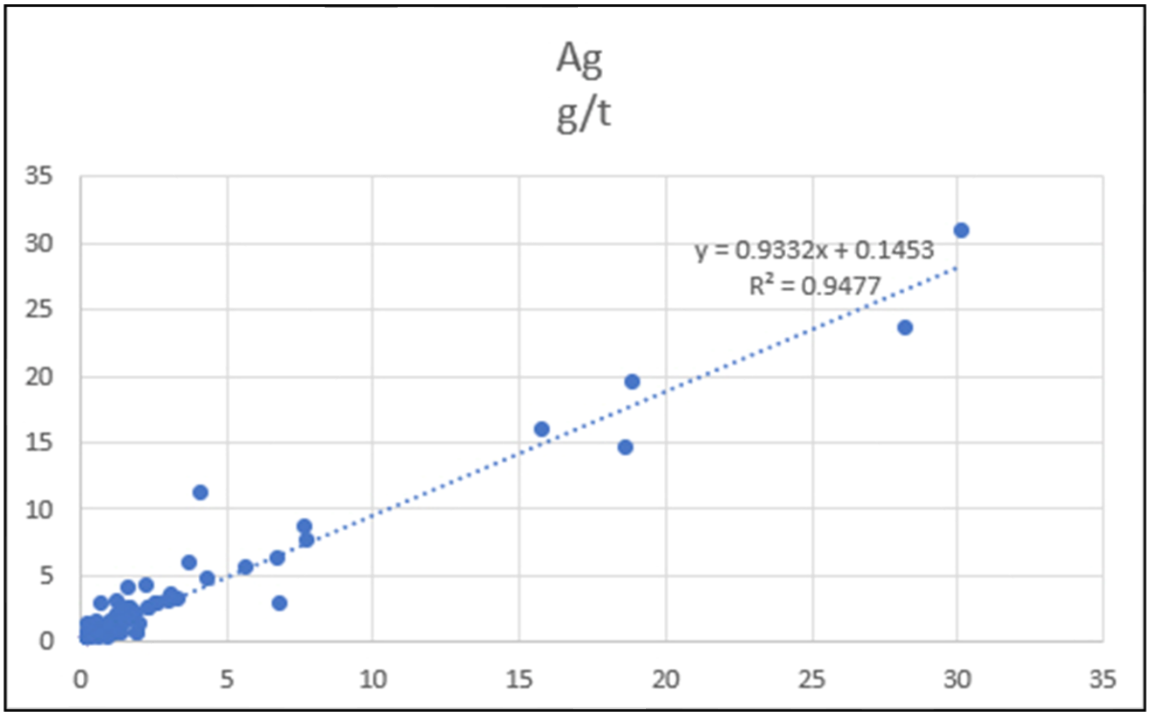
Field Duplicates are collected by taking quarter core from the same intersection from which the split half core being sent for analysis was sourced. Submitting the second half of sawn diamond core is actually a way of measuring the difference in grade between very closely adjacent different samples in the deposit. Laboratory duplicates may be produced by taking a second split after crushing diamond drill core, before the pulverizing stage or coarse duplicate. Crushing and pulverizing reduces the particle size of drill core to a nominal size (e.g., 90% passing 75 µm) and then a small subsample of this pulp is retained for assay in a pulp packet. Laboratory duplicates may be produced by taking a second sub-sample after pulverizing stage or pulp duplicate.

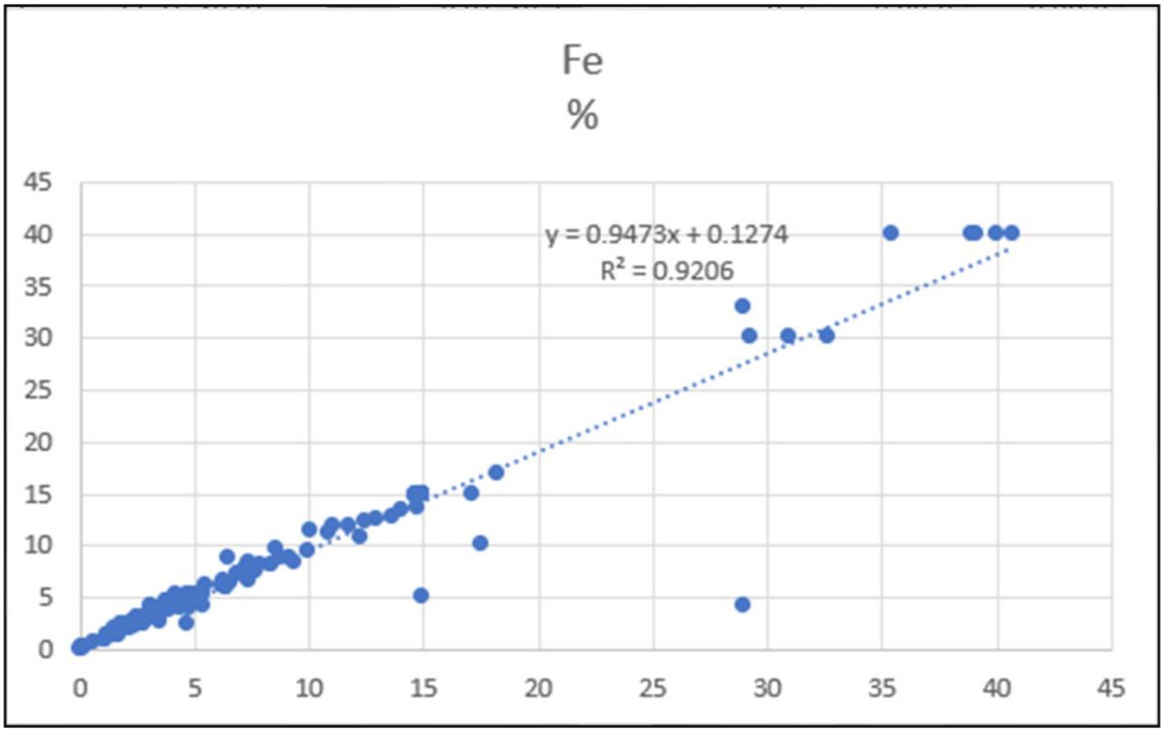
A total of 140 duplicates have been analyzed since 2011 which included 114 field, 20 coarse and 6 pulp duplicates. Of these 4 were analyzed in 2011, 16 in 2014, 9 in 2015, 21 in 2017, 11 in 2018, 7 in 2019, 46 in 2020. In totality, the duplicates for gold, copper, silver and iron illustrate relatively good accuracy and repeatability however when sample type is spilt out, potential issues are identified specifically with regard to the gold field duplicates.

Figure 11-1: All Duplicates for Gold, Copper, Silver and Iron - Original (x-axis) vs Duplicate (y-axis)









Source: Kirkham (2021)

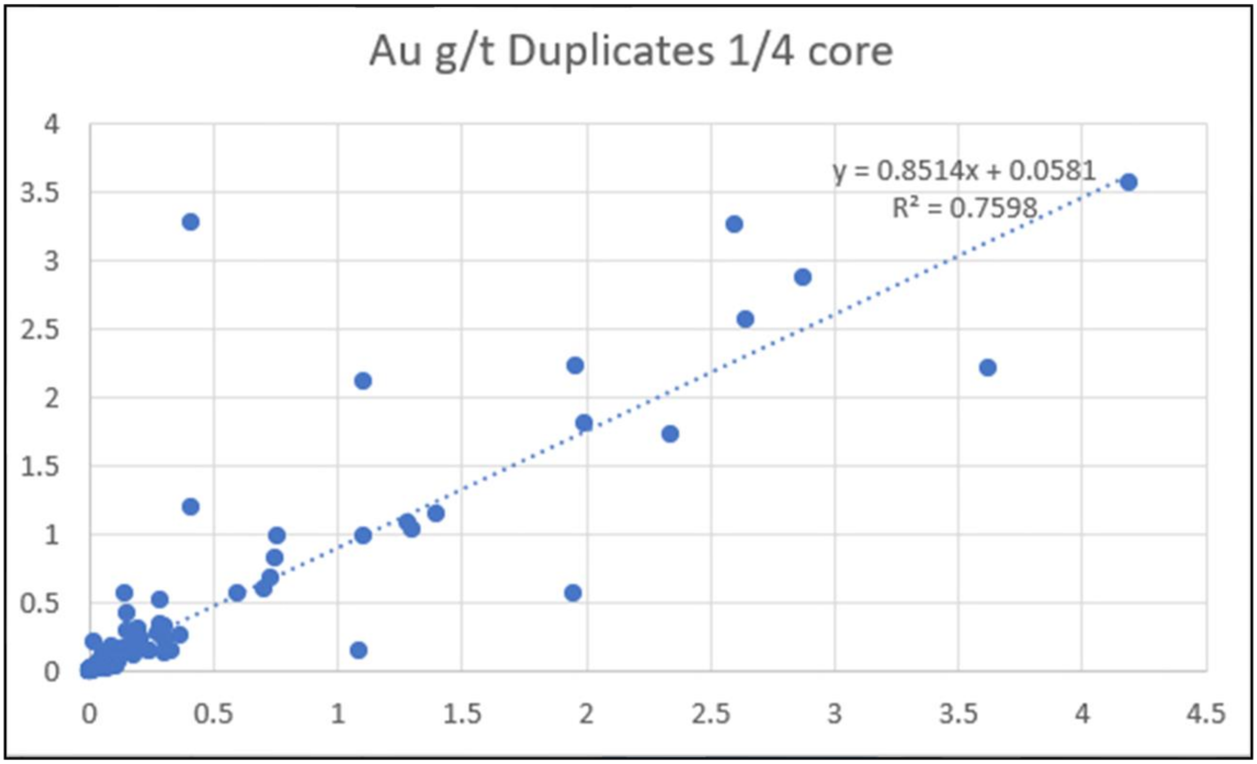
Quarter-Core Field Duplicates

Field duplicates are secondary quarter-core samples collected from the same drill interval as the original sample. They are mainly used to assess the natural grade variability of the deposit, as well as to evaluate the total subsampling variances attributable to splitting both in the field and in all subsequent subsampling steps in the laboratory.

The field duplicate sampling procedure was to take both the original and duplicate as quarter-core samples from the same half-cut core piece. A total of 114 field duplicate samples were analyzed by SGS from 2011 through 2020.

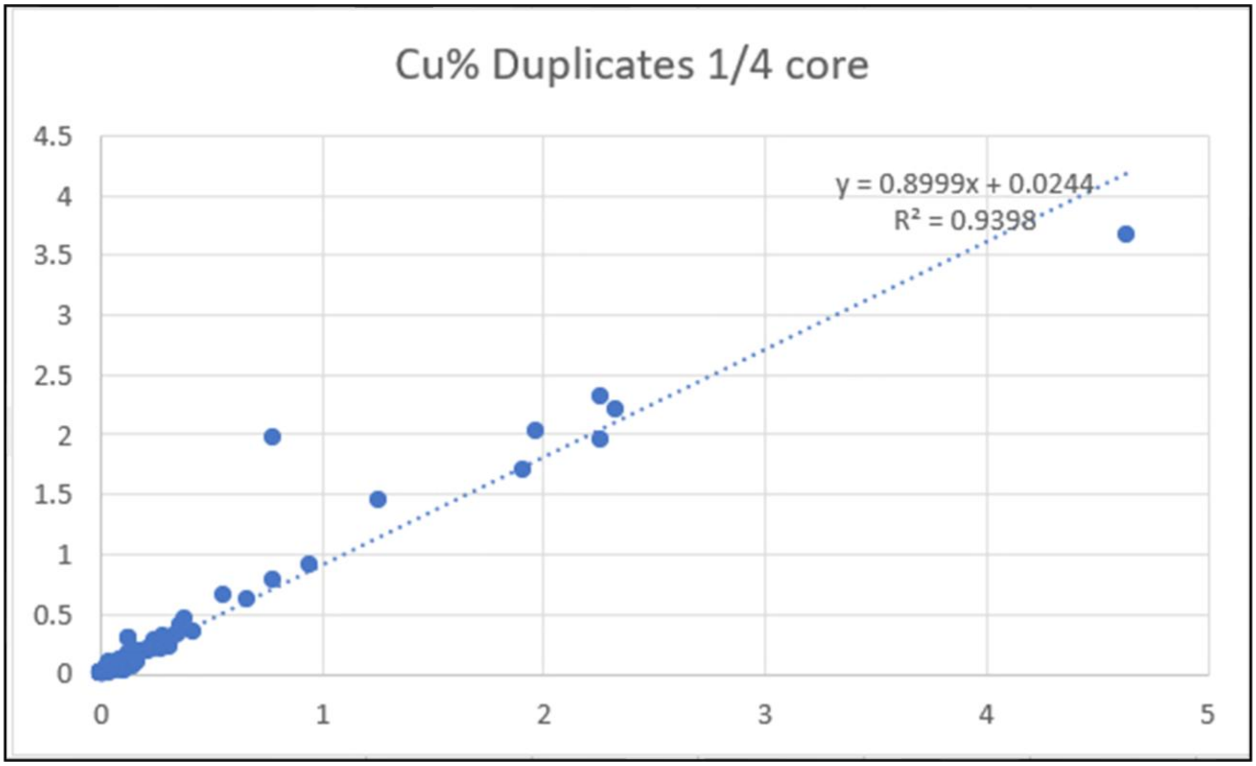
Figure 11-2 shows the field duplicates for gold to have relatively moderate bias with the correlation coefficient being 0.76 where 1.00 is perfect correlation and low precision or scatter. These issues may be the result of biased or inaccurate cutting when halving and quartering the core or sample selection issues when bagging the samples. However, the issues are more likely caused due to the variability of the gold within the samples and that the deposit is more ‘nuggety’ than anticipated. As the field duplicate analyses for copper and silver are much better behaved, the later appears to be the case for the gold variability as shown in Figure 11-3 and Figure 11-4. Low bias and high precision with the exception of one outlier.

Figure 11-2: Quarter-Core Field Duplicates for Gold - Original (x-axis) vs Duplicate (y-axis)



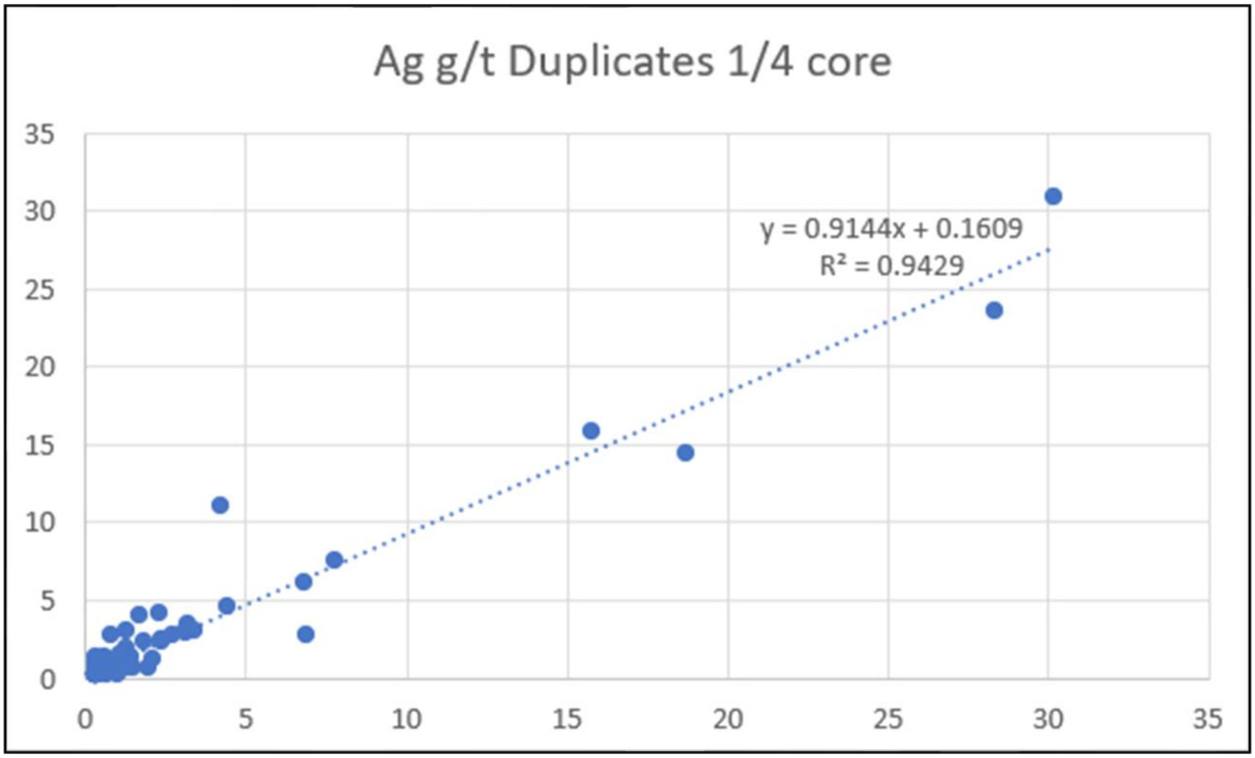
Source: Kirkham (2021)

Figure 11-3: Quarter-Core Field Duplicates for Copper - Original (x-axis) vs Duplicate (y-axis)



Source: Kirkham (2021)

Figure 11-4: Quarter-Core Field Duplicates for Silver - Original (x-axis) vs Duplicate (y-axis)

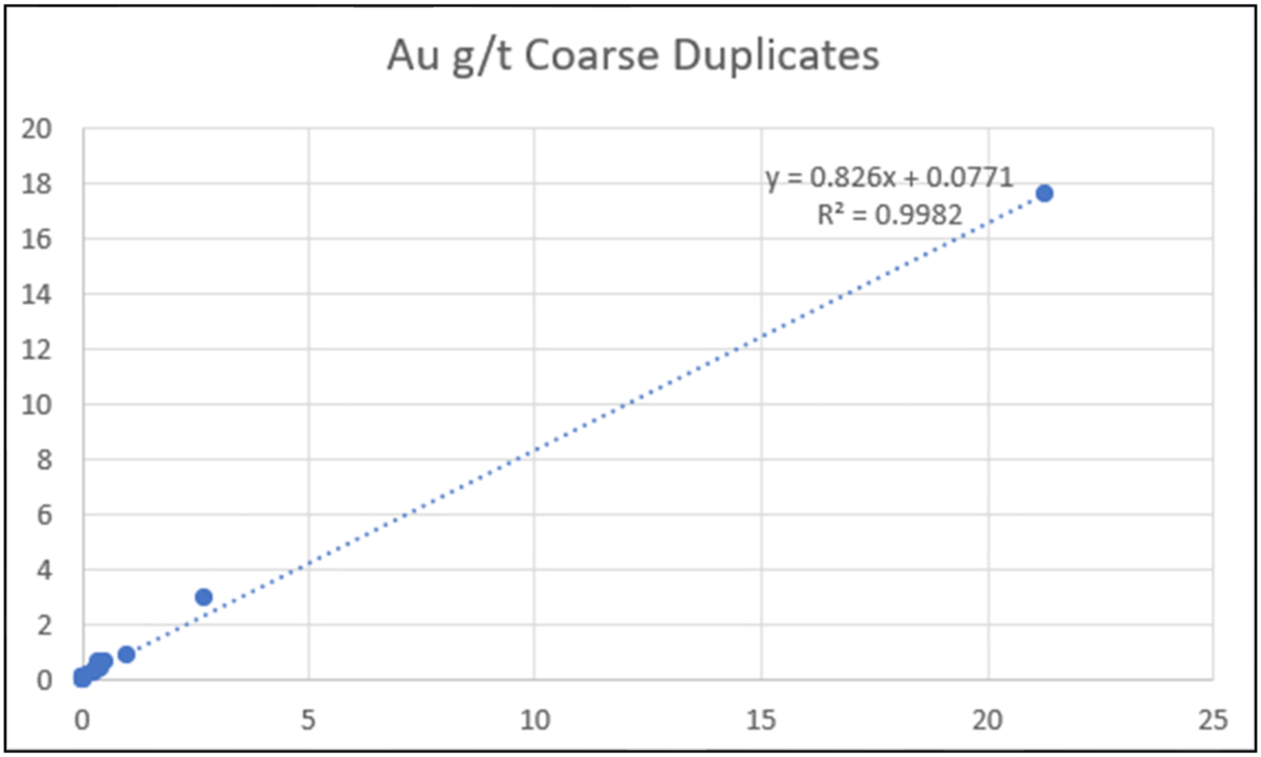


Source: Kirkham (2021)

Coarse Reject Preparation Duplicates

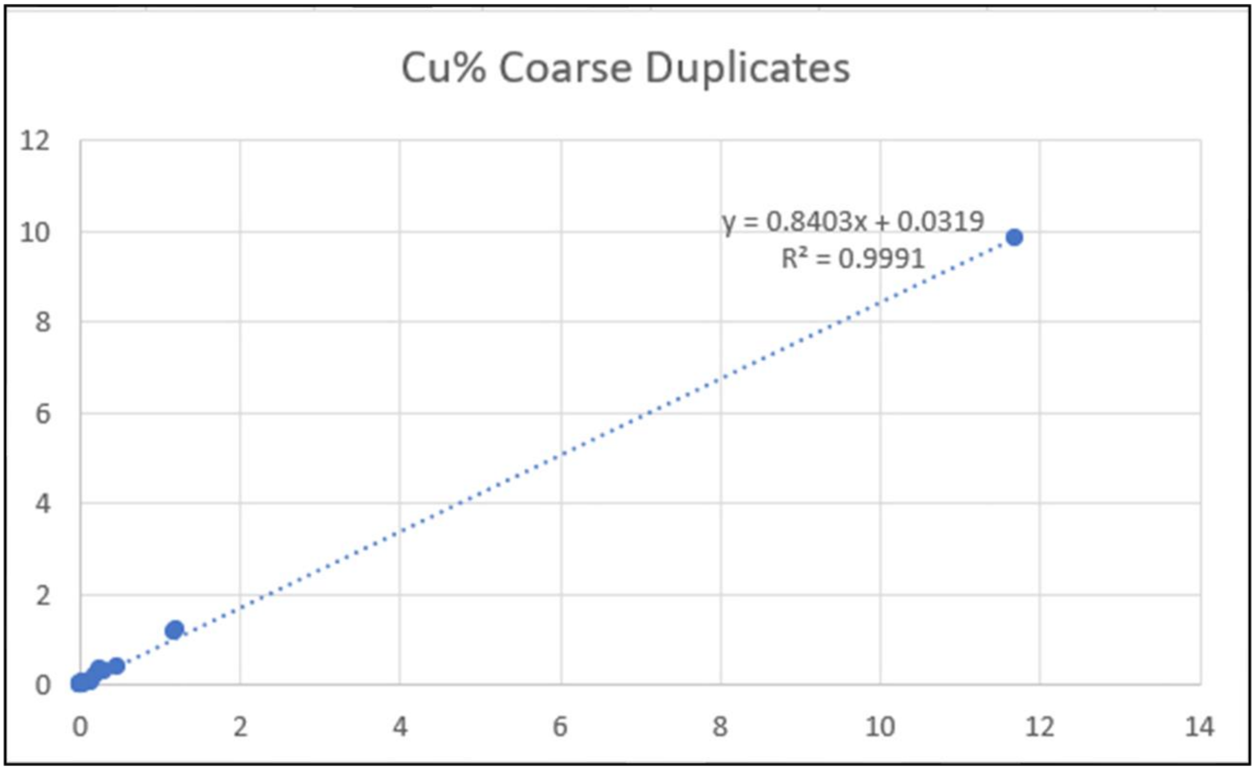
Mexican Gold conducted a limited program of coarse reject duplicate analyses. A sample was derived from the original sample coarse reject which was included within the original core sample stream. A total of 20 coarse reject duplicate analyses are within the project database. The analysis illustrates excellent accuracy and repeatability for gold, copper and silver as shown in Figure 11-5, Figure 11-6 and Figure 11-7.

Figure 11-5: Coarse Duplicates for Gold - Original (x-axis) vs Duplicate (y-axis)



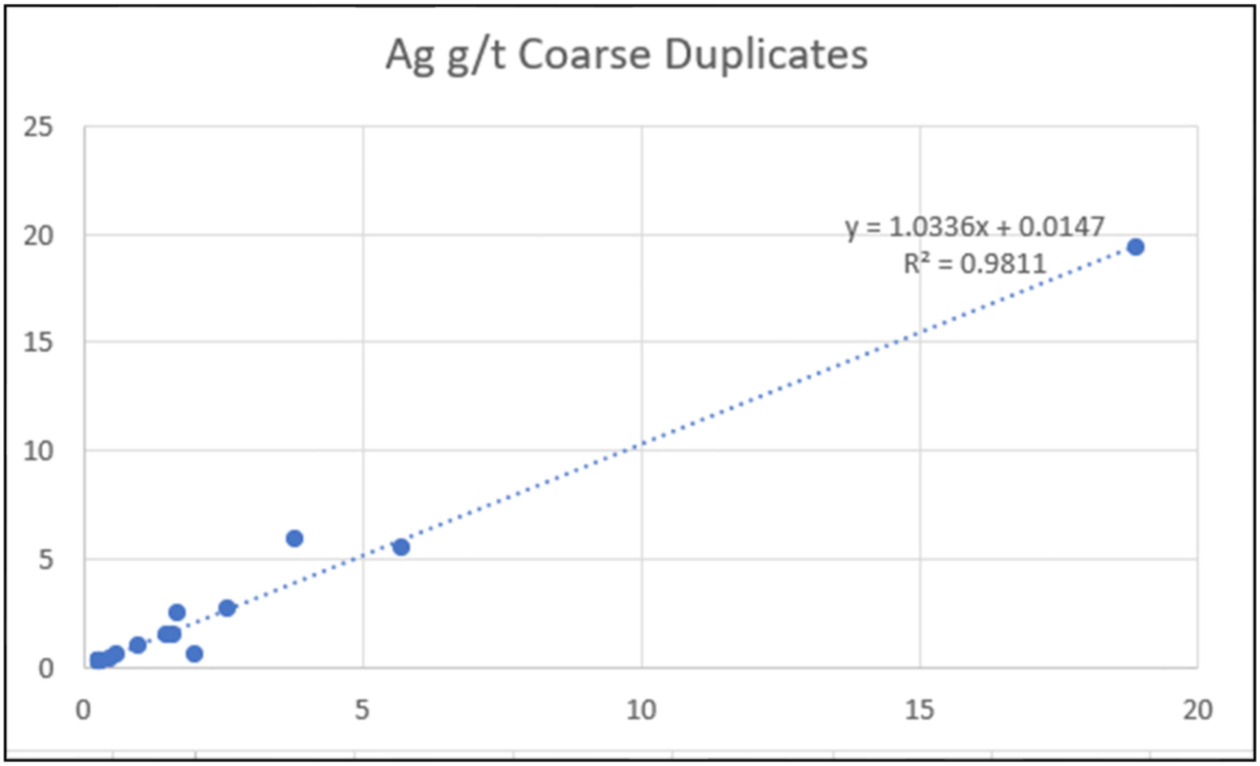
Source: Kirkham (2021)

Figure 11-6: Coarse Duplicates for Copper - Original (x-axis) vs Duplicate (y-axis)



Source: Kirkham (2021)

Figure 11-7: Coarse Duplicates for Silver - Original (x-axis) vs Duplicate (y-axis)

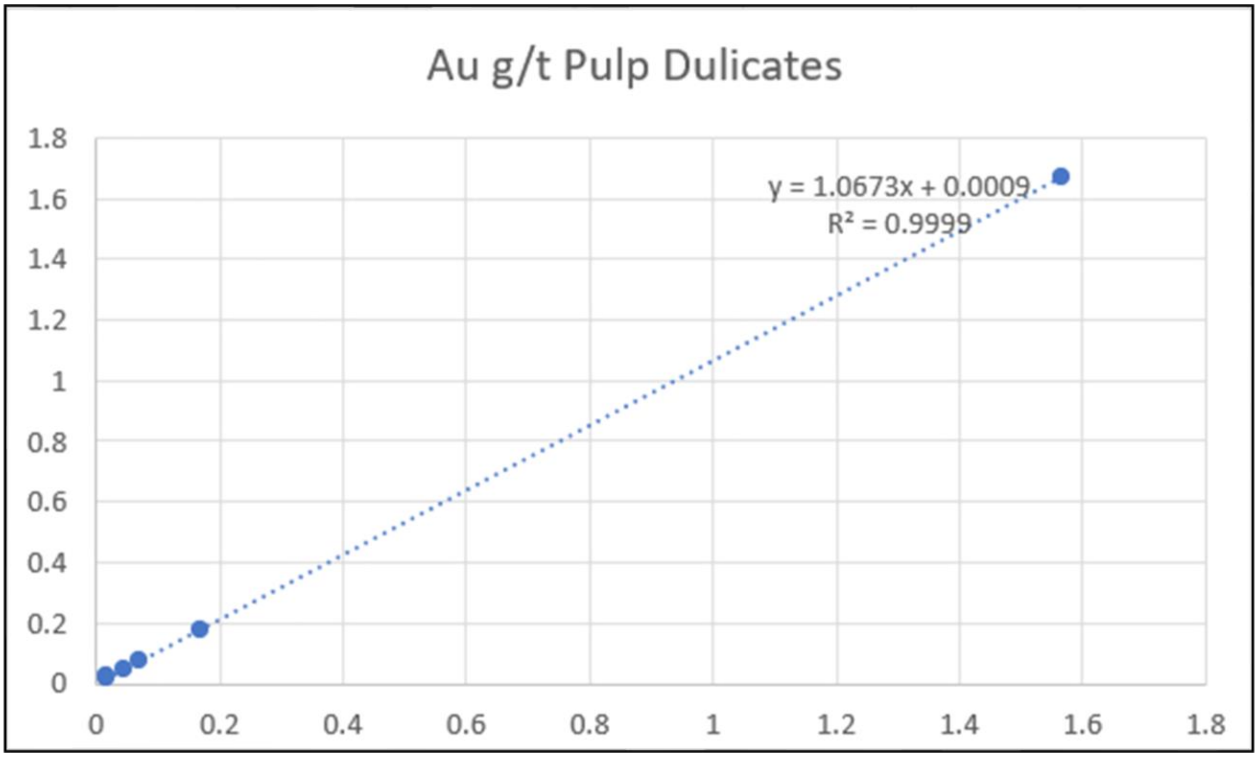


Source: Kirkham (2021)

Pulp Reject Preparation Duplicates

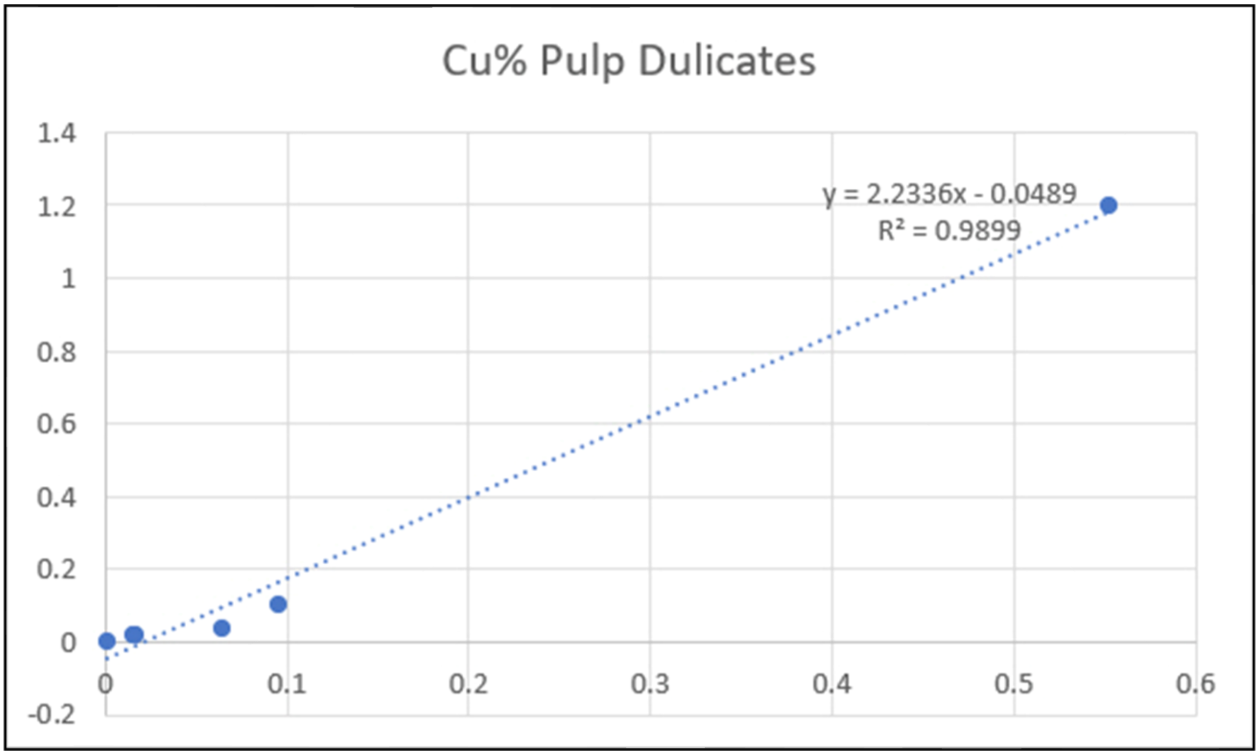
A limited program of pulp reject duplicate analyses was performed in 2014 and 2015. A second pulp from the original sample pulp reject and include that sample within the original core sample stream. Although this is a very small dataset with only 6 pulp duplicates, the results showed excellent repeatability.

Figure 11-8: Pulp Duplicates for Gold - Original (x-axis) vs Duplicate (y-axis)



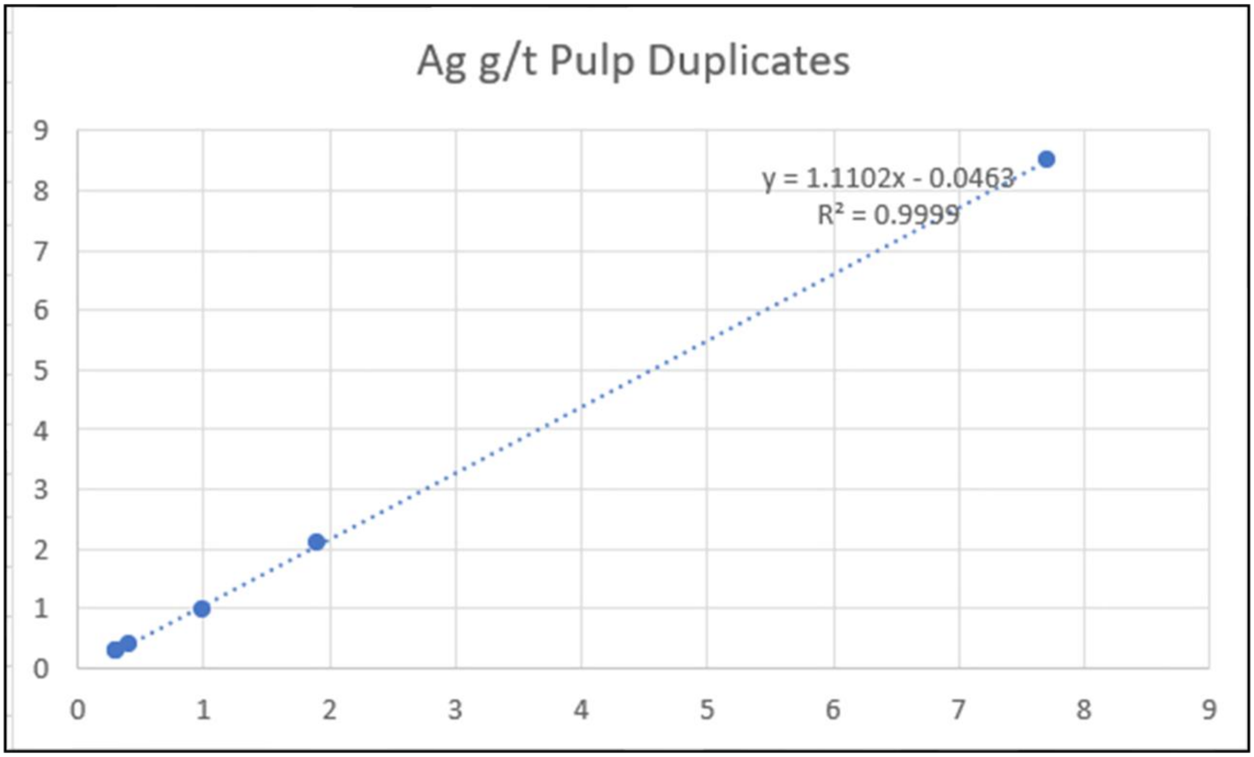
Source: Kirkham (2021)

Figure 11-9: Pulp Duplicates for Copper - Original (x-axis) vs Duplicate (y-axis)



Source: Kirkham (2021)

Figure 11-10: Pulp Duplicates for Silver - Original (x-axis) vs Duplicate (y-axis)



Source: Kirkham (2021)

Repeats (Umpire Laboratory)

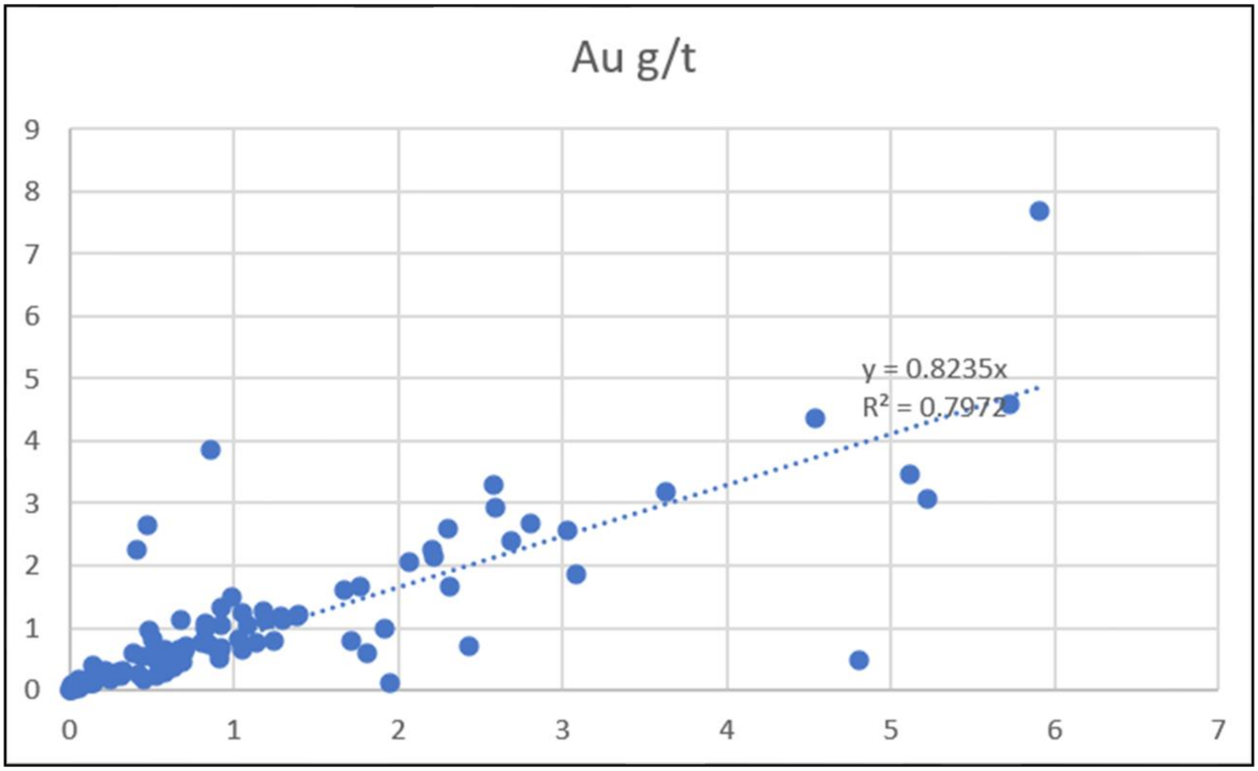
Repeats are samples that have been previously prepared and assayed (so they are already finely comminuted pulps) and that then have been re-submitted for another identical analysis at another laboratory. Comparison of the results provides a measure of the precision of a laboratory, i.e., the ability of the laboratory to get the same assay result under the same conditions.

Pairs of samples assayed at different laboratories may help to define the inter-laboratory precision and may also identify a bias between the two laboratories.

Duplicates of coarse and pulp rejects were sent to ALS Chemex for check gold, copper and silver analysis with the analysis at SGS. The comparisons between the laboratories for the gold pulp duplicates show poor correlation as shown in Figure 11-11. It appears there is a relatively high bias for the gold duplicates at SGS versus ALS and repeatability or precision is poor. However, the comparison is significantly improved if the early data series particularly that data that was collected in the 2011 drilling campaign, is excluded as shown in Figure 11-12. This illustrates further data quality issues with the 2011 drilling campaign and further supports the exclusion problematic data from the dataset as discussed in Sections 10, 12 and 14. This entailed excluding 16 out of a total of 39 frilled in 2011. These holes were predominantly within the Santa Cruz deposit area. Should they be validated and verified, they may be considered for inclusion in the future.

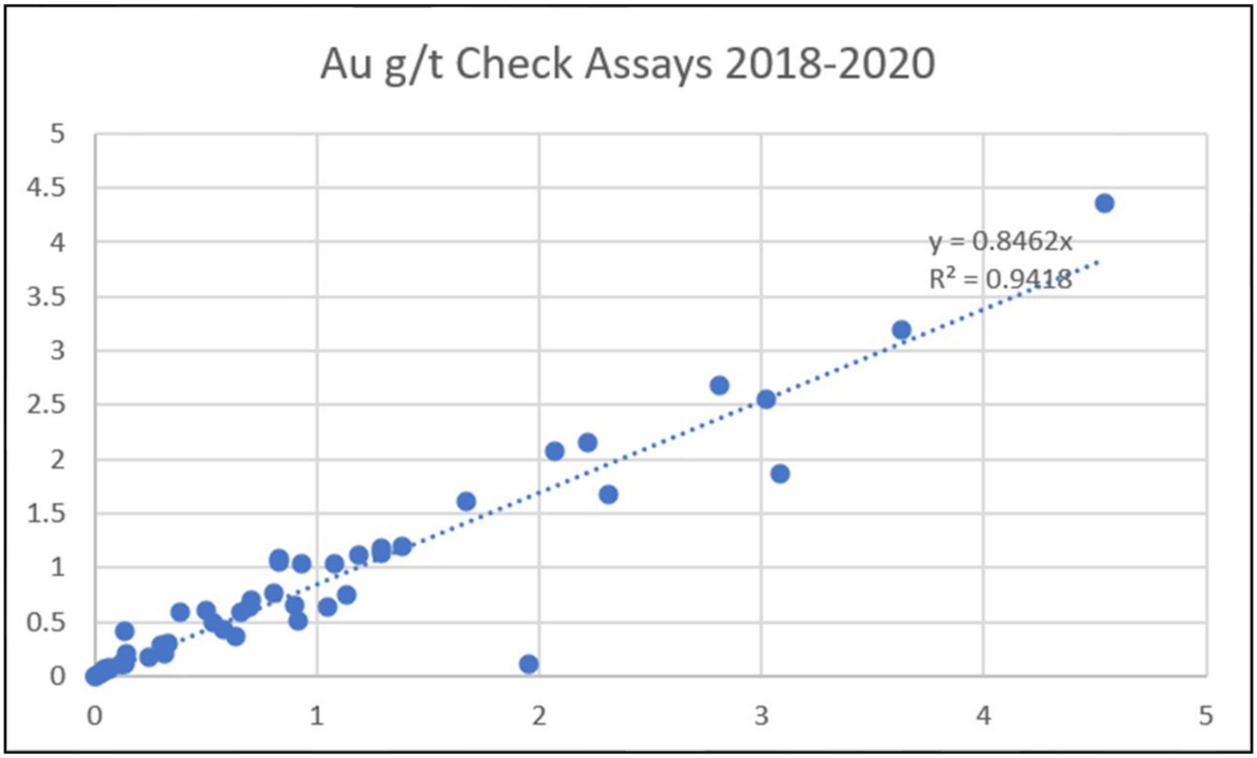
Figure 11-13 and Figure 11-14 shows the comparison for the repeat duplicate pulp rejects which illustrates excellent agreement with correlation coefficients being 0.995 and 0.959, for copper and silver respectively.

Figure 11-11: ALS (y-axis) Pulp Checks vs. SGS (x-axis) Original – Gold



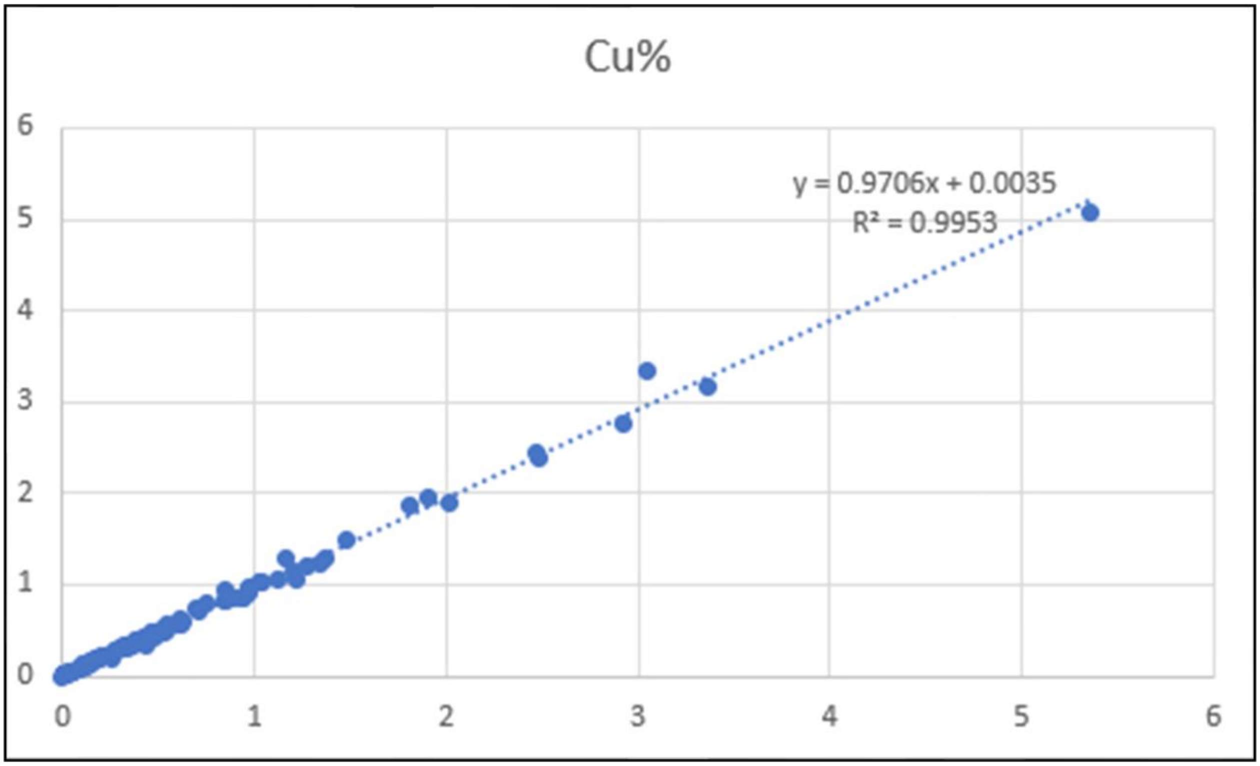
Source: Kirkham (2021)

Figure 11-12: 2018-2020 ALS (y-axis) Pulp Checks vs. SGS (x-axis) Original – Gold



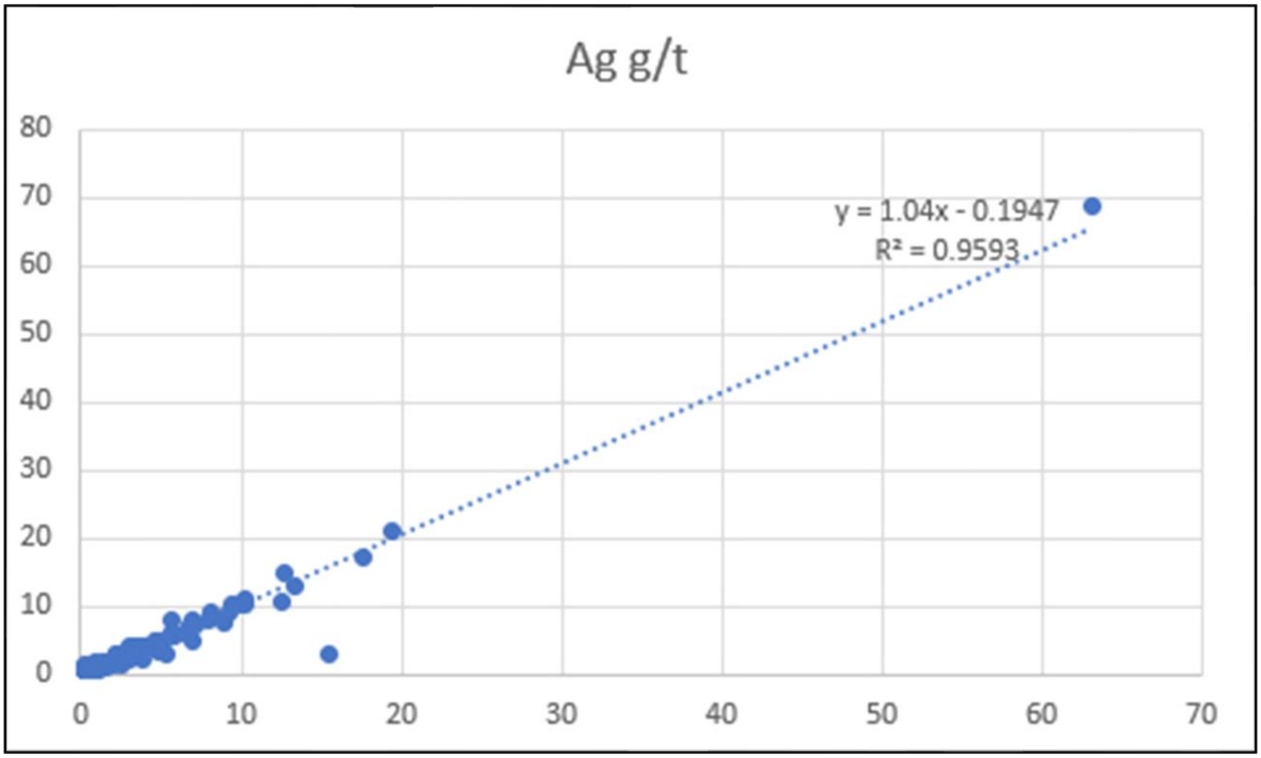
Source: Kirkham (2021)

Figure 11-13: ALS (y-axis) Pulp Checks vs. SGS (x-axis) Original – Copper



Source: Kirkham (2021)

Figure 11-14: ALS (y-axis) Pulp Checks vs. SGS (x-axis) Original – Silver



Source: Kirkham (2021)

11.2.2 QA/QC Performance & Discussion for Chip and Channel Samples

The QA/QC practices are designed to ensure that all data is correctly entered, checked and reported in a timely manner. Standards and blanks are inserted into the sample stream at one for every ten samples supervised company geoscientific professionals. The standards and blanks were obtained from CDN Laboratories. The Reference Standards used for the surface sampling program were CDN-ME-2 is 2.1 g/t Au +/- 0.11 g/t and 0.48% Cu +/- 0.018%; CDN-ME-11 is 1.38 g/t Au +/- 0.1 g/t and 2.44% Cu +/- 0.11%; and the CDN-BL-7 (blank).

A total of 99 QA/QC samples were inserted with 33 ME-2 and 30 ME-11 standards along with 36 BL-7 blanks. Results showed good performance for gold where there were no failures or warning for the ME-2 standard and no failures or warnings for the ME-11 standard. It should be noted that a failure is three standard deviations from the expected standard grade and a warning is two standard deviations. Results for copper were good for the ME-11 standards with two failures and two warnings while the performance for the ME-2 standard was poor with five failures and twelve warnings.

11.3 Conclusions and Opinion

The sampling methods and approach employed for the various sampling and drilling programs are appropriate for the deposit type and generally follow industry best practice guidelines. The analytical methods used are also appropriate for the minerals and elements being explored and are performed by a certified laboratory.

There are issues with the quality assurance and quality control within the early 2011 program which has been addressed by excluding a subset of this dataset. Further QA/QC would be required to warrant the inclusion of this data in the future. In addition, the company should review its QA/QC program and should increase the frequency for insertion of QA/QC samples.

It is the opinion of the QP, Garth Kirkham, P.Geo., that the sampling preparation, security, analytical procedures and quality control protocols used by Mexican Gold are consistent with generally accepted industry best industry best practices and are therefore reliable for the purpose of resource estimation.

12 DATA VERIFICATION

Site visits were conducted by several of the QPs, as detailed in Section 2.2. The purpose of the visits was to fulfil the requirements specified under NI 43-101 guidelines and become familiar with the property. These site visits consisted of tours of mineralized and non-mineralized headings, as well as an inspection of the surface core logging, sampling, storage areas, and existing infrastructure. There were no limitations on, or failure to conduct, the data verification for this report.

12.1 General Project

12.2 Geology, Drilling & Assaying

Garth Kirkham, P. Geo., visited the property between January 16 through 19, 2021. The site visits included an inspection of the property, offices, drill sites, outcrops, drill collars, core storage facilities, core receiving area, and tours of major centers and surrounding villages most likely to be affected by any potential mining operation.

The tour of the office and storage facilities showed a clean, well-organized, professional environment. On-site staff led the author through the chain of custody and methods used at each stage of the logging and sampling process. All methods and processes are up to industry standards and reflect best practices, and no issues were identified.

A visit to the collar locations showed that the collars were difficult to find in many cases as the area is subject to human and wildlife disturbance as well as experiencing significant weather and extensive vegetation. Four collars were identified and marked however there appeared to be discrepancies between locations, particularly elevations, and recorded locations within the database. As a result of the uncertainties encountered, a plan to resolve was agreed with the company as detailed in Section 12.2.1.

The author selected five drillholes from the database that were representative of the El Dorado and Santa Cruz deposits and represented both geological and spatial variation. These drillholes were laid out at the core storage area and inspected in detail. Site staff supplied the logs and assay sheets for verification against the core and the logged intervals. The data correlated with the physical core and no issues were identified. In addition, the author toured the complete core storage facilities, selecting and reviewing core throughout. No issues were identified, and recoveries appeared to be very good. In addition, the methods and procedures for specific gravity measurements were reviewed and approved.

A complete review of the drill core and sample chain of custody was performed and reviewed. All methods and procedures followed standard industry best practice and no issues were identified.

Based on the site visit and an inspection of all aspects of the project, the author is confident that the data and results are valid, including all methods and procedures. It is the opinion of the independent author that all work, procedures, and results have adhered to best practices and industry standards required by NI 43-101. No duplicate samples were taken to verify assay results, but the author believes that the work is being performed by a competent professional that adheres to industry best practices and standards in the opinion of the QP.

The core is accessible, and the core is stored in a secure warehouse and compound. The core facilities are clean and well organized for easy access and analysis by way of a core map.

12.2.1 Collars

As discussed previously, several issues were identified during the site visit related to collar locations and in particular, collar elevations. It was noticed in 2017 by Mexican Gold that database collar elevations for some drillholes did not coincide with available topographic mapping and a survey was competed at that time in an attempt to rectify the issues. It was clear at the time that the issues remained, and questions related to precision and accuracy persisted.

A subsequent review and inspection of the drillhole database also identified several significant issues. The company was informed of these issues and corrective plan was established which entailed performing a detailed ground survey to place known monuments and to tie in identifiable collars and landmarks. This was then followed by a LIDAR survey, which had been recommended in 2019 (Tietz 2019) in order to give accurate location and elevation tie-ins to survey and to assist in determining precise coordinates for the drillhole collars and therefore data locations in 3-dimensions. The drillhole collars were then subsequently corrected to precisely align with the LIDAR survey linked to the ground survey within the database.

12.2.2 Downhole Surveys

The QP conducted a review of the corrected drillhole collars and orientations and were compared against adjacent drillholes in addition to the lithology models and visual inspection of sections and plans. In the area of the Santa Cruz zone, there were discrepancies and conflicts with the location of intersections. It appears the 2011 drillhole series that were drilled through the Santa Cruz zone were in disagreement with the more recent drilling campaigns. Previous reports (Tietz 2019) and personal communications with on-site staff identified the use of the Maxibor downhole survey system as a source of uncertainty and potential error. The QP decided to exclude the drillholes from the 2011 drilling campaign that intersected the Santa Cruz zone from the resource estimation. Should these holes be adequately verified in the future; they may be included in the future.

12.2.3 Assay, Lithology, Geotechnical and Density Database

The QP utilized the data import facility within the MineSight^{TM} software which is used for the estimation process and is equipped with verification and validation features. The data was supplied in MS-EXCEL csv files for the assay, lithology, geotechnical and density data. The import of the data was validated and did not report any errors.

12.2.4 Geology Data

The geologic and domain models were derived from interpretations provided by Mexican Gold. Although there we no significant issues identified, these models were broad and required verification against the raw drillhole database. The QP performed extensive validation and adjustments to the mineralized domains using LeapFrog® which were then imported into MineSight^{TM} for final utilization for the resource estimation. These were then validated against the final block model results.

12.2.5 Assay Data

The 2019 Technical Report (Tietz 2019) stated “the full drill assay database, which included the drill data and all QA/QC analyses, was compared to the original SGS laboratory assay certificates. A total of thirty-four errors were noted in the gold, silver, and copper data for an error rate of about 0.16% of the total assay database. None of these errors are considered significant. Assays for two drillholes which were not in the database were added and there were also some rounding inconsistencies, which were corrected, but these would not have been considered material if left uncorrected. Less than detection values were changed in the database from a zero to one half of the detection limit. All corrections were made to the data before use in the resource estimate and the assay database is considered very clean for use in future resource studies.” (Tietz 2019).

However, the QP discovered a significant number of duplicate sample numbers and various discrepancies when comparing original assay certificates to the database. The company was notified, and it was recommended that a complete database audit be initiated. This audit was performed by independent professionals and supervised by the QP. All assays were validated against original assay certificates. The result showed that all the data validated, and discrepancies were explained and justified.

12.2.6 Magnetite

As the resource estimate that is the subject of this Technical Report also includes an estimate of the saleable iron component namely, magnetite, it is necessary to estimate the magnetite grade. The precise analysis of ferromagnetic compounds, such as magnetite, is extremely difficult to detect by chemical methods, as such, these grades were derived by measuring the total iron in addition to the magnetite fraction using the SATMAGAN process. SATMAGAN is designed to measure iron material concentrations and was performed on 421 samples which validated and verified the magnetite grades. The estimation process is discussed further in Section 14.12.

12.2.7 Statement

Mr. Kirkham is of the opinion that the geology, drilling and assaying data and results are valid and can be relied upon. Mr. Kirkham is also confident that the methods and procedures used are reliable. It is the opinion of Mr. Kirkham that all work, procedures, and results have adhered to best practices and industry standards as required by NI 43-101.

The QP is confident that the data and results are valid based on the site visit and inspection of all aspects of the project, including methods and procedures used. It is the opinion of the independent QP that all work, procedures, and results have adhered to best practices and industry standards required by NI 43-101. No duplicate samples were taken during January 2021 site visit to verify assay results as visual inspection of significant mineralization in core along with an extensive review of data, past results and certified assays satisfied the QP of the validity of the information. The QP was satisfied with the results from previous verification sampling performed on the project. In addition, there were no limitations with respect to access and response to requests. The author is of the opinion that the work was being performed that adheres to industry best practices and standards.

In conclusion, the data verification processes did not identify any material issues with the Las Minas sample/assay data or project. The author is satisfied that the assay data is of suitable quality to be used as the basis for this resource estimate.

12.3 Metallurgy

The results of the metallurgical testing were verified by reviewing the data for each of the flotation and magnetite concentration tests. The calculated head grades for each of the tests was compared to the expected value from initial head assays and to the calculated head assays for other tests on the respective composites. The assays were further verified by comparing the assays in the testwork reports to the assay certificates.

It is Mr. Crowie's opinion that the testwork conducted was performed at industry standards and the results are valid for predicting mill recoveries in this Preliminary Economic Assessment.

12.4 Mining

The mineral resource model and vein wire frames was imported into Maptek™ Vulcan™ (Vulcan) software and verified against mineral resources statements and vein volumes and tonnes and contained ounces matched original Kirkham models. In the QP's opinion, the mining data mentioned previously has been verified and is adequate for the PEA Technical Report as required by NI 43-101 guidelines.

12.5 Infrastructure

Infrastructure information was verified by the QPs physical inspection of access roads, towns and villages, powerlines, hydroelectric plant, port facilities, water sources, process plant locations, waste and water management locations.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

There have been two metallurgical testwork programs on the Las Minas deposit by Mexican Gold; the first was conducted in 2015 and the second was conducted in preparation for this report in 2021. Both programs were run at ALS Metallurgy Kamloops and were overseen internally by Robert Sloan, P.Eng. The 2021 metallurgical testwork was also partially overseen by the QP; the QP became involved approximately halfway through the 2021 testwork program.

The two programs are very similar in their approach to recovering valuable copper, gold and silver into a saleable copper concentrate. This approach is typical for chalcopyrite rich copper ores due to their strong response to this recovery methodology.

Magnetite was also recognized as potential revenue opportunity for the Las Minas deposit and magnetic separation testwork was conducted with a focus on recovering magnetite from the flotation tailings.

13.2 Copper Processing Testwork

Flotation was identified early on in the 2015 testwork program as the preferred recovery method for copper, gold, and silver due to the primary minerals present (chalcopyrite and bornite). The 2015 and 2021 programs are discussed in Sections 13.2.1 and 13.2.2.

13.2.1 2015 Metallurgical Testwork

The 2015 testwork was conducted on a single 59 kg composite of the El Dorado zone designated the ED Composite. The composite was blended and then split into subsamples to allow the following testwork:

- Chemical analysis;
- Comminution testing;
- QEMSCAN Trace Mineral Search (3 size fractions);
- Flotation testing; and
- Magnetite testwork.

A characteristic of the Las Minas mineralized material is that it has a relatively low sulphur concentration suggesting that the pyrite content is relatively small. The chemical composition of the ED sample can be seen in Table 13-1.

Table 13-1: ED Composite Chemical Composition

Sample	Assay							
	Cu (%)	CuOx (%)	CuCN (%)	CuRes (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)
ED Composite	2.16	0.099	0.16	1.90	31.5	2.81	7	1.32

Notes:
1. CuOx – copper oxide, weak acid digestion, CuCN – Cyanide soluble copper, CuRes – Calculated value of Cu - CuOx - CuCN

Source: ALS Metallurgy Kamloops (2015)

The comminution testing included Bond ball mill work index (Wibm) and Abrasion Index (Ai). The values for each of these parameters were found to be 13.3 kWh/t and 0.14 for Wibm and Ai respectively.

QEMSCAN identified that the gold in the ED Composite was mostly associated with Chalcopyrite. The mineralogy conducted on this sample did not recognize altered mineralization such as bornite, although when a mineral balance is estimated using chemical content and expected mineral composition, there must be secondary copper minerals such as bornite in the sample. This will be further discussed in Section 13.2.2.

The flotation testwork consisted of 6 rougher flotation tests, 6 cleaner flotation tests and a single Locked Cycle Test (LCT). The rougher flotation testing consisted mostly of adjusting reagent dosage with a particle size, P_{80} of 150 µm., with one test with a coarser grind size (P_{80} of 190 µm) and one test at natural pH. The cleaner flotation testing was mostly optimizing the target grind size of the rougher concentrate.

An LCT simulates a plant flotation circuit, with the cleaner circuit tailings products recirculated as would be found in a typical processing plant. An LCT is conducted in “cycles” which allows the intermediate products from the first cycle to be added to the second cycle. The LCT was performed using the conditions found in the preceding rougher and cleaner tests as shown in Table 13-2.

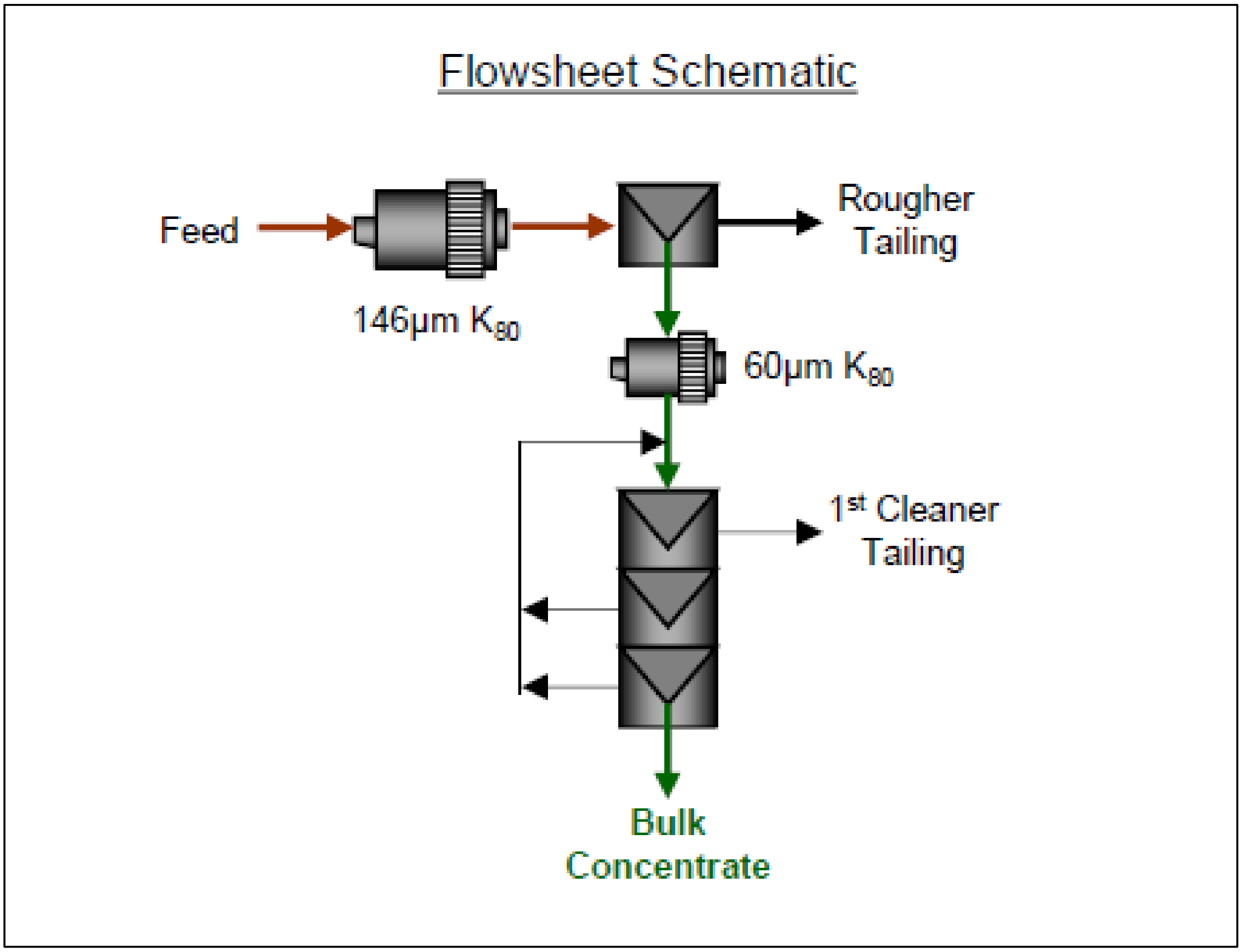
Table 13-2: Locked Cycle Test Conditions

Stage	pH	Redox mV	Reagent Addition (g/t)	
			Lime	PAX
Primary Grind	9.1	107	150	-
Bulk Rougher	8.7 - 9.1	+38 to +101	-	80
Regrind	9.1	125	50	-
Bulk Cleaner	8.6 - 9.1	+96 to 126	-	14

Source: ALS Metallurgy Kamloops (2015)

The flowsheet used for the LCT can be seen in Figure 13-1. This flowsheet recycles the second and third cleaner tailings back to the first cleaner feed. In the flowsheet developed during the rougher and cleaner flotation tests, the second and third cleaner tailings are recycled back to the first cleaner feed.

Figure 13-1: LCT Flowsheet



Source: ALS Metallurgy Kamloops (2015)

The results for cycle 4 and 5 are averaged in Table 13-3 and show that a concentrate grade of 22.3% copper and 13.4 g/t gold was produced. This would be marketed as a copper concentrate with high gold values (smelters will usually become interested in the gold in a copper concentrate when it is above 1 g/t).

Table 13-3: Locked Cycle Test Results

Product	Weight (%)	Assay					Distribution (%)				
		Cu (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Cu	Fe	S	Ag	Au
CYCLES IV and V											
Flotation Feed	100	2.43	31.6	3.16	7	1.59	100	100	100	100	100
Bulk Con	**10.4**	**22.3**	**30.4**	**28.7**	**57**	**13.4**	**95.4**	**10.0**	**94.9**	**84.2**	**88.8**
Bulk 1st Clnr Tail	6.7	0.86	25.4	1.23	10	1.16	2.4	5.4	2.6	9.9	4.9
Bulk Ro Tail	82.9	0.06	32.2	0.09	1	0.14	2.2	84.6	2.5	5.9	7.1

Source: ALS Metallurgy Kamloops (2015)

13.2.2 2021 Metallurgical Testwork

The most recent metallurgical testwork program was initiated in June of 2020 and was completed in the spring of 2021. This program consisted of 3 composite samples which had low, medium and high grades, which were then used to build a master composite that is meant to reflect the average grades expected in the overall deposit. Composite 1 was produced of lower grade samples throughout the claim, Composite 2 is from the Santa Cruz deposit, and Composite 3 is high grade from the El Dorado zone.

Table 13-4: Summary of 2020 Head Assay Results

Sample	Elements for Assay						
	Cu (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	CuOx (%)	CuCN (%)
Comp 1 Head 1	0.86	26.4	0.85	4	0.89	-	-
Comp 1 Head 2	0.93	27.3	0.89	5	0.99	-	-
Average	**0.90**	**26.9**	**0.87**	**5**	**0.94**	-	-
Comp 2 Head 1	1.62	22.1	1.27	16	2.80	0.24	0.80
Comp 2 Head 2	1.51	21.6	1.21	15	1.93	-	-
Average	**1.56**	**21.9**	**1.24**	**16**	**2.36**	-	-
Comp 3 Head 1	3.01	35.8	2.93	11	4.97	0.21	0.71
Comp 3 Head 2	3.21	37.3	3.05	10	4.46	-	-
Average	**3.11**	**36.6**	**2.99**	**11**	**4.72**	-	-
Master Comp Head 1	1.20	28.0	1.50	5	1.50	0.13	0.24
Master Comp Head 2	1.27	29.0	1.53	8	1.61	0.12	0.22
Average	**1.24**	**28.5**	**1.52**	**7**	**1.56**	0.13	0.23

Source: ALS Metallurgy Kamloops (2021)

The head assays for the 2021 program showed higher oxide and cyanide soluble copper ratios than the 2015 program indicating that there is more alteration in these samples. Oxide copper is copper that is found in minerals that do not contain sulphur and is more amenable to leaching with acid than recovery by flotation. Cyanide soluble copper assays indicate copper minerals that have been altered, such as bornite and chalcocite.

The 2021 program followed a similar approach to the testwork program conducted in 2015, with an emphasis on proving a flotation flowsheet that would produce a saleable copper concentrate.

Comminution data for the 2021 testwork was limited to Wibm tests on composites 1 and 3. This program did not conduct any abrasion index testing since it had been included in the 2015 testwork program. The Wibm results were 15.4 and 15.1 for composites 1 and 3 respectively indicating that the mineralized material was harder than the samples tested in 2015, which had a Wibm of 13.3.

A single gravity test was conducted on Composite 2 as part of a rougher flotation test. The test included a single pass through a laboratory centrifugal concentrator. The concentrate from the centrifugal concentrator was panned to produce a pan concentrate of approximately 20% of the gold. The tailings from the panning and centrifugal concentrator were combined for the flotation portion of the test.

The flotation testwork program in 2021 included a total 13 rougher tests and 15 cleaner tests and were conducted on Composites 2, 3 and the Master Composite. A LCT was completed for each of the 3 composites as well.

For Composites 2 and 3, the flotation testwork program focused primarily on an optimized reagent scheme for the composites. The samples responded mostly as expected, but with slightly lower recoveries, likely due to the increased level of oxide copper mineralization.

The results for Composite 2, which can be seen in Table 13-5, show that 86% of the copper and 86% of the gold was recovered to a concentrate grading 27% copper and 49.5 g/t gold. The lower recovery was expected with Composite 2 since the ratio of oxide copper assay to total copper assay was significantly higher for Composite 2 than the ED Composite tested in 2015.

Table 13-5: Composite 2 Locked Cycle Test Results

KM6130-15 Composite 2
CYCLES (IV+V) MASS BALANCE FLOWSHEET AND METALLURGICAL BALANCE DATA

Flotation Stream	Product	Weight	Assay					Distribution (%)				
No.		%	Cu (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Cu	Fe	S	Ag	Au
1	Copper Ro Feed	100.0	1.76	22.1	1.26	19	3.20	100.0	100.0	100.0	100.0	100.0
2	Copper Ro Tail	90.6	0.20	22.7	0.17	2	0.37	10.1	92.8	12.2	9.8	10.6
3	Copper Ro Con	9.4	16.7	16.8	11.8	177	30.3	89.9	7.2	87.8	90.2	89.4
4	Copper 1st Clnr Feed	10.2	15.8	16.7	11.2	170	28.7	92.1	7.7	90.2	93.6	91.4

Flotation Stream	Product	Weight	Assay					Distribution (%)				
No.		%	Cu (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Cu	Fe	S	Ag	Au
5	Copper 1st Clnr Tail	4.7	2.22	14.3	1.71	37	4.02	5.9	3.0	6.3	9.3	5.9
6	**Copper 1st Clnr Con**	**5.5**	**27.3**	**18.7**	**19.1**	**282**	**49.5**	**86.2**	**4.7**	**83.9**	**84.3**	**85.6**
7	Copper Clnr Scav Tail	3.9	1.69	14.2	1.28	28	3.14	3.8	2.5	3.9	5.9	3.8
8	Copper Clnr Scav Con	0.8	4.86	14.7	3.89	80	8.40	2.2	0.5	2.4	3.4	2.0
9	Final Tail	94.5	0.26	22.3	0.22	3	0.49	13.8	95.3	16.1	15.7	14.4

Source: ALS Metallurgy Kamloops (2021)

The results for Composite 3 can be found in Table 13-6. The higher feed grades and lower oxide ratio resulted in Composite 3 having a higher overall recovery, similar to the results in 2015.

Table 13-6: Composite 3 Locked Cycle Test Results

KM6130-13 Composite 3
CYCLES (IV+V) MASS BALANCE FLOWSHEET AND METALLURGICAL BALANCE DATA

Flotation Stream	Product	Weight	Assay					Distribution (%)				
No.		%	Cu (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Cu	Fe	S	Ag	Au
1	Copper Ro Feed	100.0	3.34	40.4	2.98	17	6.85	100.0	100.0	100.0	100.0	100.0
2	Copper Ro Tail	83.7	0.16	42.7	0.16	1	0.65	4.1	88.6	4.5	4.8	7.9
3	Copper Ro Con	16.3	19.7	28.3	17.5	102	38.8	95.9	11.4	95.5	95.2	92.1
4	Copper 1st Clnr Feed	17.2	19.0	28.3	16.9	99	38.9	97.7	12.1	97.3	98.1	97.7
5	Copper 1st Clnr Tail	4.0	3.43	27.6	3.06	30	13.7	4.1	2.7	4.1	6.8	8.0
6	**Copper 1st Clnr Con**	**13.2**	**23.7**	**28.6**	**21.0**	**120**	**46.5**	**93.7**	**9.4**	**93.2**	**91.4**	**90**
7	Copper Clnr Scav Tail	3.0	2.43	27.3	2.25	22	5.30	2.2	2.0	2.3	3.8	2.3
8	Copper Clnr Scav Con	1.0	6.60	28.4	5.65	54	40.4	1.9	0.7	1.8	3.0	5.6
9	Final Tail	86.8	0.24	42.2	0.23	2	0.81	6.3	90.6	6.8	8.6	10.3

Source: ALS Metallurgy Kamloops (2021)

The Master composite is a blend of composites 1, 2, and 3 with the aim of achieving an approximation of the Las Minas deposit average grade. This composite responded similarly to the previous samples with a little lower recovery, although the froth in the cleaner circuit became overloaded (too heavy). The overloaded froth caused recovery issues to the final copper concentrate.

Initially, stronger frothers and lower collector dosage were used to improve the cleaner circuit results, but this was only marginally successful. The lower collector dosage negatively affected overall recovery and the higher molecular weight frothers were not successful to get the froth to move. The heavy froth was determined to be caused by a high proportion of fast floating copper minerals in the sample. The solution to the heavy froth was to bypass the first rougher concentrate to the final concentrate and only clean the subsequent rougher concentrates.

The first rougher was found to be producing a saleable concentrate and contained approximately 60% of the copper in the feed and 55% of the gold. When the first rougher was bypassed to the final concentrate, the cleaner circuit froth became significantly more stable. The results for the Master Composite LCT can be seen in Table 13-7.

Table 13-7: Master Composite Locked Cycle Test Results

KM6130-37 Master Composite 1
CYCLES (IV+V) MASS BALANCE FLOWSHEET AND METALLURGICAL BALANCE DATA

Flotation Stream	Product	Weight	Assay					Distribution (%)				
No.		%	Cu (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Cu	Fe	S	Ag	Au
1	Copper Ro Feed	100.0	1.34	30.0	1.49	8	1.76	100	100	100	100	100
2	Copper Ro Con 1	3.6	23.8	28.0	24.0	112	27.0	63.9	3.4	57.8	51.2	55.0
3	Copper Ro 2-6 Feed	96.4	0.50	30.0	0.65	4	0.82	36.1	96.6	42.2	48.8	45
4	Copper Ro Tail	85.2	0.07	30.8	0.10	1	0.22	4.3	87.7	5.8	16.2	10.7
5	Copper Ro 2-6 Con	11.2	3.80	23.9	4.85	23	5.38	31.8	8.95	36.5	32.6	34.3
6	Copper 1st Clnr Feed	15.2	3.42	22.7	4.42	23	4.88	38.7	11.5	45.0	44.4	42.1
7	Copper 1st Clnr Tail	9.3	0.93	22.8	1.28	13	1.79	6.5	7.08	7.98	14.8	9.45
8	Copper 1st Clnr Con	5.8	7.36	22.6	9.43	40	9.82	32.2	4.41	37	29.5	32.6
9	Copper 2nd Clnr Con	2.8	13.2	26.2	16.7	60	17.3	27.7	2.46	31.4	21.4	27.6
10	Copper 2nd Clnr Tail	3.0	1.98	19.3	2.73	21	2.93	4.5	2.0	5.6	8.1	5.1
11	Copper 3rd Clnr Tail	0.9	3.51	19.6	4.87	32	5.33	2.4	0.6	2.9	3.7	2.7
12	Copper 3rd Clnr Con	1.9	17.7	29.2	22.2	73	22.9	25.3	1.9	28.5	17.7	24.9
13	Final Tail	94.5	0.15	30.1	0.22	3	0.37	10.8	94.8	13.8	31.1	20.1
14	**Overall Copper Con**	**5.5**	**21.7**	**28.4**	**23.4**	**98**	**25.5**	**89.2**	**5.2**	**86.2**	**68.9**	**79.9**

Source: ALS Metallurgy Kamloops (2021)

Although this program did not include mineralogy, it was noted that the low sulphur grades in the final concentrate suggests that there is a significant amount of gangue being recovered to the final concentrate. Based on the low sulphur, assay, it is estimated that 30% to 40% of the material in the final concentrate is non-sulphide gangue, which is a potential source of improvement to the final copper concentrate grade.

13.3 Magnetite Testwork

Magnetite testwork was conducted in both the 2015 and the 2021 metallurgical testwork programs demonstrating that there may be an opportunity to produce a magnetite concentrate from the flotation tailings.

In 2015, Davis Tube, a preliminary bench scale rougher magnetic separation test, was conducted on a sample of flotation feed and a sample of flotation tailings. The flotation feed sample had recovery of 60% of the iron into a concentrate grading 68% iron, which should be saleable to the iron material market. The Davis Tube testing on the flotation tailings identified that 65% of the iron could be concentrated into a concentrate grading 67% iron at a magnetic field strength of 4,000 gauss.

In the 2021 testwork, the focus was on flotation tailings. Davis Tube testing was again used to characterize the magnetic recovery of the sample, showing that the flotation tails had a high recovery of iron to a magnetic concentrate. The recovery determined in the Davis Tube testing was 80% of the iron, into a concentrate grade of 62.7% Fe. At this stage of the testwork, the magnetic iron was not determined.

In addition to the above 2021 testwork a larger sample made up of a composite of rougher tailings tests was introduced into the Low Intensity Magnetic Separator (LIMS) at ALS to determine the larger scale separation characteristics.

The LIMS rougher concentrate recovered 80% of the iron in the sample to a concentrate grading 59% Fe using a magnetic field strength of 1100 gauss. A sub sample of the rougher concentrate was screened, and the size fractions were assayed, Table 13-8. The results indicated that a grind size of 36µm was optimal for producing a high-grade magnetic concentrate.

Table 13-8: Magnetite Size Fractions – 2021 Testwork

Size Fraction	Wt	Metal Assays	Distribution
	%	Fe-%	%
>150µm	26.7	53.4	24.7
<150>106µm	19	50.8	16.8
<106>75µm	15.8	61.2	16.8
<75>36µm	17.2	63.8	19.1
<36µm	21.3	61.2	22.7
Total	100	57.6	100

Source: ALS (2021)

A 5 kg sub-sample of the rougher concentrate was reground in a laboratory rod mill to a P_{80} of 23 µm; finer than the target of 38µm (this is a typical issue in laboratory testwork programs due to the low mass of material to perform grind calibrations). The particle size of the ground rougher concentrate was measured using a cyclosizer prior to the cleaner magnetic separation test.

The cleaner magnetic separation test was also performed on the LIMS, with a magnetic field strength of 900 gauss to produce a final magnetic concentrate. The results from this testwork are summarized in Table 13-8.

For most of the testwork the assays conducted were for total iron, rather than magnetic iron. SATMAGAN testing, which provides a magnetic iron determination, was conducted on the LIMS feed material to differentiate iron oxide and sulphide minerals (pyrite, chalcopyrite and bornite, etc.) from magnetite. This testwork indicated that the feed to the magnetic pilot plant was 32% magnetite.

In Table 13-9 the magnetic iron and non-magnetic iron are differentiated, to illustrate the relationship between the forms of iron in the LIMS test since the block model includes magnetite (Fe3O4).

The table calculates the magnetite recovery using the magnetite feed grade determined by the SATMAGAN test and an assumption that the final concentrate grade is 90% magnetite, based on the SG of the concentrate being 4.95 and the SG of magnetite 5.16. Although SATMAGAN testing would have provided a better determination, it was not conducted at the time of the testwork. The overall magnetite recovery is calculated from the mass of concentrate in the cleaner circuit (in %) and the magnetite estimation detailed above. It should be noted that the magnetite rougher and cleaner circuit recoveries are estimates based on the calculated total recovery.

Table 13-9: Magnetite Recovery – 2021 Testwork

KM6130-32 Cu Ro Tl Composite
Overall Metallurgical Balance

Product	Weight		Assay (%)				Distribution (%)		
	%	grams	Fe	Fe_{mag}	$Fe_{non\text{-}mag}$	Fe_3O_4	Fe	Fe_{mag}	Fe_3O_4
Cleaner Magnetic Con	34.0	4468.0	70.0	65.1	4.9	90.0	77.5	95.50	95.50
Cleaner Magnetic Tails	7.8	1030.6	10.2	4.4	5.8	6.1	2.6	1.50	1.50
Rougher Magnetic Con	41.8	5498.6	58.8	53.8	5.0	74.3	80.1	97.00	97.00
Magnetic Tail	58.2	7660.0	10.5	1.2	9.3	1.6	19.9	3.00	3.00
Feed	100.0	13158.6	30.7	23.2	7.5	32.0	100	100	100

Assumptions:

1) Magnetite concentrate is 90% magnetite; and 2) The rougher recovery of magnetite is 97%.

The magnetite recovery shown in Table 13-8 is based on magnetite determinations from the flotation rougher tailings. Since SATMAGAN testing was not conducted on the composite feed, an estimation of the magnetite content in the final flotation concentrate was estimated at 5.5%, which is the mass pull of the flotation concentrate. This value would be considered the highest likely loss to the flotation concentrate since magnetite in the flotation concentrate would be due to carryover with other non-sulphide gangue. This results in a total recovery of magnetite to the magnetic concentrate of 90%.

13.4 Metallurgical Assumptions

The metal recoveries and concentrate grades assumed for the PEA can be found in Table 13-10. The copper, gold and silver recoveries are taken directly from the LCT.

Table 13-10: Recovery and Concentrate Grade Estimates

Parameter	Unit	Value
Process Recovery		
Cu Recovery	%	90
Au Recovery	%	80
Ag Recovery	%	70
Magnetite Recovery	%	90.3
Concentrate Grade		
Cu	%	22
Au	g/t	25
Ag	g/t	95
Magnetite	%	90

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

This section describes the work undertaken by Kirkham Geosystems, including key assumptions and parameters used to prepare the mineral resource models for El Dorado and Santa Cruz Zones, together with appropriate commentary regarding the merits and possible limitations of such assumptions.

In the opinion of the QP, the mineral resource estimates reported herein are a reasonable representation of the mineral resources found within the Project at the current level of sampling. The mineral resources were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy **(CIM)** "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (CIM, Nov. 2019) and are reported in accordance with the Canadian Securities Administrators' NI 43-101 and are classified in accordance with the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves. It is important to note that mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mine-ability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated.

The mineral resource evaluation reported herein for the Las Minas Deposit is current and supersedes earlier mineral resource estimates completed on the property including:

- NI 43-101 Technical Report Mineral Resource Estimate Las Minas Gold-Copper Project (Read, Shoemaker 2017); and
- Technical Report and Estimated Resources for the Las Minas Project, Veracruz, Mexico (Tietz 2019).

Mexican Gold field work on the Project from 2010 to 2021, including drilling, was carried out under the supervision of Sonny Bernales, P.Geo. who is a senior geologist for Mexican Gold.

The mineral resource estimation methodology involved the following procedures:

- Generation of updated geological models and review of structural controls on mineralization;
- Database verification and validation;
- Exploration data analysis, compositing and evaluation of outliers;
- Construction of estimation domains for gold, copper and silver;
- Spatial statistics and geostatistical analysis;

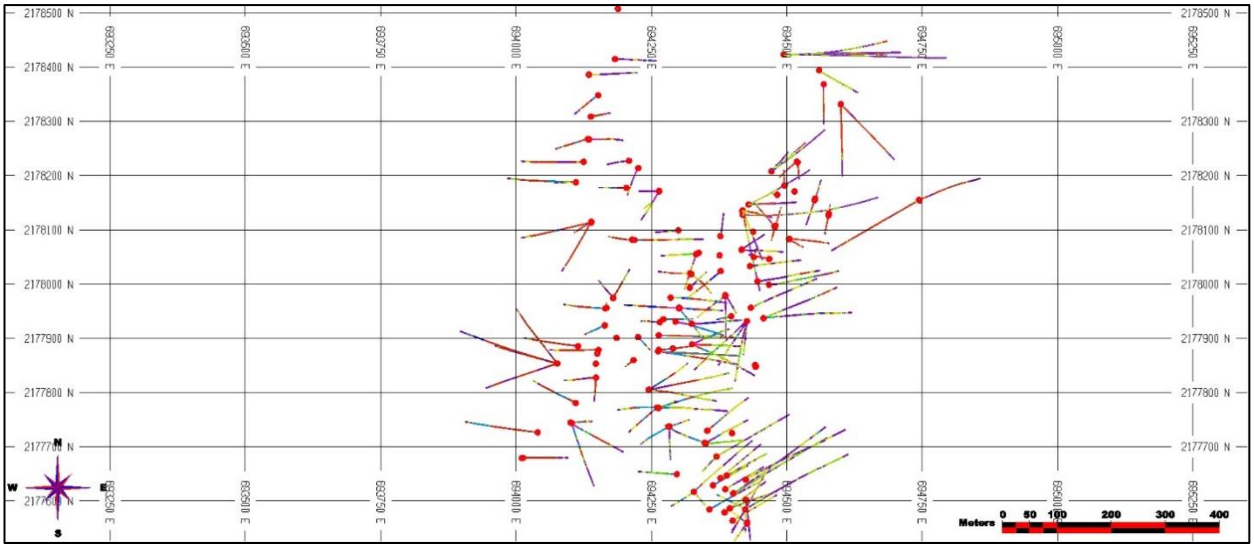
- Block modeling and grade interpolation;
- Mineral resource classification and validation;
- Assessment of “reasonable prospects for eventual economic extraction”; and
- Preparation of the mineral resource statement.

Within the Las Minas Project, 206 drillholes (32,058 m) supports the mineral resource estimate. The deposit was segregated into multiple estimation domains based on geologic models for each of the mineralized units. The estimated mineral resources occur within the Las Minas gold-copper-silver-magnetite skarn deposit, which consists of the mineralized endo-skarn and exoskarn units within the El Dorado and Santa Cruz zones. The mineral domains were then used to code the block model, and assays within the modeled domains were evaluated geostatistically to establish estimation parameters. Assays were composited into 2 m lengths. MineSight^{TM}, a commercially available geologic modeling and mine planning software package, was used to produce a three-dimensional block model while LeapFrog® Software was utilized to produce the solids models for the estimation domains.

14.2 Drillhole Data

The 206 drillholes in the database were supplied in electronic format by Mexican Gold. This included collars, downhole surveys, lithology data and assay data (i.e., Au g/t, Ag g/t, Cu%, Total Fe%). Validation and verification checks were performed during importation of data to ensure there were no overlapping intervals, typographic errors or anomalous entries. Minor discrepancies were corrected. It should be noted that based on issues identified with respect to the downhole surveys during the 2011 campaign, 16 of the drillholes that intersected the Santa Cruz zone have been excluded. These issues were confirmed via visual inspection and modelling. It also must be noted that if these holes can be sufficiently validated and verified, they may be brought back in to inform future work. Figure 14-1 shows a plan view of the supplied drillholes.

Figure 14-1: Plan View of Las Minas Drillholes



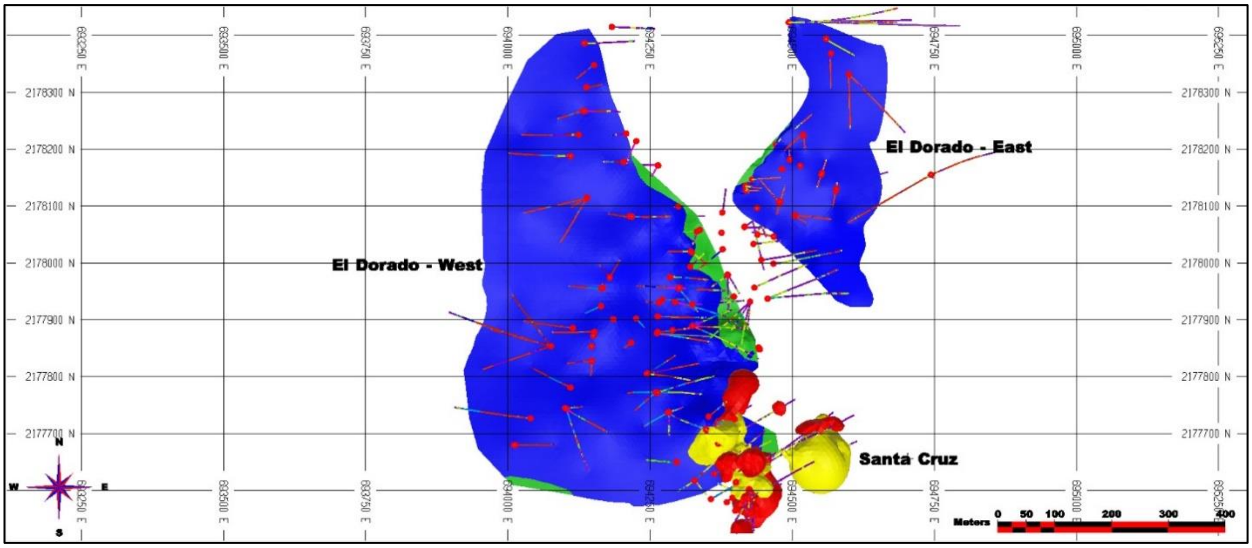
Source: Kirkham (2021)

14.3 Geology Model

Solid models of the mineralized zones (Figure 14-2 and Figure 14-3) were created from sections and based on a combination of lithology and site knowledge.

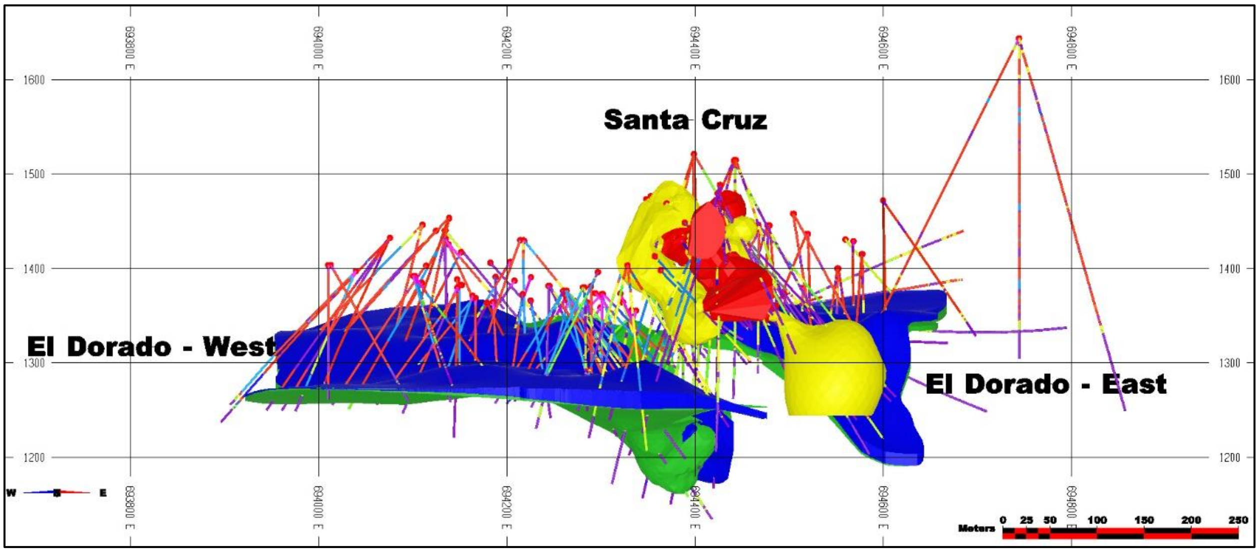
Every intersection was inspected and the solid was then manually adjusted to match the drill intercepts. Once the solid model was created, it was used to code the drillhole assays and composites for subsequent statistical and geostatistical analysis. The solid zone was used to constrain the block model by matching assays to those within the zones. The orientation and ranges (distances) used for search ellipsoids in the estimation process were derived from strike and dip of the mineralized zone, site knowledge and on-site observations by Mexican Gold geological staff.

Figure 14-2: Plan View of Las Minas Mineralized Zones and Drillholes



Source: Kirkham (2021)

Figure 14-3: Section View of Las Minas Mineralized Zones and Drillholes Looking 325 Degrees Azimuth



Source: Kirkham (2021)

14.4 Data Analysis

The drillhole database was numerically coded by solids for the El Dorado and Santa Cruz endo-skarn and exoskarn mineralized zones. The database was then manually adjusted, drillhole by drillhole, to ensure accuracy of zonal intercepts. Table 14-1 shows the drillhole statistics for the gold, silver, copper and iron assays.

Table 14-1: Statistics for Gold, Silver, Copper and Iron by Zone

Metal	Mineralized Unit	#	Length (m)	Maximum	Mean	SD	CV
Au g/t	SC Endoskarn	452	760.7	9.61	0.34	0.79	2.3
	SC Exoskarn	596	1,008.3	21.30	0.89	1.80	2.0
	ED East Endoskarn	181	268.6	2.54	0.17	0.30	1.8
	ED West Endoskarn	446	864.2	14.72	0.36	1.05	3.0
	ED East Exoskarn	353	512.8	27.05	0.76	1.98	2.6
	ED West Exoskarn	881	1,643.0	47.40	1.38	2.75	2.0
	Total	**2,909**	**5,057.6**	**47.40**	**0.83**	**2.00**	**2.4**
	All	**8,980**	**14,683.7**	**47.40**	**0.43**	**1.44**	**3.4**
Ag g/t	**SC Endoskarn**	452	760.7	59.10	2.07	5.72	2.8
	SC Exoskarn	596	1,008.3	126.00	4.52	10.54	2.3
	ED East Endoskarn	181	268.6	8.00	0.83	1.37	1.7
	ED West Endoskarn	446	864.2	29.00	1.21	2.90	2.4
	ED East Exoskarn	353	512.8	127.00	3.13	6.44	2.1
	ED West Exoskarn	881	1,643.0	50.20	4.36	6.12	1.4
	Total	**2,909**	**5,057.6**	**127.00**	**3.20**	**6.85**	**2.1**
	All	**8,980**	**14,683.7**	**127.00**	**1.66**	**4.64**	**2.8**
Cu %	SC Endoskarn	452	760.7	6.05	0.20	0.53	2.7
	SC Exoskarn	596	1,008.3	11.70	0.56	1.19	2.1
	ED East Endoskarn	181	268.6	1.79	0.13	0.29	2.2
	ED West Endoskarn	446	864.2	2.98	0.21	0.41	1.9
	ED East Exoskarn	353	512.8	5.37	0.57	0.66	1.2
	ED West Exoskarn	881	1,643.0	9.28	0.84	1.08	1.3
	Total	**2,909**	**5,057.6**	**11.70**	**0.51**	**0.92**	**1.8**
	All	**8,980**	**14,683.7**	**17.80**	**0.25**	**0.67**	**2.7**
Fe %	SC Endoskarn	452	760.7	45.30	5.71	6.73	1.2
	SC Exoskarn	596	1,008.3	53.92	11.81	11.66	1.0
	ED East Endoskarn	181	268.6	25.40	6.73	5.08	0.8
	ED West Endoskarn	446	864.2	57.50	9.52	8.23	0.9

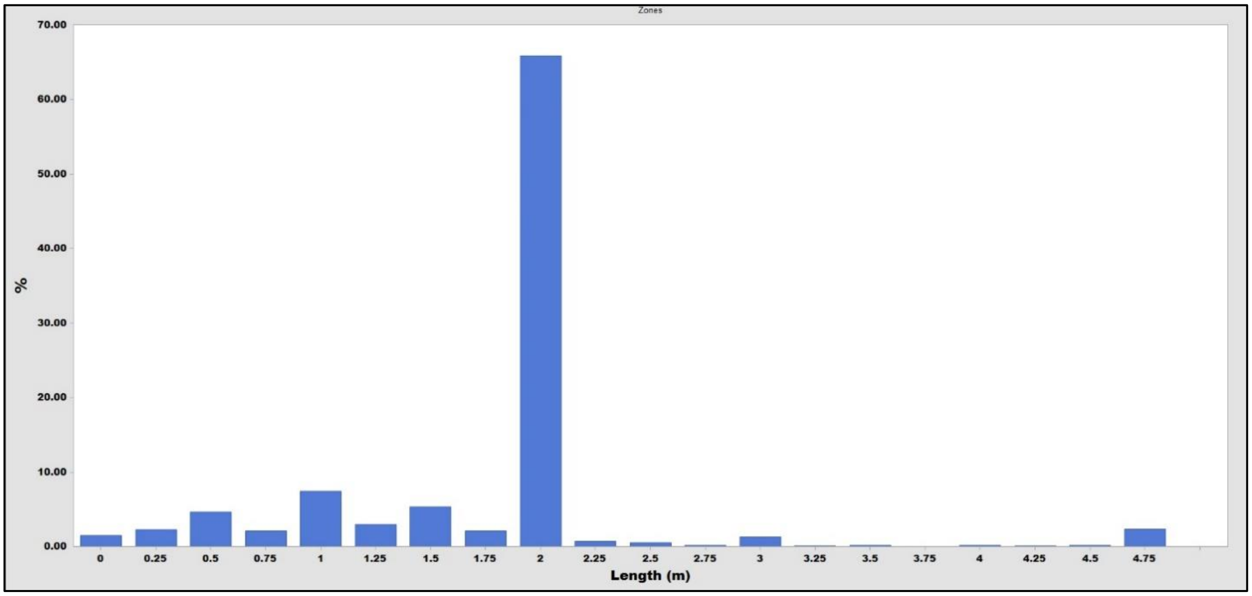
Metal	Mineralized Unit	#	Length (m)	Maximum	Mean	SD	CV
	ED East Exoskarn	353	512.8	58.44	16.45	11.97	0.7
	ED West Exoskarn	881	1,643.0	61.38	22.33	15.52	0.7
	Total	**2,909**	**5,057.6**	**61.38**	**14.12**	**13.43**	**1.0**
	All	**8,980**	**14,683.7**	**61.38**	**8.13**	**10.09**	**1.2**

Source: Kirkham (2021)

14.5 Composites

It was determined that a 2.0 m composite length offered the best balance between supplying common support for samples and minimizing the smoothing of the grades. The 2.0 m sample length also was consistent with the distribution of sample lengths within the mineralized domains as shown in the histogram of assay lengths in Figure 14-4.

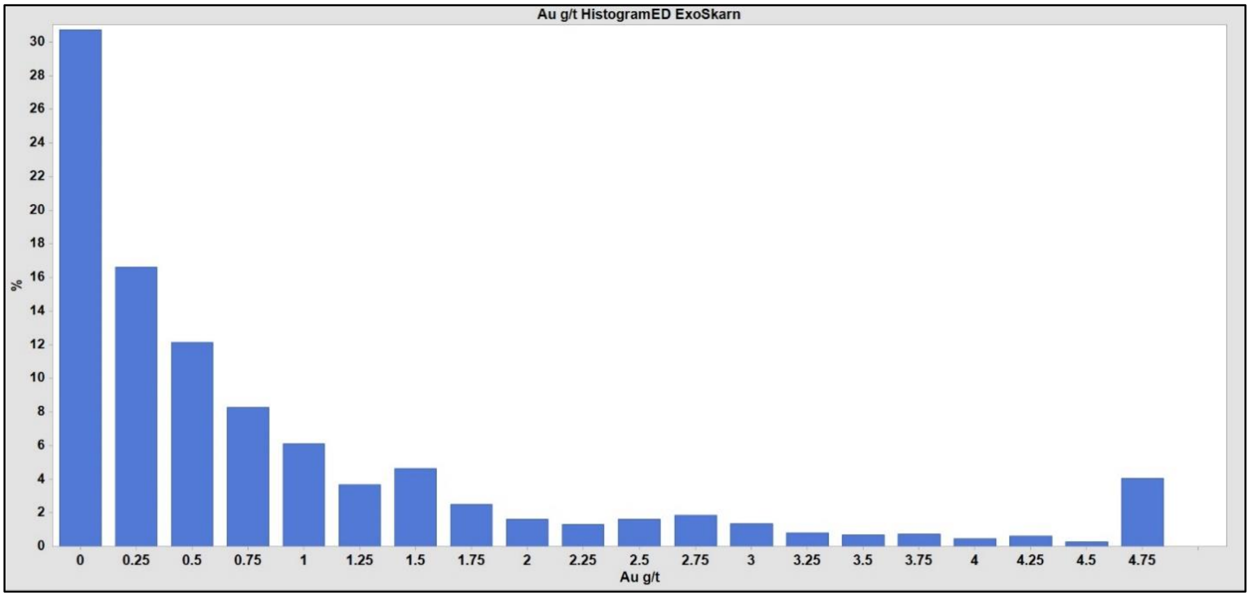
Figure 14-4: Assay Interval Lengths

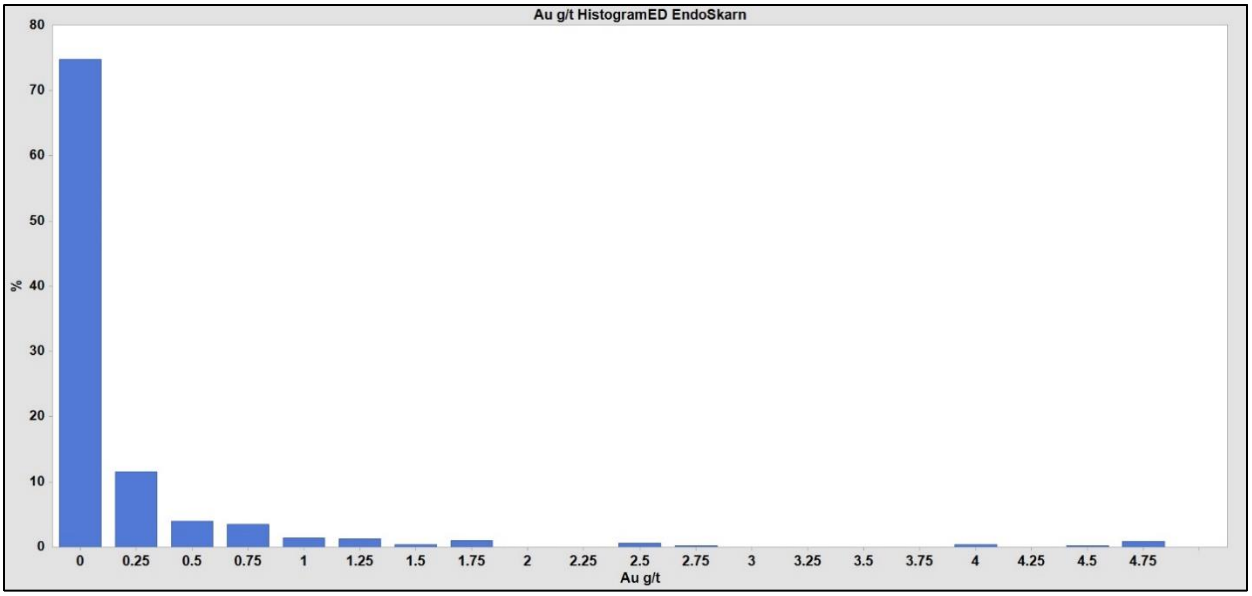


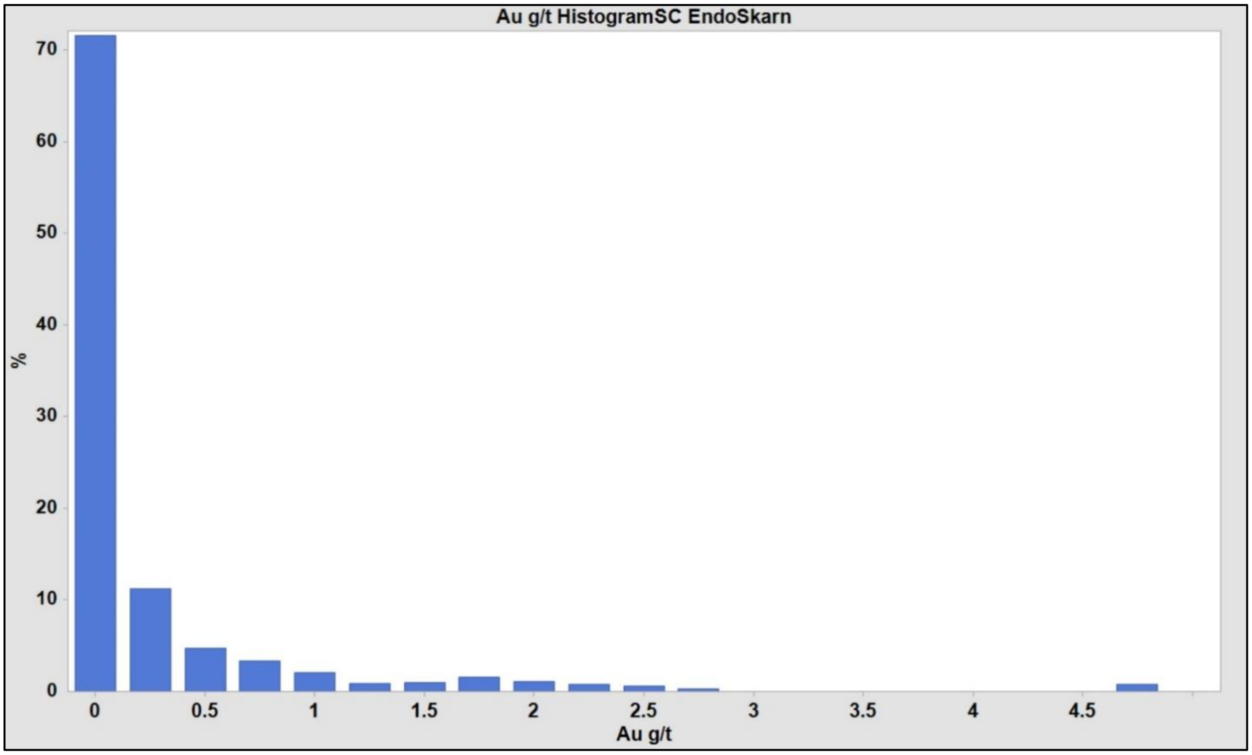
Source: Kirkham (2021)

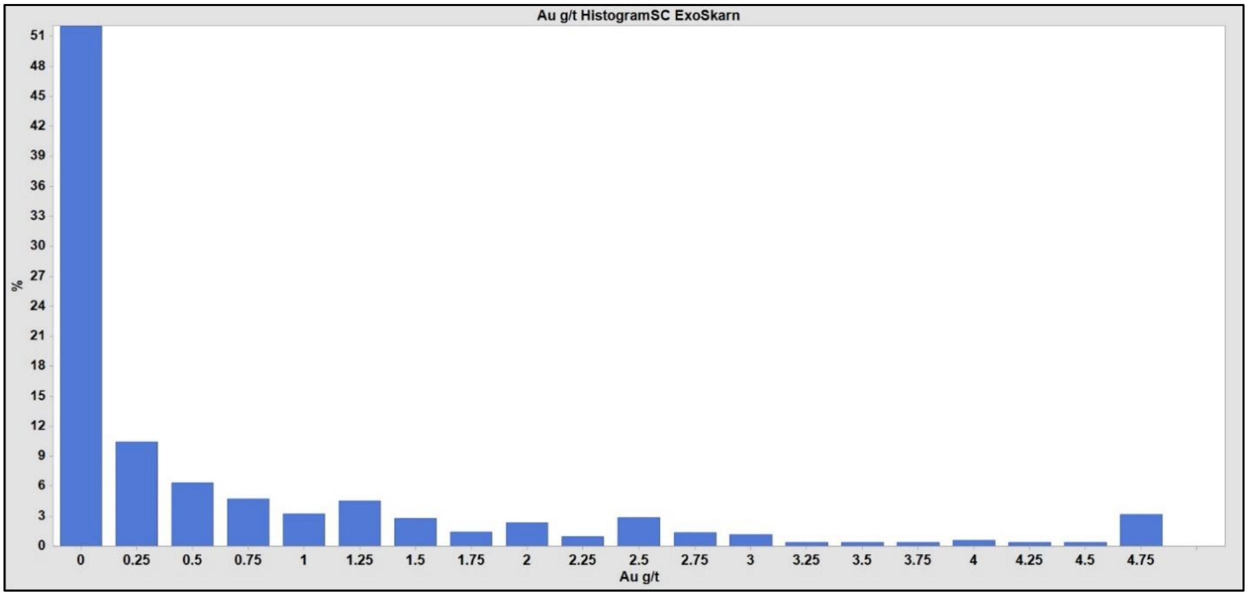
Figure 14-5 through Figure 14-8 show the histograms for gold, silver, copper and iron assays, respectively, within the mineralized solids for all zones which demonstrate well-formed log-normal distribution for all metals.

Figure 14-5: Histograms of Gold Composite Grades within the El Dorado (ED) and Santa Cruz (SC) Endo- and Exo-skarn Zones



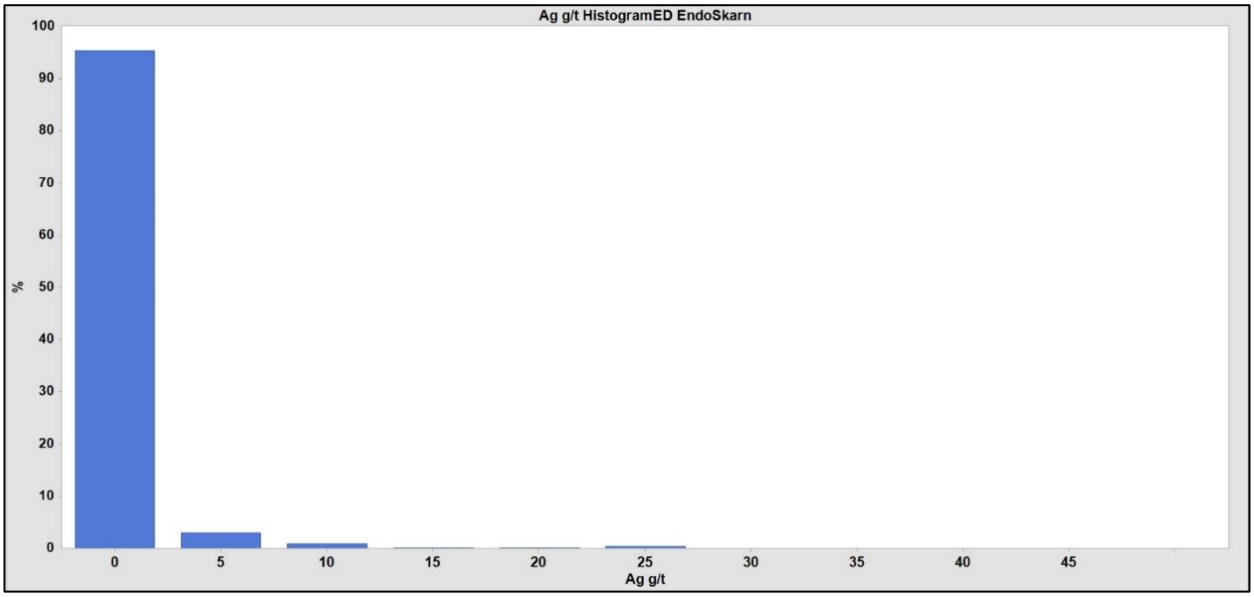


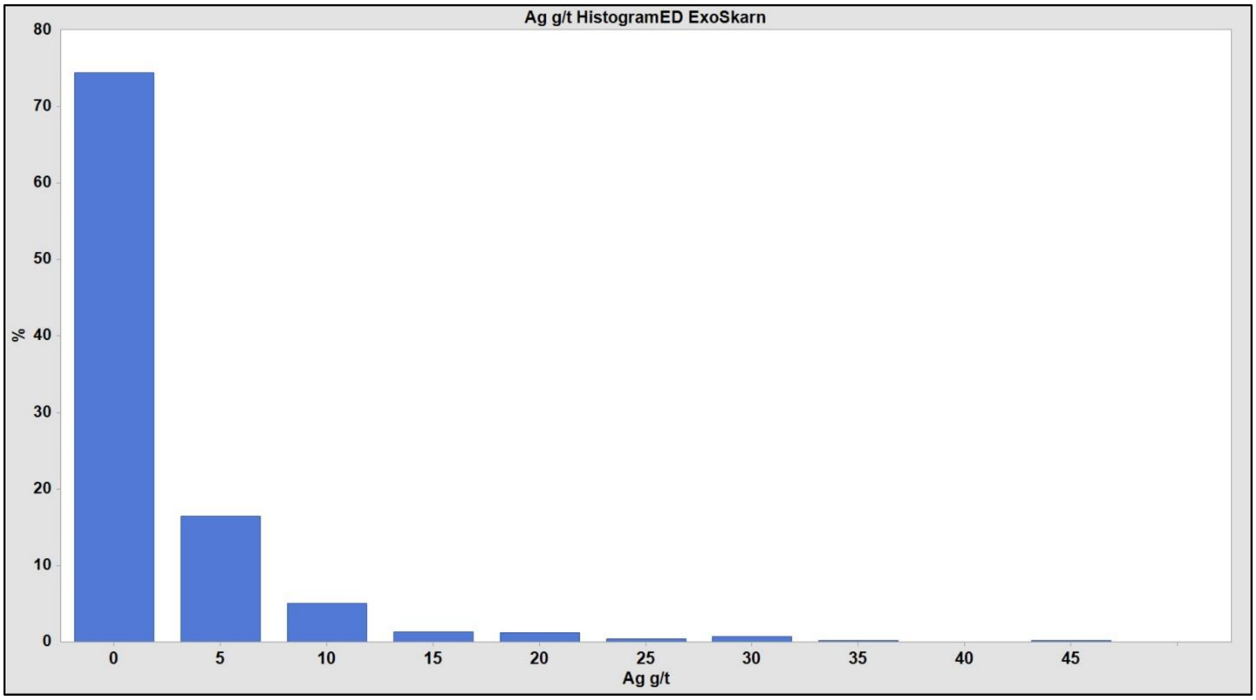


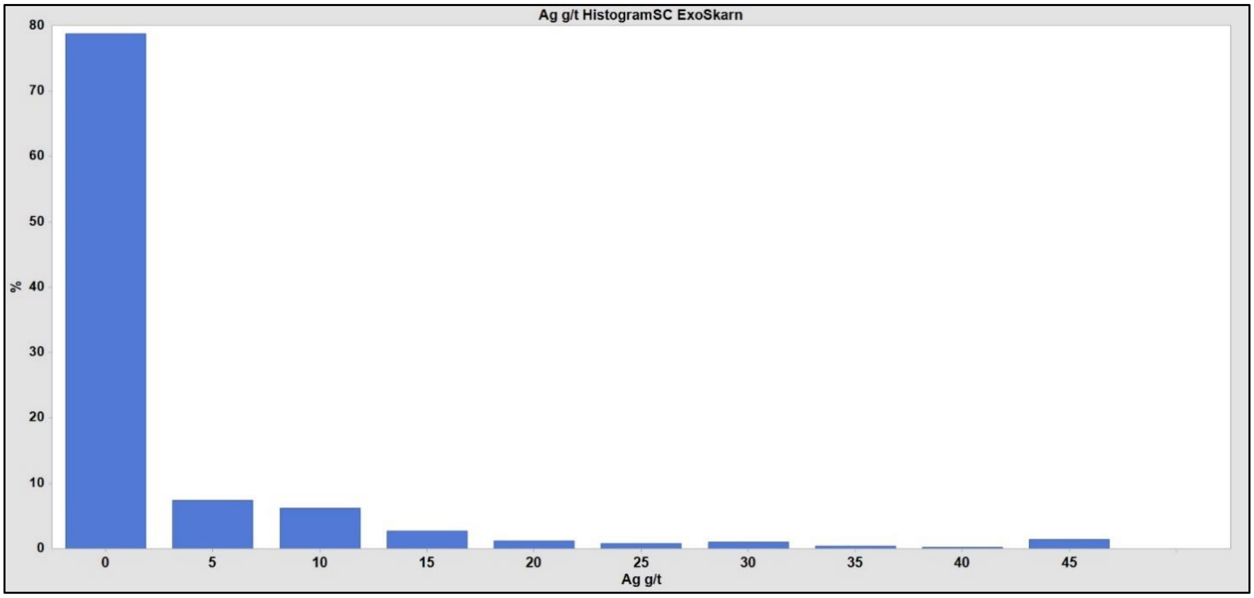


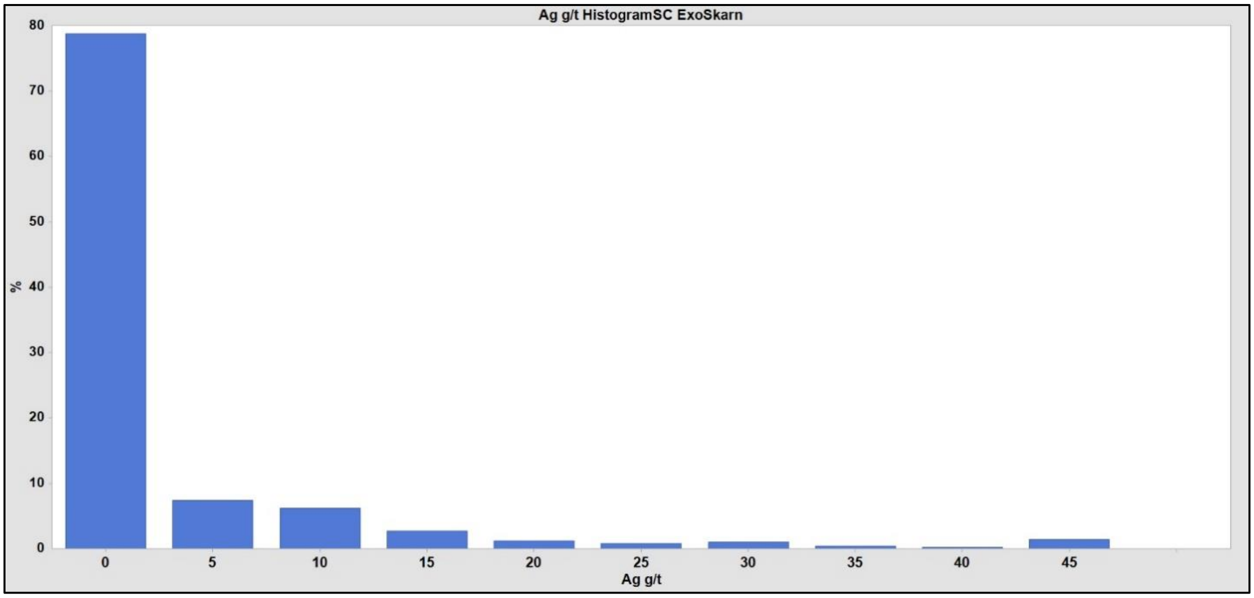
Source: Kirkham (2021)

Figure 14-6: Histograms of Silver Composite Grades within the El Dorado (ED) and Santa Cruz (SC) Endo- and Exo-skarn Zones



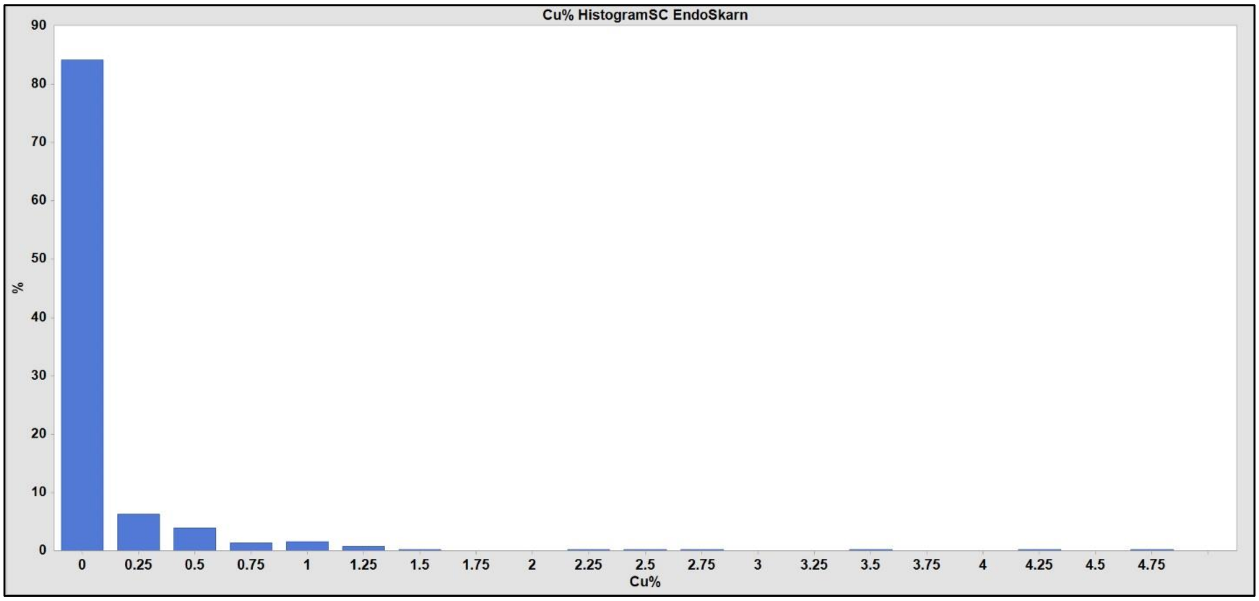


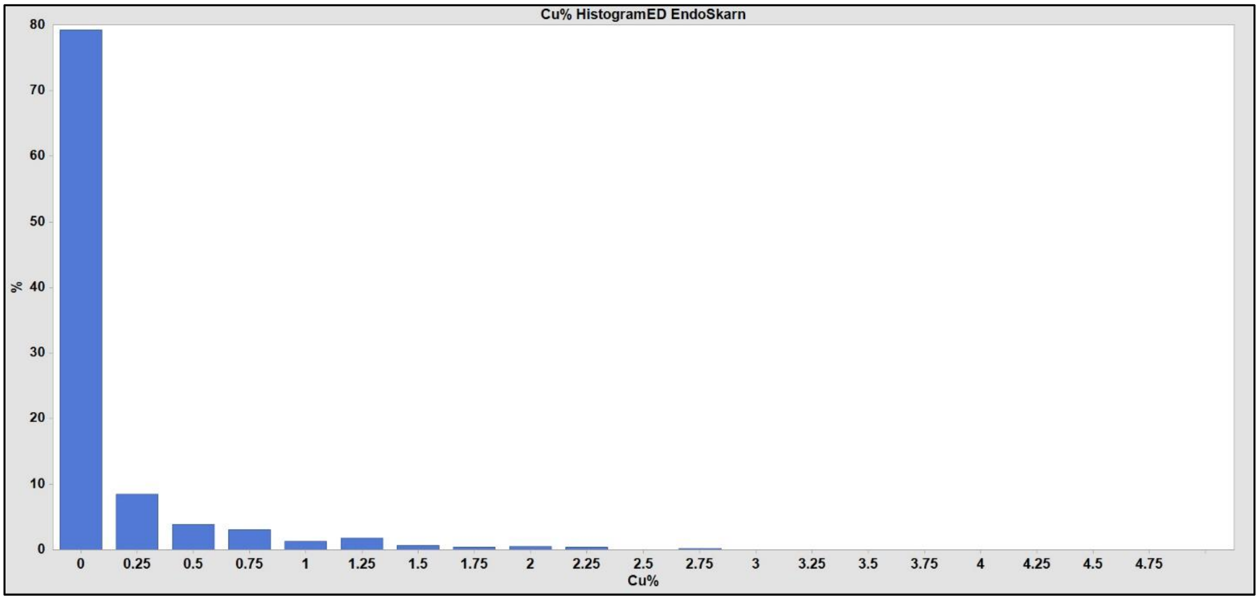


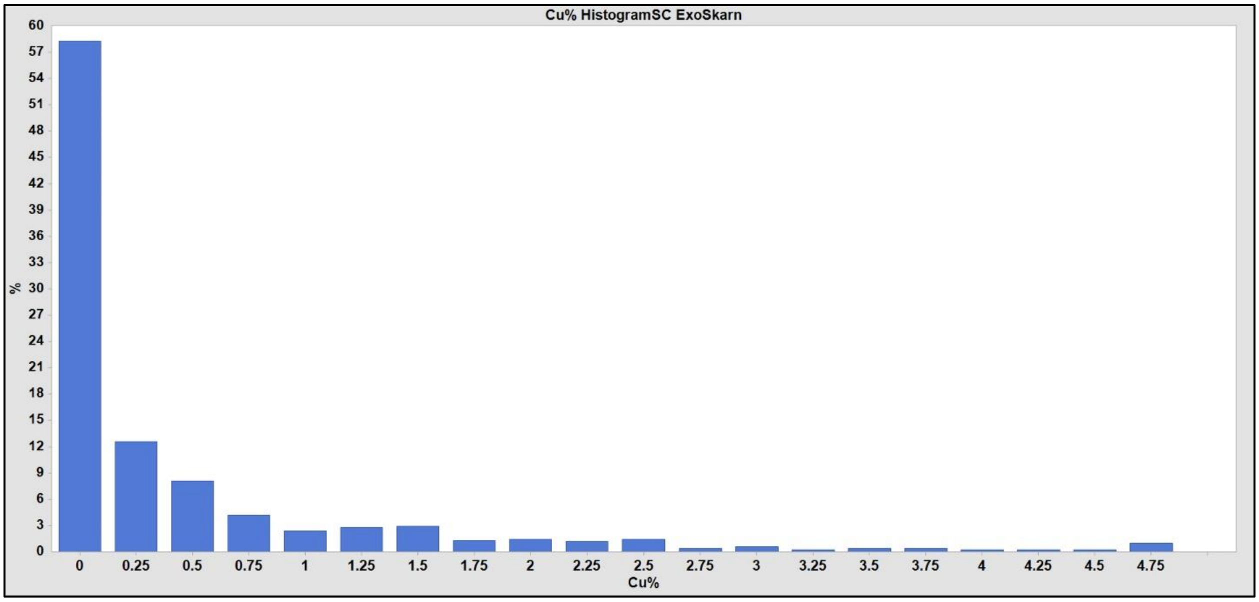


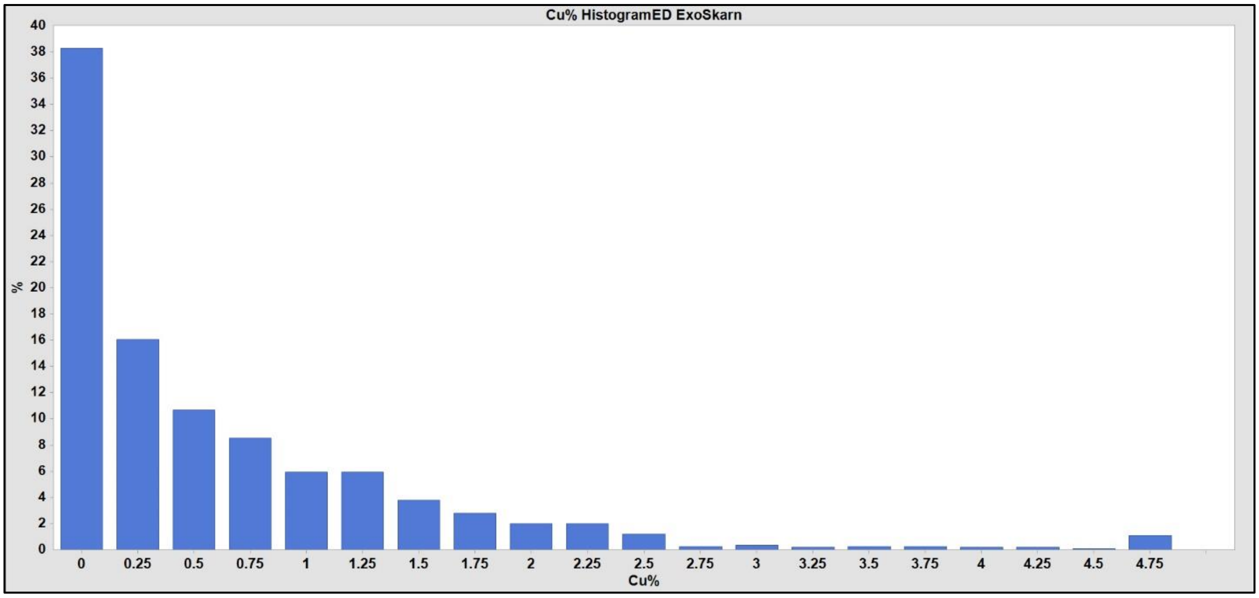
Source: Kirkham (2021)

Figure 14-7: Histograms of Copper Composite Grades within the El Dorado (ED) and Santa Cruz (SC) Endo- and Exo-skarn Zones



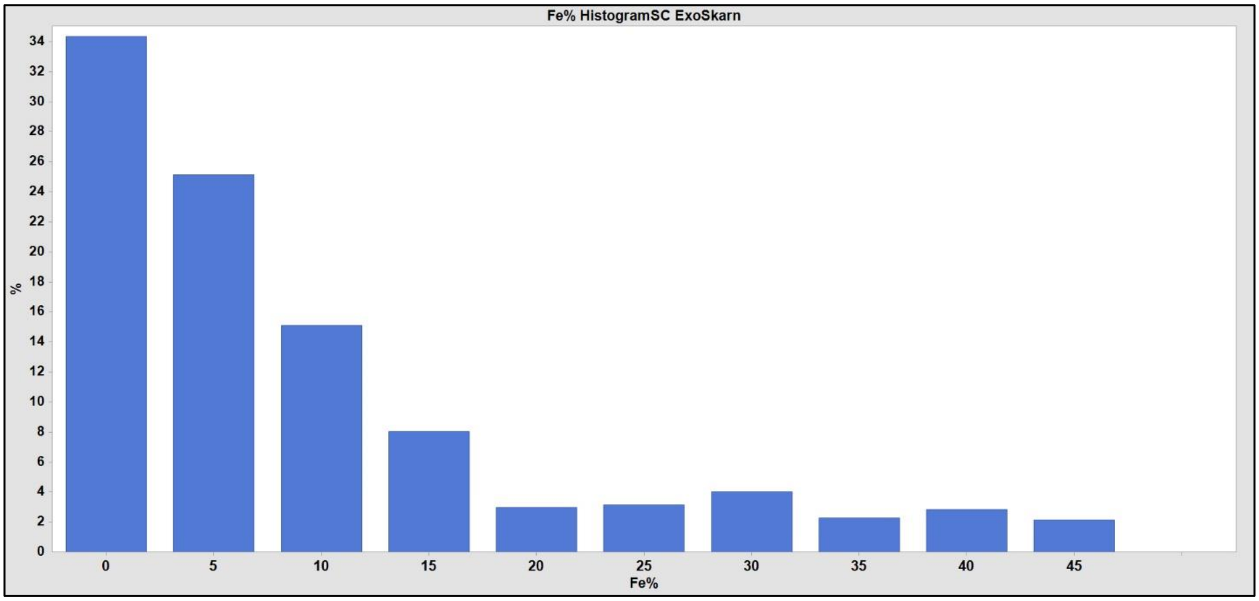


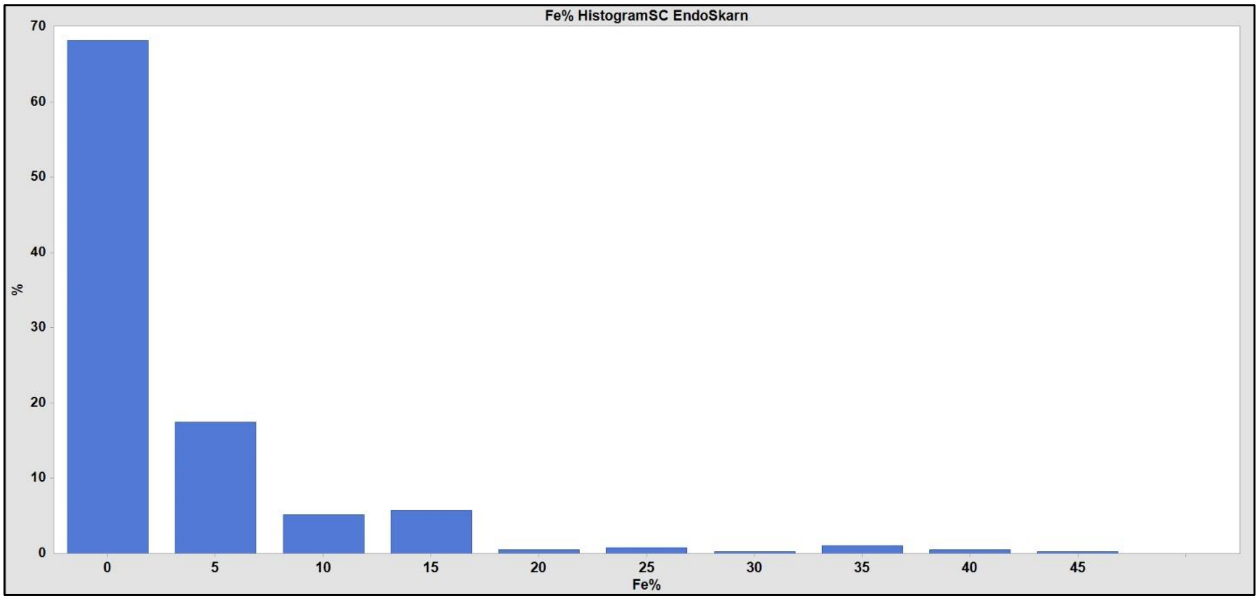


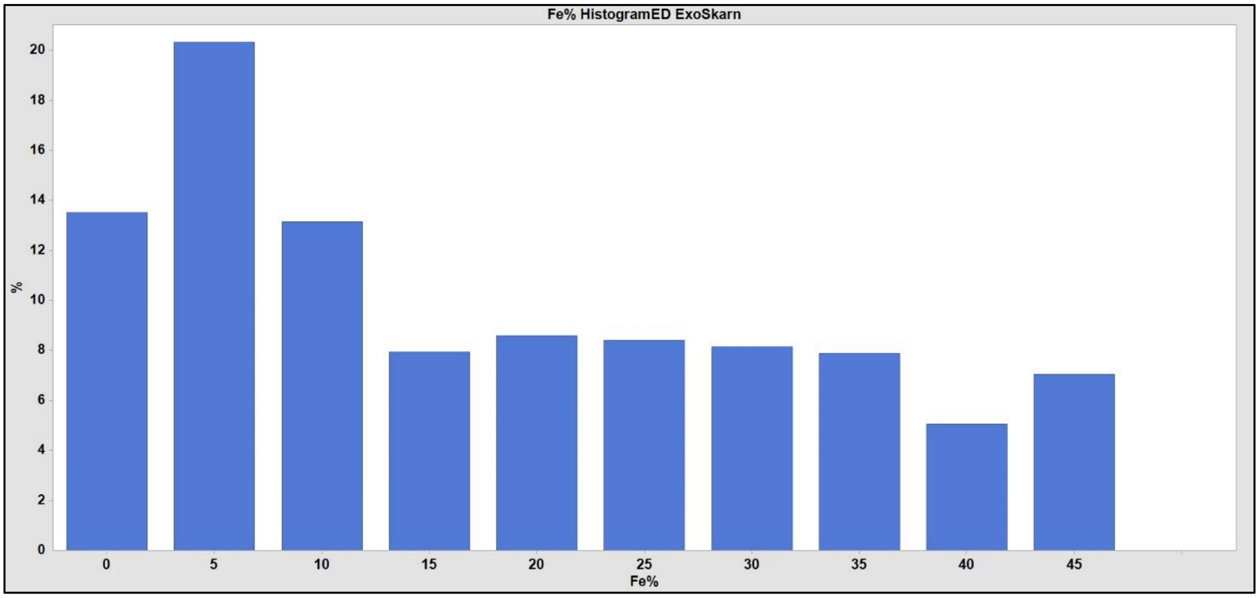


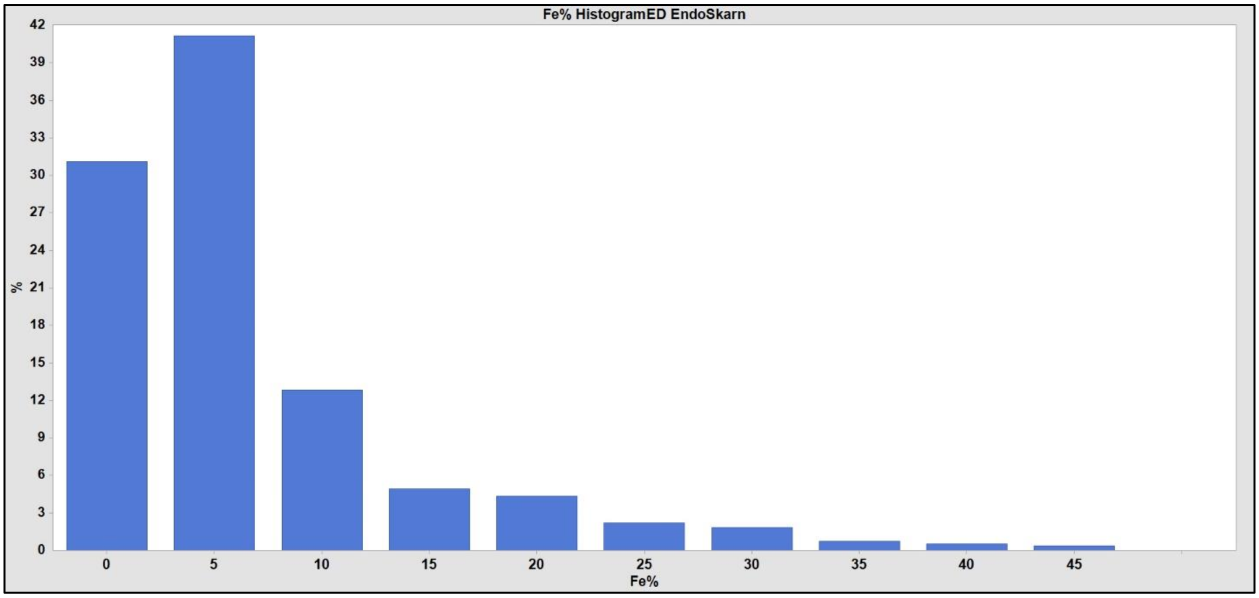
Source: Kirkham (2021)

Figure 14-8: Histograms of Iron Composite Grades within the El Dorado (ED) and Santa Cruz (SC) Endo- and Exo-skarn Zones









Source: Kirkham (2021)

For reference, Table 14-2 lists the mineralized zone names along with the numeric codes utilized for coding of the assays, composites and subsequent block model.

Table 14-2: Mineralized Zone Numeric Codes

ZONE	ED East Endo-skarn	ED West Endo-skarn	ED East Exo-skarn	ED West Exo-skarn	SC Endo-skarn	SC Exo-skarn
Code	31	32	41	42	3	4

Source: Kirkham (2021)

Table 14-3 shows the basic statistics for the 2.0 m copper composite grades within the mineralized domains. It should be noted that although 2.0 m is the composite length, any residual composites of lengths greater than 1.0 m and less than 2.0 m were retained to represent a composite, while any composite residuals less than 1.0 m were combined with the previous composite.

There is a total of 2,462 (4,861 m) composites, with mean gold, silver, copper and iron grades for all zones of 0.83 g/t gold, 3.25 g/t silver, 0.52% copper and 14.38% total iron shown in Table 14-3. In all cases, the grades of the exoskarn units are significantly greater than those of the endo-skarn units. In addition, the west units are approximately 2 times the values for those of the east units within the El Dorado zone.

The Coefficient of Variation (CV), the standard deviation divided by the mean (σ/μ), is a unit independent measure of variability. It shows the extent of variability in relation to the mean of the population. Values great than 1 indicate a high degree of variability however for data that has outlier sub-populations values up to 2 are reasonable. The standard methodology to reduce the effect of these outliers is to cut the outliers to an appropriate threshold which will be addressed in the following section.

The CVs for gold range from 1.7 to 2.9, silver range from 1.3 to 2.7, copper range from 1.1 to 2.5 and total iron range from 0.6 to 1.1. These ranges illustrate a moderate to high variability with the Santa Cruz zones with the West Endo-skarn being the zones that require addressing.

Table 14-3: Composite Statistics Weighted by Length

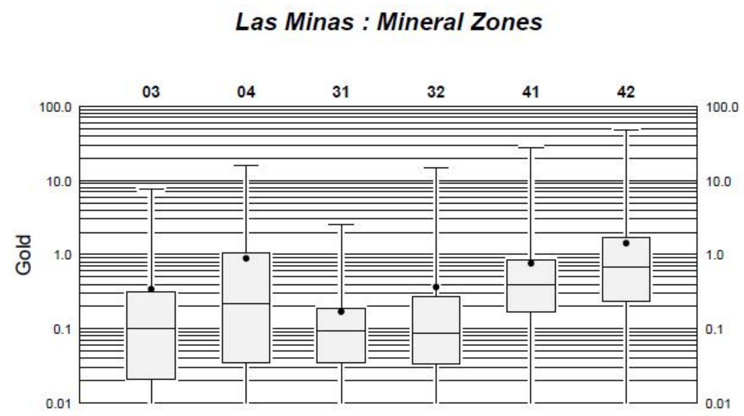
Metal	Mineralized Unit	#	Length (m)	Maximum	Mean	SD	CV
AU	SC Endoskarn	394	760.7	7.68	0.34	0.74	2.2
	SC Exoskarn	508	1,008.3	15.94	0.89	1.64	1.8
	ED East Endoskarn	136	268.6	2.54	0.17	0.29	1.7
	ED West Endoskarn	409	815.2	14.72	0.37	1.08	2.9
	ED East Exoskarn	262	512.8	27.05	0.76	1.98	2.6
	ED West Exoskarn	753	1,495.6	47.40	1.43	2.73	1.9
	Total	2,462	4,861.2	47.40	0.83	1.94	2.3
	All	8,047	14,683.7	47.40	0.43	1.37	3.2

Metal	Mineralized Unit	#	Length (m)	Maximum	Mean	SD	CV
AG	SC Endoskarn	394	760.7	59.10	2.07	5.64	2.7
	SC Exoskarn	508	1,008.3	89.37	4.52	9.81	2.2
	ED East Endoskarn	136	268.6	8.00	0.83	1.28	1.5
	ED West Endoskarn	409	815.2	29.00	1.21	2.94	2.4
	ED East Exoskarn	262	512.8	127.00	3.13	6.40	2.0
	ED West Exoskarn	753	1,495.6	50.20	4.59	6.14	1.3
	Total	2,462	4,861.2	127.00	3.25	6.67	2.1
	All	8,047	14,683.7	127.00	1.66	4.44	2.7
CU	SC Endoskarn	394	760.7	5.15	0.20	0.50	2.5
	SC Exoskarn	508	1,008.3	11.40	0.56	1.09	1.9
	ED East Endoskarn	136	268.6	1.73	0.13	0.26	2.0
	ED West Endoskarn	409	815.2	2.98	0.21	0.40	1.9
	ED East Exoskarn	262	512.8	4.40	0.57	0.63	1.1
	ED West Exoskarn	753	1,495.6	9.28	0.89	1.06	1.2
	Total	2,462	4,861.2	11.40	0.52	0.89	1.7
	All	8,047	14,683.7	17.80	0.25	0.64	2.5
FE	SC Endoskarn	394	760.7	45.30	5.71	6.56	1.1
	SC Exoskarn	508	1,008.3	53.36	11.81	11.47	1.0
	ED East Endoskarn	136	268.6	25.40	6.73	4.96	0.7
	ED West Endoskarn	409	815.2	57.50	9.86	8.31	0.8
	ED East Exoskarn	262	512.8	58.22	16.45	11.44	0.7
	ED West Exoskarn	753	1,495.6	61.38	23.65	14.99	0.6
	Total	2,462	4,861.2	61.38	14.38	13.29	0.9
	All	8,047	14,683.7	61.38	8.13	9.90	1.2

Source: Kirkham (2021)

The box plots for each of gold, silver, copper and total iron are shown in Figure 14-9 through Figure 14-12. As would be expected, the endo-skarn units illustrate a statistical relationship to each other. However, the box plots show that all of the zones are statistically different and therefore it would not be prudent to group zones for the purpose of estimation. Therefore, the zones are to be estimated individually using hard boundaries, meaning samples that are to inform the blocks in one zone or domain, will not be allowed to inform any other domain.

Figure 14-9: Box Plot of Gold Composites by Zone

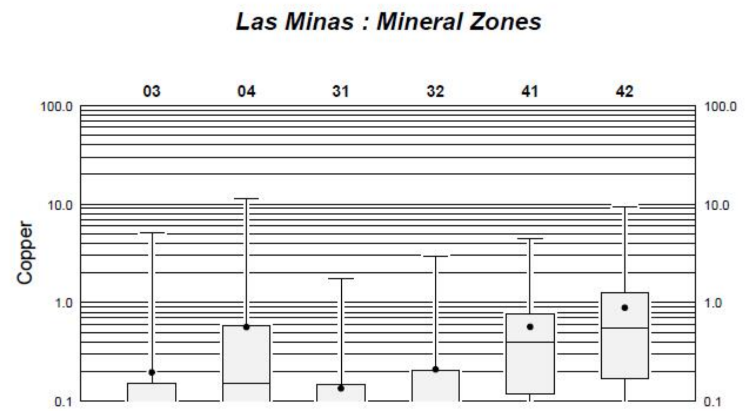


	03	04	31	32	41	42	
Number of data	394	508	136	409	262	751	**Number of data**
Mean	0.343	0.891	0.171	0.367	0.765	1.431	**Mean**
Variance	0.548	2.691	0.086	1.167	3.904	7.445	**Variance**
Skewness	5.352	4.206	5.452	8.04	10.217	8.963	**Skewness**
Coef of Variation	2.159	1.84	1.711	2.945	2.582	1.906	**Coef of Variation**
Maximum	7.68	15.94	2.542	14.72	27.05	47.4	**Maximum**
Upper quartile	0.31	1.075	0.189	0.271	0.843	1.672	**Upper quartile**
Median	0.1	0.215	0.094	0.086	0.393	0.693	**Median**
Lower quartile	0.021	0.035	0.035	0.034	0.17	0.234	**Lower quartile**
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	**Minimum**

Variable: Gold
Acceptable range: 0.0 to 10000.0
Weights: LNGTH

Source: Kirkham (2021)

Figure 14-10: Box Plot of Copper Composites by Zone

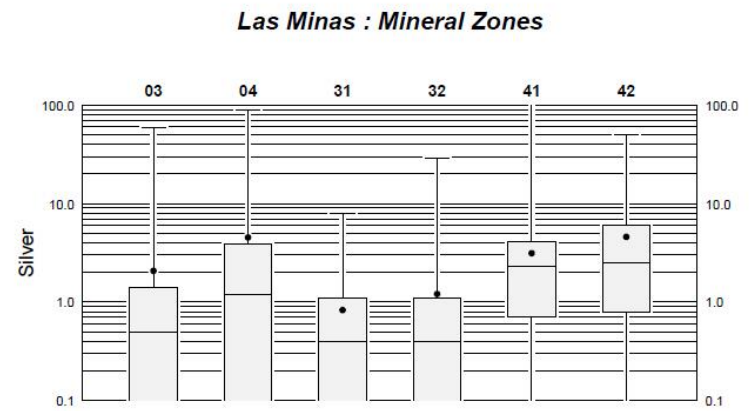


	03	04	31	32	41	42	
Number of data	394	508	136	409	262	751	**Number of data**
Mean	0.196	0.565	0.135	0.21	0.569	0.889	**Mean**
Variance	0.25	1.195	0.07	0.161	0.392	1.132	**Variance**
Skewness	6.032	4.516	3.944	3.505	2.408	2.953	**Skewness**
Coef of Variation	2.548	1.936	1.96	1.905	1.1	1.197	**Coef of Variation**
Maximum	5.15	11.4	1.73	2.98	4.4	9.28	**Maximum**
Upper quartile	0.15	0.577	0.146	0.207	0.773	1.274	**Upper quartile**
Median	0.05	0.15	0.04	0.06	0.402	0.551	**Median**
Lower quartile	0.01	0.01	0.01	0.02	0.12	0.17	**Lower quartile**
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	**Minimum**

Variable: Copper
Acceptable range: 0.0 to 100.0
Weights: LNGTH

Source: Kirkham (2021)

Figure 14-11: Box Plot of Silver Composites by Zone

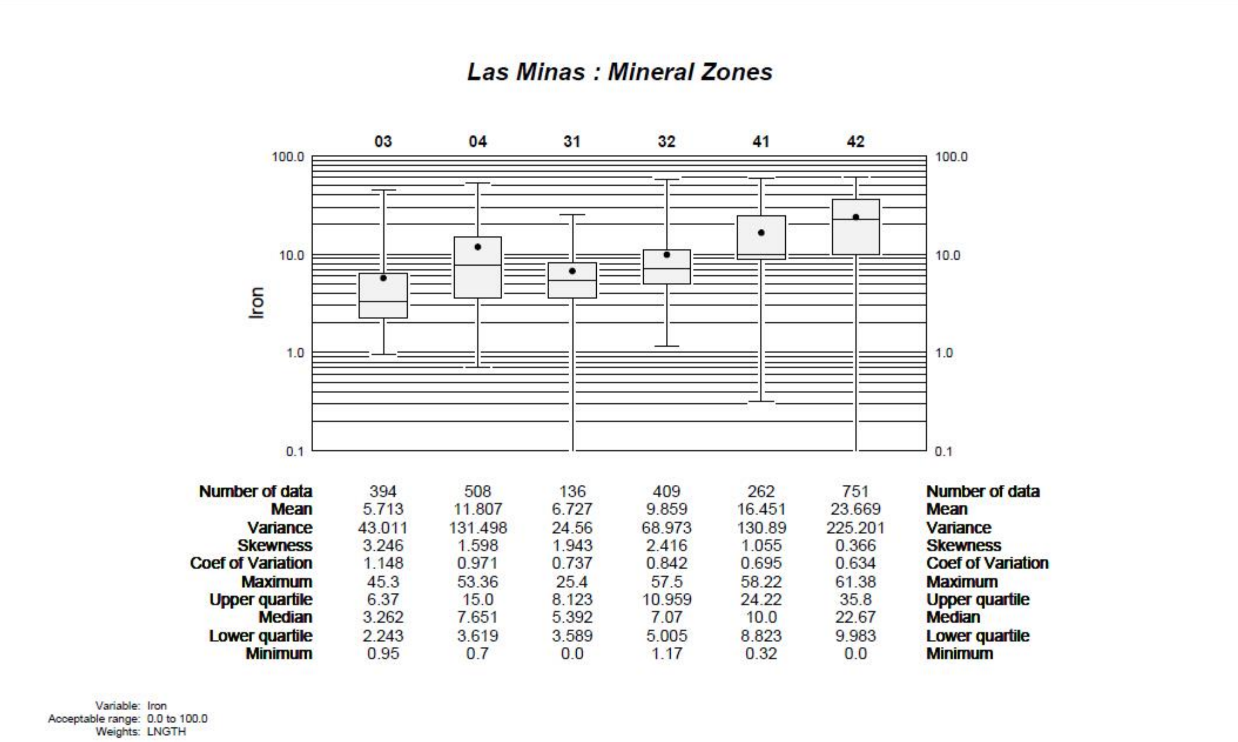


	03	04	31	32	41	42	
Number of data	394	508	136	409	262	751	**Number of data**
Mean	2.071	4.525	0.829	1.207	3.132	4.59	**Mean**
Variance	31.762	96.346	1.631	8.622	40.912	37.797	**Variance**
Skewness	5.983	4.641	3.096	5.966	15.705	2.785	**Skewness**
Coef of Variation	2.721	2.169	1.54	2.432	2.042	1.339	**Coef of Variation**
Maximum	59.1	89.4	8.0	29.0	127.0	50.2	**Maximum**
Upper quartile	1.4	3.9	1.1	1.1	4.087	5.95	**Upper quartile**
Median	0.5	1.2	0.4	0.4	2.29	2.5	**Median**
Lower quartile	0.0	0.0	0.0	0.0	0.7	0.8	**Lower quartile**
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	**Minimum**

Variable: Silver
Acceptable range: 0.0 to 10000.0
Weights: LNGTH

Source: Kirkham (2021)

Figure 14-12: Box Plot of Iron Composites by Zone



	03	04	31	32	41	42	
Number of data	394	508	136	409	262	751	**Number of data**
Mean	5.713	11.807	6.727	9.859	16.451	23.669	**Mean**
Variance	43.011	131.498	24.56	68.973	130.89	225.201	**Variance**
Skewness	3.246	1.598	1.943	2.416	1.055	0.366	**Skewness**
Coef of Variation	1.148	0.971	0.737	0.842	0.695	0.634	**Coef of Variation**
Maximum	45.3	53.36	25.4	57.5	58.22	61.38	**Maximum**
Upper quartile	6.37	15.0	8.123	10.959	24.22	35.8	**Upper quartile**
Median	3.262	7.651	5.392	7.07	10.0	22.67	**Median**
Lower quartile	2.243	3.619	3.589	5.005	8.823	9.983	**Lower quartile**
Minimum	0.95	0.7	0.0	1.17	0.32	0.0	**Minimum**

Source: Kirkham (2021)

14.6 Evaluation of Outlier Assay Values

An evaluation of the probability plots suggests that there may be outlier assay values that could result in an overestimation of resources. Although it is believed that this risk is relatively low, it was considered prudent to cut the gold, copper, silver and iron composites within the El Dorado zones to reduce the effects of outliers. Santa Cruz was not subjected to cutting as these zones are very tightly constrained which limits smearing and spreading of high-grade values.

Table 14-4: Outlier Cutting Thresholds and Subsequent Impact

Metal	Mineralized Unit	Mean	CV	Cut Grade	Mean	CV	Mean %Diff	CV %Diff
Au g/t	East Endoskarn	0.17	1.7	1.50	0.16	1.4	-6%	-18%
	West Endoskarn	0.37	2.9	1.50	0.25	1.5	-32%	-48%
	East Exoskarn	0.76	2.6	8.00	0.67	1.5	-12%	-40%

Metal	Mineralized Unit	Mean	CV	Cut Grade	Mean	CV	Mean %Diff	CV %Diff
	West Exoskarn	1.43	1.9	8.00	1.30	1.2	-10%	-35%
Cu%	East Endoskarn	0.13	2.0	0.75	0.11	1.5	-15%	-23%
	West Endoskarn	0.21	1.9	0.75	0.16	1.4	-24%	-28%
	East Exoskarn	0.57	1.1	3.30	0.56	1.0	-1%	-4%
	West Exoskarn	0.89	1.2	3.30	0.84	1.0	-6%	-17%
Ag g/t	East Endoskarn	0.83	1.5	4.50	0.77	1.3	-7%	-13%
	West Endoskarn	1.21	2.4	4.50	0.88	1.4	-27%	-41%
	East Exoskarn	3.13	2.0	20.00	2.90	1.0	-7%	-51%
	West Exoskarn	4.59	1.3	20.00	4.31	1.2	-6%	-13%
Fe%	East Endoskarn	6.73	0.7	25.40	6.73	0.7	0%	0%
	West Endoskarn	9.86	0.8	35.00	9.67	0.8	-2%	-7%
	East Exoskarn	16.45	0.7	50.00	16.40	0.7	0%	-1%
	West Exoskarn	23.65	0.6	50.00	23.44	0.6	-1%	-2%

Source: Kirkham (2021)

14.7 Specific Gravity Estimation

Bulk densities were based on a total of 3,179 measurements with 1,386 individual measurements taken within the mineralized zones by Mexican Gold field personnel. These density values ranged from 2.2 t/m^3 to 8.1 t/m^3 as shown in Table 14-5. Specific gravities were calculated on a block-by-block basis by interpolating the SG measurements using an inverse distance to the second power and limited within the individual mineralized material zone solids. A default density of 3.49 t/m^3 was assigned to any blocks that were not assigned a calculated value.

Table 14-5: Statistics for Specific Gravity Measurements by Zone

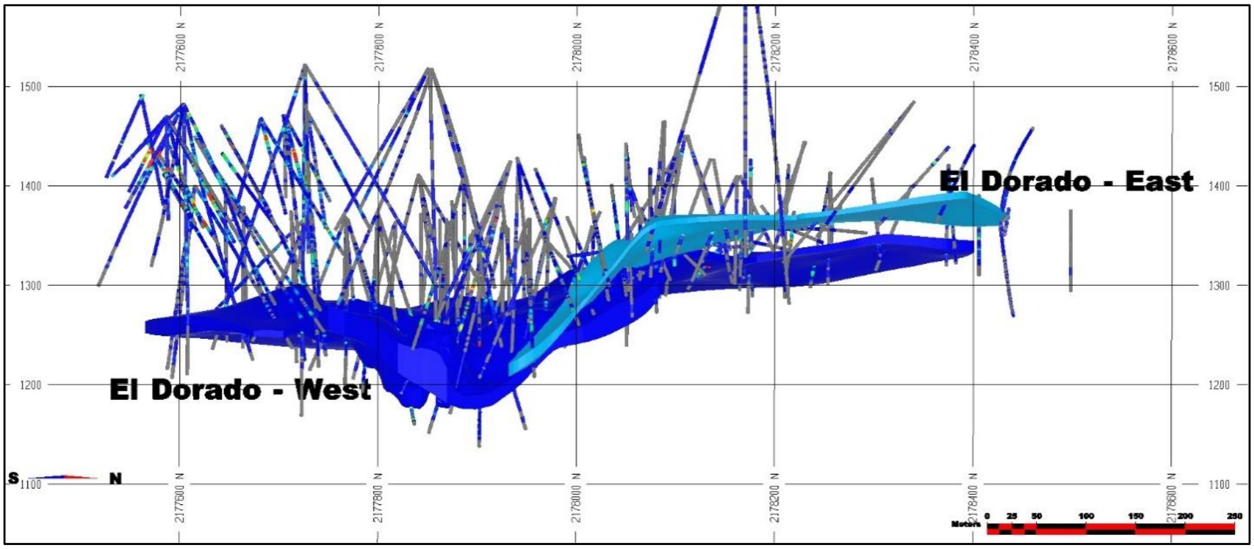
	Mineralized Unit	#	Length (m)	Minimum	Maximum	Mean	SD	CV
SG	SC Endoskarn	76	5.9	2.84	4.88	3.59	0.50	0.1
	SC Exoskarn	57	4.5	2.50	4.70	3.33	0.38	0.1
	ED East Endoskarn	88	7	2.20	4.60	3.32	0.47	0.1
	ED West Endoskarn	347	27.5	2.40	8.10	3.55	0.51	0.1
	ED East Exoskarn	100	7.6	2.60	4.50	3.36	0.47	0.1
	ED West Exoskarn	718	55.2	2.51	6.00	3.51	0.49	0.1
	Total	1,386	107.7	2.20	8.10	3.49	0.50	0.1
	All	3,179	244.76	2.10	8.30	3.31	0.49	0.1

Source: Kirkham (2021)

14.8 Relative Elevation Estimation for El Dorado Exo- and Endo-skarn Zones

The El Dorado endo-skarn and exoskarn zones in the deposit pose a number of challenges with respect to modeling and interpolation. The first challenge is that, based on data and observations, the mineralization, and more importantly, the grade, appears to be layered or banded. In addition, due to the abrupt change in strike of the deposit and undulations as shown in Figure 14-13, using a standard oriented ellipse to guide the estimation process does not account for, nor does it adequately deal with, significant changes in dip, and more importantly, the layered deposits that are angled.

Figure 14-13: Plan View of El Dorado Exo-skarn Zones Illustrating Estimation Challenges

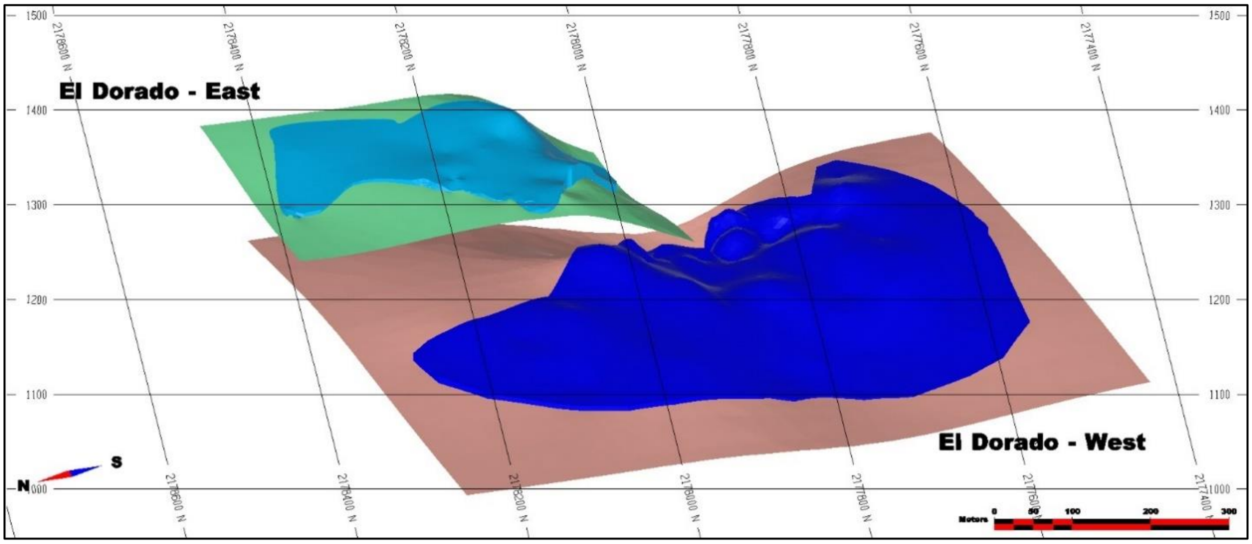


Source: Kirkham (2021)

Grades in the model have been estimated using ordinary kriging as discussed in Section 14.11. However, in an attempt to adequately account for the changes in strike and dip, a relative elevation modeling approach has been used. This method measures distances relative to the footwall contact of the domains (Figure 14-14) which are stored in both model blocks and composited drillhole samples. These Footwall Distance Values (FWDIS) are linked during interpolation to ensure that samples will only correlate with data within its relative stratigraphic position. These relative elevations essentially flatten out the deposit for interpolation. Using relative elevations is a reflection of the continuity of the mineralization in relation to the orientation of the deposit.

The grade models have been developed using the relative elevation approach and omni-directional, anisotropic search ellipses.

Figure 14-14: Plan View of El Dorado Exo-skarn Zones Illustrating Estimation Challenges



Source: Kirkham (2021)

14.9 Variography

Experimental variograms and variogram models in the form of correlograms were generated for gold, silver, copper and total iron grades. Table 14-6 shows the correlogram models using for the ordinary kriging estimation process. Note that the El Dorado correlogram models are oriented in one direction which reflets the utilization of the ‘relative elevation’ methodology where they are essentially flattened to the footwall plane.

Table 14-6: Geostatistical Parameters Used for Estimation

Lithology Unit	Geostatistical Parameters	ED East Endo-skarn	ED West Endo-skarn	ED East Exo-skarn	ED West Exo-skarn	SC Endo-skarn	SC Exo-skarn
Lithology Code		31	32	41	42	3	4
AU g/t	Nugget (C0)	0.1	0.119	0.122	0.41	0.356	0.457
	First Sill (C1)	0.133	0.851	0.456	0.295	0.466	0.366
	Second Sill (C2)	0.767	0.03	0.422	0.295	0.178	0.178
1st Structure	Range along the Z'	19.4	27.1	17.5	24.1	6.5	11.6
	Range along the X'	3.7	57.4	72.9	70.9	59	61.7
	Range along the Y'	7.1	9.3	8.5	13.3	23.1	20.6
	R1 about the Z	27	53	-3	71	53	-50

Lithology Unit	Geostatistical Parameters	ED East Endo-skarn	ED West Endo-skarn	ED East Exo-skarn	ED West Exo-skarn	SC Endo-skarn	SC Exo-skarn
	R2 about the X'	0	0	0	0	44	-11
	R3 about the Y'	0	0	0	0	-7	0
2nd Structure	Range along the Z'	1.2	64.7		160.7	60.7	55
	Range along the X'	30.3	28.7	18.3	7	6.1	17.4
	Range along the Y'	7.1	14.3	13.6	15	182.1	60.8
	R1 about the Z	-56	-30	-79	8	-19	-26
	R2 about the X'	0	0	0	0	-1	93
	R3 about the Y'	0	0	0	0	0	81
AG g/t	Nugget (C0)	0.241	0.09	0.28	0.32	0.082	0.346
	First Sill (C1)	0.021	0.589	0.156	0.542	0.427	0.471
	Second Sill (C2)	0.738	0.321	0.564	0.138	0.49	0.183
1st Structure	Range along the Z'	3.8	6.6	13.1	8.6	16.6	52.2
	Range along the X'	55.8	30.9	192.3	14	18.2	98.5
	Range along the Y'	16.9	8.4	2.4	17.5	4.6	9.4
	R1 about the Z	-18	-7	1	-113	33	1
	R2 about the X'	0	0	0	0	-33	41
	R3 about the Y'	0	0	0	0	-32	-47
2nd Structure	Range along the Z'	178	125.2	14.9	1865.3	31.7	113.5
	Range along the X'	4.5	55.3	25.9	124.6	2.9	13.5
	Range along the Y'	6.9	10	14.9	363.2	36	385
	R1 about the Z	-43	22	-118	52	-59	3
	R2 about the X'	0	0	0	0	-65	53
	R3 about the Y'	0	0	0	0	60	-56
CU%	Nugget (C0)	0.1	0.128	0.047	0.198	0.163	0.14
	First Sill (C1)	0	0.502	0.603	0.645	0.501	0.411
	Second Sill (C2)	0.9	0.37	0.35	0.156	0.336	0.449
1st Structure	Range along the Z'	5.3	9.5	10.6	31.7	5.2	19.9
	Range along the X'	117.3	82.4	75.4	66	15.5	25.2
	Range along the Y'	5.4	10.4	13.7	10.4	8.2	7.8
	R1 about the Z	-17	52	23	25	67	28
	R2 about the X'	0	0	0	0	35	8
	R3 about the Y'	0	0	0	0	68	-7
2nd Structure	Range along the Z'	4	1973.7	4.3	74.3	139.6	51.8
	Range along the X'	53.4	37.4	25.8	2295.1	2.1	9.7
	Range along the Y'	5.4	11	16.8	10.4	162.6	72.8
	R1 about the Z	-30	23	89	-54	-13	-11
	R2 about the X'	0	0	0	0	78	47

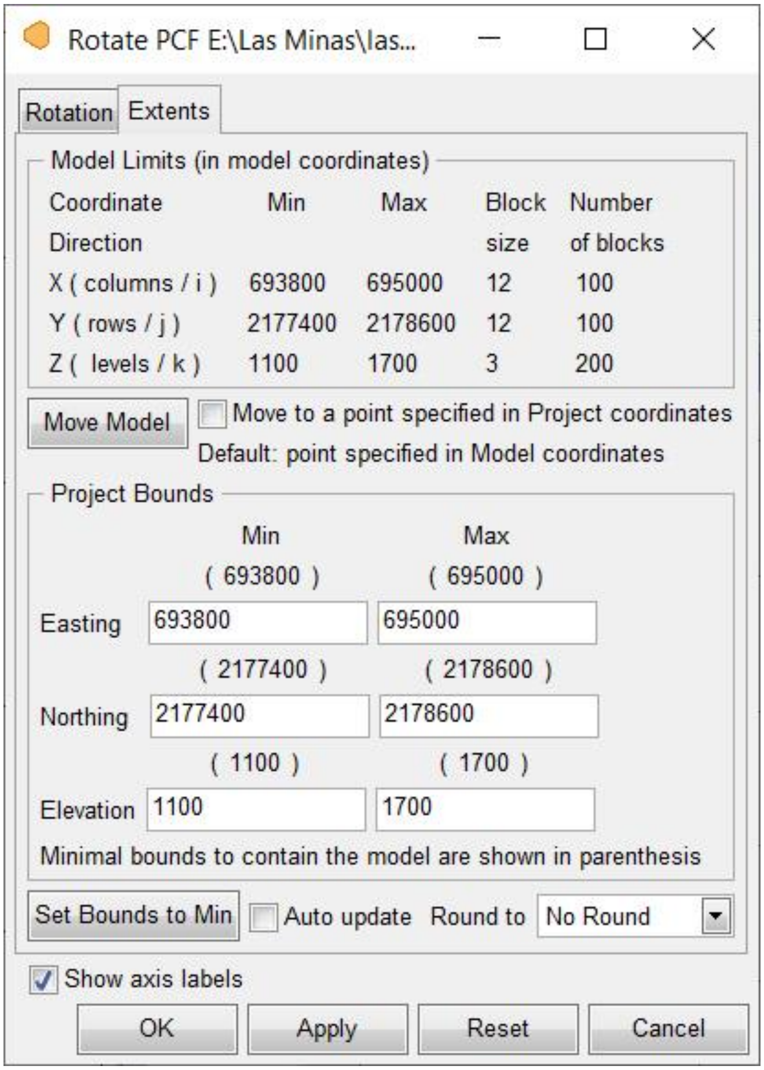
Lithology Unit	Geostatistical Parameters	ED East Endo-skarn	ED West Endo-skarn	ED East Exo-skarn	ED West Exo-skarn	SC Endo-skarn	SC Exo-skarn
	R3 about the Y'	0	0	0	0	17	52
FE%	Nugget (C0)	0.119	0.35	0.003	0.154	0.219	0.084
	First Sill (C1)	0.133	0.211	0.676	0.703	0.416	0.55
	Second Sill (C2)	0.747	0.439	0.321	0.143	0.365	0.366
1st Structure	Range along the Z'	50.4	30	3.4	18.8	47.5	13.7
	Range along the X'	345.9	6.5	68.5	26.1	75.5	42
	Range along the Y'	8.5	6.8	7.8	13	8.4	4.1
	R1 about the Z	-7	20	1	-32	53	-41
	R2 about the X'	0	0	0	0	-7	-6
	R3 about the Y'	0	0	0	0	40	-47
2nd Structure	Range along the Z'	166.8	222.4	83.7	123.5	756.2	113.9
	Range along the X'	7.7	93.7	36.2	3481.2	73.4	9.6
	Range along the Y'	8.9	7	8	13	129.3	45.4
	R1 about the Z	-51	47	7	-46	52	14
	R2 about the X'	0	0	0	0	28	17
	R3 about the Y'	0	0	0	0	2	-26

Source: Kirkham (2021)

14.10 Block Model Definition

The block model used to estimate the resources was defined according to the limits specified in Figure 14-15 and Figure 14-16. The block model is orthogonal and non-rotated, reflecting the general orientation of the deposit. The chosen block size was 12 m by 12 m by 3 m, roughly reflecting the drillhole spacing (i.e., 4–6 blocks between drillholes) which are spaced at approximately 50 m centers. Note: MineSight^{TM} uses the centroid of the blocks as the origin. Furthermore, the parent blocks are subsequently sub-blocked in all three directions to 0.5 m for the purpose on mine design and mine planning.

Figure 14-15: Origin and Orientation for the Las Minas Block Model



Model Limits (in model coordinates)

Coordinate Direction	Min	Max	Block size	Number of blocks
X (columns / i)	693800	695000	12	100
Y (rows / j)	2177400	2178600	12	100
Z (levels / k)	1100	1700	3	200

☐ Move to a point specified in Project coordinates
Default: point specified in Model coordinates

Project Bounds

	Min	Max
	(693800)	(695000)
Easting	693800	695000
	(2177400)	(2178600)
Northing	2177400	2178600
	(1100)	(1700)
Elevation	1100	1700

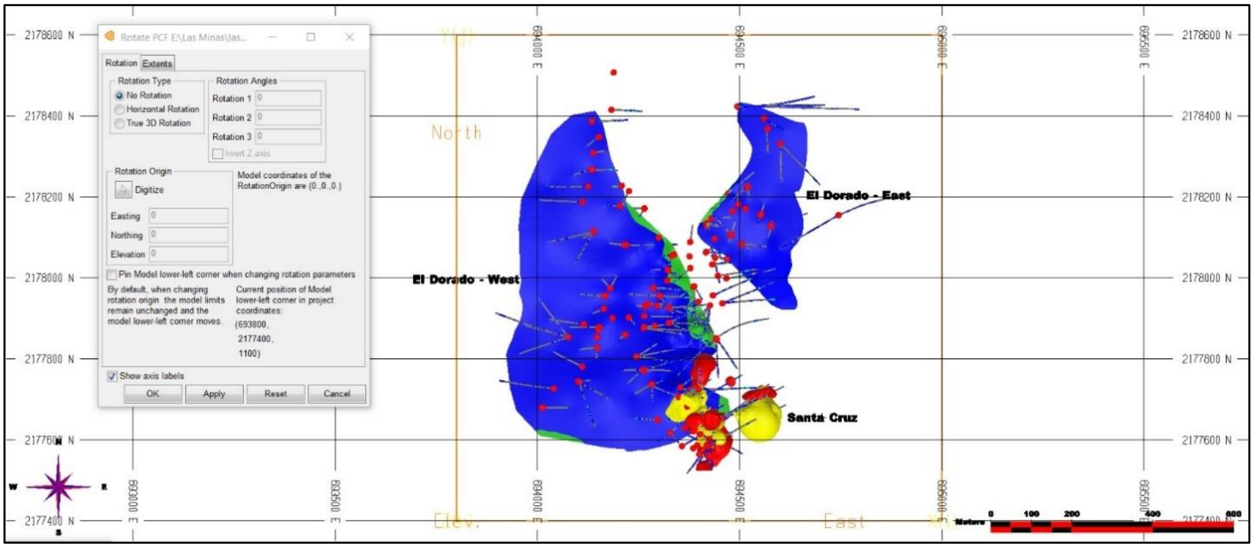
Minimal bounds to contain the model are shown in parenthesis

☐ Auto update Round to No Round

☑ Show axis labels

Source: Kirkham (2021)

Figure 14-16: Dimensions for the Las Minas Block Model



Rotation Type: ● No Rotation ○ Horizontal Rotation ○ True 3D Rotation

Rotation Angles: Rotation 1: 0; Rotation 2: 0; Rotation 3: 0; ☐ Invert Z axis

Rotation Origin: Digitize; Easting: 0; Northing: 0; Elevation: 0

Model coordinates of the RotationOrigin are (0.,0.,0.)

☐ Pin Model lower-left corner when changing rotation parameters

By default, when changing rotation origin the model limits remain unchanged and the model lower-left corner moves.

Current position of Model lower-left corner in project coordinates: (693800, 2177400, 1100)

☑ Show axis labels

Source: Kirkham (2021)

14.11 Resource Estimation Methodology

Gold, copper, silver and total iron block grades were estimated from capped composited samples in a single pass. The mineral resources were estimated using ordinary kriging interpolation for the continuous mineralized domains. Due to the highly constrained nature of the zones in addition to the employment of relative elevation techniques, an omni-directional, anisotropic search ellipse was used in the individual zones.

The resource estimation plan includes the following items:

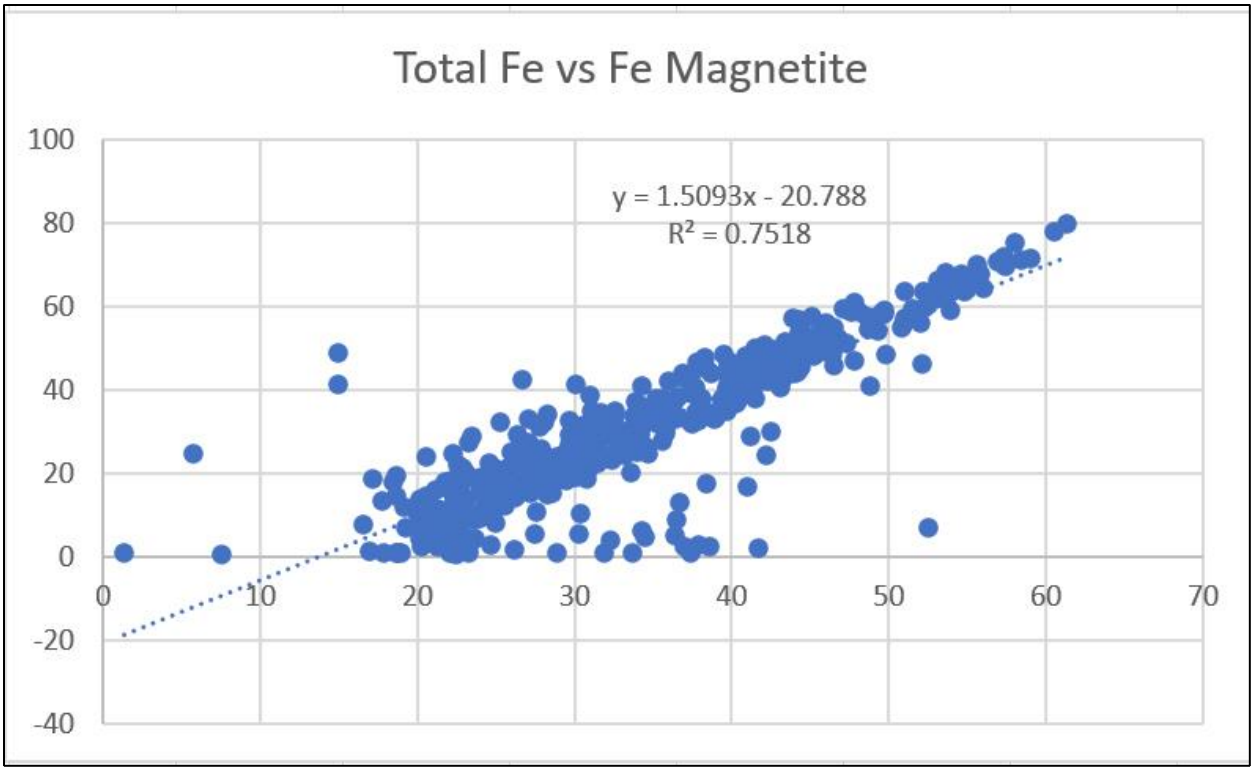
- Mineralized zone code and percentage of modelled mineralization in each block;
- Estimated block gold, silver, copper, and iron grades by ordinary kriging, using a one-pass estimation strategy for the mineralized zone;
- Relative estimation techniques were employed for the El Dorado zones but not for the Santa Cruz zones; and
- Minimum of 4 composites and a maximum of 12 composites were allowed per block while a maximum of 3 composites were allowed per drillhole.

14.12 Estimation of Magnetite

The estimation of the magnetite was performed by applying a regression formula to the estimated Total Fe%. As stated in Section 14.11, Total Fe% was estimated using ordinary kriging with relative elevation methods. In order to estimate the proportion that is magnetite, the two methods considered were to calculate the ratio by molecular weight or from actual magnetite measurements, as determined using SATMAGAN testing, and applying a regression analysis to determine the magnetite grade. The regression method derived by physical measurements was selected to provide the most reliable, defensible result.

A total of 421 SATMAGAN test results were derived from samples within the endo-skarn and exoskarn zones, selected throughout so as to have a representative population and distribution. The values supplied for SATMAGAN testing are Total Fe% and magnetite as a percentage. These can be plotted against each other, and a regression line can be created as shown in Figure 14-17 to apply to samples that do not have SATMAGAN testing but do have Total Fe% thereby filling in the complete dataset.

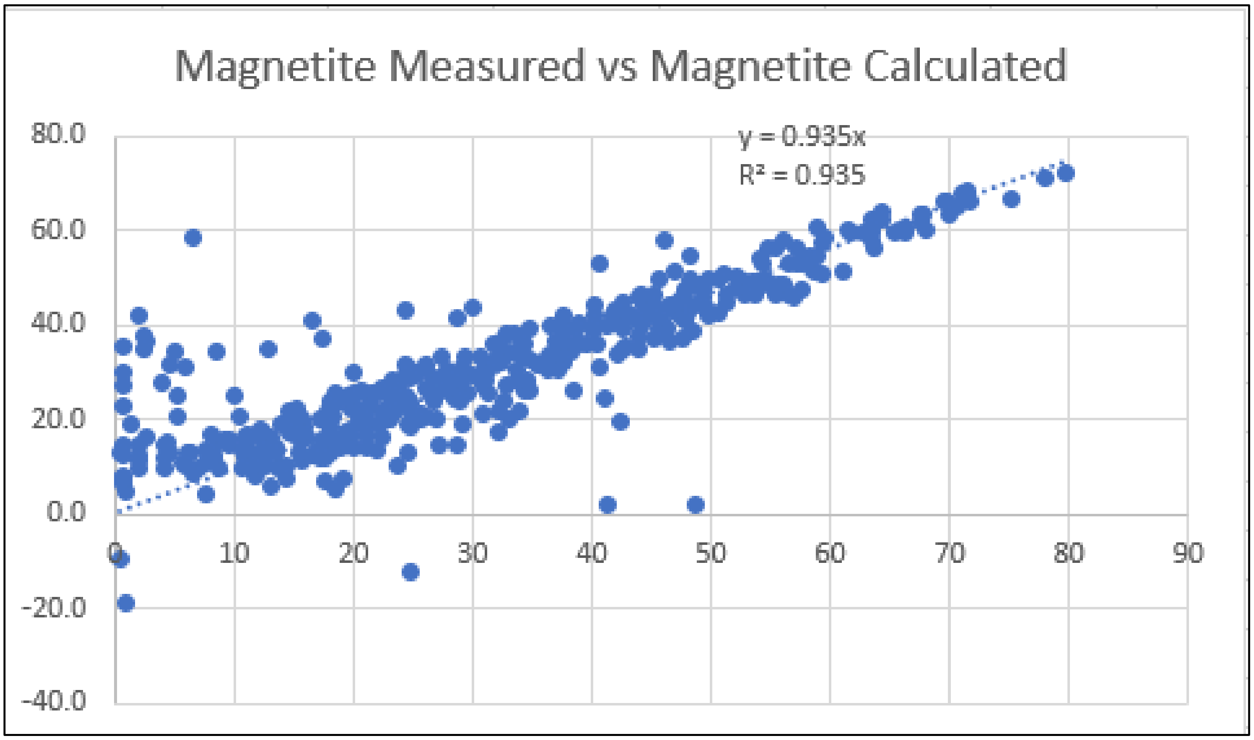
Figure 14-17: Regression Analysis



Source: Kirkham (2021)

To validate the results, a scatterplot comparison was performed comparing the measured SATMAGAN magnetite and the calculated magnetite as shown in Figure 14-18. This comparison illustrates an excellent correlation coefficient of 0.935 where 1.000 being a perfect correlation.

Figure 14-18: Validation of Regression – SATMAGAN Magnetite vs Regression Calculated Magnetite



Source: Kirkham (2021)

14.13 Resource Validation

A graphical validation was completed on the block model. This type of validation serves the following purposes:

- Checks the reasonableness of the estimated grades based on the estimation plan and the nearby composites;
- Checks that the general drift and the local grade trends compare to the drift and local grade trends of the composites;
- Ensures that all blocks in the core of the deposit have been estimated;

- Checks that topography has been properly accounted for;
- Checks against manual approximate estimates of tonnages to determine reasonableness; and
- Inspects for and explains potentially high-grade block estimates in the neighborhood of the extremely high assays.

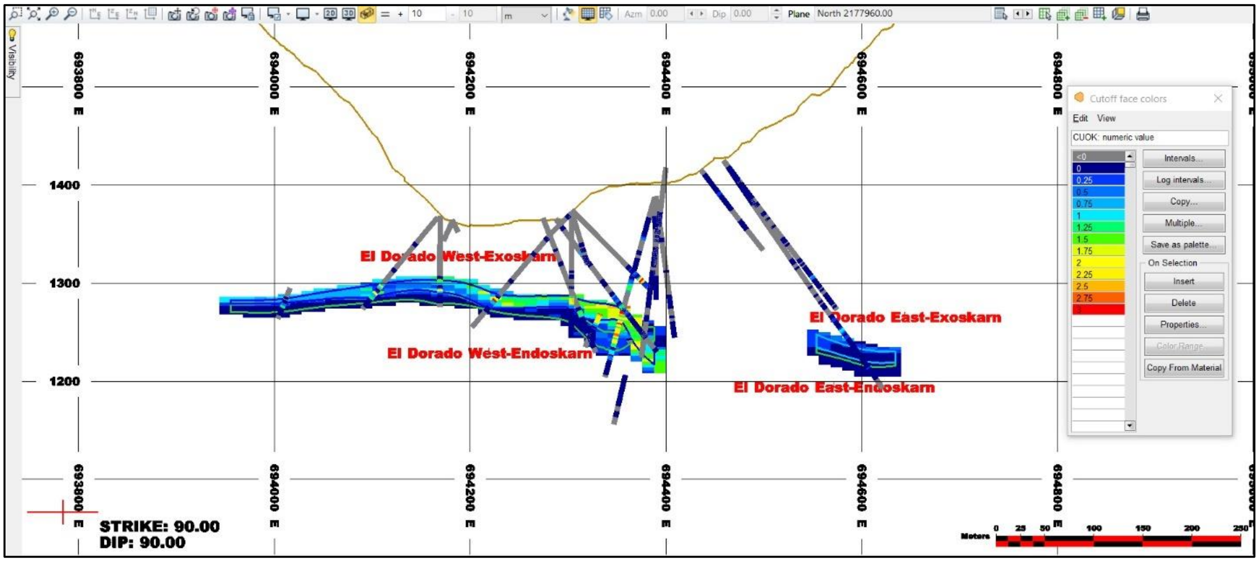
A full set of cross sections, long sections and plans showing the block grades and composites were used to digitally check the block model. There was no indication that a block was wrongly estimated, and it appears that every block grade could be explained as a function of the surrounding composites and the applied estimation plan.

The validation techniques included the following:

- Visual inspections on a section-by-section and plan-by-plan basis;
- Use of grade-tonnage curves;
- Swath plots comparing kriged estimated block grades with inverse distance and nearest neighbour estimates; and
- Inspection of histograms showing distance from first composite to nearest block, and average distance to blocks for all composites (this gives a quantitative measure of confidence that blocks are adequately informed in addition to assisting in the classification of resources).

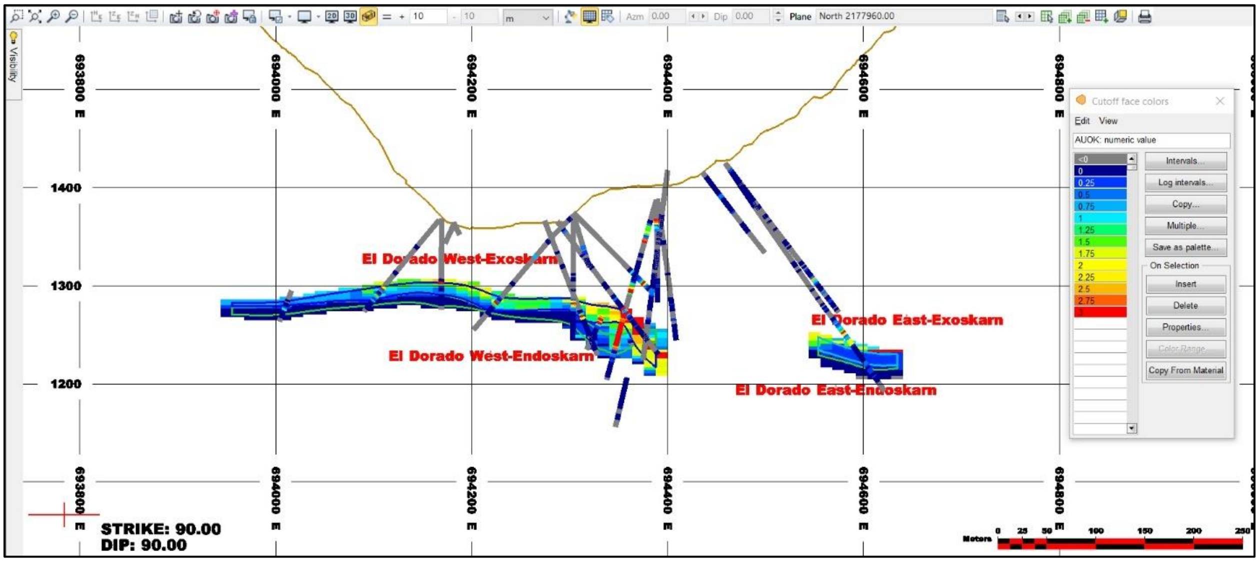
Figure 14-19 through Figure 14-22 illustrates the validated block model shown with composited drillholes and grades for Cu and Au, respectively.

Figure 14-19: Section at 2177960n of Cu% Block Model with Drillholes


Azm 0.00
Dip 0.00
Plane North 2177960.00
CUOK: numeric value

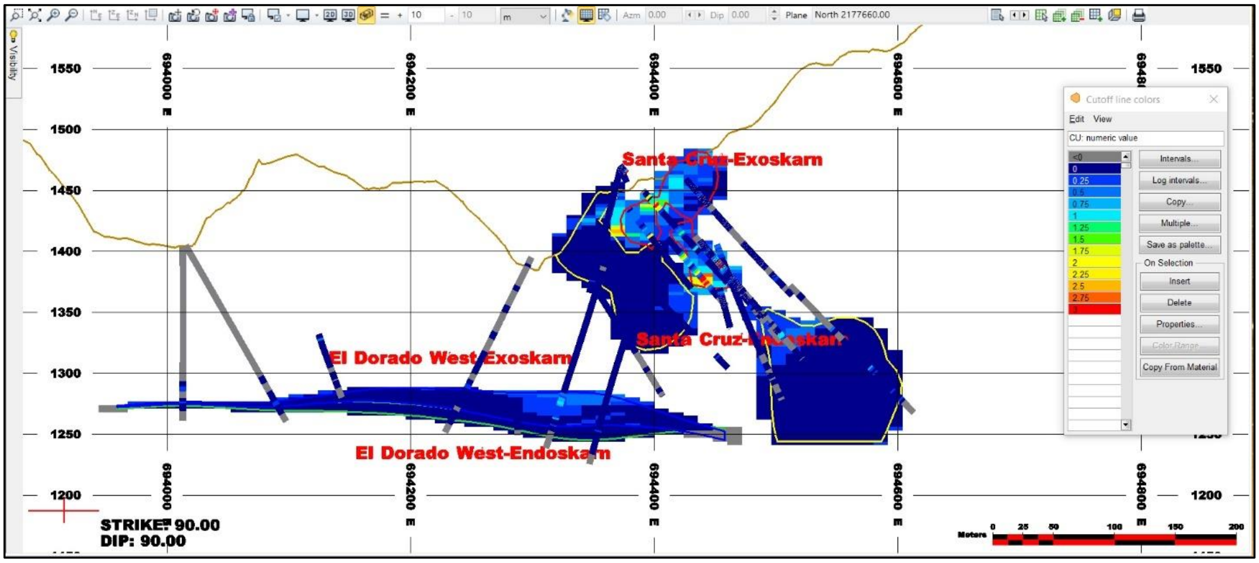
Source: Kirkham (2021)

Figure 14-20: Section at 2177960n of Au g/t Block Model with Drillholes


Azm 0.00
Dip 0.00
Plane North 2177960.00
AUOK: numeric value

Source: Kirkham (2021)

Figure 14-21: Section at 2177660n of Cu% Block Model with Drillholes

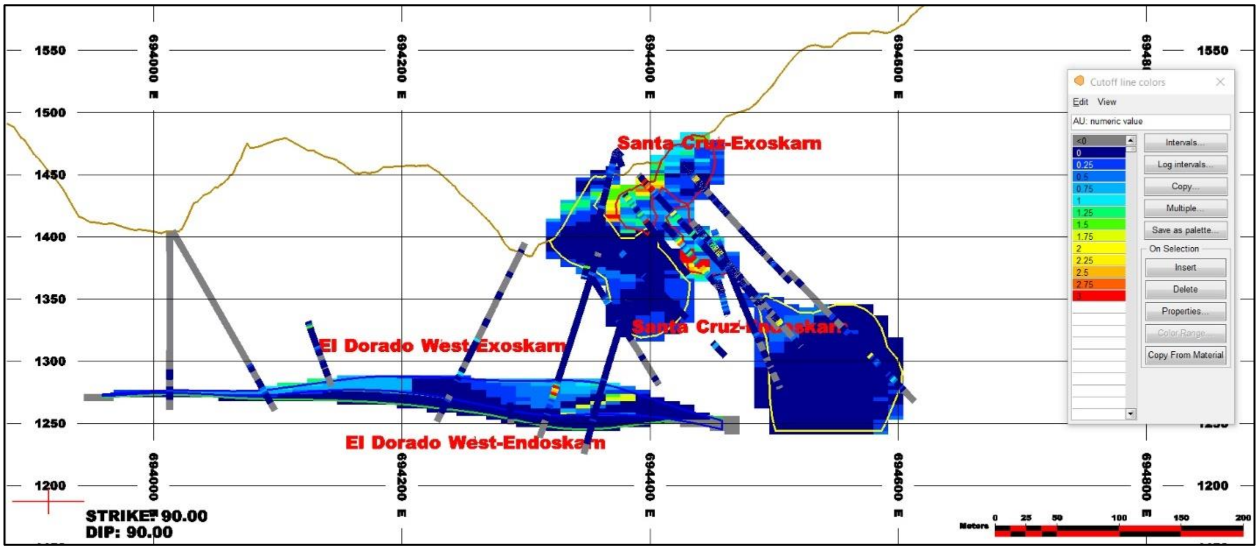


Azm 0.00 Dip 0.00 Plane North 2177660.00

CU: numeric value

Source: Kirkham (2021)

Figure 14-22: Section at 2177660n of Au g/t Block Model with Drillholes



AU: numeric value

Source: Kirkham (2021)

14.14 Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines* (2019). Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Mineral resources for the Las Minas deposit were classified according to the *CIM Definition Standards for Mineral Resources and Mineral Reserves* (2014) by Garth Kirkham, P.Geo., an “independent qualified person” as defined by National Instrument 43-101.

Drillhole spacing in the Las Minas deposit is sufficient for preliminary geostatistical analysis and evaluating spatial grade variability. The QP is, therefore, of the opinion that the amount of sample data is adequate to demonstrate very good confidence in the grade estimates for the deposit.

The estimated blocks were classified according to the following:

- Confidence in interpretation of the mineralized zones;
- Number of data used to estimate a block;
- Number of composites allowed per drillhole; and
- Distance to nearest composite used to estimate a block.

The classification of resources was based primarily on distance to the nearest composite; however, all of the quantitative measures, as listed here, were inspected and taken into consideration. In addition, the classification of resources for each zone was considered individually by virtue of their relative depth from surface and the ability to derive meaningful geostatistical results.

Mineral Resources are classified under the categories of Indicated and Inferred according to CIM guidelines. Mineral Resource classification was based primarily on drillhole spacing and on continuity of mineralization. There are no measured resources at Las Minas. Indicated resources were defined as blocks with a distance to three drillholes of less than ~30 m to nearest composite and occurring within the estimation. Inferred resources were defined as those with a drillhole spacing of less than ~60 m.

Blocks were classified as Indicated if they were within approximately 30 m of a composite and were interpolated with a minimum of two drillholes. Note: There were no blocks classified as Measured resources. Blocks were classified as Inferred if the nearest composite was less than 60 m from the block being estimated. Furthermore, an interpreted boundary was created for the Indicated and inferred threshold in order to exclude orphans and reduce “spotted dog” effect. The remaining blocks were unclassified and may be considered as geologic potential for further exploration.

14.15 Cut-off Grade and RPEEE Evaluation

The mineral resources are as per the requirement to report resource that have a "Reasonable Prospect of Eventual Economic Extraction (RPEEE)". To this end, an appropriate, realistic cut-off grade was selected based on reasonable operating costs as shown Table 14-7.

Table 14-7: NSR Cut-off Grade Calculation

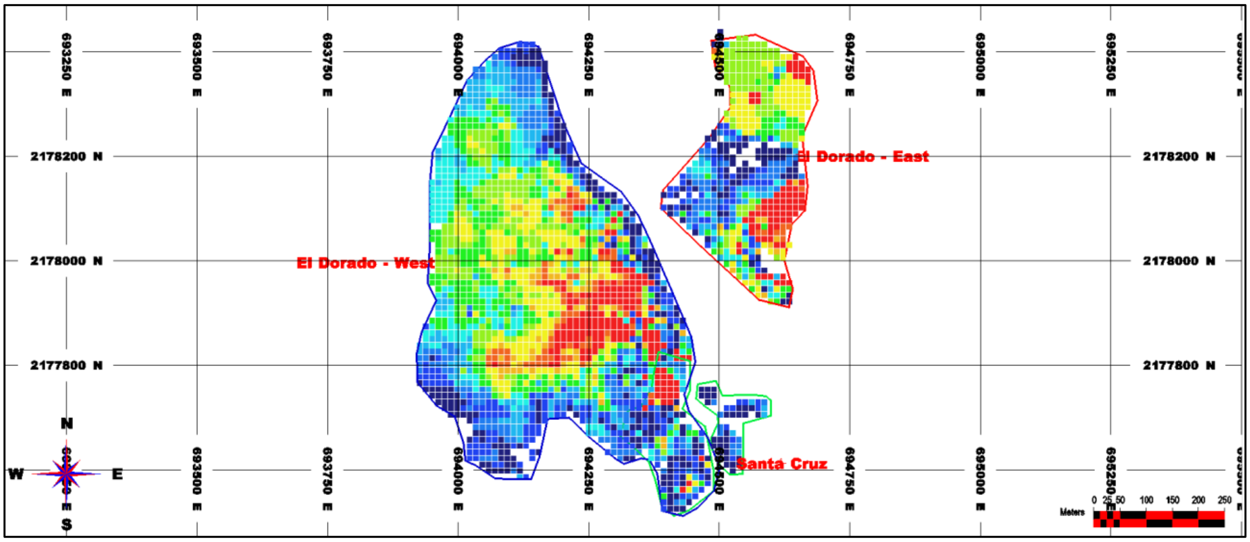
Cut-off Grade	Unit	2021 Resources
On Site Costs		
Mining Cost (UG)	US$/t milled	22.00
Milling Cost	US$/t milled	18.17
G&A Cost	US$/t milled	9.00
Sustaining CAPEX	US$/t milled	12.50
Total Site Costs (UG)	**US$/t milled**	**61.67**
Mining Losses and Dilution		
Mining Recovery (UG)	%	95%
Dilution (UG)	%	20%
Cut-off Grade		
Insitu Cut-off NSR (w/ dilution)	US$	**$78.53**

Source: Kirkham (2021)

Note that the base case resource cut-off grade takes into account the mineable resource cut-off calculation presented in Section 16.4.3 which has been calculated to be NSR$90 and adjusted to allow for consideration of incremental resources mining cost (i.e., operating costs less some or all of the mining cost) resulting in a base case cut-off of US$80 NSR.

Furthermore, an interpreted boundary (Figure 14-23) was created to ensure continuity by providing approximate mining shapes for the indicated and inferred resources in order to exclude orphans and to insure incremental and 'must-take' resources are not fully excluded.

Figure 14-23: RPEEE NSR Cut-off Grades



Source: Kirkham (2021)

14.16 Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 14-8 shows the total resources for all metals at varying NSR cut-off thresholds. The reader is cautioned that these values should not be misconstrued as a mineral reserve. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grades.

Note that the base case cut-off grades presented in Table 14-8 are based on potentially underground, mineable resources at the base case of US$80 NSR.

Table 14-8: Sensitivity of Las Minas Mineral Resources

Class	NSR COG (US$)	Tonnes	NSR (US$)	Au (g/t)	Au ('000 ounces)	Ag (g/t)	Ag ('000 ounces)	Cu (%)	Cu ('000 lbs)	Fe Magnetite (%)	Fe Magnetite ('000 tonnes)	AuEQ (g/t)	AuEq ('000 ounces)
Indicated	>=60	5,431	122.00	1.71	299	4.27	746	0.95	114,341	13.84	752	2.94	514
	>=70	4,750	130.25	1.83	280	4.44	678	1.02	106,373	14.35	682	3.14	479
	>=80	**4,133**	**138.58**	**1.96**	**260**	**4.64**	**617**	**1.08**	**98,311**	**14.77**	**610**	**3.34**	**443**
	>=90	3,549	147.47	2.09	239	4.87	555	1.14	89,467	15.31	543	3.55	405
	>=100	3,009	156.99	2.24	217	5.12	495	1.21	80,326	16.19	487	3.77	365
	>=110	2,572	165.96	2.38	197	5.36	444	1.27	72,146	16.86	434	3.98	329

Class	NSR COG (US$)	Tonnes	NSR (US$)	Au (g/t)	Au ('000 ounces)	Ag (g/t)	Ag ('000 ounces)	Cu (%)	Cu ('000 lbs)	Fe Magnetite (%)	Fe Magnetite ('000 tonnes)	AuEQ (g/t)	AuEq ('000 ounces)
Inferred	>=60	6,769	102.84	1.32	287	5.49	1,195	0.86	128,586	16.23	1,099	1.97	428
	>=70	6,012	107.69	1.38	266	5.73	1,108	0.91	119,959	16.95	1,019	2.06	398
	>=80	**5,200**	**112.83**	**1.44**	**241**	**5.97**	**997**	**0.95**	**108,802**	**17.54**	**912**	**2.16**	**361**
	>=90	4,228	119.33	1.54	209	6.19	842	1.00	93,057	18.00	761	2.29	311
	>=100	3,226	127.04	1.67	173	6.44	668	1.05	74,354	18.24	589	2.44	253
	>=110	2,106	138.88	1.84	125	7.07	479	1.14	52,930	18.42	388	2.66	180

Notes:

1. Mineral Resource Statement prepared by Garth Kirkham (Kirkham Geosystems Ltd.) in accordance with NI 43-101.
2. Effective date: September 18, 2021. All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under NI 43-101.
3. Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under NI 43-101. Mineral resources are not Mineral Reserves and do not have demonstrated economic viability.
4. Underground Mineral Resources are reported at a cut-off grade of US$80 NSR. Cut-off grades are based on a price of US$1,700/oz gold, US$20/oz silver, US$3.50/lb copper and US$100/t magnetite concentrate and a number of operating cost and recovery assumptions, including a reasonable contingency factor.
5. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
6. The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.
7. Mineral resources are not mineral reserves until they have demonstrated economic viability. Mineral resource estimates do not account for a resource's mineability, selectivity, mining loss, or dilution.
8. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.

Source: Kirkham (2021)

14.17 Mineral Resource Statement

Table 14-9 shows the Mineral Resource Statement for the Las Minas deposit.

The author evaluated the resource in order to ensure that it meets the condition of "reasonable prospects of eventual economic extraction" as suggested under NI 43-101. The criteria considered were confidence, continuity and economic cut-off. The resource listed below is considered to have "reasonable prospects of eventual economic extraction".

The Mineral Resource Estimate which updates the previously reported October 2019 estimate, incorporates data from new drilling conducted in 2020-2021 that successfully delineated a new deposit on the project and increased the resource base in the Inferred Resource category.

Table 14-9: Las Minas Deposit Indicated and Inferred Mineral Resource Estimate at US$80 NSR Cut-off

Class	Tonnes	NSR (US$)	Au (g/t)	Au ('000 ounces)	Ag (g/t)	Ag ('000 ounces)	Cu (%)	Cu ('000 lbs)	Fe Magnetite (%)	Fe Magnetite ('000 tonnes)	AuEQ (g/t)	AuEq ('000 ounces)
Indicated	4,133	138.58	1.96	260	4.64	617	1.08	98,311	14.77	610	3.34	443
Inferred	5,200	112.83	1.44	241	5.97	997	0.95	108,802	17.54	912	2.16	361

Notes:

1. Mineral Resource Statement prepared by Garth Kirkham (Kirkham Geosystems Ltd.) in accordance with NI 43-101.
2. Effective date: September 18, 2021. All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under NI 43-101.
3. Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under NI 43-101. Mineral resources are not Mineral Reserves and do not have demonstrated economic viability.
4. Underground Mineral Resources are reported at a cut-off grade of US$80 NSR. Cut-off grades are based on a price of US$1,700/oz gold, US$20/oz silver, US$3.50/lb copper and US$100/t magnetite concentrate and a number of operating cost and recovery assumptions, including a reasonable contingency factor.
5. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
6. The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.
7. Mineral resources are not mineral reserves until they have demonstrated economic viability. Mineral resource estimates do not account for a resource's mineability, selectivity, mining loss, or dilution.
8. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.

Source: Kirkham (2021)

14.18 Comparison to 2019 Resource Estimation

The following is a comparison between the previous resource estimate performed in 2019 and the current estimate as stated within this report. Table 14-10 shows the total Indicated and Inferred resources stated in 2019 as compared to the current Indicated and Inferred resources stated herein. As shown, Indicated resources have decreased, and Inferred resources have increased between the two estimates.

The 2020 - 2021 drilling campaigns contributed to the revisions and these are the reasons for the change in resources. These activities were focused on developing a better understanding of geology and structure, more accurately defining the mineralized zones both in the and lithology units, revising the models and domains, targeting of additional resources.

The significant differences from the 2019 Resource Estimate (Tietz 2019) and the current 2021 Resource Estimate are as follows:

- The addition of the 2020 and 2021 Drilling;

- Revised Domains and Lithology Solids;
- Revised drillholes selection criteria;
- Revised estimation methodology and parameters;
- Cut-off – of $80 NSR in 2021 vs 1.5 g/t AuEq in 2019; and
- Classification schema is based on drill spacing and current CIM best practice guidelines.

Table 14-10 shows that there is a 24% decrease in indicated resources whilst there is a significant increase on 107% in inferred resources.

Table 14-10: 2019 Mineral Resources Compared to the 2021 Base Case Mineral Resources

Year	Class	Tonnes	NSR ($)	Au (g/t)	Au ('000 ounces)	Ag (g/t)	Ag ('000 ounces)	Cu (%)	Cu ('000 lbs)	Fe Magnetite (%)	Fe Magnetite ('000 tonnes)	AuEQ (g/t)	AuEq ('000 ounces)
2019*	Indicated	5,457		1.78	313	6.5	1,148	1.25	150,319			3.67	645
	Inferred	2,514		1.25	101	5.5	446	0.94	51,965			2.68	217
2021**	Indicated	4,133	138.58	1.96	260	4.64	617	1.08	98,311	14.77	610	3.34	443
	Inferred	5,200	112.83	1.44	241	5.97	997	0.95	108,802	17.54	912	2.16	361
Difference	**Indicated**	**-24%**		**10%**	**-17%**	**-29%**	**-46%**	**-14%**	**-35%**			**-9%**	**-31%**
	Inferred	**107%**		**15%**	**139%**	**8%**	**124%**	**1%**	**109%**			**-19%**	**66%**

Source: Tietz (2019)*; Kirkham (2021)**

15 MINERAL RESERVE ESTIMATE

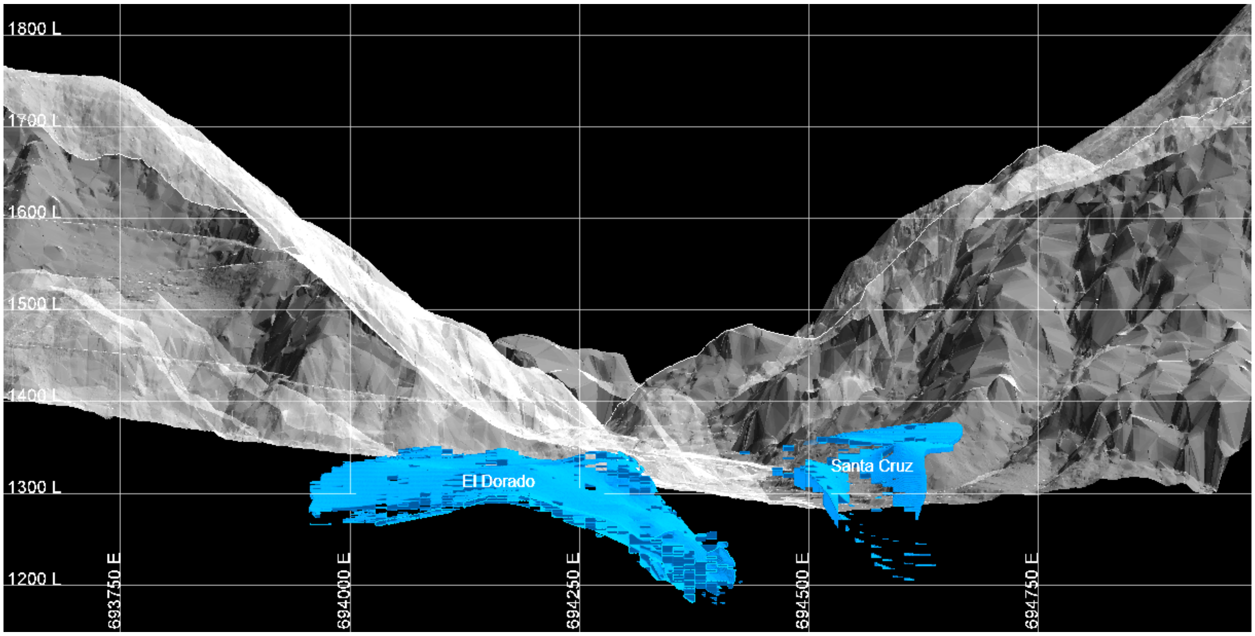
Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary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

16 MINING METHODS

16.1 Introduction

Mineralization consists of two deposits: El Dorado and Santa Cruz (Figure 16-1). Due to its proximity to surface, the El Dorado East Zone will be targeted early in mine life. This material will be mined via Room and Pillar (RPL) methods using three access points and will be hauled directly to the processing plant using underground haul trucks.

Figure 16-1: Isometric View of Mineralized Zones - Looking North



Prior to completion of Zone A, the Main Portal is developed to recover the deeper El Dorado West (Zones C, D, and E) material. Santa Cruz (Zone B) is developed using internal ramping, mined via longhole stoping methods, and hauled to the crusher. This deposit will be mined using a combination of Long-Hole Stoping (LHS) and room and pillar methods. All mineralized material will be hauled via truck to an underground crusher at 1367L, where it will be crushed and conveyed to the processing plant at surface. For LHS, levels will be located throughout the mine at 20 m vertical increments which will be connected by the main ramp sized at 5.0 m W x 5.0 m H. Where the deposit becomes shallow dipping, RPL is utilized to recover mineralized material. These areas will be accessed utilizing existing LHS development.

Longhole stopes will be backfilled utilizing a cemented paste backfill comprised of tailings which will be pumped from surface. Run of mine material will be backfilled where possible, particularly in empty room and pillar voids, to avoid additional haulage cost to place at surface.

16.2 Geotechnical Analysis and Recommendations

16.2.1 Rock Mass Characterization

Geotechnical specific drilling and testing programs have not yet been carried out for the site. As such, core photographs for a total of 19 resource drillholes (15 from West deposit, 2 at East deposit and 2 at SC deposit) were reviewed to assess ground conditions across the mine area. High level estimates of rock mass quality were made according to the Barton Q' rock mass rating system (Barton & Grimstad, 1994) for the immediate hanging wall, footwall and mineralized material zones. The estimates were made using available RQD data where possible and visual estimates of the number of joint sets (Jn). Reasonably conservative estimates of joint condition parameters (Jr and Ja) were made based on the core photographs and experience in similar geologic environments.

Overall, the hanging wall, footwall and mineralized material zones all appear to be of good geomechanical quality with high intact rock strengths and low fracture frequencies. There are no major fault structures known or interpreted within the deposits; however, drillhole LM-14-JB-03 appears to have intersected a significant fault at a distance of approximately 20 m above the hanging wall. Additional interpretation of the potential structure should be carried out as the project advances.

Rock Quality Designation (RQD) values typically ranged from 75 to 95 (20th and 80th percentiles, respectively) with a mean value of 85, classifying mostly as 'Good' to Excellent rock mass quality according to Deere (1989). Rock hardness was logged according to ISRM (1978) guidelines typically ranging between R4 and R5 indicating Strong to Very Strong intact rock strengths.

Based on the Barton (2002) Q-system rock mass rating system, the overall rock mass quality classifies as 'Fair' to 'Good' with Q' values ranging between 5.9 and 15.8 (20th and 80th percentiles, respectively) and a mean value of 11.5. The estimates of Q' for each zone are summarized in Table 16-1.

Table 16-1: Summary of Rock Mass Quality Data

Logging Zone	Core Runs	Avg. Core Recovery %	Avg. RQD %	Q'			
				20%	Avg.	75%	Classification
Hanging Wall	74	96	83	6.1	15.3	22.5	Fair to Good
Ore Zone	98	97	90	6.7	10.6	15.3	Fair to Good
Footwall	69	97	84	4.0	7.9	10.5	Fair to Good
All Runs Combined	271	97	85	5.5	11.3	15.8	Fair to Good

16.2.2 Stability Analysis

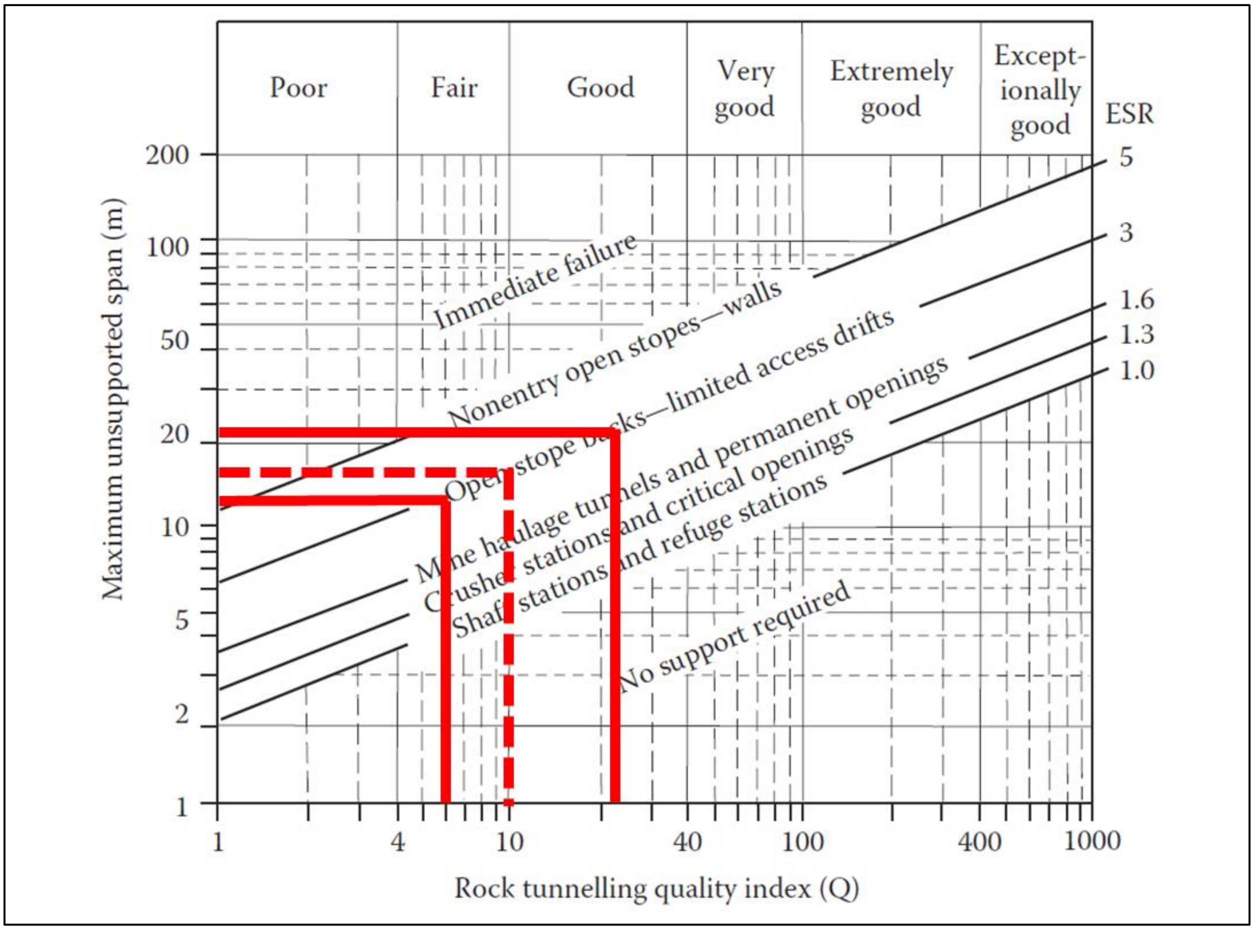
Mining of the Las Minas deposit will be done with a combination of room and pillar for flatter portions and transverse long hole stoping for the steeper portions. Given the deposit is shallow, minimal waste rock is available for backfilling long hole stopes. The steep natural terrain in the area limits potential for tailings storage on surface. As such, tailings will be placed back underground as cemented paste and the long hole stopes will be mined in a primary/secondary sequence.

16.2.2.1 Room and Pillar

The minimum room and pillar mining height will be 4 m which represents approximately 40% of all the room and pillar stopes. The maximum height is 21 m with stopes greater than 10 m in height representing only 10% of all room and pillar stopes. Where over 6 m in height, mining will be done in multiple passes working off waste rock backfill similar to post pillar cut and fill. The average room and pillar height is 6.7 m.

The maximum stable unsupported span for stope backs was checked first using the Q-system diagram developed by Hutchinson & Diederichs (1996). As shown in Figure 16-2, unsupported spans of 10 m to approximately 20 m are anticipated to be stable for the anticipated lower and upper bound Q’ values. The average Q’ value of 11.5 would be anticipated to allow an unsupported (non-entry) span of approximately 18 m based on Figure 16-3.

Figure 16-2: Unsupported Span Limits



Source: Hutchinson & Diederichs (1996)

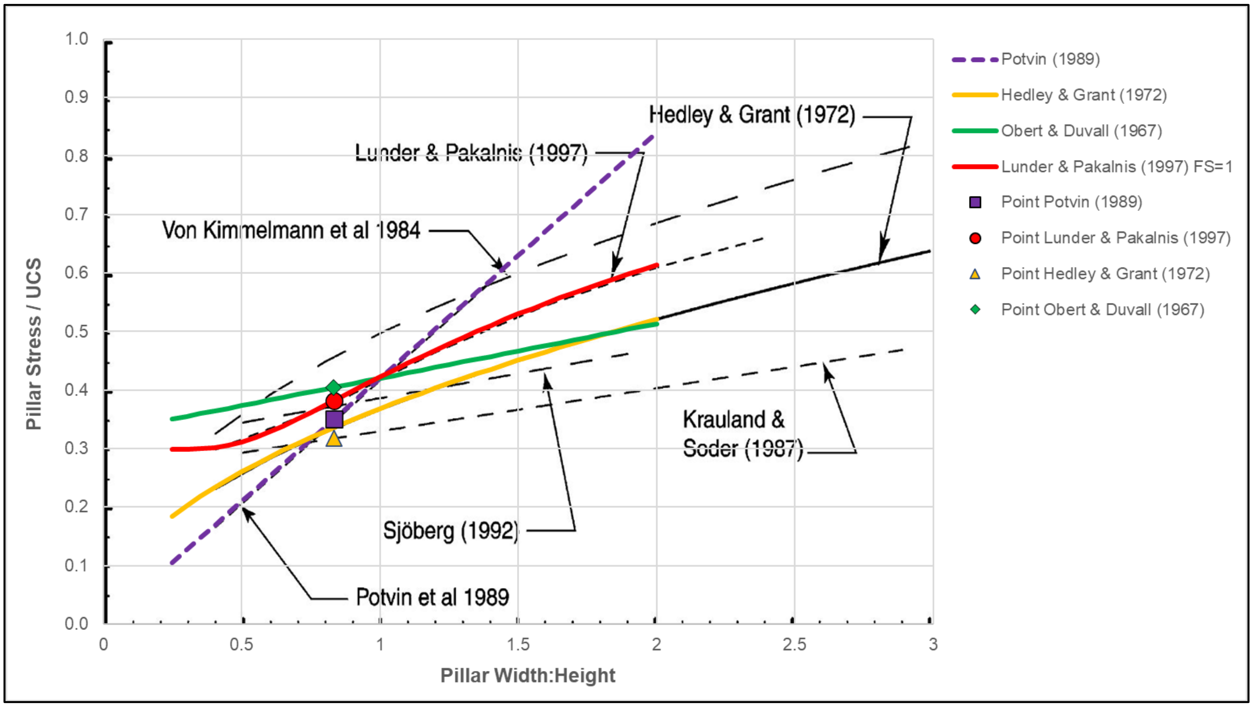
The tributary area method of analysis was subsequently used to confirm pillar stability based on a 7 m room (10 m maximum span measured diagonally). Tributary area pillar stability analysis is most applicable to large and deeper mining areas with consistent pillar and stope dimensions. For smaller mining areas such as the mining proposed at Las Minas, vertical stresses from the rock mass above tend to arch over the area thus the pillars do not experience the full overburden load as is assumed by the tributary area analysis method. As the mining area increases, pillar loads typically increase towards the tributary area calculated values.

A vertical pillar stress of approximately 23 MPa was calculated using tributary area for 5 m x 5 m pillars, 7 m rooms and a depth of 150 m below ground surface. The extent of the planned mining area suggests that most of the proposed mining areas are small enough that some level of stress arching will likely occur. As such, the results are considered to be conservative.

The strength of slender pillars (width-to-height ratios < 0.8) can be estimated using empirically developed equations; however, variable results can be produced when the width-to-height ratios approach 0.5 and below. Some of the commonly used empirical pillar stability equations are plotted on Figure 16-3 along with the pillar stresses and width-to-height ratios calculated from the Las Minas analyses.

Based on a conservatively assumed pillar UCS of 75 MPa, the room width may be 7 m with 5 m x 5 m pillars at the maximum 6 m height. The results of the empirical analyses indicate safety factors ranging from 1.1 (Potvin 1989) up to 1.5 (Hedley & Grant 1972). Mining of rooms greater than 6 m in height will be done with multiple passes, working off the top of waste rock or cemented paste backfill. Backfill will provide confinement on lower portions of pillars greater than the 6 m in height increasing their strength/stability. Pillars should not be greater than 6 m in height without backfilling.

Figure 16-3: Slender Pillar Stability Chart



Source: Martin & Maybee (2000)

16.2.2.2 Transverse Longhole Stope Dimensions

Empirical stope design analyses are based on stability graphs where the stability number (N') is plotted on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the stope face on the horizontal axis. The stability number is calculated based on the rock mass quality (Q'), and three empirical factors: A (induced stress conditions), B (geologic structure orientation) and C (dip angle of the stope face).

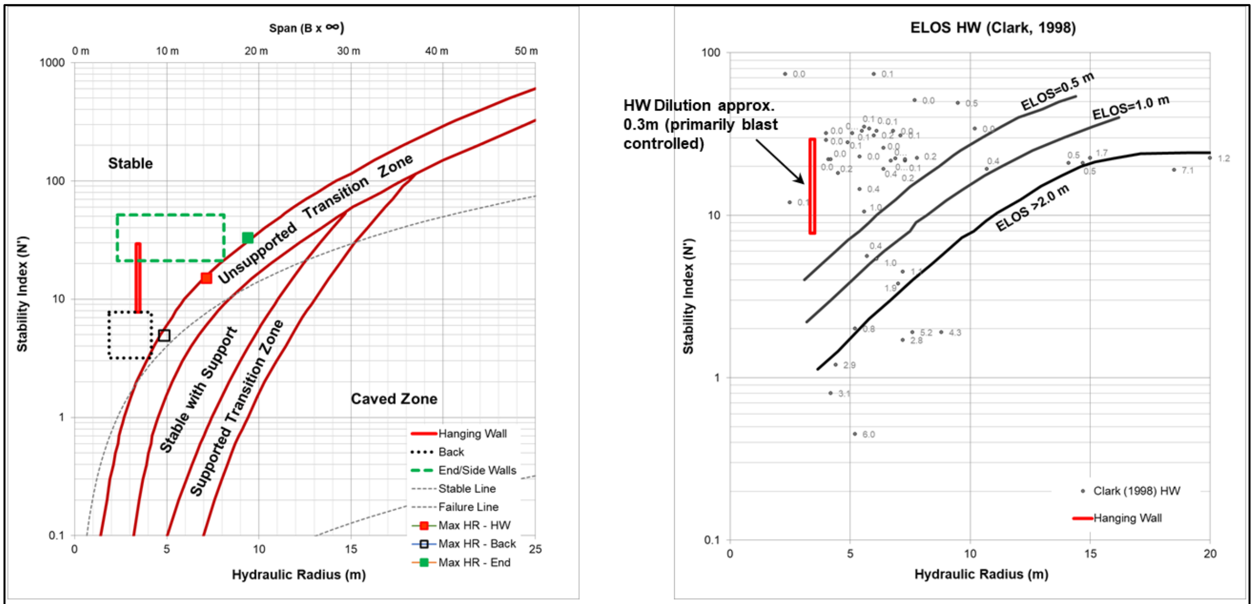
Maximum stope dimensions were estimated using the Potvin (2001) method for the anticipated range of rock mass conditions and stope sizes. The Trueman & Mawdesley (2003) 'Stable' line was also used as a second check against the Potvin (2001) results. Upper and lower bound estimates of stope face dimensions and rock quality (Q') were analyzed.

Empirical factors for calculation of the stability number (N') were based on the following assumptions:

- Induced stress parameter, A equal to 1 given the relatively strong UCS (75 to 100 MPa) and low anticipated horizontal stress given the shallow depth of the stoping areas (<260 m bgs);
- Joint orientation factor, B equal to 0.3 for the hanging wall, footwall and back based on the assumption that the dominant discontinuity orientation will be sub-parallel to the hanging wall; and
- Gravity factor, C equal to 5.5 for the hanging wall corresponding to a dip angle of 65º, C equal to 2.0 for the flat stope backs and C of 8 for the vertical stope walls.

Based on a 20 m level spacing and 10 m stope widths (measured parallel to strike) the maximum stable length (measured perpendicular to strike) is recommended to be 50 m. A 5 m minimum thickness pill should be left between any two stopes that are not tightly backfilled. Dilution will be controlled mostly by the quality of blasting practices and to a lesser degree, geotechnical conditions.

Figure 16-4: Empirical Stope Stability and Dilution Charts



Wall	Dip	Q'	Empirical Parameters			N'
			A (Stress)	B (Joint Orientation)	C (Dip Angle)	
HW	65	5.9	0.8	0.3	5.5	7.7
Back	0	6.6	0.8	0.3	2.0	3.2
FW	65	3.8	0.8	0.3	8.0	7.3
End/Side	90	6.6	0.8	0.5	8.0	21.1
HW	65	22.4	0.8	0.3	5.5	29.4
Back	0	16.1	0.8	0.3	2.0	7.7
FW	65	10.3	0.8	0.3	8.0	19.8
End/Side	90	16.1	0.8	0.5	8.0	51.5

Max. and Min. Ranges		
Width	Height	Length
50.0	24.0	10.0
6.0	20.0	10.0

16.3 Mining Methods

The El Dorado deposit is separated into east and west mineralized zones by a roughly 150 m wide diorite dyke. The sub-horizontal skarn on the west side of the dyke has an 800 m northwest strike length, extends roughly 400 m in width and is 10 to 15 m in thickness on average. Towards the dyke, this zone forms an east-dipping keel extending up to 100 m with an average dip between 55 to 60 degrees. The portion of the El Dorado deposit on the east side of the dyke is sub-horizontal and has a strike length of 250 m, a width of 200 m, and is 5 to 10 in thickness.

The Santa Cruz deposit lies near surface, roughly 0.5 km south of the Las Minas pueblo. The deposit is near vertical, has a 200 m northwest strike length and extends up to 200 m down dip.

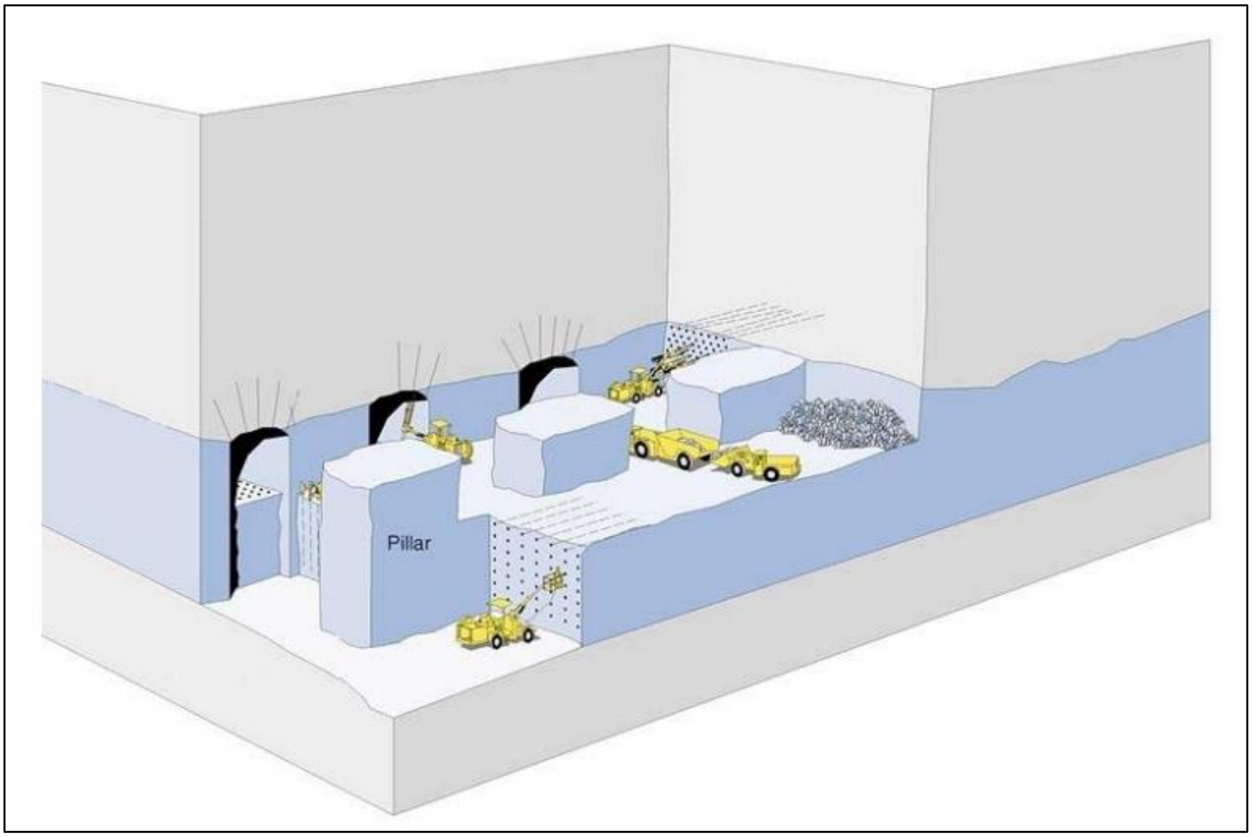
Initially, the deposit was reviewed for both open pit and underground potential. Though the deposit is located near surface, open pit assessment was eliminated due to the steep topography, limited external waste storage areas, and high annual precipitation.

Due to the deposit geometry, it will be developed using both longhole stoping and room and pillar mining. Long hole stopes account for 52% of production tonnes and will be utilized where the deposit is steeply dipping. Where the deposit becomes almost horizontal, room and pillar mining will be utilized as it allows for better extraction and lower backfill requirements. Room and pillar mining accounts for 41% of production tonnes. The remaining 7% of the mill feed tonnes is derived from mineralized development.

16.3.1 Room and Pillar

Room and pillar mining (Figure 16-5) is applicable in the extraction of flat-bedded deposits of limited thickness. This method is used to recover resources in open stopes. Pillars are left behind to support the hanging wall and contain mineralized material which is non-recoverable. Rooms and pillars are arranged in regular patterns. The spans are 7 m and the pillars are 5 m x 5 m. This makes for a mining extraction ratio of 83%.

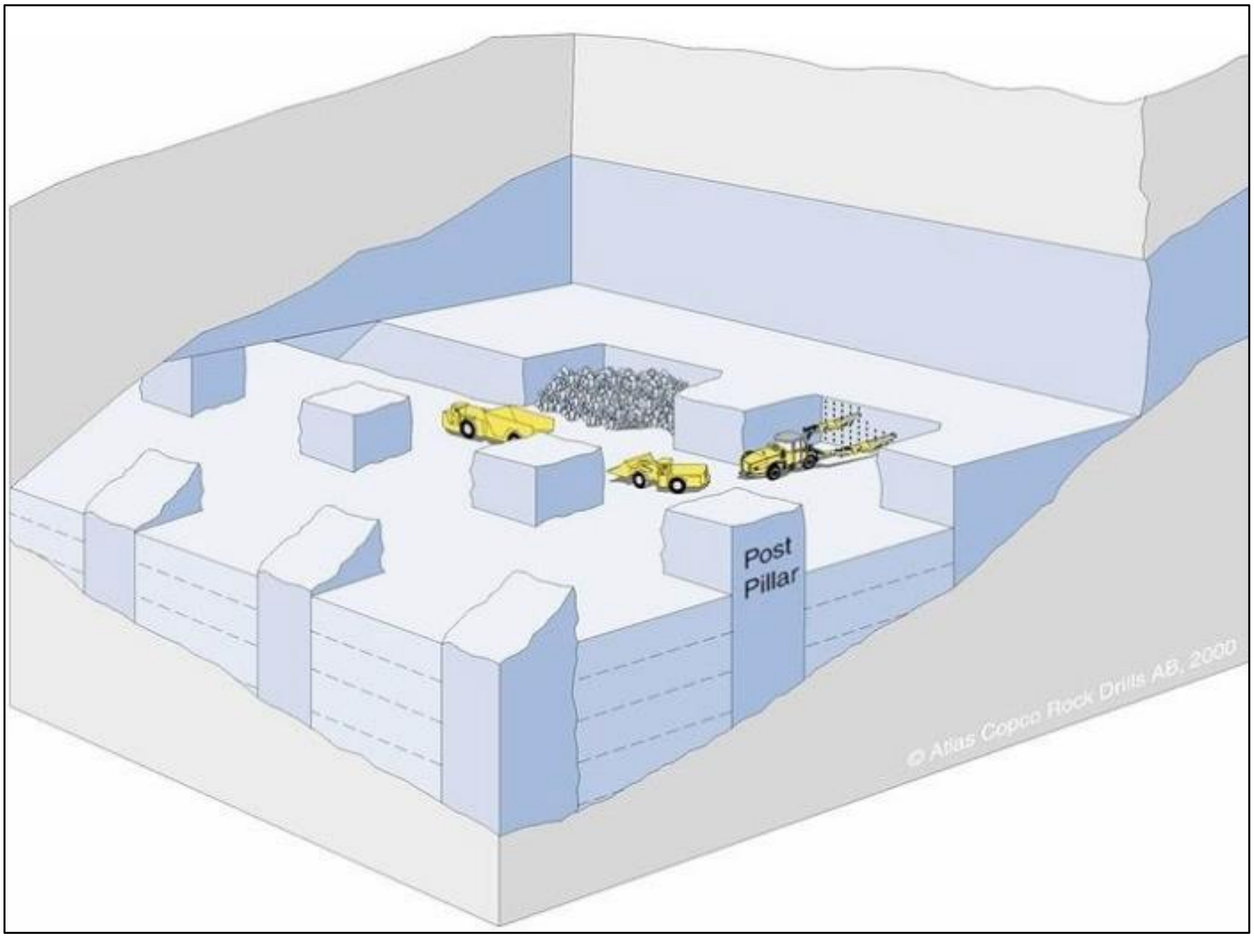
Figure 16-5: Classic Room and Pillar Mining Conceptual Schematic



Source: Atlas Copco (2014)

Rooms do not typically require backfill, but where deposit thickness is greater than 6 m, mining will be completed in multiple passes working off mine rock fill, similar to post room and pillar mining (Figure 16-6). Rooms will be backfilled using a combination of waste development material and lightly cemented tailings. The tailings are cemented to prevent movement of material. The waste rock is stored underground to decrease the surface impact and decrease the size of the tailings storage facility.

Figure 16-6: Post Room and Pillar Mining Conceptual Schematic



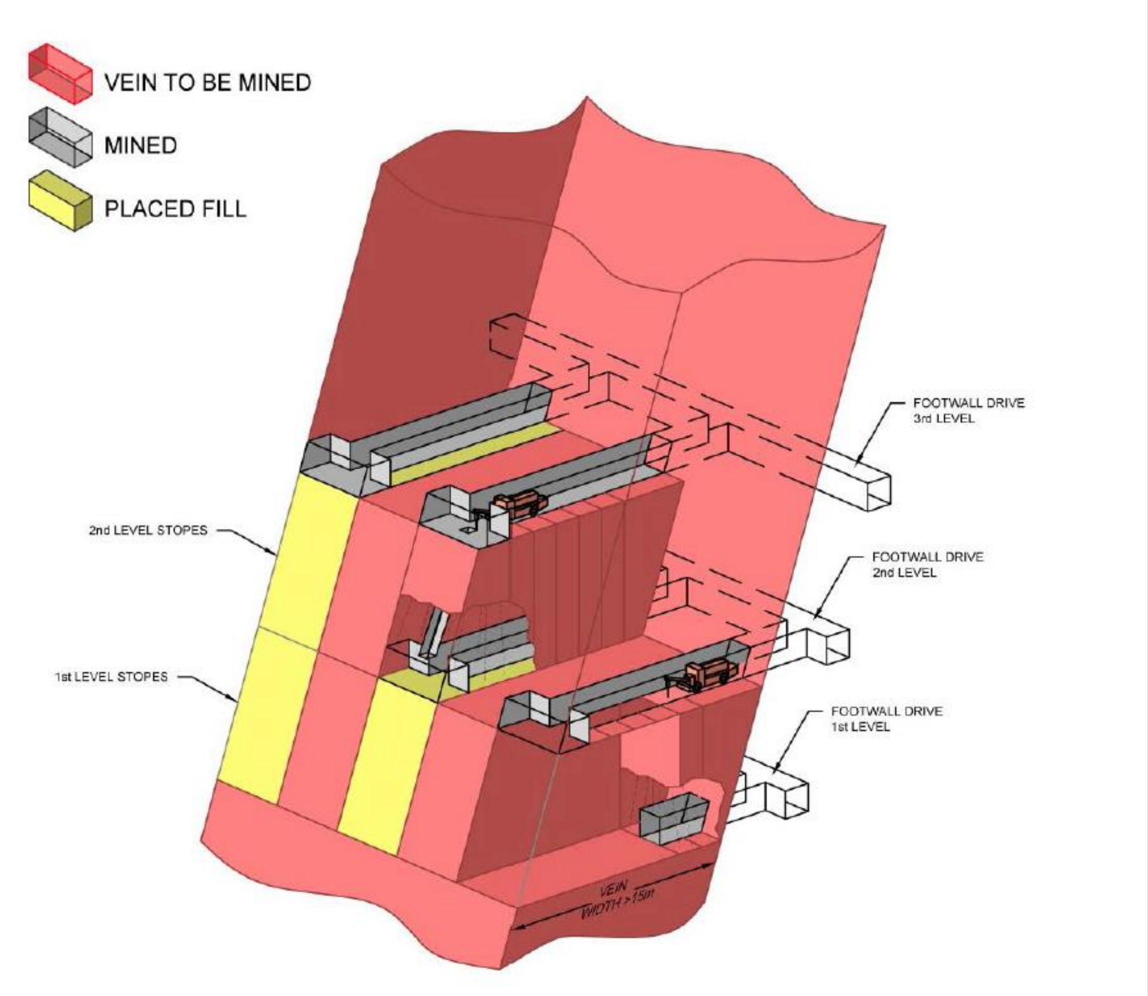
Source: Atlas Copco (2014)

16.3.2 Longhole Stoping

Longhole stoping (Figure 16-7) is utilized in steeply dipping deposits. A top and bottom drift delineate the stope and blast holes are drilled between the two levels using a Longhole Drill. The stope is blasted, and material is extracted from the bottom drift by conventional remote mucking with LHDs.

Stopes are mined in a primary/secondary fashion where primary stopes are mined and backfilled with a cemented paste. Secondary stopes are also backfilled using lightly cemented paste and waste rock. Footwall access is driven parallel to the resource with cross-cut entries evenly spaced along strike. This allows for increased productivity and the ability to mine several stopes at a given time.

Figure 16-7: Longhole Stoping Conceptual Schematic



16.4 Mine Design Parameters

To determine mineable shapes, the following design process was utilized:

- Analyze the geologic resource model for geometric properties including mineralized zone width, depth, length, and continuity;
- Select the mining methods best suited for the deposit based on geometry, economics, and geotechnical parameters;
- Determine the economic cut-off grade (COG) based on estimated operating cost, mine recovery, dilution, and commodity price assumptions;
- Identify the blocks within the geologic model that are above COG (including mine dilution and recovery), and produce optimized stope shapes; and
- Develop a mine plan around the economically viable production stope and complete economic analysis.

These steps are discussed in detail in the sections below.

16.4.1 Dilution and Mine Recovery

Assumptions for mine dilution and recovery are presented in Table 16-2. Dilution varies by mining method. Design dilution is derived from overbreak, while the source of mine dilution is due to mucking or backfill. Design recovery refers to mineralized material left behind as pillars.

Table 16-2: Mine Dilution and Recovery Inputs

Mining Method	Dilution			Recovery		
	Design	Mine	Net	Design	Mucking	Net
Longhole Stope	33%	10%	43%	100%	95%	95%
Room & Pillar	5%	0%	5%	83%	95%	79%
Development	14%	0%	14%	100%	95%	95%

16.4.2 Net Smelter Return

As the deposit is polymetallic, producing a copper, gold, silver and a magnetite concentrate, economic analysis was conducted using a calculated Net Smelter Return (NSR). Inputs for the NSR calculation are included in Table 16-3. NSR was calculated for each block and stored within the 3D block model for evaluation.

Table 16-3: NSR Input Parameters

Parameter	Unit	Las Minas
COPPER CONCENTRATE		
Metal Prices		
Cu Price	US$/lb	3.25
Au Price	US$/oz	1,625
Ag Price	US$/oz	20.00
Exchange Rate	C$:US$	0.76
Royalties	% NSR	0.0
Recovery		
Copper Concentrate		
Cu Recovery	%	90.0
Au Recovery	%	80.0
Ag Recovery	%	70.0
Concentrate Grade		
Copper Concentrate		
Cu	%	21.7
Au	g/t	-
Ag	g/t	-
Moisture Content	%	8
Magnetite Concentrate		
Magnetite	% Magnetite	90.0
Smelter Payables		
Cu Payable	%	96.1
Min. Cu deduction	% Cu/tonne	1.1
Au Payable	%	97.2
Min. Au deduction	g/t concentrate	1
Ag Payable	%	60.9
Min. Ag deduction	g/t concentrate	30
Treatment & Refining Costs		
Cu TC	US$/dmt con	80.00
Cu RC	US$/payable lb	0.08
Au RC	US$/payable oz	5.00
Ag RC	US$/payable oz	0.40
Transport Costs		
Transport Costs	US$/wmt	56.20
Total Transport to Smelter	US$/dmt	61.09

Parameter	Unit	Las Minas
MAGNETITE CONCENTRATE		
Magnetite Price (Iron Smelter Feed)	US$/tonne	100.00
Recovery		
Magnetite Feed Reporting to Tailings	%	94.5
Rougher Mag Separator Recovery	%	97.0
Estimated Magnetite Mineralized Material Recovery	%	98.5
Overall Magnetite Recovery	%	90.3
Concentrate Grade		
Magnetite	% Magnetite	90.0
Transport Costs		
Transportation Cost	US$/$t_{con}$	61.09

16.4.3 Cut-off Grade

Economic stope material is identified from the 3D block model utilizing Maptek Vulcan™ Stope Optimizer software. NSR cut-off was determined based on parameters contained in Table 16-4.

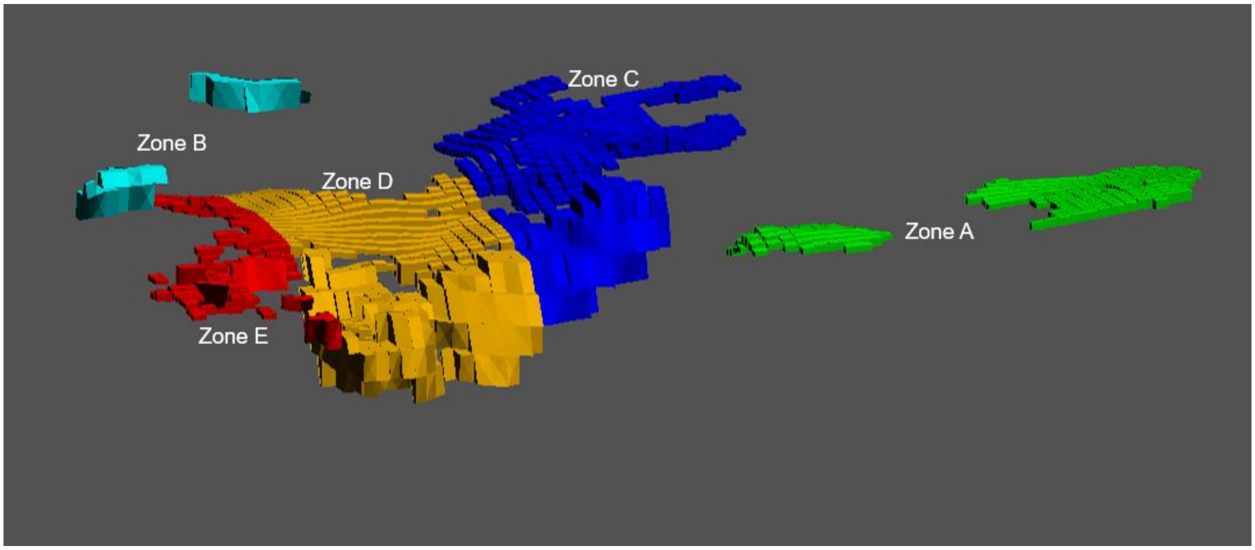
Table 16-4: Cut-off Grade Parameters

Estimated Operating Cost	Units	Cost
Underground Mining Cost	$/t	38.00
Processing Cost	$/t	18.17
G&A Cost	$/t	9.00
Sustaining Cost	$/t	12.50
Total	**$/t**	**77.67**
Mine Losses and Dilution		
Mucking Recovery	%	95
Mucking Dilution	%	10
Calculated COG		
In-situ NSR Cut-off Including Dilution	$/t	88.52

Notes:

Operating parameters may differ from those used in the economic model due to subsequent, more detailed estimation work. These differences are not considered to have a material impact on PEA results.

Figure 16-8: Oblique View - Mining Zone Definition – Looking Northwest

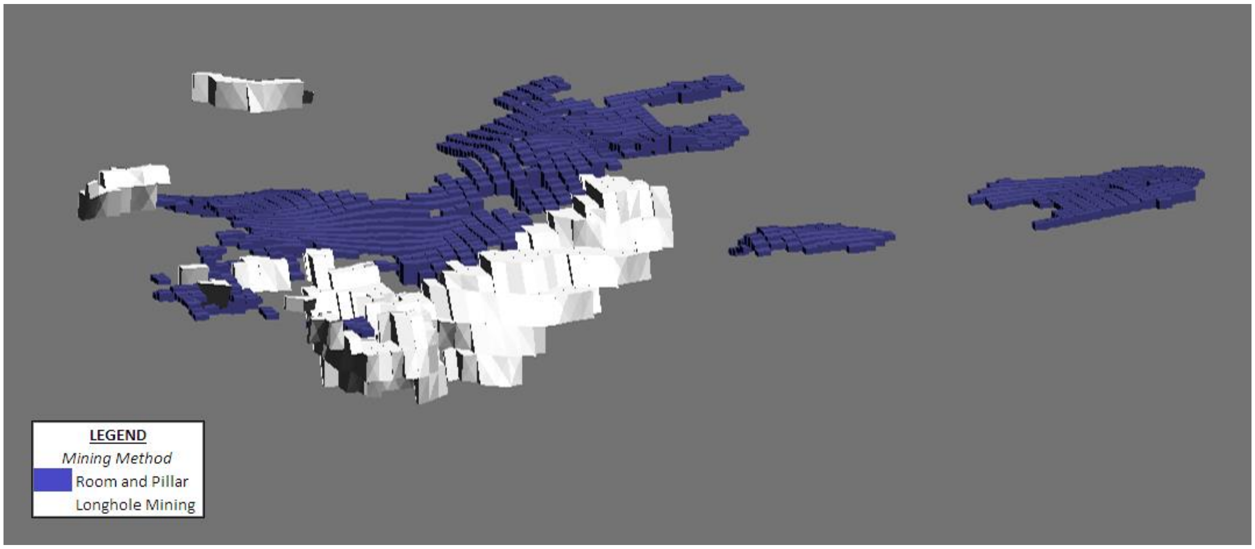


Mining begins with Zone A. This area is part of the El Dorado East deposit which is shallow dipping and outcrops at surface. Minimal work is required to access this area allowing mining to begin while the main ramp is developed to depth.

Zone B is developed using an access off the main ramp at the 1330L. While mining of Zone B progresses, the main ramp is extended to the 1200L where the mining of the remaining zones may begin. Here the deposit is steeply dipping, so LHS extraction is used. Mining proceeds sequentially by zone from the 1200L up. When the deposit becomes shallower, generally above the 1280L, Room and Pillar extraction methods are used.

Figure 16-9 depicts the LHS and RPL mining areas. The following parameters were used to design the Min/max/average height LHS and RPL.

Figure 16-9: Stopes by Mining Method – Looking Northwest

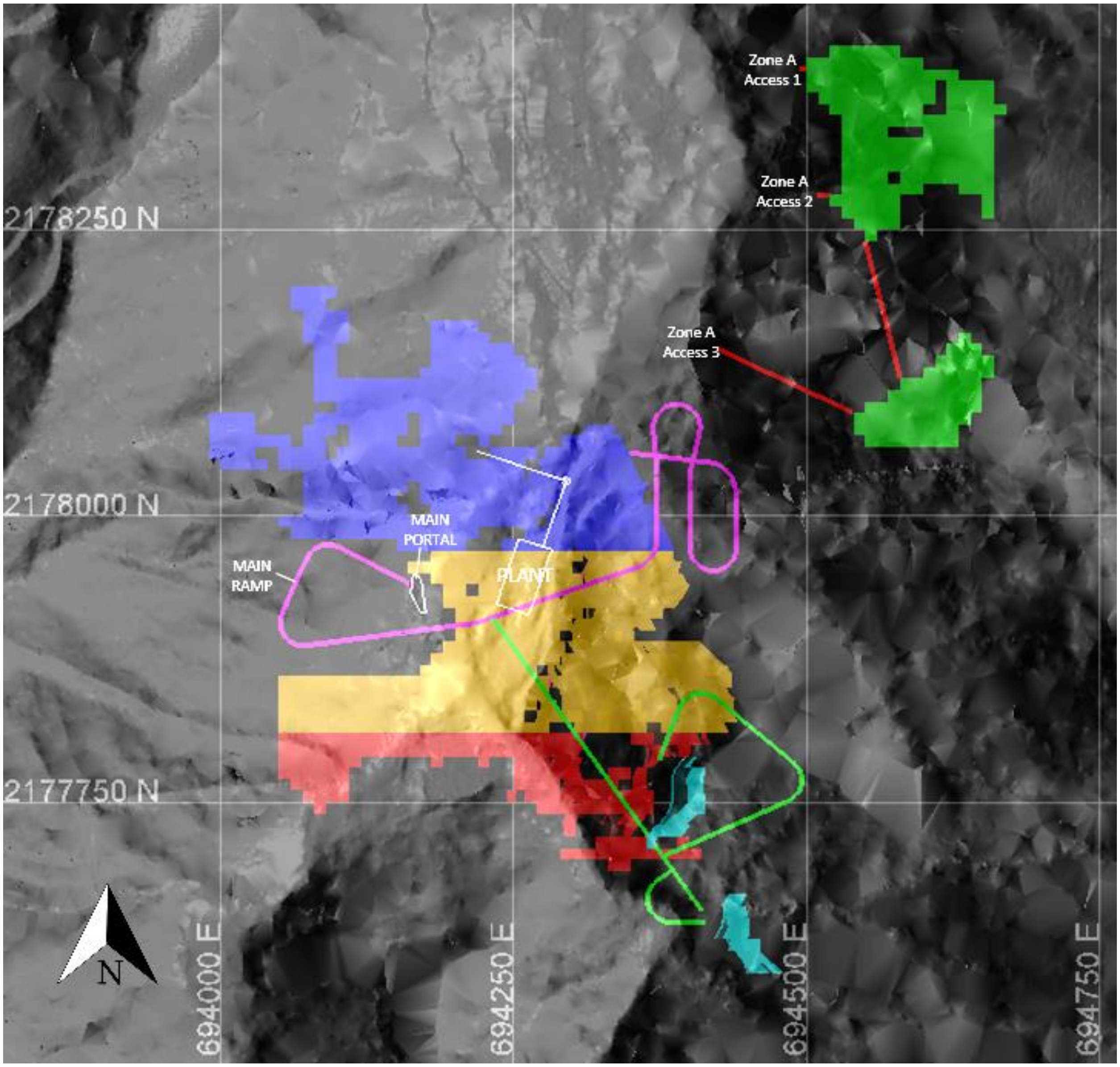


16.4.3.1 Access

A depiction of mine access is included as Figure 16-10. Mining begins initially in Zone A which outcrops at surface and is accessed separately from the rest of the deposit. Three access points will be developed using existing surface roads which will be upgraded for mine haulage.

All other zones utilize the main ramp for material haulage, mineralized material conveyance, personnel/equipment access, and services. The ramp will be 5.0 m wide and 5.0 m high; this was chosen to accommodate ventilation ducting, services, and mobile equipment. Ramps and access development will be driven at a maximum gradient of 15%. The portal exit, at 1370 L, was selected for its proximity to the plant location and surface constraints for conveying. A depiction showing access layout is included in Figure 16-10.

Figure 16-10: Mine Access



16.4.3.2 Lateral Development

The main ramp will be driven from the Portal at 1370 masl to the 1200 L, a linear distance of 1,400 m and will be used for all haulage from Zones B through E.

Zone B is separated vertically from Zones C through E. It is accessed by a secondary ramp which begins from the Main Ramp at the 1330L to meet Zone B at the 1380L. Zone B is mined in three 20 m levels from 1360 to 1380L and 1420L.

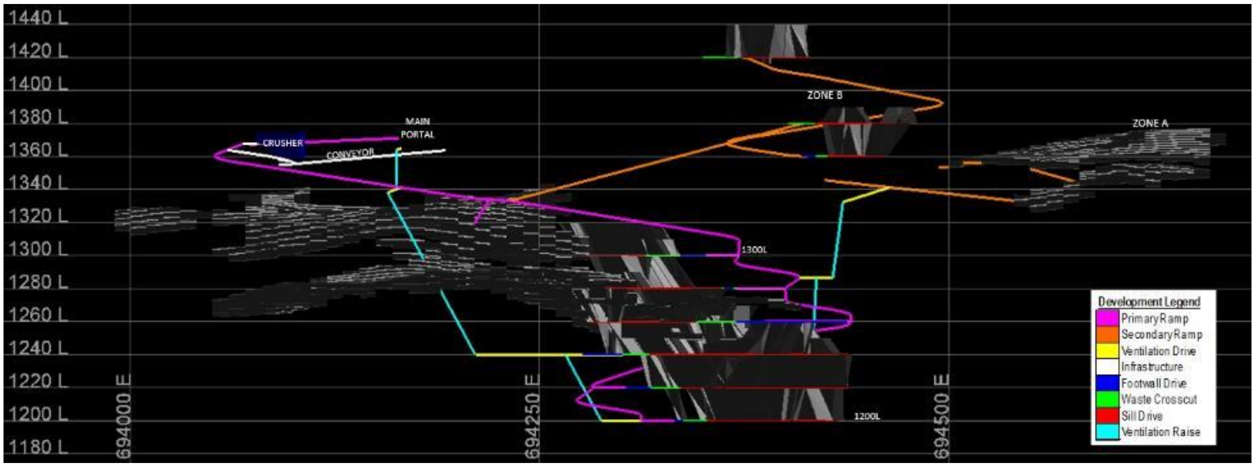
For the remaining Zones C through E, levels are located at 20 m vertical increments from 1200 to 1300L. These areas are mined from the bottom-up beginning via LHS until the dip of the deposit becomes shallower (above the 1280L). Footwall drives are located at a minimum offset of 20 m from the deposit. Re-muck bays and pullout areas will be placed every 150 m along ramps and footwall drives. Allowance for sumps and electrical substations has been included at 250 m intervals. Crosscuts are located at 10 m intervals along the footwall drifts, connecting them to the stoping blocks. Sills will be driven from the crosscuts will be driven through the mineralized material body.

Where the deposit begins to shallow and mining method transitions to RPL, existing LHS development will be used with rooms extending to follow the footwall of the deposit. A summary of lateral development by type is included in Table 16-5 and Figure 16-11.

Table 16-5: Lateral Development Summary

Type	Units	Width	Height	Type	Total Planned
Ramp	m	5.0	5.0	CAPEX	2,419
Ventilation Drive	m	4.0	4.0	CAPEX	271
Re-muck Bay	m	4.0	4.0	CAPEX	434
Infrastructure	m	5.0	5.0	CAPEX	313
Sump	m	5.0	5.0	CAPEX	184
Substation	m	5.0	5.0	CAPEX	184
Footwall Drive	m	4.0	4.0	OPEX	1,352
Level Access	m	4.0	4.0	OPEX	258
Waste Crosscut	m	3.5	4.0	OPEX	3,012
Sill Drive	m	3.5	4.0	OPEX	5,471
Total Lateral Development	**m**				**13,898**

Figure 16-11: Las Minas Development – Looking North



16.4.3.3 Vertical Development

Vertical development will be used to provide fresh air for Zones B through E. A fresh air raise will be driven from surface to 1175L by raise bore and will act as a primary fresh air raise and escape way. A secondary raise will be driven from 1255L to 1330L, where it will connect with the Zone A access. A summary of vertical development is shown in Table 16-6 as well as in Figure 16-12 of Section 16.5.1.

Table 16-6: Vertical Development Summary

Type	Units	Width	Height	Type	Total Planned
Ventilation Raise - Primary	m	2.4	2.4	CAPEX	180
Ventilation Raise - Secondary	m	2.4	1.6	CAPEX	80
Total Vertical Development	**m**				**260**

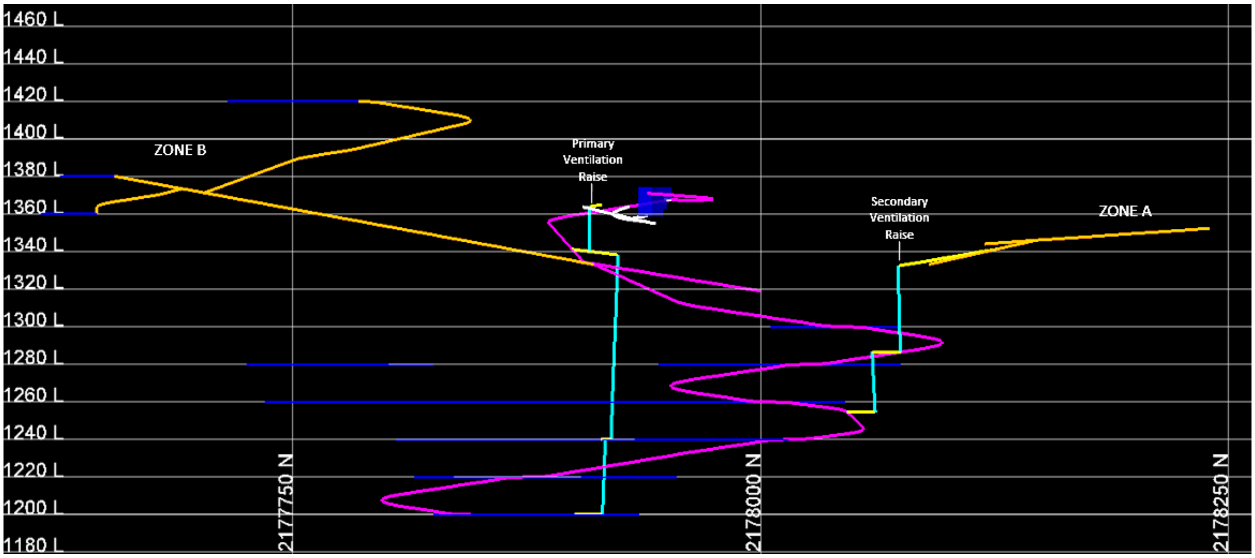
16.5 Mine Services

16.5.1 Mine Ventilation

The current mine configuration allows for flexibility in ventilation planning. Initially, fans located at the northern part of Zone A on the Access 2 and Access 3 portals. This will pull air from Access 1 and exhaust out the hill side. Later in the mine life when Zone B is developed and mined, air will be exhausted from the hillside and the primary portal will function as the intake. The deeper zones in the mine (Zones C, D, and E), will be ventilated from a series of raises connecting to surface via lateral development. The primary exhaust is the conveyor drift, and the airflow is in the same direction the conveyor is operating. The secondary vent raise will function as an intake to the lower levels of the mine. A minimum of 200kcfm with an operating power rating of 125hp will be required underground to adequately ventilate the working areas and remove equipment exhaust. Auxiliary fans and ducting will be used to ventilate active working areas.

The ventilation design is shown in Figure 16-12.

Figure 16-12: Ventilation Schematic Looking West



16.5.2 Water Supply

To control mine inflows, sumps and pumping stations are placed at regular intervals throughout the mine.

Water used for underground operations will be distributed using a combination of steel and polyurethane piping. As much as possible, water will be drawn from collected inflows after solid settlement and filtration. Surface collection may be used to supplement water needs if required.

16.5.3 Electrical Distribution

Electrical power consumption from the mine is largely attributed to the following sources:

- Main and auxiliary ventilation fans;

- Mine air compressors;
- Crushing and conveying;
- Drilling, explosives loading, and ground support equipment;
- Dewatering pumps; and
- Refuge Stations.

High voltage cables would enter the mine via the main portal and would be distributed to electrical sub-stations near the active mining zones. High-voltage power would be supplied at 4160 V and reduced to 480 V at electrical sub-stations. Each working level will include a primary substation and power panel near the ramp entrance where power will be further reduced and distributed to working faces.

16.6 Material Handling

16.6.1 Mineralized Material

A combination of three 10-t Load haul Drive (LHD) units and four 30-t trucks are used for mineralized material and waste haulage. Broken mineralized material is mucked by LHD’s to re-muck bays or loaded directly into trucks. During the first year of operation, the material is trucked to surface, after this period the material will be hauled to the underground crusher.

16.6.2 Waste Material

When possible, waste rock will be trammed and placed in mined out areas as backfill. Waste rock which cannot be used as backfill will be mucked and hauled to surface using the 30-t trucks. It will then be placed in a stockpile and rehandled by 3.4 m^3 Front End Loader (FEL) into 20-t surface haul trucks for placement in the TSF.

16.7 Backfill

Backfill consists of cemented paste and run of mine (ROM) waste material. For longhole stopes, it is assumed that 90% of primary and secondary stope voids will be filled with a cemented paste composed of tailings which will be pumped to working areas from surface. ROM waste will be placed in Room and Pillar areas or any available open voids. It is estimated that 60% of the voids left from room and pillar mining will be backfilled. Annual backfill placement schedule is shown in Table 16-9.

16.8 Mine Equipment

Diesel and electric hydraulic equipment will be employed throughout the mine. The primary haulage fleet will consist of 30-t haul trucks and 10-t LHDs for the mineralized material, waste handling, secondary tasks, and backfill. Development and room and pillar drilling will be conducted using two-boom jumbos. Longhole drilling will be conducted using Sandvik DL311 or equivalent drills. Smaller LHDs will be utilized around the mine for miscellaneous tasks and final stope mucking.

A 3.4 m^3 FEL and 20-t rock trucks will be used construction of the TSF and surface haulage of waste rock.

Table 16-7: Mine Equipment Fleet

Units in Operation	Pre-Production	Peak
30-t Truck	1	4
4-t LHD	1	1
7-t LHD	1	1
10-t LHD	1	3
3.4 m^3 FEL	1	1
Jumbo - 1 Boom	-	1
Jumbo - 2 Boom	1	2
Longhole Drill - Top Hammer	-	4
Jackleg/Stoper	3	5
Infill Drill	1	1
Small Explosives Truck	1	1
Large Explosives Truck	-	1
Bolter	1	2
Shotcrete Sprayer - Manual	1	1
Grout Pump	1	1
Scissor Lift	1	2
Boom Truck	1	1
Backhoe	1	1
Tractor	1	1
Telehandler	1	1
Utility Vehicle	1	1
Grader	-	1
Mechanics Truck	1	1
Fuel/Lube Truck	1	1
Electrician Truck	1	1
Supervisor Truck	1	2
Crew Van / Ambulance	1	1

Units in Operation	Pre-Production	Peak
20-t Surface Haul Trucks	1	1
Track Dozer - CAT D6	-	1

16.9 Mine Personnel

Mining personnel for the Las Minas project fall into two categories: salaried and hourly employees. Salaried employees include management, administrative and technical staff. Hourly employees follow a 2 week-in x 2 week-out schedule working 12 hours per day, day and night, with two crews on site, and two crews off at any given time.

Personnel requirements were estimated based on a production rate of 1,400 t/d. A summary of average and peak workforce requirement by category is included below (Table 16-8).

Table 16-8: Summary of Mine Personnel Requirements

Workforce - Total Employed	Average	Peak
Mine General	13	15
Drill and Blast	43	52
Load and Haul	38	42
Support Services	37	41
Backfill	2	2
Mine Maintenance	48	55
Technical Services	18	19
Grand Total UG Mining	199	226

16.10 Mine Schedule

The objective of the mine schedule is to develop the mine to maintain mill throughput of 1,400 t/d. Zone A is targeted initially due to its proximity near surface. This allows access to mineralized material immediately, while development to remaining zones is established at depth. This also minimizes initial development capital requirement. As LH stoping was sequenced from the bottom of the deposits upward, access to high-grade material is somewhat limited, however this was prioritized where possible. The annual material movement and development schedules are presented in Table 16-9 and Table 16-10 below.

Table 16-9: Annual Mineralized Material, Waste and Backfill Schedule

Annual Mine Schedule	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Mineralized Material	**kt**	**4,043**	**-**	**392**	**490**	**490**	**490**	**490**	**490**	**490**	**490**	**221**
Copper Grade	%	1.06	-	0.52	1.20	0.99	0.91	1.16	1.20	1.19	1.10	1.27
Gold Grade	g/t	1.84	-	2.56	1.85	2.02	2.16	1.95	1.68	1.48	1.45	1.21
Silver Grade	g/t	5.53	-	4.54	6.08	4.84	4.15	5.34	5.80	6.87	5.86	6.80
Magnetite Grade	%	15.7	-	8.2	16.2	13.4	12.6	13.5	17.6	21.0	19.1	21.6
Waste	**kt**	**429**	**106**	**112**	**67**	**29**	**24**	**27**	**30**	**22**	**11**	**-**
Backfill Paste	k m^3	740	-	19	99	111	112	108	97	77	80	37
Backfill Waste Rock	k m^3	119	-	24	25	14	12	13	15	11	5	0
Total Backfill	**k m^3**	**859**	**-**	**43**	**124**	**125**	**123**	**121**	**111**	**88**	**86**	**37**

Table 16-10: Annual Development Schedule

Annual Development Schedule	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Ramp Drive	m	2,419	1,029	1,059	331	-	-	-	-	-	-	-
Ventilation Drive	m	271	-	179	92	-	-	-	-	-	-	-
Footwall Drive	m	1,352	36	237	233	160	184	242	119	139	2	-
Re-muck	m	434	138	137	61	19	19	25	14	18	4	-
Substation	m	184	55	62	28	7	8	10	6	7	1	-
Infrastructure	m	313	313	-	-	-	-	-	-	-	-	-
Sump	m	184	55	62	28	7	8	10	6	7	1	-
Level Access	m	258	-	76	43	27	10	10	19	39	35	-
Sill	m	5,471	-	159	864	915	889	1,096	942	317	285	4
Crosscut	m	3,012	-	82	553	502	353	349	599	335	238	2
Ventilation Raise	m	260	-	155	104	-	-	-	-	-	-	-

17 PROCESS DESCRIPTION / RECOVERY METHODS

17.1 Introduction

The processing plant at Las Minas is designed to treat 1,400 t/d of mineralized material and produce 2 concentrates: a copper/gold concentrate and a magnetite concentrate. The process plant will include:

- Crushing circuit;
- Primary grinding;
- Flotation circuit
- Rougher flotation;
- Regrind;
- Flotation cleaner;
- Magnetite recovery;
- Concentrate dewatering; and
- Tailings dewatering and paste plant.

The crushing circuit has been designed to operate 350 days per year with 70% availability, while the grinding circuit was designed for 92% availability. The design criteria is shown in Table 17-1.

17.2 Plant Design Criteria

Table 17-1: Process Design Criteria

Criteria	Description	Units	Design
Plant Throughput		t/d	1,400
		t/a	490,000
Crusher Availability		%	70
Crusher Throughput		t/h	83
Crusher Selection		Size	C106
		Number	1
		Size	HP300
		Number	1

Criteria	Description	Units	Design
Mill/Flotation Availability		%	92
Mill Throughput		t/h	63
Physical Characteristics	BWI	kWh/t	13.3
Primary Grind Size	P_{80}	µm	150
Copper Concentrate Grind Size	P_{80}	µm	25
Magnetite Concentrate Grind Size	P_{80}	µm	25
Head Grade (Ave)		% Cu	1.06
		g/t Au	1.84
		g/t Ag	5.5
		% Magnetite	15.7
Gravity Recovery	Gold	%	20
Flotation Recovery	Copper	%	90
(Average recovery in Cu Concentrate	Gold	%	60
	Silver	%	70
Concentrate Grade	Copper	%	22
Total Recovery	Copper	%	90
	Gold	%	80
	Silver	%	70
	Magnetite	%	90
Magnetite Concentrate Grade	Iron	%	70
Cu Circuit Residence time	1st Rougher	mins	6.1
	2nd Rougher	mins	22
Cu Circuit Residence time	Cleaner 1 & Scav.	mins	21
	Cleaner 2	mins	24.7
	Cleaner 3	mins	33.7
Concentrate Filtration Rate		kg/m^2/h	500
Tailings Filtration Rate		kg/m^2/h	220
Concentrates Thickening Flux		t/m^2/h	0.1
Tailings Thickening Flux		t/m^2/h	0.5
Tailings Thickener Underflow Density		% w/w	70

17.3 Plant Design

Figure 17-1: Process Flowsheet – Copper and Gold Circuit

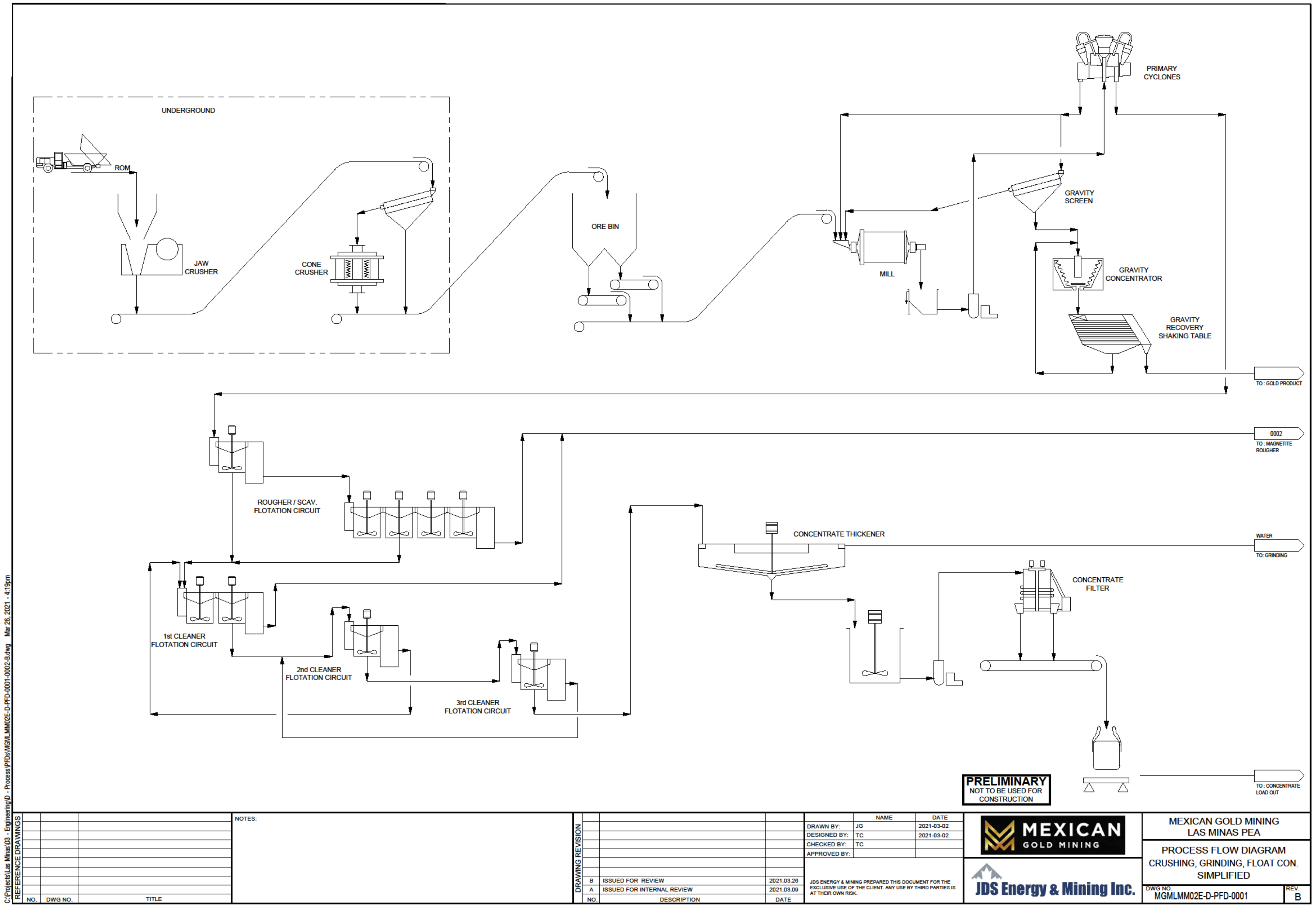


Figure 17-2: Process Flowsheet – Magnetite Circuit

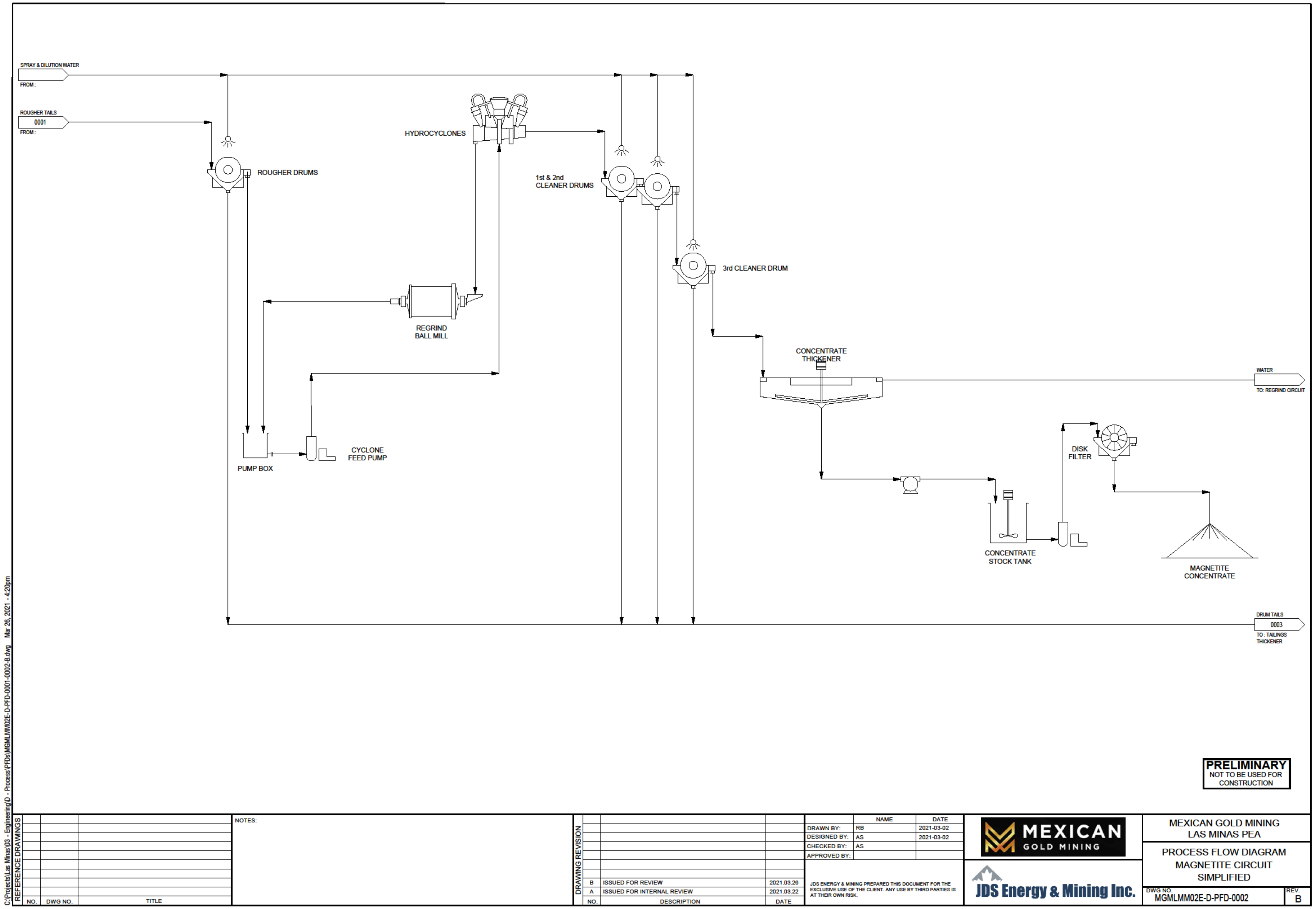
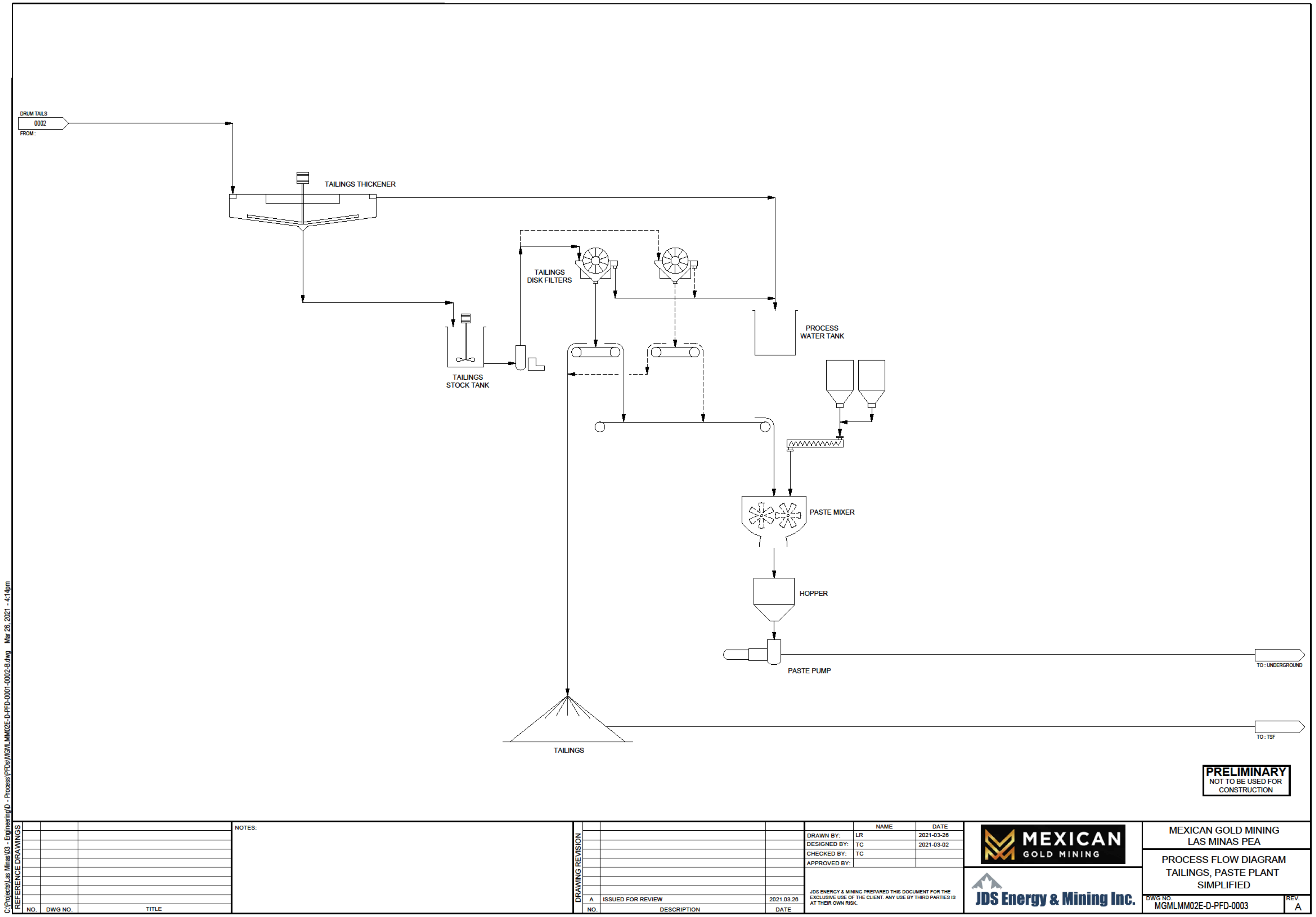


Figure 17-3: Process Flowsheet – Paste and Tailings Circuits



17.4 Process Plant Description

17.4.1 Crushing

Due to limited space on surface, the crushing circuit will be installed underground. The crushing circuit will consist of a VF43x20-1V vibrating grizzly feeder which feeds mineralized materials into a C106 jaw crusher with a closed size setting (CSS) of 40 mm. The crusher will discharge on to a conveyor belt which feeds a double deck vibrating screen. The screen decks are initially set at 50 mm on the top deck and 10mm on the bottom deck. The screen oversize will become the cone crusher feed. The cone crusher is an HP300 crusher with a CSS of 13 mm. The crusher discharge will be combined with the screen undersize to form the mill feed.

The crushing circuit can be seen in Figure 17-1.

17.4.2 Grinding

The grinding consists of a single stage 3.6 m dia. x 5.4 m long ball mill with an 1100 kW motor in closed circuit with three 250 mm cyclones (2 operating and 1 standby). The mineralized materials are fed into the ball mill at a particle size P_{80} of 13.35 mm. The grinding circuit product has a P_{80} of 150 µm.

The grinding mill discharge will flow by gravity into a pumpbox which will supply feed to one of two cyclone feed pumps. The cyclone feed pumps will be 8x10 pumps (one operating and one standby) with 75 kW motors.

Due to available mill sizes, the chosen ball mill will have capacity to increase throughput while maintaining the target grind size P_{80} of 150 µm if future throughput increases are desired. It is estimated that the grinding circuit could handle up to 1,700 t/d without requiring additional grinding power assuming the ore hardness is 15 kWh/t as found in the 2021 testwork program.

A gravity circuit consisting of a single centrifugal concentrator, shaking table and associated tanks and pumps is included to recover coarse gold particles early in the circuit to a higher payable product. This circuit also acts to prevent mis-sized gold particles from escaping the grinding circuit to the flotation circuit. With only 1 gravity test conducted and because it is difficult to quantify the effect of gravity recovery on final recovery, the total recovery was maintained as the recovery found in the LCT but was split into 20% gravity recovery and 60% flotation recovery.

17.4.3 Flotation

The flotation circuit consists of first rougher, second rougher, first cleaner, second cleaner, and third cleaner circuits.

The first rougher circuit pulls fast floating gold and copper directly to the final concentrate. The circuit consists of two 10 m^3 flotation cells. The concentrate from these cells is greater than 20% copper and therefore considered concentrate grade. A bypass to the cleaner cells will be installed in case grade or mineralogy fluctuations. The first rougher circuit recovers greater than 60% of the copper and 50% of the gold in the feed to a concentrate grade of >23% copper and >27 g/t gold.

The second rougher consists of seven 10 m^3 flotation tank cells and produces a concentrate in the range of 3-5% copper. The second rougher concentrate is pumped to a regrind circuit consisting of a Vertimill in closed circuit with cyclones. The regrind circuit grinds the concentrate to a particle size P_{80} of 25 µm. The regrind cyclone overflow reports, by gravity, to the first cleaner circuit.

The first and second cleaner circuits both consist of two 5 m^3 cells, while the 3rd cleaner circuit is a single 5 m^3 tank cell. The first cleaner tailings report to the final tails, while the 2nd and 3rd cleaner tailings report to the stage immediately preceding. The final concentrate grade targets >21% copper and >25 g/t gold.

17.4.4 Magnetite Recovery Circuit

The magnetite recovery circuit consists of 2 single drum 1100 gauss rougher magnetic concentrators. The rougher concentrate is reground in a Vertimill regrind mill in closed circuit with cyclones to a product size P_{80} of 25 µm. The cyclone overflow is directed to the magnetic cleaner circuit which consists of a single drum 900 gauss cleaner magnetic concentrator. The product from the magnetite circuit will achieve a concentrate grade of >70% iron.

17.4.5 Reagents

The reagents used in the Las Minas mill will consist of lime, potassium amyl xanthate (PAX), 2 types of frothers, flocculent, and antiscalent. The reagents, with the exception of antiscalent which is typically added to the circuit as shipped, will be made up and distributed via the reagent handling system.

The expected reagent consumptions are summarized in Table 17-2.

Table 17-2: Reagents and Process Consumables

Description	Delivered Form	Daily Usage (kg/day)
PAX	25 kg bags	66
MIBC	200 litre Drum	77
W31	200 litre Drum	139
Lime	1,000 kg Bag	455
Antiscalent	200 litre Drum	14
Flocculent	200 litre Drum	14
Ball Mill Grinding Media - 50 mm balls	1,000 kg Bag	1,532

17.4.6 Concentrates

The plant will produce two separate concentrates: a copper concentrate and a magnetite concentrate. Both concentrates will be dewatered separately. The copper concentrate will be trucked to the port at Veracruz to be sold internationally. The magnetite concentrate will be preferentially sold locally to reduce transportation costs.

17.4.7 Concentrate Dewatering

Concentrate dewatering circuits will be built for both the copper/gold and the magnetite concentrates.

The copper concentrate dewatering circuit will consist of an 8 m diameter thickener and a dedicated pressure filter. The copper concentrate will be bagged for shipping to the smelter.

The magnetite concentrate dewatering circuit will consist of a 6 m diameter thickener and a dedicated pressure filter. The magnetite concentrate will be shipped as a bulk product.

17.4.8 Tailings

The tailings will be thickened in a 15 m diameter thickener and then filtered in 1 of 2 pressure plate filters. The tailings will then either be deposited into a stockpile which will then be loaded in trucks and placed in the tailings impoundment or will be transferred to the paste plant.

The paste plant will receive filtered tailings from the mill and combine them with water, cement and fly ash to make paste to be deposited underground.

17.4.9 Water Management

Water will be reclaimed from the tailings dewatering circuit. The mill will utilize 100% of the tailings water for re-use in the circuit. Supplemental water will be added from underground dewatering.

17.4.10 Process Plant Personnel

The process plant expected staffing is for 94 people including maintenance, operations, and technical support. This is higher than a typical copper concentrator due to the filtered tailings and magnetite production circuits. A summary of the personnel can be seen in Table 17-3.

Table 17-3: Process Plant Personnel

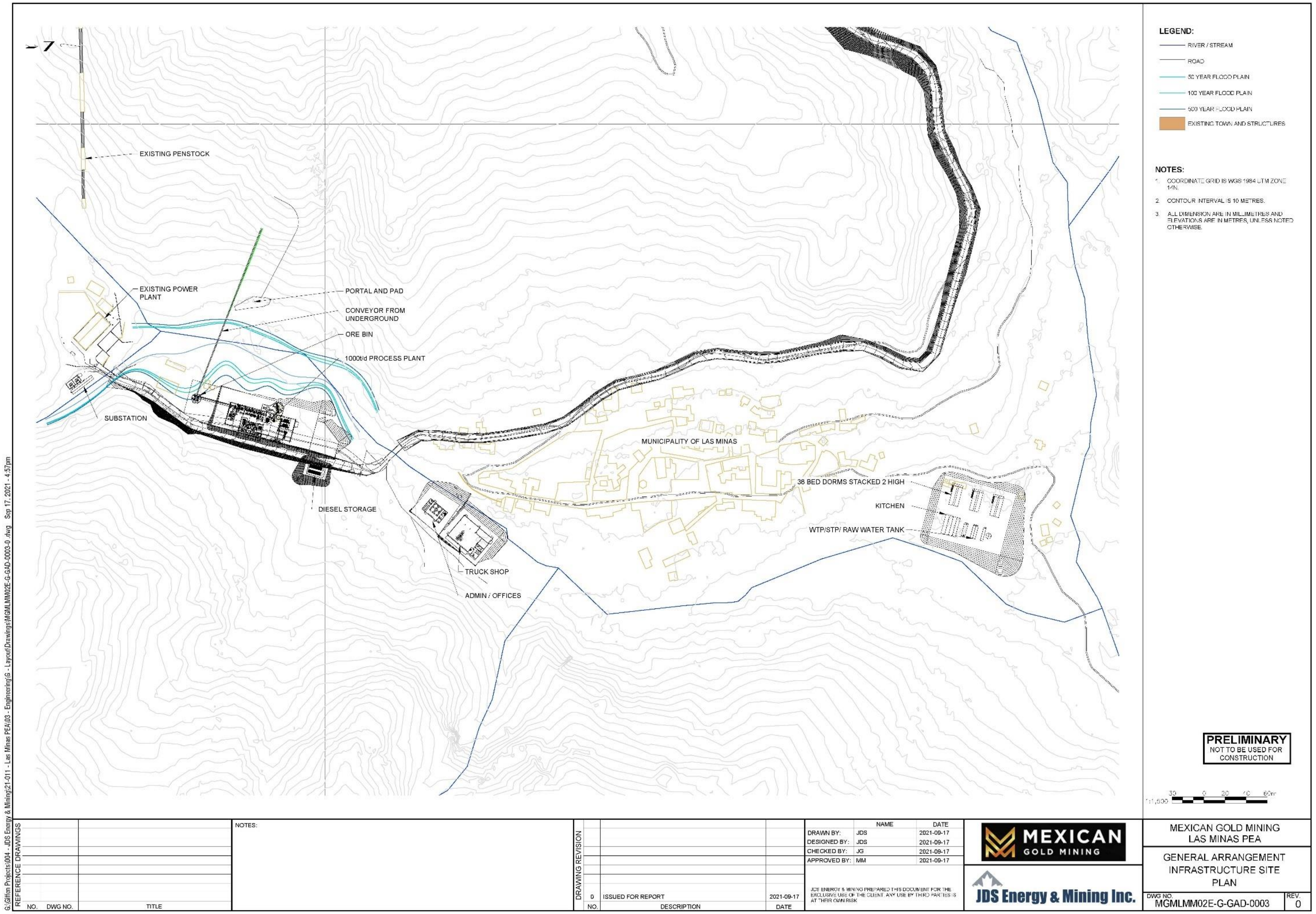
Position	Staff/Hourly	Quantity
Operations & Maintenance		
Mill Manager	Staff	1
Mill General Foreman	Staff	1
Administration Assistant	Staff	1
Mill Shift Foreman	Staff	4
Crushing Operators	Hourly	4
Grinding Mill Operators	Hourly	4
Control Room Operators	Hourly	4
Flotation Operators	Hourly	4
Magnetite Circuit Operators	Hourly	8
Tailings and Paste	Hourly	8
Reagent Prep	Hourly	4
General Labourers (Shared)	Hourly	4
Maintenance Superintendent	Staff	1
Maintenance Foreman	Staff	4
Maintenance Planner	Staff	4
Electrician	Hourly	4
Millwrights	Hourly	8
Instrumentation	Hourly	2
Maintenance Apprentice	Hourly	8
Subtotal - Operations & Maintenance		**78**
Process Technical Services		
Chief Metallurgist	Staff	1
Senior Metallurgist	Staff	1
Chief Chemist	Staff	1
Senior Chemist	Staff	1
Assay Technicians	Hourly	12
Subtotal - Process Technical Services		**16**
Total Process Plant Personnel		**94**

18 PROJECT INFRASTRUCTURE AND SERVICES

The project infrastructure is designed to support the operation of a 1,400 t/d mine and processing plant, operating on a 24 hour per day, seven day per week basis. It has been developed for the most economical operation at this production rate. The overall site layout showing the proposed location of the processing plant, tailings storage facilities, power plant and supporting infrastructure is provided in Figure 18-2.

18.1 General Site Arrangement

Figure 18-1: Site Layout



18.2 Roads

Improvements to the existing gravel road will be constructed as necessary including proper gradients, widening and general cut / fill operations.

Haul roads are planned to be upgrades of existing roads, and new roads planned for transporting mineralized material, waste rock and tailings to their designated destinations. The roads will connect the mine portals, plant site, and TSF for the transport of mined and processed material as well as mining equipment. Mine haul roads are planned to be constructed to accommodate 30-tonne underground trucks. Waste and tailings will be hauled to the TSF via 20-tonne surface haul trucks.

18.3 Buildings and Structures

18.3.1 Foundations

Structural foundations will be primarily reinforced concrete on top of either bedrock, structural back fill, or on native till with topsoil and organics stripped off. All foundations will be designed by Civil Engineers, for specific loading and Project lifespan, and will take into account the geotechnical conditions and site conditions relative to each application.

The Project facilities that will require constructed (structural) foundations can be separated into three main groups:

- Light buildings, such as administrative buildings, and laboratories;
- Heavy industrial facilities such as the crusher, conveyors, and other ancillary structures such as large storage tanks; and
- Light industrial facilities such as warehouses and truck maintenance facilities.

18.3.2 Light Buildings

Light buildings will be constructed on concrete foundations with concrete slab floors. The foundations will be designed and stamped by an engineer. Generally, materials under foundation slabs should be reworked as follows:

- Remove vegetative cover and any organic material;
- Rework the exposed soil and compress the material in place to a depth of at least 30 cm; and
- In some instances, the areas will be over-excavated, partially backfilled, and compacted.

The site will be graded to allow for positive drainage, which will minimize the risk of saturating the soil under the foundations.

Where appropriate, the recommendations of the manufacturers of modular or prefabricated housing units will be followed.

18.3.3 Heavy Industrial Facilities

Heavy structures can vibrate and can be susceptible to foundation settlement. These structures will generally be founded on bedrock which is typically at a depth of 1 m to 2 m. Overburden will be removed to expose the bedrock.

The primary crusher, which can induce vibrations, will be located underground. This will provide a solid foundation and also reduce noise in the area. The grinding mill will be located on surface and the top of bedrock will be examined to assess the level of weathering, if any. Any zone of excessive weathering, as assessed by the Geotechnical Engineer, will be removed to the extent possible. If needed, and based on the recommendations from the manufacturer, foundations may be anchored into the bedrock to increase the overall foundation mass participating in the dynamic response of the crusher and grinding mills foundation systems.

Other structures such as conveying equipment and above-ground storage tanks will also be founded on bedrock unless the depth to bedrock is such that the excavation is impractical or cost prohibitive. In the latter case, the structures will be placed on fill compacted to a dry density equal to at least 95% of the maximum density as obtained from ASTM D 1557 to provide adequate bearing capacity and so that estimated settlements meet the equipment manufacturers requirements.

18.4 Ancillary Facilities

18.4.1 Assay Laboratory

This facility will serve the plant’s assay, environmental, metallurgical requirements, and grade control needs. The laboratory will consist of pre-fabricated modules and ancillary equipment, such as drying ovens, dust and fume control, and heating equipment.

18.4.2 Warehouse and Maintenance Shop Building

A truck shop will be built to service the mine fleet mobile equipment. It will be designed and built to accommodate the planned fleet including 30 t haul trucks. The truck shop will be a pre-engineered structure with concrete foundations and floor slab.

The truck shop will be located on its own prepared earthworks pad, separate from the plant site, and in closer proximity to the underground and haul route from the plant to the tailings facility. It will have space allocated for equipment scheduled for repair, a ready line, wash facilities and room for the equipment to maneuver. The pad will be graded to ensure surface water drainage is collected in a containment pond.

18.4.3 Fuel Storage

There will be two 10,000 L storage tanks and a dispensing station for the mine mobile equipment fleet located across from the plant pad. The facility will be complete with the requisite spill storage capacity and will meet the fuel storage requirements.

18.4.4 Mine Dry

The mine dry will be a modular pre-fabricated structure shipped to site for final assembly and will be connected to water and sewage systems.

18.4.5 Power Supply

Power for the site will be provided by the adjacent existing 15 MW hydroelectric facility supplied by steel penstock tubes from a reservoir several hundred metres up the ridge. Stepdown transformers will be installed including switchgear throughout where required.

An electrical load list was developed for the project operations, based on the process design and mechanical equipment list. The connected load is approximately 5 MW.

18.5 Camp and Administration Offices

The construction/permanent camp and administration office complex will include the following:

- Dormitory units;
- Kitchen and food storage;
- Dining room;
- Arrivals/departure building including reception and first aid;
- Recreation facilities and gymnasium;
- Administrative office building including mine engineering offices, mine dry and lunchroom;
- Utility rooms (mechanical, electrical, domestic potable/hot water, fire protection);
- Laundry;
- Maintenance workshop;
- IT/server room;
- Water treatment plant; and

- Sewage treatment plant.

The complex will be pre-fabricated modular-type construction.

There will be several dormitories, each organized with bedroom blocks for a total of 250 fully furnished rooms. Each dormitory will include wash and laundry facilities. The complex will service both the construction and operation phases of the project.

The kitchen layout and equipment design will satisfy the requirements for continuous operation. The kitchen will include areas for a bakery, vegetable preparation, meat cutting, general food preparation and cooking, dish and pot washing, and waste storage. A dedicated food storage area will serve the kitchen.

The main dining room will include tables, comfortable seating for 250 people. The main dining room will serve as the emergency assembly point for the camp.

The mine engineering offices, mudroom and lunchroom will be combined into one single module. The primary function of the mud and lunchroom is to provide amenities for the construction crews. At the completion of the construction phase, this area will be converted for use as meeting room / training area.

18.6 Waste Rock Management

Waste rock from underground development will primarily be disposed of as backfill in mined out underground stopes. Waste rock from the pre-production period will be disposed of on surface and used for construction of the embankment of the TSF. Over the life of the mine, nearly 120,000 m^3 of waste rock will be placed underground as backfill and 91,000 m^3 of waste rock is placed on the TSF.

18.7 Tailings Storage Facility

18.7.1 Design Basis

The principal objectives for the design of the Tailings Storage Facility (TSF) are safe and economic storage of tailings, protection of the regional groundwater and surface waters during operations and in the long-term (post-closure), and to achieve effective reclamation at mine closure. The design of the TSF addresses the following requirements:

- Permanent, secure, and total confinement of tailings materials within an engineered facility;
- Dewatered (filtered) tailings technology for removal of free draining liquids from the tailings during operations;
- The inclusion of monitoring features to confirm the quantitative performance objectives (QPOs) are achieved and the design intent is met;
- Staged development of the facility over the life of the project; and

- An end land use that meets the objectives of the project stakeholders and Communities of Interest.

Mining will yield approximately 3.4 Mt of mineralized material over approximately an 11-year mine life. The mill will operate at a nominal throughput of approximately 1,000 t/d and the generated tailings will include the following:

- 16% of the mill tailings will be removed by a magnetite circuit;
- 36% of mill tailings will be used for underground paste backfill; and
- 48% of mill tailings will be dewatered (filtered) and stored on surface in the TSF.

Over the life of mine approximately 1,725,000 t of filtered tailings will be placed in the TSF, requiring approximately 860,000 m^3 of storage capacity. The design storage capacity has a provision for a ramp-up in Year 1 with the mill operating at 50% throughput and 50% of the paste backfill tailings used underground with the remaining 50% placed on surface in the TSF.

The planned tailings schedule is presented on Table 18-1.

Table 18-1: Tailings Schedule

Year	Surface Tailings				Paste Tailings to Surface				Total Tailings to Surface			
	Tonnes	Tonnes (Cum.)	m^3	m^3 (Cum.)	Tonnes	Tonnes (Cum.)	m^3	m^3 (Cum.)	Tonnes	Tonnes (Cum.)	m^3	m^3 (Cum.)
0.5	37,752	37,752	18,737	18,737	31,500	31,500	15,634	15,634	69,252	69,252	34,372	34,372
1.0	37,752	75,505	18,737	37,475	31,500	63,000	15,634	31,268	69,252	138,505	34,372	68,743
1.5	75,505	151,010	37,475	74,950		63,000	-	31,268	75,505	214,010	37,475	106,218
2.0	75,505	226,515	37,475	112,425	-	63,000	-	31,268	75,505	289,515	37,475	143,693
2.5	75,505	302,019	37,475	149,899	-	63,000	-	31,268	75,505	365,019	37,475	181,168
3.0	75,505	377,524	37,475	187,374	-	63,000	-	31,268	75,505	440,524	37,475	218,643
3.5	75,505	453,029	37,475	224,849	-	63,000	-	31,268	75,505	516,029	37,475	256,117
4.0	75,505	528,534	37,475	262,324	-	63,000	-	31,268	75,505	591,534	37,475	293,592
4.5	75,505	604,039	37,475	299,799	-	63,000	-	31,268	75,505	667,039	37,475	331,067
5.0	75,505	679,544	37,475	337,274	-	63,000	-	31,268	75,505	742,544	37,475	368,542
5.5	75,505	755,049	37,475	374,748	-	63,000	-	31,268	75,505	818,049	37,475	406,017
6.0	75,505	830,553	37,475	412,223	-	63,000	-	31,268	75,505	893,553	37,475	443,492
6.5	75,505	906,058	37,475	449,698	-	63,000	-	31,268	75,505	969,058	37,475	480,966
7.0	75,505	981,563	37,475	487,173	-	63,000	-	31,268	75,505	1,044,563	37,475	518,441
7.5	75,505	1,057,068	37,475	524,648	-	63,000	-	31,268	75,505	1,120,068	37,475	555,916
8.0	75,505	1,132,573	37,475	562,123	-	63,000	-	31,268	75,505	1,195,573	37,475	593,391
8.5	75,505	1,208,078	37,475	599,597	-	63,000	-	31,268	75,505	1,271,078	37,475	630,866
9.0	75,505	1,283,583	37,475	637,072	-	63,000	-	31,268	75,505	1,346,583	37,475	668,341
9.5	75,505	1,359,087	37,475	674,547	-	63,000	-	31,268	75,505	1,422,087	37,475	705,815
10.0	75,505	1,434,592	37,475	712,022	-	63,000	-	31,268	75,505	1,497,592	37,475	743,290
10.5	75,505	1,510,097	37,475	749,497	-	63,000	-	31,268	75,505	1,573,097	37,475	780,765
11.0	75,505	1,585,602	37,475	786,972	-	63,000	-	31,268	75,505	1,648,602	37,475	818,240
11.5	75,505	**1,661,107**	37,475	824,446	-	**63,000**	-	31,268	75,505	**1,724,107**	37,475	**855,715**

Source: KP (2021)

Four different tailings samples were tested for Acid-Base Accounting (ABA) to support the PEA and all test results confirmed that the tailings were non-Potentially Acid Generating (Non-PAG). This has been factored into the TSF design.

The tailings will be dewatered by filtering, and most of the process water will be removed and re-used. The dewatered filter cake will be transported by trucks to the TSF, where it will be placed by spreading and compacting.

18.7.2 TSF Alternatives

Several (6) potential TSF sites were assessed to demonstrate the capability of the property to accommodate the infrastructure necessary for tailings management. The sites were inspected during a site visit that was conducted to review the local geological conditions and assist in confirming a preferred TSF location for the PEA. Input was provided by Mexican Gold regarding any areas that may have been considered not suitable for various reasons. The selected TSF location has adequate capacity, with suitable foundation conditions, and is relatively close to the plant site thereby allowing for truck haulage of the filtered tailings.

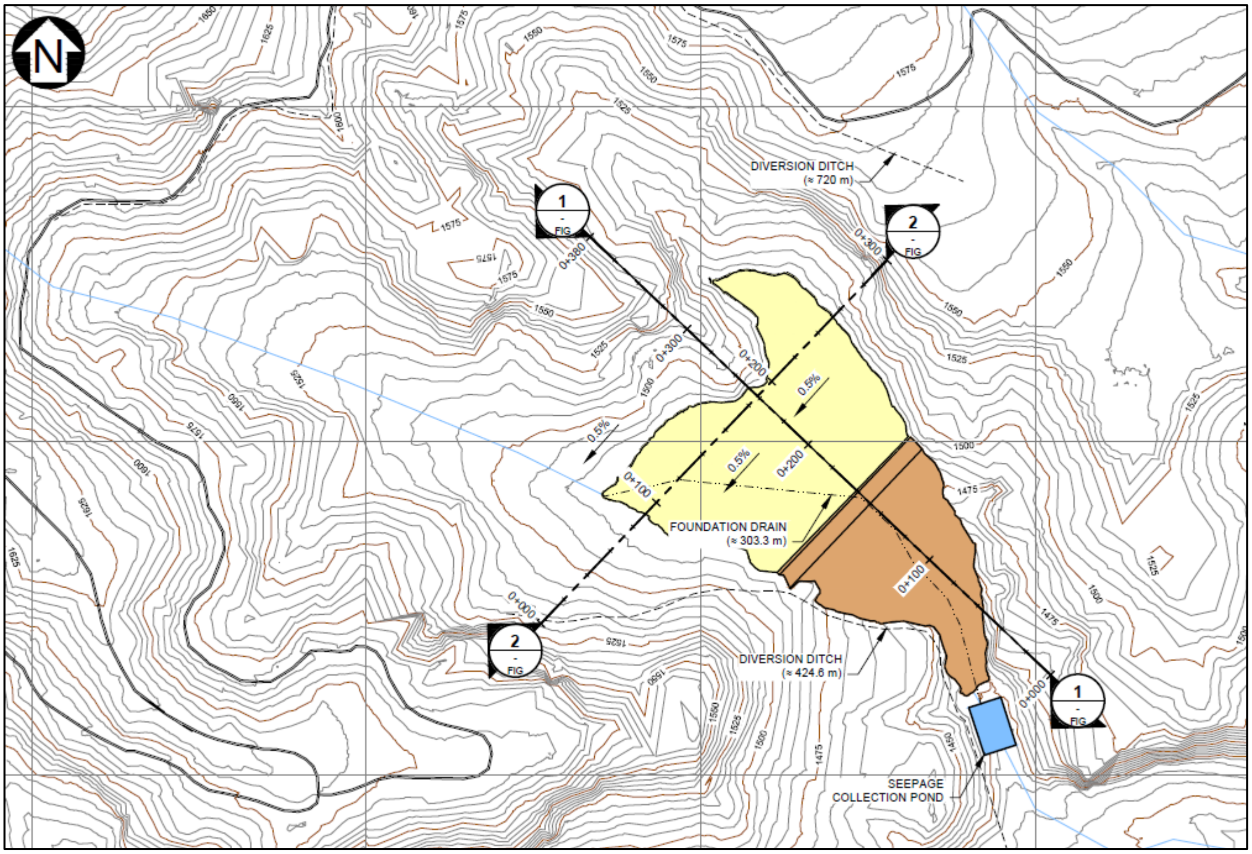
18.7.3 TSF Staging and Filling Schedule

The PEA design includes TSF development in stages over the mine life. This staged approach offers the ability to refine design, construction, and operating methodologies as experience is gained with local conditions and constraints. It also offers the potential to reduce initial capital costs and defer capital expenditure relating to TSF construction to the operations phases.

The TSF design has capacity to store 1.7 Mt of tailings. The placed (compacted) dry density of the tailings was estimated at 2.0 tonnes per cubic metre (t/m^3), reflecting a high SG of 3.4. Placed tailings will be confined by an initial Stage 1 (starter) embankment with a structural shell zone to provide the containment in later years of operations.

The Stage 1 general arrangement is shown on Figure 18-2. Stage 1 is designed to provide containment for the first 2 years of operations, with a maximum filling level at El. 1487 m. Tailings will be placed with a slope from northeast to southwest to promote runoff to the drainage at the west side of the TSF.

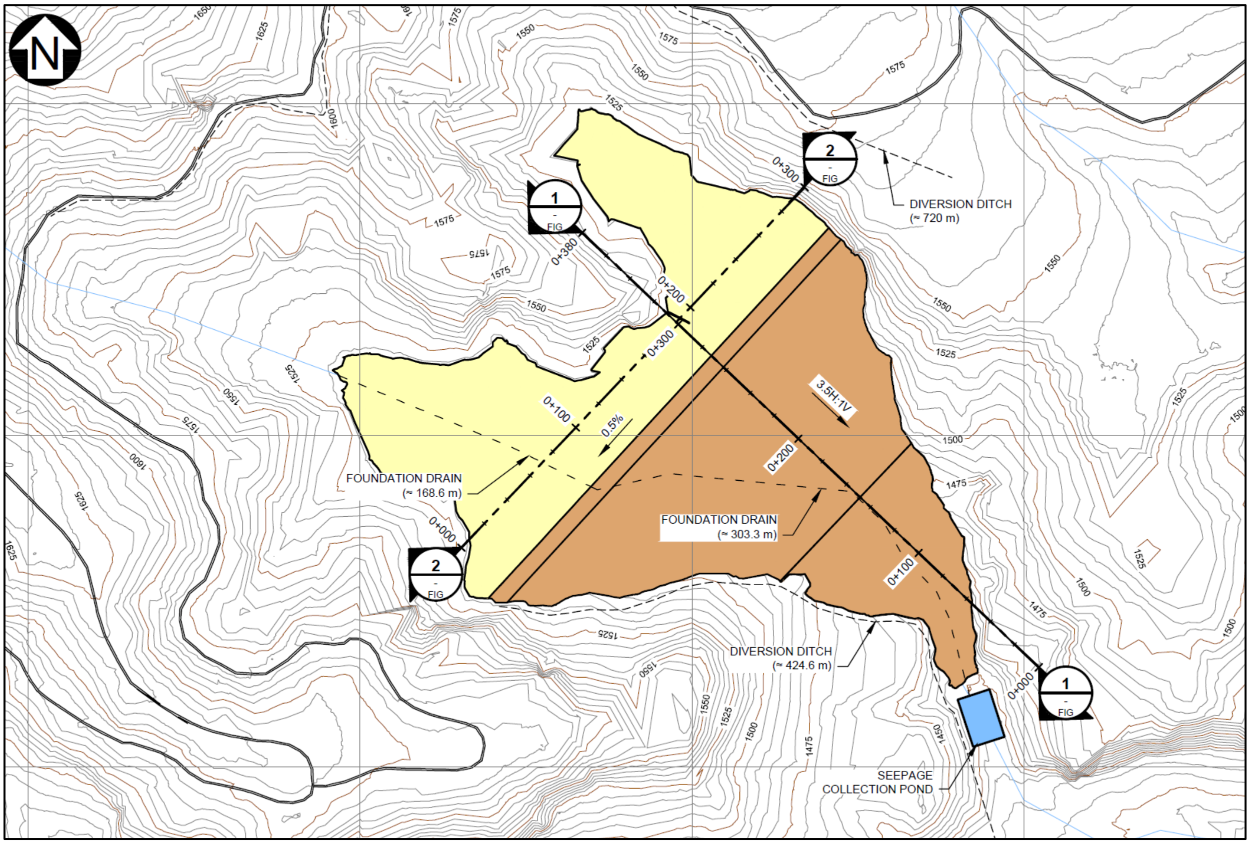
Figure 18-2: TSF Stage 1 General Arrangement



Source: KP (2021)

The final general arrangement is shown on Figure 18-3. The final arrangement is designed to provide containment for the remaining 9 years of operations, with a maximum filling level at El. 1520 m. Tailings will continue be placed with a slope from northeast to southwest to promote runoff to the drainage at the west side of the TSF.

Figure 18-3: TSF Final General Arrangement

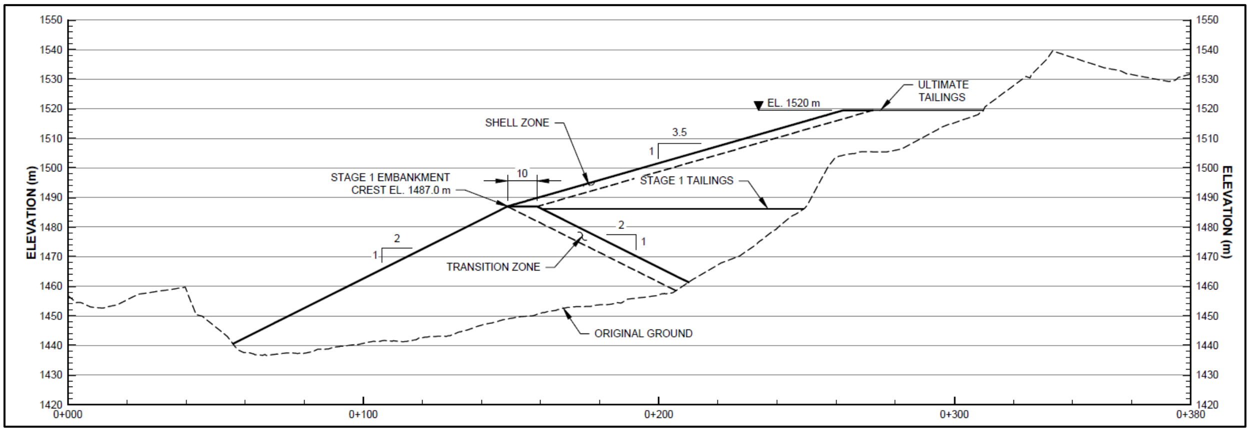


Source: KP (2021)

18.7.4 TSF Design

A cross section of the final general arrangement for the TSF is shown on Figure 18-4.

Figure 18-4: TSF Final General Arrangement Cross Section



Source: KP (2021)

Main elements of the TSF design are described below:

- Foundation Preparation
  - The embankment foundation will require clearing and topsoil stripping in preparation for fill placement; and
  - The TSF basin will be cleared and grubbed to support efficient tailings placement.
- Embankment Construction
  - At El. 1487 m, Stage 1 (2-year capacity) has a maximum height of 40 to 50 m from the bottom of the drainage. Most of the Stage 1 embankment fill will comprise well graded rockfill, and a transition zone consisting of finer material will be placed on the upstream face of the embankment, between the rockfill and placed tailings. Waste rock from underground mining may be used as embankment fill, depending on the mine development schedule;
  - The Ultimate arrangement extends to El. 1520 m and will be raised progressively over the remaining 9 years of operations. Tailings will be confined by a structural shell zone constructed from granular (free draining) material; and
  - A foundation drain will be placed in the bottom of the TSF basin to provide a drainage path for water that may infiltrate into the compacted tailings, as well as runoff that may collect in the basin. The foundation drain will discharge into a seepage collection pond downstream of the embankment. The foundation drain will consist of a drainage layer with a filter sand zone and a geotextile wrap.

- Geotechnical Instrumentation
  - Instrumentation will be installed for monitoring the performance of the TSF embankments and placed tailings. Instrumentation may include piezometers, inclinometers, survey monuments, etc.

18.7.5 TSF Operations

Filtered tailings will be delivered by truck to the TSF from a stockpile at the filter plant, located at the mill. The tailings will be spread with a bulldozer and compacted with a smooth drum roller. A 0.5 % slope will be maintained on the tailings surface (from northeast to southwest) to promote runoff to the drainage at the west side of the TSF.

18.7.6 Water Management

The Stage 1 embankment will be constructed during the pre-production period so that the facility is ready to receive filtered tailings at mill commissioning. The tailings will be dewatered to a moisture content in the range of 15-20% for placement and compaction. Process water will be removed from tailings during dewatering (filtering) and will be recirculated for use in milling. Make-up water will be sourced from the nearby river.

Runoff may occasionally accumulate in the Stage 1 TSF, and this will be minimized by constructing fit-for-purpose diversion ditches with the road above the TSF and on the west side of the facility to limit runoff into the basin. Collected runoff would drain out of the facility via the foundation drain and the free draining embankment itself, or with temporary pumps if necessary. This water would report to a seepage collection pond downstream of the embankment and would be discharged after clarification. It could also be pumped to the mill, if needed.

After Stage 1 is filled, the tailings and confining embankments will be raised simultaneously, and the 0.5% slope maintained to promote runoff to the drainage at the west side of the TSF.

18.7.7 TSF Closure

The TSF will be constructed with a 0.5% slope on the tailings surface (from northeast to southwest). This will manage runoff during operations and will facilitate closure and long-term water management. At closure, the tailings surface will be covered with a suitable growth media (topsoil) layer and revegetated. The diversion channels will be upgraded, if needed. Also, the embankment slopes could be revegetated if desired.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No market studies have been completed for the project at this time. All commodities considered in this Study are regularly sold commercially on vast international markets. As the concentrates are clean, that is relatively free of contaminants, competitive treatment charges are anticipated with relatively easy sales to open markets. The copper and iron concentrate terms that are used in the economic analysis are listed in Table 19-1 and Table 19-2.

Table 19-1: Copper Concentrate Smelter Terms

Copper Concentrate	Unit	Value
Cu Recovery	%	90
Au Recovery	%	80
Ag Recovery	%	70
Concentrate Grade		
Cu	%	21.7
Au	g/t	33.5
Ag	g/t	88
Moisture Content	%	8
Smelter Payables		
Cu Payable	%	96.1
Min. Cu deduction	% Cu/tonne	1.1
Au Payable	%	97.2
Min. Au deduction	g/t conc	1
Ag Payable	%	60.9
Min. Ag deduction	g/t conc	30
Treatment & Refining Costs		
Cu TC	US$/dmt conc	65
Cu RC	US$/payable lb	0.065
Au RC	US$/payable oz	5.00
Ag RC	US$/payable oz	0.40
Calculated Penalties	US$/dmt	0
Transport Costs	US$/wmt	56.20

Table 19-2: Iron Concentrate Terms

Iron Concentrate	Unit	Value
Fe Recovery	%	90.3
Concentrate Grade		
Fe	%	90
Moisture Content	%	8
Smelter Payables		
Fe Payable	%	100
Transport		
Transport Costs	US$/wmt	56.20

19.2 Contracts

No agreements for the sale of concentrates are currently in place.

19.3 Royalties

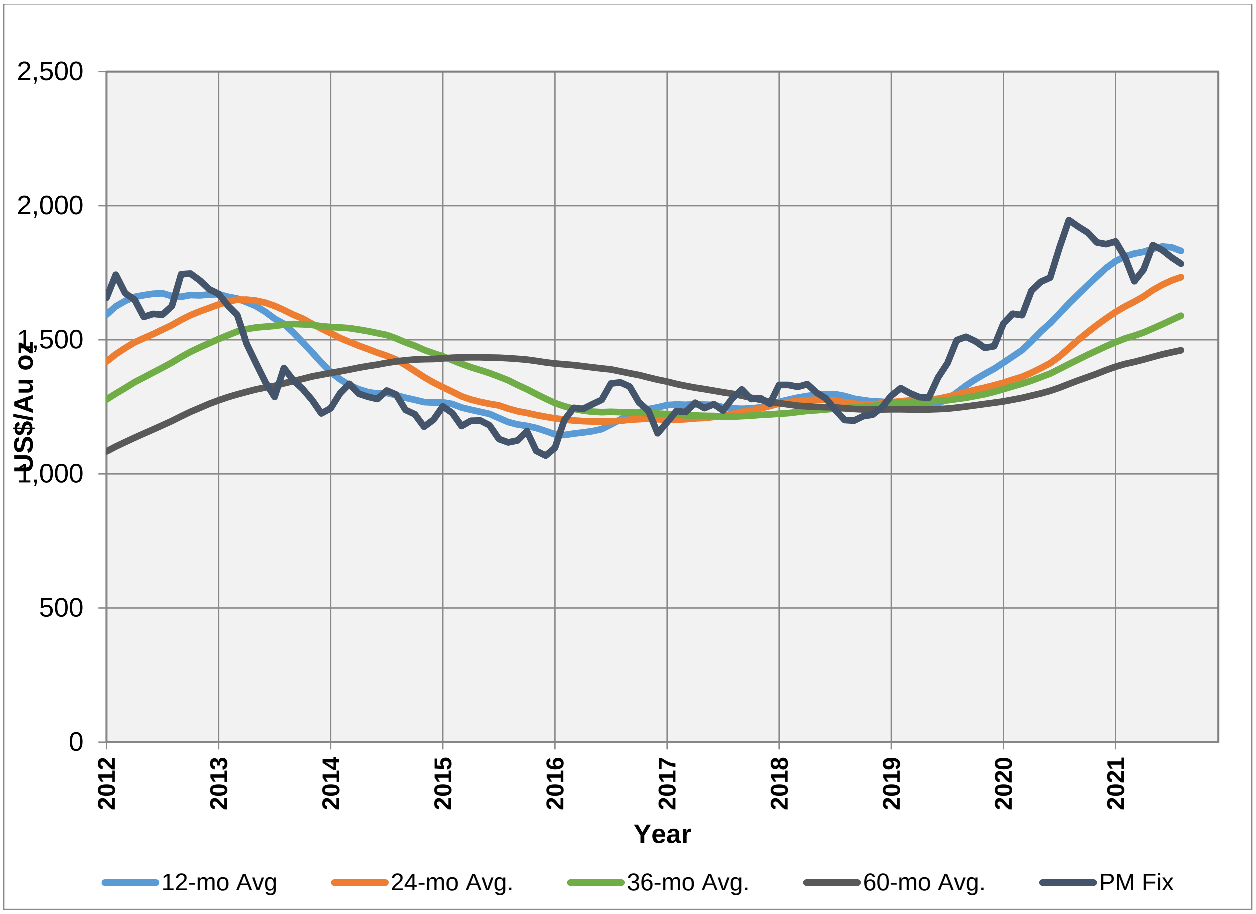
The property is not subject to any royalties.

19.4 Metal Prices

The precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong). Historical gold, copper, silver, and iron ore prices are shown in Figure 19-1, Figure 19-2, Figure 19-3 and Figure 19-4.

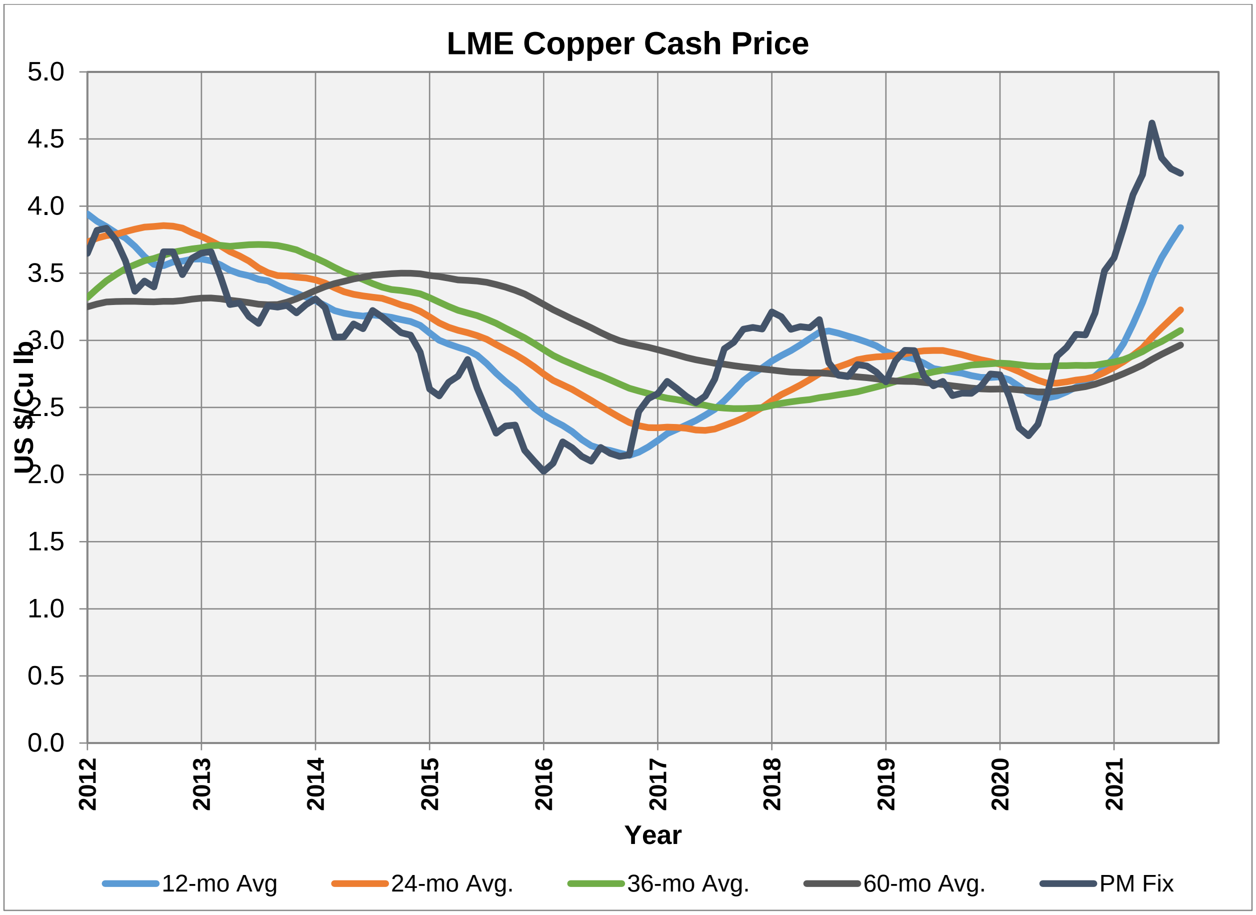
Metals prices used in the base case economic model are in line with other recently released technical reports.

Figure 19-1: Historical Gold Price



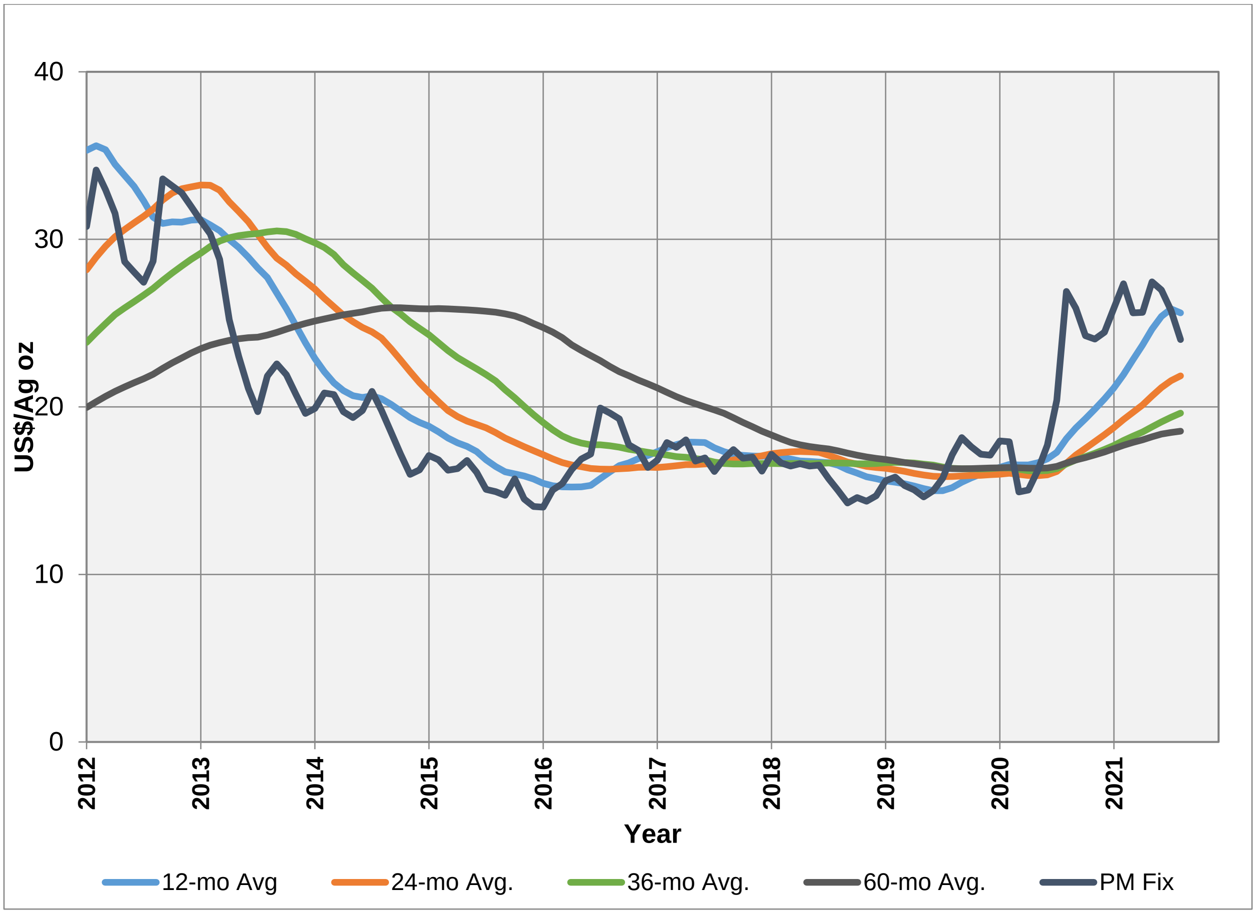
Source: Kitco (2021)

Figure 19-2: Historical Copper Price



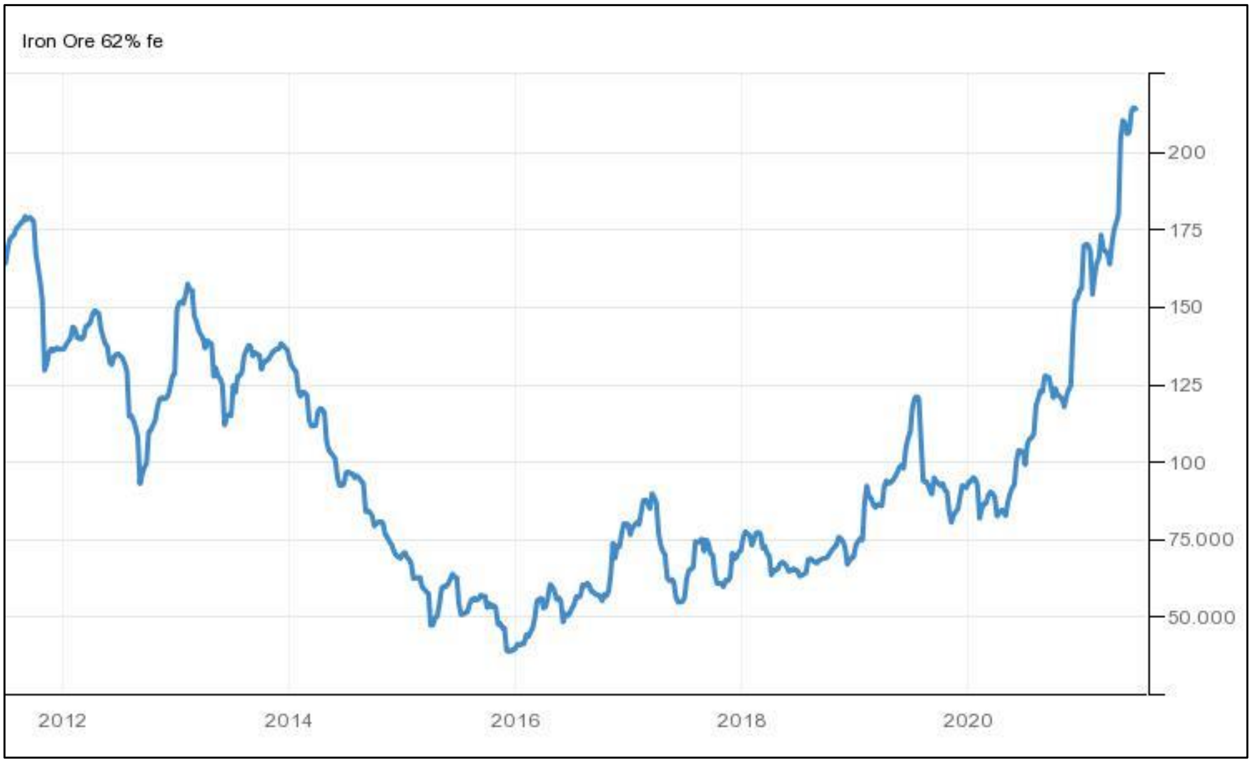
Source: London Metals Exchange (2021)

Figure 19-3: Historical Silver Price



Source: Kitco (2021)

Figure 19-4: Historical Iron Ore Price



Source: Tradingeconomics (2021)

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACTS

Mexican Gold has conducted environmental studies in the project area in order to initiate development of a defensible baseline. Exploration work is conducted in a transparent manner with the local communities, supported with a strong community outreach and support program.

Current exploration activity is fully permitted and in good standing. Mine development will require the successful conclusion of an Environmental Impact Assessment and permitting. This is a recognized and regulated process in Mexico. There are no known environmental issues that could materially impact the ability of Mexican Gold to extract the mineral resources at the Las Minas Project.

20.1 Legal Framework

Mine permitting in Mexico is administered by the federal government body Secretaría de Medio Ambiente y Recursos Naturales (SEMARNAT). Guidance for the federal environmental requirements is derived from the Ley General del Equilibrio Ecológico y la Protección al Ambiente (LGEEPA). Article 28 of the LGEEPA specifies that SEMARNAT must issue prior approval to parties intending to develop a mine and mineral processing plant. An Environmental Impact Assessment (Manifestación de Impacto Ambiental (MIA) by Mexican regulations) is the mechanism whereby approval conditions are specified where works or activities have the potential to cause ecological imbalance or have adverse effects on the environment. This is supported by Article 62 of the Reglamento de la Ley Minera. Article 5 of the LGEEPA authorizes SEMARNAT to provide the approvals for the works specified in Article 28.

The LGEEPA also contains articles that are relevant to conservation of soils, tailings management, water quality, flora and fauna, noise emissions, air quality, and hazardous waste management. The Ley de Aguas Nacionales provides authority to the Comisión Nacional de Agua (CONAGUA), an agency within SEMARNAT, to issue water abstraction concessions, and specifies certain requirements to be met by applicants.

Another important piece of environmental legislation is the Ley General de Desarrollo Forestal Sustentable (LGDFS). Article 117 of the LGDFS indicates that authorizations must be granted by SEMARNAT for land use changes to industrial purposes. An application for change in land use or Cambio de Uso de Suelo (CUS), must be accompanied by a Technical Supporting Study (Estudio Técnico Justificativo, or ETJ).

Guidance for implementation and adherence to many of the stipulations of environmental legislation is provided in a series of Normas Oficiales Mexicanas (NOM). These NOM provide specific procedures, limits, and guidelines, and carry the force of law. The relevant permit application will be developed as the Project progresses.

20.1.1 Current Permitting

All exploration activity permitting is current and reviewed quarterly SEMARNAT.

Additionally, as a matter of formality, although not required by government, "permits" were requested and received from the Mayors of Las Minas and nearby Tatatila municipalities. This has strengthened the recognition of respect for the local leaders and reinforced the notion of partnership.

20.2 Environmental Studies

The project is in an area of steep relief with altitudes ranging from 1,800 to 3,000 masl, in the vegetation zone designated as Mountain Mesophilic Forest (MMF). The MMF is dominated by deciduous trees in various strata, with an abundance of ferns and epiphytes, fed by high humidity, mist, and cloud cover.

The primary drainage of the site is the Las Minas River, which is fed by headwater streams upstream of the project, including the Tenepanoya, El Sauce, Frio, and Puerco Rivers. The Las Minas River drains to the Bobos River, or Nautla River watershed, of the North Veracruz hydrological region.

Mexican Gold has initiated environmental studies, as described in CTA (2020), including: Surface Water Quality, Vegetation, and Wildlife. Samples were retrieved in June 2020.

20.2.1 Surface Water Quality

Two surface water monitoring sites were established: both on the Frio River which travels through the project area before joining the Las Minas River. One site (SW-01) was located upstream of any anticipated project effects; the other site (SW-02) was located downstream of the project area, but upstream from the town of Las Minas.

Samples were retrieved from the field and forwarded to Bureau Veritas laboratory in Canada for analysis of:

- Total metals;
- Dissolved metals;
- Mercury;
- pH;
- Hardness;
- Sulphates;
- Chlorides;

- Colour;
- Dissolved oxygen;
- Total suspended solids; and
- Conductivity.

Laboratory analysis results are summarized in Table 20-1, Table 20-2, and Table 20-3 below. Values were compared to the Mexican NOM-001-SEMARNAT-1996, which specifies surface discharge concentration maxima for various parameters. None of the results exceeded the maxima designated for the protection of aquatic life, the strictest standard.

Table 20-1: Surface Water Physico-chemical Parameters

Parameter	Units	Detection Limit	Site SW-01	Site SW-02
pH	u.e.	N.A.	8.06	8.12
Electrical conductivity	µmho/cm	1.0	230	190
Dissolved oxygen	mg/L	N.A.	8.77	8.75
Total Suspended Solids	mg/L	10	<10	<10
Chlorides	mg/L	1.0	8.6	2.3
Sulphates	mg/L	1.0	11.0	14.0
Hardness	mg/L	1.0	79	74
Color	TCU	2.0	9.0	9.0

Source: CTA (2020)

Table 20-2: Surface Water Total Metals Concentrations

Parameter	Units	Detection Limit	Site SW-01	Site SW-02
Aluminum (Al)	mg/L	0.0049	0.430	0.400
Antimony (Sb)	mg/L	0.0005	<0.0005	<0.0005
Arsenic (As)	mg/L	0.001	<0.001	<0.001
Barium (Ba)	mg/L	0.002	0.014	0.015
Beryllium (Be)	mg/L	0.0004	<0.0004	<0.0004
Bismuth (Bi)	mg/L	0.001	<0.001	<0.001
Boron (B)	mg/L	0.01	0.092	<0.01
Cadmium (Cd)	mg/L	0.00009	<0.00009	<0.00009

Parameter	Units	Detection Limit	Site SW-01	Site SW-02
Calcium (Ca)	mg/L	0.2	20	22
Chromium (Cr)	mg/L	0.005	<0.005	<0.005
Cobalt (Co)	mg/L	0.0005	<0.0005	<0.0005
Copper (Cu)	mg/L	0.0009	0.0019	0.0031
Iron (Fe)	mg/L	0.10	0.25	0.27
Lead (Pb)	mg/L	0.00050	0.00150	0.00094
Lithium (Li)	mg/L	0.005	0.0099	<0.005
Magnesium (Mg)	mg/L	0.05	7.7	5.9
Manganese (Mn)	mg/L	0.002	0.0078	0.0084
Molybdenum (Mo)	mg/L	0.00050	0.00100	0.00068
Nickel (Ni)	mg/L	0.001	<0.001	<0.001
Phosphorus (P)	mg/L	0.1	<0.1	<0.1
Potassium (K)	mg/L	0.2	3.9	2.9
Selenium (Se)	mg/L	0.002	<0.002	<0.002
Silicon (Si)	mg/L	0.05	21.00	19.00
Silver (Ag)	mg/L	0.00009	0.00015	0.00016
Sodium (Na)	mg/L	0.1	13.0	5.8
Strontium (Sr)	mg/L	0.001	0.096	0.088
Tellurium (Te)	mg/L	0.001	<0.001	<0.001
Thallium (Tl)	mg/L	0.00005	<0.00005	<0.00005
Tin (Sn)	mg/L	0.001	<0.001	<0.001
Titanium (Ti)	mg/L	0.005	0.03	0.025
Tungsten (W)	mg/L	0.001	<0.001	<0.001
Uranium (U)	mg/L	0.0001	0.00041	0.00018
Vanadium (V)	mg/L	0.0005	0.0066	0.004
Zinc (Zn)	mg/L	0.005	0.0063	<0.005
Mercury (Hg)	mg/L	0.00010	0.00120	0.00090
Zirconium (Zr)	mg/L	0.001	<0.001	<0.001

Source: CTA (2020)

Table 20-3: Surface Water Dissolved Metals Concentrations

Parameter	Units	Detection Limit	Site SW-01	Site SW-02
Aluminum (Al)	mg/L	0.0049	0.2000	0.1300
Antimony (Sb)	mg/L	0.0005	<0.0005	<0.0005
Arsenic (As)	mg/L	0.001	<0.001	<0.001
Barium (Ba)	mg/L	0.002	0.014	0.016
Beryllium (Be)	mg/L	0.0004	<0.0004	<0.0004
Bismuth (Bi)	mg/L	0.001	<0.001	<0.001
Boron (B)	mg/L	0.010	0.099	<0.010
Cadmium (Cd)	mg/L	0.00009	<0.00009	<0.00009
Calcium (Ca)	mg/L	0.2	19.0	20.0
Chromium (Cr)	mg/L	0.005	<0.005	<0.005
Cobalt (Co)	mg/L	0.0005	<0.0005	<0.0005
Copper (Cu)	mg/L	0.0009	0.0014	0.0029
Iron (Fe)	mg/L	0.10	0.12	0.12
Lead (Pb)	mg/L	0.00050	0.00074	0.0006
Lithium (Li)	mg/L	0.0050	0.0098	<0.0050
Magnesium (Mg)	mg/L	0.05	7.80	6.00
Manganese (Mn)	mg/L	0.002	0.0066	0.0076
Molybdenum (Mo)	mg/L	0.00050	0.00088	0.00070
Nickel (Ni)	mg/L	0.001	<0.001	<0.001
Phosphorus (P)	mg/L	0.1	<0.1	<0.1
Potassium (K)	mg/L	0.2	4.0	3.0
Selenium (Se)	mg/L	0.002	<0.002	<0.002
Silicon (Si)	mg/L	0.05	21.00	18.00
Silver (Ag)	mg/L	0.00009	<0.00009	0.00018
Sodium (Na)	mg/L	0.1	13.0	5.5
Strontium (Sr)	mg/L	0.001	0.093	0.085
Tellurium (Te)	mg/L	0.001	<0.001	<0.001
Thallium (Tl)	mg/L	0.00005	<0.00005	<0.00005
Tin (Sn)	mg/L	0.001	<0.001	<0.001
Titanium (Ti)	mg/L	0.005	0.0099	0.0064
Tungsten (W)	mg/L	0.001	<0.001	<0.001
Uranium (U)	mg/L	0.0001	0.00037	0.00016
Vanadium (V)	mg/L	0.0005	0.0061	0.0036
Zinc (Zn)	mg/L	0.005	<0.005	<0.005
Zirconium (Zr)	mg/L	0.001	<0.001	<0.001

Source: CTA (2020)

20.2.2 Vegetation

The vegetation in the project area is largely in its natural state, owing to the inaccessibility of much of the zone. Average slope gradients are in excess of 120%, including many vertical rock faces. Lower gradient slopes are characterized by complex humid forests with a well-developed understory and epiphyte growth (Figure 20-1). Higher gradient sites are typically drier, characterized by pine and oak forest with limited understory (Figure 20-2).

Figure 20-1: Typical Vegetation at Moderate Gradients

Source: CTA (2020)

Figure 20-2: Typical Vegetation at Higher Gradients

Source: CTA (2020)

Six vegetation study plots were established in the project area, where a total of 225 species were identified. The Mexican NOM-059-SEMARNAT-2010 identifies species considered to be at risk. Six plant species listed on the NOM were identified in the study, including:

- Aporocactus flagelliformis (Rattail cactus);
- Nephelea mexicana (Fern);
- Licaria triandra (Laurel);
- Tilia americana (Basswood);
- Cedrela odorata (Cedar); and
- Ceratozamia mexicana (Cycad).

Typically, as part of environmental management in Mexico, in advance of clearing for project construction, a plant rescue program is implemented. Particular attention will be required in future surveys to identify the locations of these species such that they can be relocated and propagated.

20.2.3 Wildlife

Wildlife surveys for amphibians, reptiles, and birds were completed at site (CTA, 2020). Findings included:

- 5 species of amphibians;
- 2 species of snakes;
- 4 species of lizards; and
- 19 species of birds.

Of the species present at site, 8 species are listed on NOM-059-SEMARNAT-2010, including:

- *Craugastor decorates* (Frog, Figure 20-3);
- *Charadrahyla taeniopus* (Frog, Figure 20-3);
- *Sceloporus salvini* (Lizard);
- *Thamnophis sumichrasti* (Sumachrast Garter Snake, Figure 20-3);
- *Thamnophis scalaris* (Longtail Garter Snake);
- *Myadestes occidentalis* (Brown-backed Solitaire, Figure 20-3);
- *Turdus infuscatus* (Black Thrush); and
- *Turdus migratorius* (American Robin).

Figure 20-3: Photos of Select Wildlife from the Project Area

Source: CTA (2020)

As is done for plants, a rescue and relocation program will be required for amphibians and reptiles, as well as mammals prior to clearing for construction. Ongoing monitoring and management of wildlife will be incorporated to mine processes.

20.3 Social and Community Requirements

20.3.1 Local Socio-Economy

The project is located in a sparsely populated valley, centered near the small town of Las Minas, located approximately 30 km by road from the state capital Xalapa. As implied by its name, Las Minas has historically relied on local mining activity as its principal economic driver. The local topography limits ranching and agriculture to select areas. Services to the town are basic. There is no public transportation system to larger population centers. Many residents rely on remittances from family that live outside of the valley.

20.3.2 Outreach Activities

Accessibility and visibility of Mexican Gold staff have been maintained through keeping the residence and office within the village. Everything possible including lumber, foodstuffs, furniture, skilled labor, etc. is sourced locally.

Community members are hired for work during exploration programs, and contractors (e.g., drillers) likewise source employment locally.

Mexican Gold is very supportive of the community, with formalized support programs for the town and its schools. Contributions are typically made in kind in order to ensure proper disposition of funds and to avoid any perceived favoritism.

The schools have enjoyed support with sports and computer equipment as well as maintenance items such as paint, plumbing supplies, electrical work, and other items. Regular, small but needed donations are also given for school graduations, dances, and uniforms for the local soccer team. Small ‘stocking stuffer’ gifts such as toys, calculators etc. are donated for school children at Christmas (Figure 20-4).

Figure 20-4: Donation of Christmas Gifts to Kindergarten School

Source: Las Minas (2019)

The municipal office has received several sets of tools for road maintenance and garbage collection, suites of proper office furniture to replace their plastic lawn chairs and tables, and other items (Figure 20-5).

Figure 20-5: Donation of Maintenance Tools to the Municipality of Las Minas in 2020

Source: Las Minas (2020)

The Mexican Gold operations have integrated well to the community and are recognized as responsible employers and community members.

20.3.3 Land Use

Voluntary surface land use agreements have been negotiated with landowners within the exploration area prior to the start of exploration activities. All lands within the current project area are privately held. Land Use Agreements are held for a renewable annual, three-year, or five-year terms, at the preference of the owner. Outside of the project area, there is a small communally-held parcel (ejido) that is used for a playground at a school, and there is federally-owned land to the south associated with a hydroelectric project.

20.4 Mine Closure

Mexico does not have detailed reclamation legislation but has national environmental laws and is developing more specific mine closure requirements. Guidance for the construction, operation, and closure of tailings impoundments is included in a national regulation NOM-141-SEMARNAT-2003. Post operation criteria are presented in Section 5.7 of NOM-141-SEMARNAT-2003 and include the following:

- Dust is not emitted into the atmosphere as a result of the loss of moisture from the surface of the tailings dam or from the curtain wall, among others;
- Run-off does not affect surface water and groundwater;
- The tailings storage facility does not fail;
- The surface of the dump shall be covered with the recovered soil, when applicable, or with materials that allow plant species to take root; and
- The plant species that are used to cover the dump shall be native to the region, in order to guarantee their success and permanence with a minimum of conservation.

No mine reclamation bond is required in Mexico.

20.4.1 Closure Planning

The Closure Plan approach will be designed to ensure long term stability of both physical and chemical properties of the site, and to return the landscape to its pre-mining capability where possible. Specific closure items will include:

- All openings to underground workings will be sealed with cement plugs or barricades;
- The tailings storage facility will be a capped and vegetated landform capable of managing runoff from storm events;
- Reagents and supplies will be removed and will be returned to suppliers, sold to other operations, disposed of in approved waste facilities, or transported to a certified company for disposal;
- Equipment, conductors and other above ground facilities for the electrical supply will be dismantled or demolished; and
- All foundations will be demolished and covered to approximate as closely as possible the pre-mining landscape topography.

The approach will also incorporate community involvement to ensure that remaining infrastructure closure methods end land use objectives and are socially acceptable and in keeping with the broader land use planning of the area.

Progressive rehabilitation is currently integrated to the exploration phase and will be an important aspect of concurrent programs during operations in order to minimize final disturbance areas upon cessation of mining. The current program of successfully rehabilitating drill pads and other unused disturbance areas will form the basis of the approach for revegetation during operations. Rehabilitation of drill platforms is undertaken as part of the contract with the drilling company, who carry ISO environmental certification.

21 CAPITAL COST ESTIMATE

21.1 Capital Cost Summary

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

Total life of mine capital costs is estimated to be $145.1M Pre-production capital costs amount to $90.4M. Capital costs during production years, less closure total $44.7M. Closure costs have been estimated at $10M. These costs are summarized in Table 21-1. The project carried a blended contingency rate of 20%.

Table 21-1: Capital Cost Summary

Capital Costs	Pre-Production (M$)	Sustaining / Closure (M$)	Total (M$)
Mining	12.3	34.7	47.0
On-site Development	4.2	-	4.2
Ore Crushing & Handling	3.2	-	3.2
Tailings Management	2.3	1.1	3.4
Mineral Processing Plant	22.6	1.4	24.0
Infrastructure	12.0	-	12.0
Project Indirects	7.3	-	7.3
Engineering & EPCM	5.4	-	5.4
Owner's Costs	6.2	-	6.2
Closure	-	10.0	10.0
Subtotal	**75.3**	**47.2**	**122.6**
Contingency	15.1	7.4	22.5
Total Capital Costs	**90.4**	**54.7**	**145.1**

21.2 Basis of Estimate

Costs are expressed in US$ with no escalation unless stated otherwise. Foreign exchange rates of C$$1.32:US$1.00 and MX20.00:US$1.00 are used where applicable. The cost estimate base date is Q3 2021 and consists of direct costs and indirect costs:

- Direct costs: Costs of all permanent equipment and bulk materials and the installation costs for all permanent facilities including contractor’s supervision and management costs, contractor's travelling costs and contractor’s administration and profits;
- Indirect costs: Costs of EPCM services, construction accommodation, temporary construction facilities and services, construction equipment, freight, vendor erection supervision, commissioning and start-up, first fills and spares; and
- Owner’s costs: Costs associated with owner’s facilities and services during construction, owner’s project management, ramp up and general fees.

Owner’s costs include the following:

- Owner’s team including construction, start-up and commissioning;
- Recruiting, training and site visits;
- IT and communications; and
- Insurance, finance, legal and office.

Contingency: A construction contingency to cover necessary work within the defined scope of the Project which cannot be identified or itemized at this stage of the Project development but is expected to be incurred.

The estimate conforms to +/- 30% which represents a Preliminary Economic Assessment level estimate.

21.3 Mine Capital Cost Estimate

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases, and comparison to similar mines in the project area. Table 21-2 summarizes the mine capital cost estimate for the Las Minas Project.

Table 21-2: Mining CAPEX Summary

Description	Unit	Initial	Sustaining	LOM
Mine Equipment	$M	$ 7.26	$ 27.36	**$ 34.62**
Mine Development	$M	$ 3.99	$ 4.29	**$ 8.28**
Mine Infrastructure	$M	$ 1.02	$ 3.07	**$ 4.10**
Total	**$M**	**$ 12.27**	**$ 34.72**	**$ 46.99**

21.3.1 Mobile Equipment Purchase and Replacement

Underground mining equipment quantities and costs were determined from mine schedule, material handling requirements, and estimated equipment utilization. Budgetary quotes were utilized to determine equipment unit cost. Due to the mine life, it is not anticipated that equipment will be replaced, however sustaining cost for rebuilds has been included.

Table 21-3: Mine Equipment Capital Cost Estimate

Mine Equipment	Unit	Initial	Sustaining	LOM
Load & Haul	$M	$ 3.46	$ 4.68	**$ 8.15**
Drilling	$M	$ 1.32	$ 5.37	**$ 6.69**
Charging	$M	$ 0.07	$ 0.55	**$ 0.61**
Ground Support	$M	$ 0.92	$ 0.92	**$ 1.83**
Services	$M	$ 1.38	$ 1.78	**$ 3.16**
Auxiliary	$M	$ 0.11	$ 0.11	**$ 0.22**
Equipment Rebuilds	$M	$ 0.00	$ 13.96	**$13.96**
Equipment Replacement	$M	$ 0.00	$ 0.00	**$ 0.00**
Equipment Total	**$M**	**$ 7.26**	**$ 27.36**	**$ 34.62**

21.3.2 Lateral and Vertical Capital Waste Development

Capital waste development represents the mine's permanent infrastructure and includes the main access ramp, ventilation raise accesses, level accesses, sumps, ore pass accesses and permanent explosive storage cut-outs, as well as main ventilation raises. Underground capital lateral and capital vertical waste development is capitalized as a sustaining cost for time after pre-production and will not appear as an operating cost. Mineralized development and short-term access are included as an operating cost.

With the exception of waste crosscuts into mineralization, lateral development in waste rock has been capitalized. Underground infrastructure, including development necessary to place the underground crusher, has been considered capital projects. Capitalized operating expenses are defined as mine operating costs (development, material extraction, mine maintenance, and mine general costs) incurred prior to and during commissioning and ceasing at the commencement of commercial operations and general project revenues. Once plant feed is processed, these costs transition to operating expenses.

Table 21-4: Development Cost Estimate

Mine Development	Unit	Initial	Sustaining	LOM
Capital Lateral Development	$M	$ 2.00	$ 3.56	**$ 5.57**
Capital Vertical Development	$M	$ 0.00	$ 0.73	**$ 0.73**
Capitalized Operating Expenses	$M	$ 1.98	$ 0.00	**$ 1.98**
Mine Development Total	**$M**	**$ 3.99**	**$ 4.29**	**$ 8.26**

21.3.3 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling. Allowances have been made for miscellaneous items such as PPE, radios, water supply, refuge stations, and geotechnical investigations.

Table 21-5: Mine Infrastructure Cost Estimate

Mine Infrastructure	Unit	Initial	Sustaining	LOM
Portal	$M	$ 0.40	$ 0.15	**$ 0.55**
Dewatering	$M	$ 0.03	$ 0.66	**$ 0.69**
Electrical Distribution	$M	$ 0.05	$ 1.51	**$ 1.56**
Ventilation	$M	$ 0.50	$ 0.73	**$ 1.23**
Paste Backfill Borehole	$M	$ 0.04	$ 0.03	**$ 0.07**
Mine Infrastructure Total	**$M**	**$ 1.02**	**$ 3.07**	**$ 4.10**

21.4 TSF Capital Cost Estimate

The PEA level design was used to prepare cost estimates for initial capital and sustaining capital/ operating expenses for the TSF. Quantities were developed by KP using layouts generated in AutoCAD Civil 3D and topography provided by JDS. Unit rates were provided by PMICSA in US$ and have been reviewed by JDS and KP. Some adjustments have been made to certain PMICSA rates. The cost estimate is summarized as follows and shown on Table 21-6.

- Initial Capital: US$2.3M; and
- Sustaining Capital/ Operating Costs: US$ 1.1M.

Table 21-6: TSF Cost Estimate

Item Number	Description	Units	Unit Cost	Initial Capital (Stage 1 Embankment)		Sustaining Capital / Operating Costs (Expansions)	
				Quantity	Cost	Quantity	Cost
1000	**Tailings Storage Facility**						
1100	**TSF Earthworks**						
1110	Foundation Preparation						
1111	Clearing/Grubbing of TSF Embankment and Basin Footprint	m^2	$0.70	37,500	$26,250	50,500	$35,350
1112	Topsoil Stripping of TSF Embankment Footprint	m^3	$6.33	20,500	$129,765	2,000	$12,660
1120	Material Development and Fill Placement						
1121	Rockfill - Excavate, Load, Haul, Spread and Compact (6-inch minus)	m^3	$9.51	109,651	$1,042,781	0	$ -
1122	Transition Zone - Excavate Load, Haul, Place, Spread and Compact (3-inch minus)	m^3	$11.55	20,500	$236,775	0	$ -
1124	Structural Shell Zone Fill Zone - Excavate, Load, Haul, Spread and Compact	m^3	$11.55	0	$ -	68,000	$785,400
1140	Geotechnical Instrumentation	LS	$50,000	1	$50,000	1	$50,000
1150	Foundation Drain System						
1152	Drainage Layer - Excavate, Process, Load, Haul, Place, Spread and Compact (1-inch to 3-inch)	m^3	$13.34	1,500	$20,010	875	$11,673
1153	Filter Sand Zone - Excavate, Process, Load, Haul, Place, Spread and Compact	m^3	$13.34	1,500	$20,010	875	$11,673
1154	Geotextile Wrap for Filter Sand Zone - Supply and Install	m^2	$1.75	7,200	$12,600	28,800	$50,400
	SUB-TOTAL ITEM 1100				**$1,538,191**		**$957,155**

Item Number	Description	Units	Unit Cost	Initial Capital (Stage 1 Embankment)		Sustaining Capital / Operating Costs (Expansions)	
				Quantity	Cost	Quantity	Cost
1300	**TSF Water Management**						
1310	Diversion and Runoff Collection Channels	m	$185.92	1,150	$213,808	400	$74,368
1330	Seepage Collection Pond (Seepage recovery and recycle system)	LS	$312,035	1	$312,035	0	$ -
1340	Construction Dewatering Allowance	LS	$98,743	1	$98,743	0	$ -
1350	Sediment and Erosion Control BMPs	LS	$195,574	0.5	$97,787	0.5	$97,787
	SUB-TOTAL ITEM 1300				**$722,373**		**$172,155**
1500	**Operating Costs**						
1540	Filtered Tailings Placement						
1541	Load, Haul, Place and Compact Filtered Tailings	m^3	$1.16	-	$ -	-	$ -
	SUB-TOTAL ITEM 1500				**$ -**		**$ -**
	SUBTOTAL				**$2,260,564**		**$1,129,310**

22 OPERATING COST ESTIMATE

22.1 Operating Cost Summary

The operating cost estimate was prepared using first principles, applying project experience, and avoiding the use of general industry factors. Inputs are derived from engineers, contractors and suppliers who have provided similar services to other projects. The operating cost is based on owner owned and operated mining/services fleets and minimal use of permanent contractors except where value is provided through expertise and/or packages efficiencies/skills.

Operating costs in this section of the report include mining, processing, tailings, and administration up to the production of concentrate from the site. Mine operating costs incurred during the construction phase (pre-production Years -2 and -1) are capitalized and form part of the capital cost estimate. Concentrate transportation, treatment and refining charges, and royalties are discussed in Section 19.

Operating costs are presented in 2021 US dollars on a calendar year basis. No escalation or inflation is included. Average annual operating costs over the life of mine are $28M and are summarized in Table 22-1. Labour requirements have been estimated for the major cost areas and the annual average is presented in Table 22-2.

Table 22-1: Breakdown of Estimated Operating Costs

Operating Costs	$/t Milled	LOM M$
Mining	35.83	145
Processing	14.55	59
G&A	7.37	30
Total	**57.76**	**234**

Table 22-2: Summary of Personnel Requirements

Position	Annual Average
Mining	
Mine General	13
Mining Operations	120
Mine Maintenance	48
Technical Services	18
Total Mining Personnel	**199**

Position	Annual Average
Processing	
Process Plant	78
Technical Services	16
Total Process Plant Personnel	**94**
G&A	
Total General & Administration	**150**
Total Site	
Total Personnel - All Areas	**443**

22.2 Basis of Estimate

Costs are expressed in US$ with no escalation unless stated otherwise. Foreign exchange rates of C$$1.32:US$1.00 and MX20.00:US$1.00 are used where applicable. The cost estimate base date is Q3 2021. The estimate conforms to +/- 30% which represents a Preliminary Economic Assessment (PEA) level estimate. Key operating cost component assumptions are listed in Table 22-3.

Table 22-3: Key OPEX Component Assumptions

Item	Unit	Value
Electrical power cost	$/kWh	0.104
Average power consumption	MW	4.8
Diesel cost (delivered)	$/litre	0.97
LOM average manpower (including contractors, excluding corporate)	employees	443

22.3 Mine Operating Cost Estimate

Mine operating costs refer to expenses incurred including all activities directly related to the drilling, blasting, loading, and hauling of mill feed to the process plant or underground crushing facility, as well as waste and tailings storage.

The mine operating costs include the following functional areas:

- Production - costs associated directly with the drilling, blasting, and mucking of the mineable resource. This includes lateral development through mineralization;

- Lateral Waste Development – costs related to non-capitalized development. A total of 4,622 m of lateral waste is classified as operating costs over the LOM;

- Materials Handling – Costs associated with haulage of material. Mineralized haulage to crushing facility, waste to backfill or surface storage;

- Backfill – direct cost related to paste backfill;

- Tailings Construction & Handling – includes costs to haul and place rockfill from the mine to construct TSF as well as cost to haul and place dewatered tailings within TSF;

- Mine Maintenance – direct costs related to the maintenance of underground fixed and mobile equipment; and

- Mine General – costs related to mine support activities such as supervision, technical services, shared infrastructure, support equipment, power, and material delivery underground.

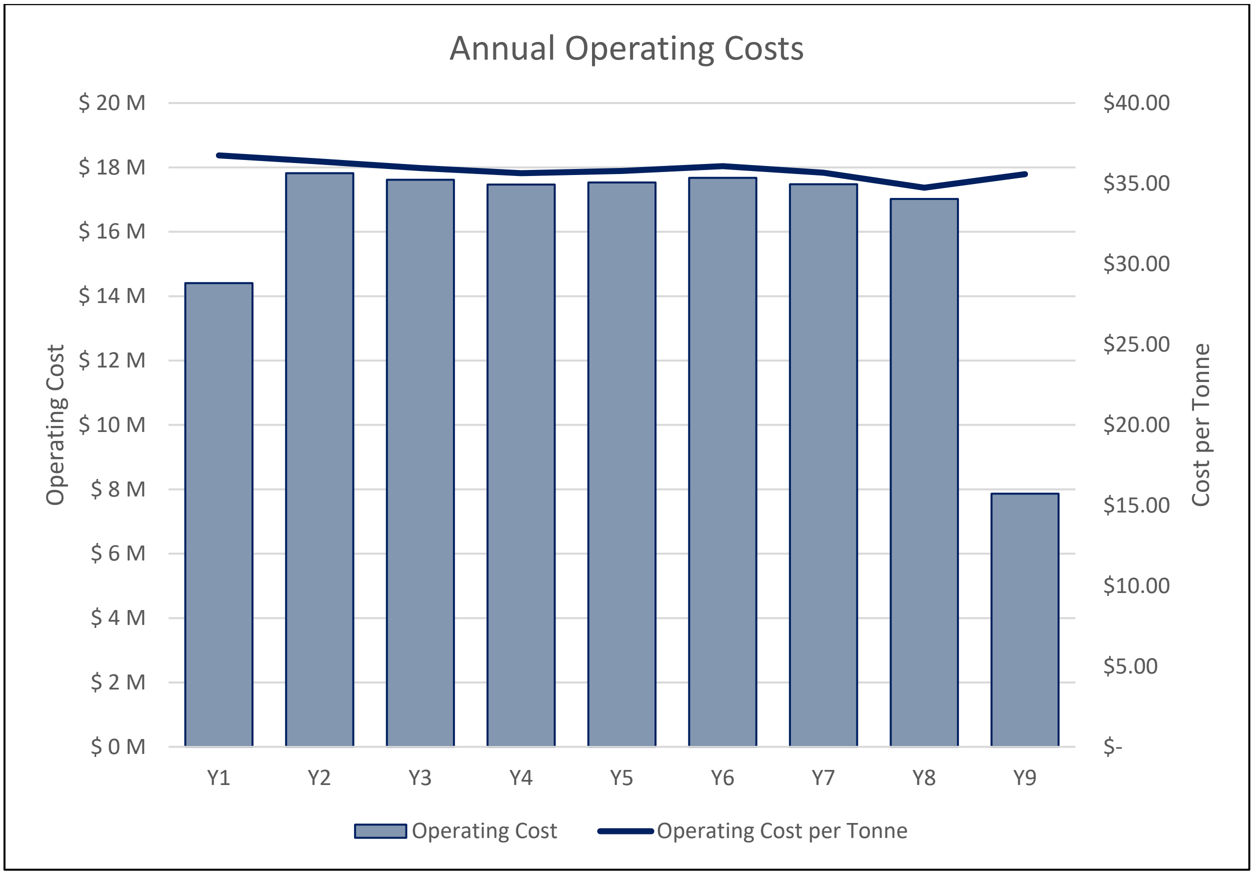
Capital waste development represents the mine's permanent infrastructure and includes the main access ramp, ventilation raise accesses, level accesses, sumps, ore pass accesses and permanent explosive storage cut-outs, as well as main ventilation raises. Underground capital lateral and capital vertical waste development is capitalized as a sustaining cost and will not appear as an operating cost. Mineralized development and short-term access will be included as an operating cost.

Total mine operating costs are summarized in Table 22-4 and presented graphically by year in Figure 22-1.

Table 22-4: Summary of Mine OPEX Estimate

Description	Production ($M)	Unit Rate ($/t)
Production	72.7	17.98
Lateral Waste Development	5.4	1.34
Materials Handling	23.5	5.80
Backfill	7.2	1.79
Tailings Construction & Handling	4.1	1.02
Mine Maintenance	5.6	1.39
Mine General	26.3	6.50
Total	**144.9**	**35.83**

Figure 22-1: Annual Mine Operating Cost Estimate



22.4 Processing Operating Cost Estimate

The mill operating costs for the 1,400 t/d mill have been estimated to be US$14.55/t of mill feed. The breakdown of the costs can be found in Table 22-5.

The costs for processing include crushing, grinding, flotation, magnetic separation, dewatering flotation and magnetic separation concentrates. The costs cover operations, maintenance, and technical support labour as well as consumables, supplies and contingency.

Table 22-5: Summary of Processing OPEX Estimate

Processing Operating Costs	Production ($M)	$/t Processed	% of Total
Labour	12.4	3.08	21%
Power & Fuel	23.9	5.92	41%
Maintenance	10.4	2.58	18%
Operations	9.1	2.26	15%
Assay Lab	2.9	0.71	5%
Total	**58.8**	**14.55**	**100%**

22.5 General and Administration Operating Cost Estimate

The costs of general and administrative (G&A) expenses include administration, accounting, human resources, community relations, health and safety, environment, communications, procurement and logistics, security, warehousing, site services, camp catering and cleaning, water treatment and insurance. Average camp requirements are expected to be approximately 233 people. The total G&A costs are summarized in Table 22-6.

Table 22-6: Summary of General & Administrative OPEX Estimate

Processing Operating Costs	Production ($M)	$/t Processed	% of Total
Labour	13.7	3.38	46%
Expenses	12.3	3.04	41%
Vehicle Operations	2.0	0.50	7%
Power	0.6	0.15	2%
Supplies	1.2	0.30	4%
Total	**29.8**	**7.37**	**100%**

23 ECONOMIC ANALYSIS

An economic model was developed for the Las Minas project to estimate annual cash flows and sensitivities. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. 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 variation in metal price, foreign exchange rate, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

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

This technical report contains forward-looking information regarding commodity price assumptions, projected mine production rates, construction schedules and forecasts of resulting cash flows. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labor 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.

23.1 Basis of Analysis

All costs, commodity pricing and economic results are reported in United States dollars ($), unless otherwise noted. Table 23-1 outlines the planned LOM production and grade estimates.

Table 23-1: LOM Production Summary

Parameter	Unit	Value
Mine Life	Years	8.5
Resources Processed	M t	4.0
Average Au Head Grade	g/t	1.84
Average Au Head Grade	g/t	5.5
Average Cu Head Grade	%	1.06
Average Magnetite Head Grade	%	15.7
Au Payable	k oz	180
Ag Payable	k oz	202
Cu Payable	M lbs	81
Iron Concentrate Payable	k dmt	637

Other economic factors used in the economic analysis include the following:

- Nominal 2021 dollars;
- No inflation;
- Taxes (discussed in Section 23.2);
- Numbers are presented on a 100% ownership basis and do not include financing costs;
- Revenues were modeled to occur in the period when the material was processed;
- Costs and taxes are calculated for each period in which they occur;
- Costs from operations incurred in the pre-production period have been capitalized;
- Exclusion of sunk costs (i.e., exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, construction etc.) However, pre-development costs are utilized for tax deductions; and
- Exclusion of any servicing of the debt incurred to finance the Project.

Mine revenue is derived from the sale of a copper concentrate containing gold and silver, and an iron concentrate into the international marketplace.

Metal prices used in the economic model are between the current trailing 2-year and 3-year averages and were held constant over the LOM. The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely 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. It is expected that there will be variability in the metal prices throughout the planned mine life. For the purposes of the model, this variability was not incorporated and has been disregarded.

Table 23-2 outlines the market terms used in the economic analysis. Figure 23-1 illustrates the annual and cumulative payable metal in terms of payable gold equivalent by project year. The project is expected to produce an approximate gold equivalent of 47k oz per year for a total of 383 k oz over the LOM. Gold and Copper generate over 95% of the project net revenues. The total Net Smelter Return by commodity is shown in Figure 23-2. The project is not currently subject to any third-party corporate royalties.

Table 23-2: NSR Assumptions used in the Economic Analysis

Assumptions	Unit	Value
Gold Price	US$/oz	1,625
Silver Price	US$/oz	20
Copper Price	$US/lb	3.25
Iron Concentrate Price	$US/dmt	100
Au Payable	%	94
Au Refining Charge	$/oz	5.00
Ag Payable	%	40
Ag Refining Charge	$/oz	0.40
Cu Payable	%	95
Cu Treatment Charge	$/dmt	65
Cu Refining Charge	$/lb	0.065
Cu Transportation Charge	$/wmt	56
Fe Concentrate Payable	%	100
Fe Transportation Charge	$/wmt	56

Figure 23-1: Annual and Cumulative Payable Gold Equivalent Production

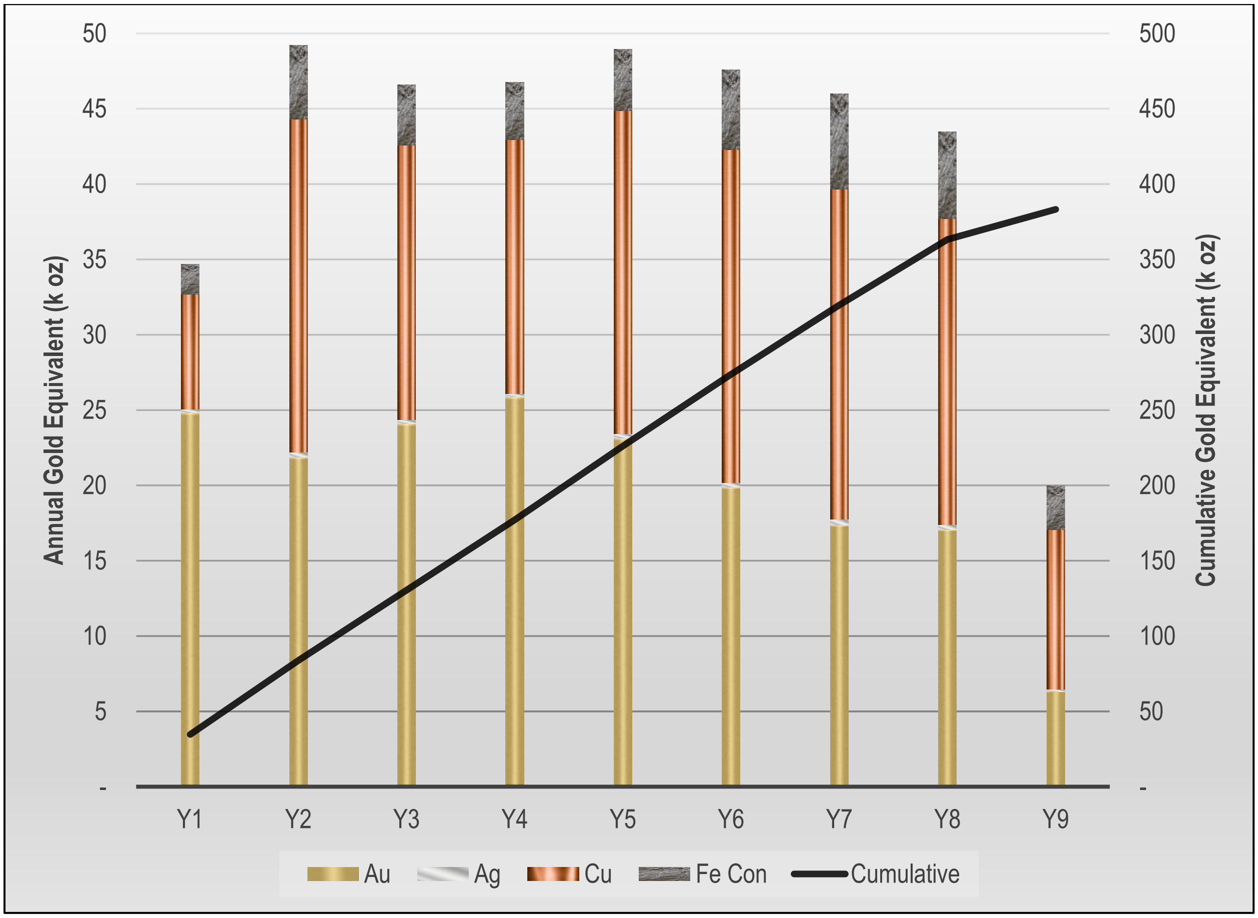
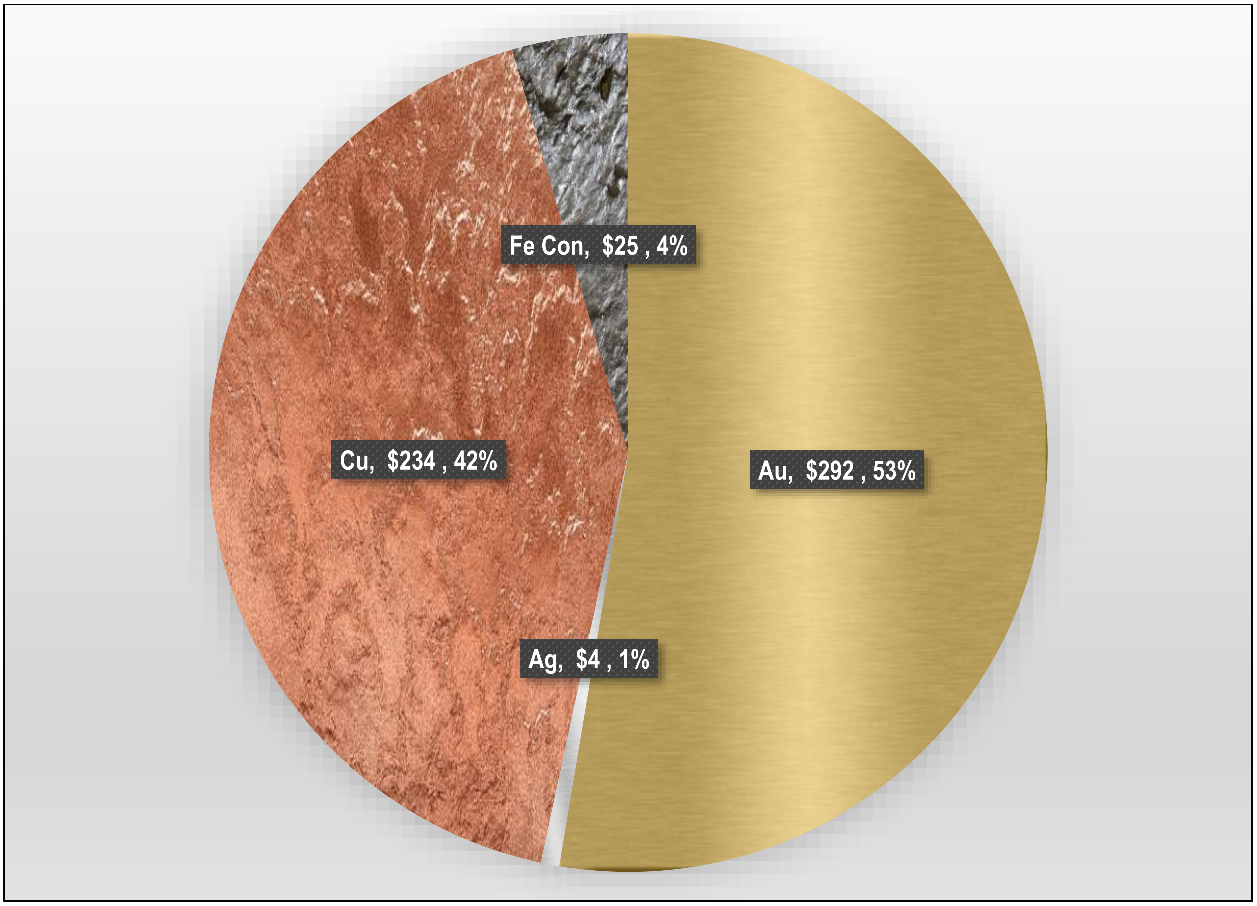


Figure 23-2: Total Net Smelter Return by Commodity (US$M)



23.2 Taxes

Mexican taxes include a 7.5% Special Mining Duty (SMD) on EBITDA, a 0.5% Extraordinary Mining Duty on income from Gold and Silver, and a Corporate Income tax rate of 30% after deductions for the SMD, EMD and asset depreciation. Depreciation allowances for Mexico are based on a straight-line methodology and typically vary from 5% to 12% allowance per year. It has been estimated that approximately US$77M will be paid in taxes over the project life.

23.3 Results

The Las Minas project has an after-tax net IRR 16% of and a net present value at 8% of $35 M. Figure 23-3 shows the projected annual cash flows used in the economic analysis. Table 23-3 summarizes the overall economic results. The cash flow model is shown in Table 23-4.

Figure 23-3: Annual and Cumulative After-Tax Cash Flows

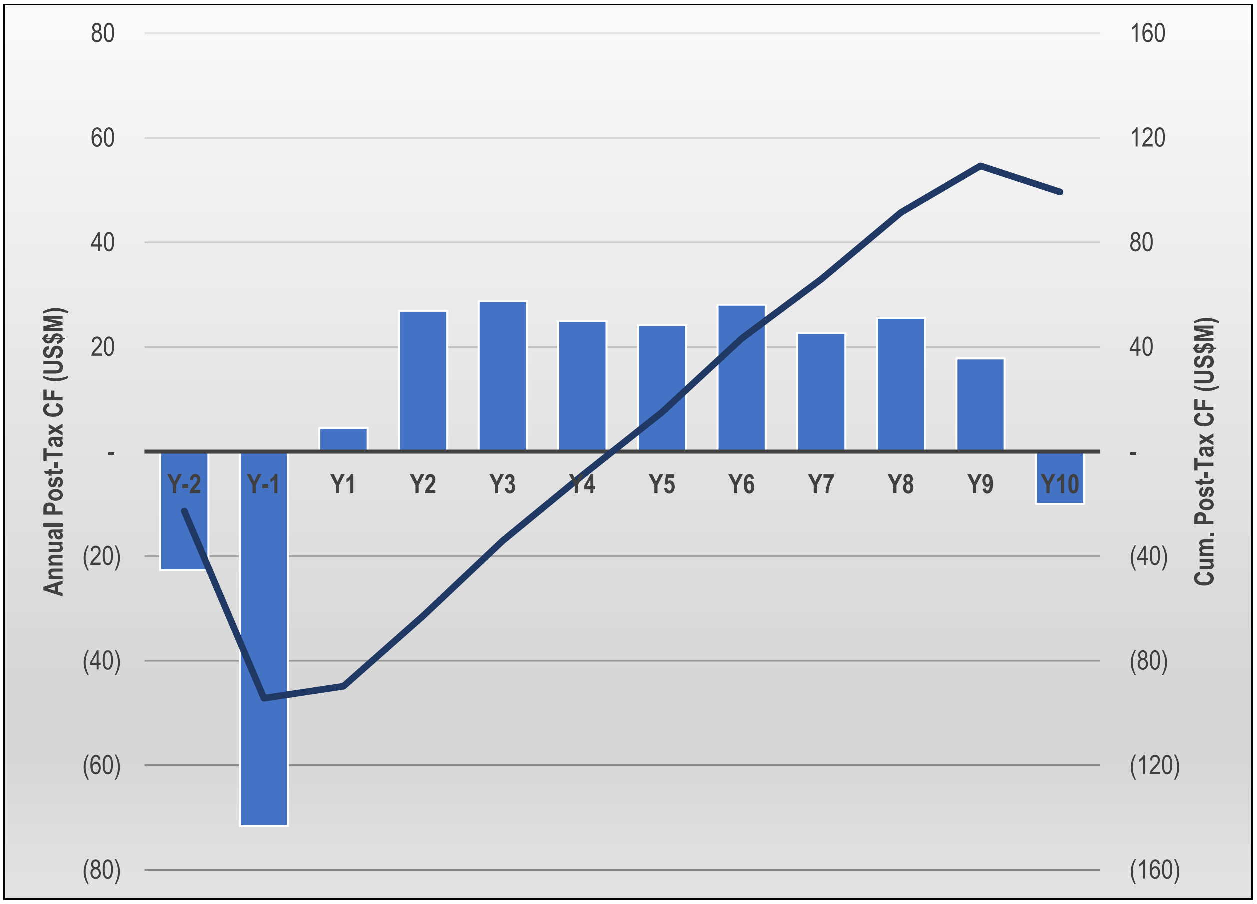


Table 23-3: Summary of Economic Results

Category	Unit	Value
Revenues	M $	623
Operating Costs	M $	234
Treatment/Refining/Transportation	M $	68
Cash Flow from Operations	M $	322
Initial Capital Costs	M $	90
Sustaining and Closure	M $	55
All-in Sustaining Cost# (net of by-product credits)	$/oz Au	145
All-in Sustaining Cost° (gold equivalent)	$/oz AuEq	928
Net Pre-Tax Cash Flow	M $	177
Pre-Tax NPV$_{5\%}$	**M $**	**114**

Category	Unit	Value
Pre-Tax $NPV_{8\%}$	**M $**	**86**
Total Taxes	M $	77
Net After-Tax Cash Flow	M $	99
Net After-Tax $NPV_{5\%}$	**M $**	**55**
Net After-Tax $NPV_{8\%}$	**M $**	**35**

Notes:

[#] AISC formula: (Operating Costs + Refining Costs + Sustaining Capital + Closure – Net by-product credits) / Payable Au oz

° AISC formula: (Operating Costs + Refining Costs + Sustaining Capital + Closure) / Payable AuEq oz

Table 23-4: Cash Flow Model

	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
METAL PRICES & F/X RATE														
Au	US$/oz	**1,625**	1,625	1,625	1,625	1,625	1,625	1,625	1,625	1,625	1,625	1,625	1,625	1,625
Ag	US$/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
Cu	US$/lb	**3.25**	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25
Fe Con	US$/dmt	**100.00**	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
MILL SCHEDULE														
Resources Processed	**k tonnes**	**4,043**	-	-	392	490	490	490	490	490	490	490	221	-
Au	g/t	**1.84**	-	-	2.56	1.85	2.02	2.16	1.95	1.68	1.48	1.45	1.21	-
Ag	g/t	**5.53**	-	-	4.54	6.08	4.84	4.15	5.34	5.80	6.87	5.86	6.80	-
Cu	%	**1.06**	0.00	0.00	0.52	1.20	0.99	0.91	1.16	1.20	1.19	1.10	1.27	0.00
Magnetite	%	**15.7**	0.00	0.00	8.23	16.17	13.37	12.63	13.51	17.57	20.96	19.11	21.58	0.00
Contained Metal														
Au	k oz	**239**	-	-	32	29	32	34	31	27	23	23	9	-
Ag	k oz	**719**	-	-	57	96	76	65	84	91	108	92	48	-
Cu	M lbs	**94**	-	-	4	13	11	10	13	13	13	12	6	-
Magnetite	k tonnes	**635**	-	-	32	79	66	62	66	86	103	94	48	-
RECOVERED AND PAYABLE METALS														
Recovery to Cu Concentrate														
Au	%	**80.0**	0.0	0.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	0.0
Ag	%	**70.0**	0.0	0.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	0.0
Cu	%	**90.0**	0.0	0.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	0.0
Metal Recovered														
Au	k oz	**191**	-	-	26	23	25	27	25	21	19	18	7	-
Ag	k oz	**503**	-	-	40	67	53	46	59	64	76	65	34	-
Cu	M lbs	**85**	-	-	4	12	10	9	11	12	12	11	6	-
Copper Concentrate														
Dry Mass	dmt	**177,389**	-	-	8,391	24,342	20,061	18,550	23,613	24,330	24,111	22,363	11,629	-
Au Grade	g/t Au	**33.5**	-	-	95.7	29.8	39.4	45.6	32.4	27.1	24.0	25.4	18.4	-
Ag Grade	g/t Ag	**88.2**	-	-	148.4	85.6	82.7	76.8	77.5	81.8	97.7	89.8	90.5	-
Cu Grade	% Cu	**21.7**	0.0	0.0	21.7	21.7	21.7	21.7	21.7	21.7	21.7	21.7	21.7	0.0
Au Payable														
Payable Metal	k oz	**180**	-	-	25	22	24	26	23	20	17	17	6	-
Revenues	US$M	**293.1**	-	-	40.4	35.5	39.1	42.0	37.6	32.3	28.2	27.7	10.3	-
Ag Payable														
Payable Metal	k oz	**202**	-	-	19	27	21	17	22	25	32	26	14	-
Revenues	US$M	**4.0**	-	-	0.4	0.5	0.4	0.3	0.4	0.5	0.6	0.5	0.3	-

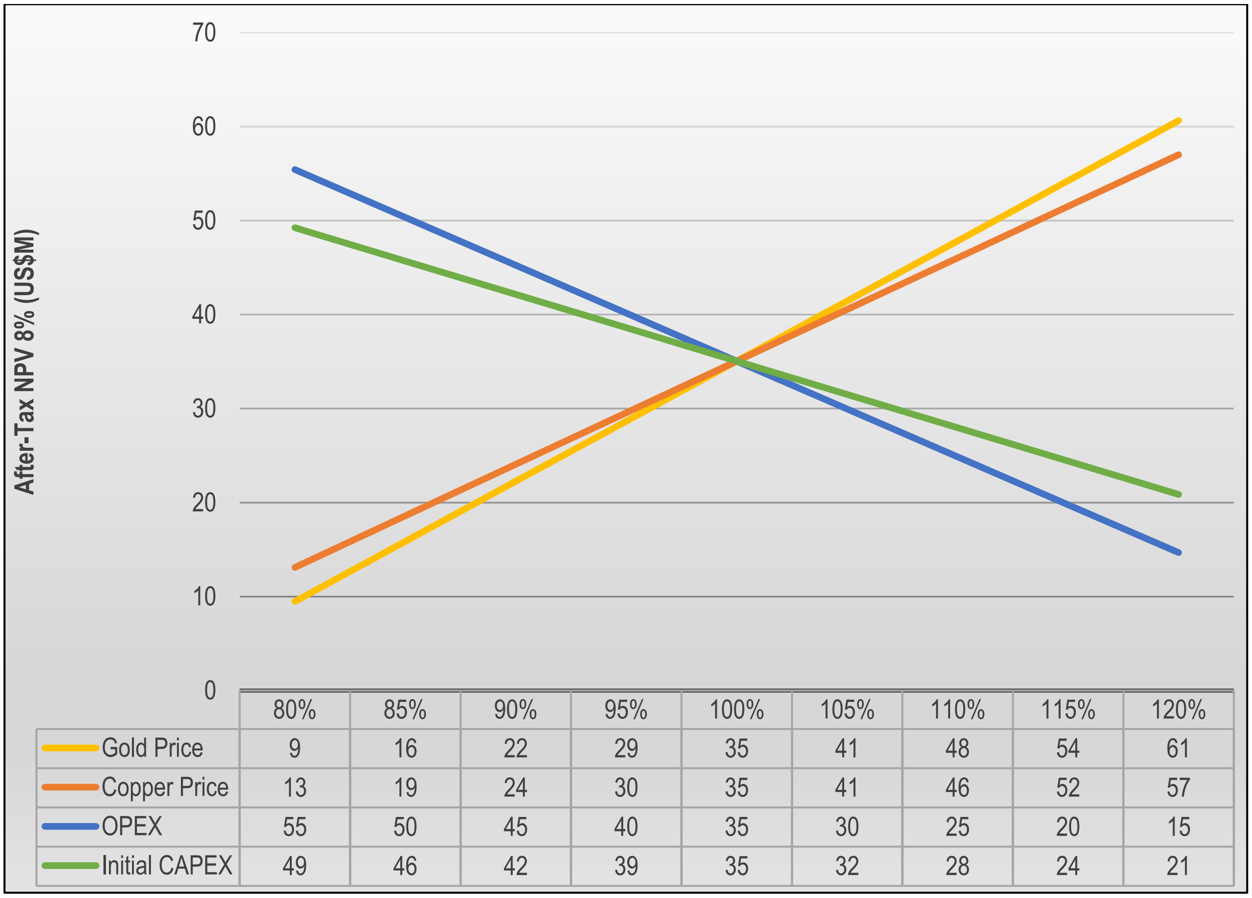
	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Cu Payable														
Payable Metal	M lbs	**81**	-	-	4	11	9	8	11	11	11	10	5	-
Revenues	US$M	**261.8**	-	-	12.4	35.9	29.6	27.4	34.9	35.9	35.6	33.0	17.2	-
Total Payables	US$M	**559.0**	**-**	**-**	**53.1**	**72.0**	**69.2**	**69.8**	**72.9**	**68.7**	**64.4**	**61.2**	**27.7**	**-**
Treatment Charge	US$M	**11.5**	-	-	0.5	1.6	1.3	1.2	1.5	1.6	1.6	1.5	0.8	-
Au Refining Charge	US$M	**0.9**	-	-	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	-
Ag Refining Charge	US$M	**0.1**	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-
Cu Refining Charge	US$M	**5.2**	-	-	0.2	0.7	0.6	0.5	0.7	0.7	0.7	0.7	0.3	-
Concentrate Transportation Cost	US$M	**10.8**	-	-	0.5	1.5	1.2	1.1	1.4	1.5	1.5	1.4	0.7	-
Total TC/RC & Transport	**US$M**	**28.6**	**-**	**-**	**1.4**	**3.9**	**3.3**	**3.0**	**3.8**	**3.9**	**3.9**	**3.6**	**1.8**	**-**
Royalties	US$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-
Copper Concentrate NSR	**US$M**	**530.4**	**-**	**-**	**51.7**	**68.1**	**65.9**	**66.7**	**69.1**	**64.8**	**60.6**	**57.7**	**25.9**	**-**
	US$/t	**131.2**	**-**	**-**	**131.9**	**139.0**	**134.5**	**136.2**	**141.0**	**132.2**	**123.6**	**117.7**	**117.0**	**-**
Iron Concentrate														
Magnetite Recovery	%	**90.3**	0	0	90	90	90	90	90	90	90	90	90	0
Magnetite Recovered	k tonnes	**574**	-	-	29	72	59	56	60	78	93	85	43	-
Concentrate Grade	% Magnetite	**90**	0	0	90	90	90	90	90	90	90	90	90	0
Dry Mass	dmt	**637,335**	-	-	32,385	79,495	65,717	62,065	66,435	86,391	103,028	93,961	47,857	-
Revenues	US$M	**63.7**	-	-	3.2	7.9	6.6	6.2	6.6	8.6	10.3	9.4	4.8	-
Concentrate Transportation Cost	US$M	**38.9**	-	-	2.0	4.9	4.0	3.8	4.1	5.3	6.3	5.7	2.9	-
Royalties	US$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-
Iron Concentrate NSR	**US$M**	**24.8**	**-**	**-**	**1.3**	**3.1**	**2.6**	**2.4**	**2.6**	**3.4**	**4.0**	**3.7**	**1.9**	**-**
	US$/t	**6.1**	**-**	**-**	**3.2**	**6.3**	**5.2**	**4.9**	**5.3**	**6.9**	**8.2**	**7.5**	**8.4**	**-**
Total NSR	**US$M**	**555.2**	**-**	**-**	**52.9**	**71.2**	**68.5**	**69.2**	**71.7**	**68.2**	**64.6**	**61.3**	**27.7**	**-**
	US$/tonne	**137.33**	**0.00**	**0.00**	**135.07**	**145.29**	**139.74**	**141.13**	**146.29**	**139.11**	**131.79**	**125.15**	**125.41**	**0.00**
OPEX														
Mining	US$M	**144.9**			14.4	17.8	17.6	17.5	17.5	17.7	17.5	17.0	7.9	-
	US$/t processed	**35.83**	-	-	36.74	36.36	35.95	35.64	35.78	36.08	35.67	34.74	35.58	-
Processing	US$M	**58.8**	-	-	5.7	7.1	7.1	7.1	7.1	7.1	7.1	7.1	3.2	-
	US$/t processed	**14.55**	-	-	14.55	14.55	14.55	14.55	14.55	14.55	14.55	14.55	14.55	-
Tailings & Backfill (Covered by Mining)	US$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-
	US$/t processed	**0.00**	-	-	-	-	-	-	-	-	-	-	-	-
G&A	US$M	**29.8**	-	-	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	1.6	-
	US$/t processed	**7.37**	-	-	8.99	7.19	7.19	7.19	7.19	7.19	7.19	7.19	7.19	-
Total OPEX	**US$M**	**233.5**	**-**	**-**	**23.6**	**28.5**	**28.3**	**28.1**	**28.2**	**28.3**	**28.1**	**27.7**	**12.7**	**-**
	US$/t processed	**57.76**	**-**	**-**	**60.29**	**58.11**	**57.70**	**57.39**	**57.53**	**57.83**	**57.42**	**56.49**	**57.32**	**-**

	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Net Operating Income	US$M	321.7	-	-	29.3	42.7	40.2	41.0	43.5	39.8	36.4	33.6	15.0	-
	US$/tonne	79.57	-	-	74.78	87.18	82.04	83.74	88.77	81.28	74.38	68.67	68.08	-
CAPEX														
Initial & Sustaining Capital														
Mining	US$M	**47.0**		12.3	15.2	3.5	0.4	4.0	6.2	1.2	3.9	0.2	-	-
On-Site Development	US$M	**4.2**	1.2	2.9										
Ore Crushing & Handling	US$M	**3.2**	1.0	2.2										
Tailing Management	US$M	**3.4**	0.7	1.6		0.2	0.2	0.2	0.2	0.2	0.2	0.2	-	-
Mineral Processing Plant	US$M	**24.0**	6.8	15.8		0.2	0.2	0.2	0.2	0.2	0.2	0.2	-	-
Infrastructure	US$M	**12.0**	3.6	8.4										
Indirects	US$M	**7.3**	2.2	5.1										
EPCM	US$M	**5.4**	1.6	3.8										
Owners Costs	US$M	**6.2**	1.9	4.4										
Subtotal Pre-Contingency	US$M	**112.6**	18.9	56.4	15.2	3.9	0.8	4.4	6.6	1.6	4.3	0.6	-	-
Contingency	US$M	**22.5**	3.8	11.3	3.0	0.8	0.2	0.9	1.3	0.3	0.9	0.1	-	-
Total - Initial & Sustaining Capital	**US$M**	**135.1**	**22.7**	**67.7**	**18.3**	**4.7**	**0.9**	**5.3**	**7.9**	**1.9**	**5.2**	**0.7**	**-**	**-**
Mine Closure & Monitoring Costs	US$M	**10.0**				-	-	-	-	-	-	-	-	10.0
Total CAPEX	**US$M**	**145.1**	**22.7**	**67.7**	**18.3**	**4.7**	**0.9**	**5.3**	**7.9**	**1.9**	**5.2**	**0.7**	**-**	**10.0**
Pre-Production	**US$M**	**90.4**	**22.7**	**67.7**										
Sustaining	**US$M**	**54.7**			**18.3**	**4.7**	**0.9**	**5.3**	**7.9**	**1.9**	**5.2**	**0.7**	**-**	**10.0**
Working Capital	US$M	**0.0**		3.9	-	-	-	-	-	-	-	-	(3.9)	-
Net Pre-Tax Cash Flow	US$M	176.6	- 22.7	- 71.6	11.1	38.1	39.3	35.8	35.6	38.0	31.3	33.0	19.0	- 10.0
Cumulative Net Pre-Tax Cash Flow	US$M	176.6	- 22.7	- 94.3	- 83.3	- 45.2	- 5.9	29.8	65.4	103.4	134.7	167.6	186.6	176.6
Taxes	US$M	**77.4**	-	-	6.5	11.1	10.5	10.8	11.4	9.8	8.6	7.4	1.2	-
Net Post-Tax Cash Flow	US$M	99.3	- 22.7	- 71.6	4.5	26.9	28.8	25.0	24.2	28.1	22.7	25.6	17.8	- 10.0
Cumulative Net Post-Tax Cash Flow	US$M	99.3	- 22.7	- 94.3	- 89.8	- 62.9	- 34.1	- 9.1	15.1	43.2	65.9	91.5	109.3	99.3

23.4 Sensitivities

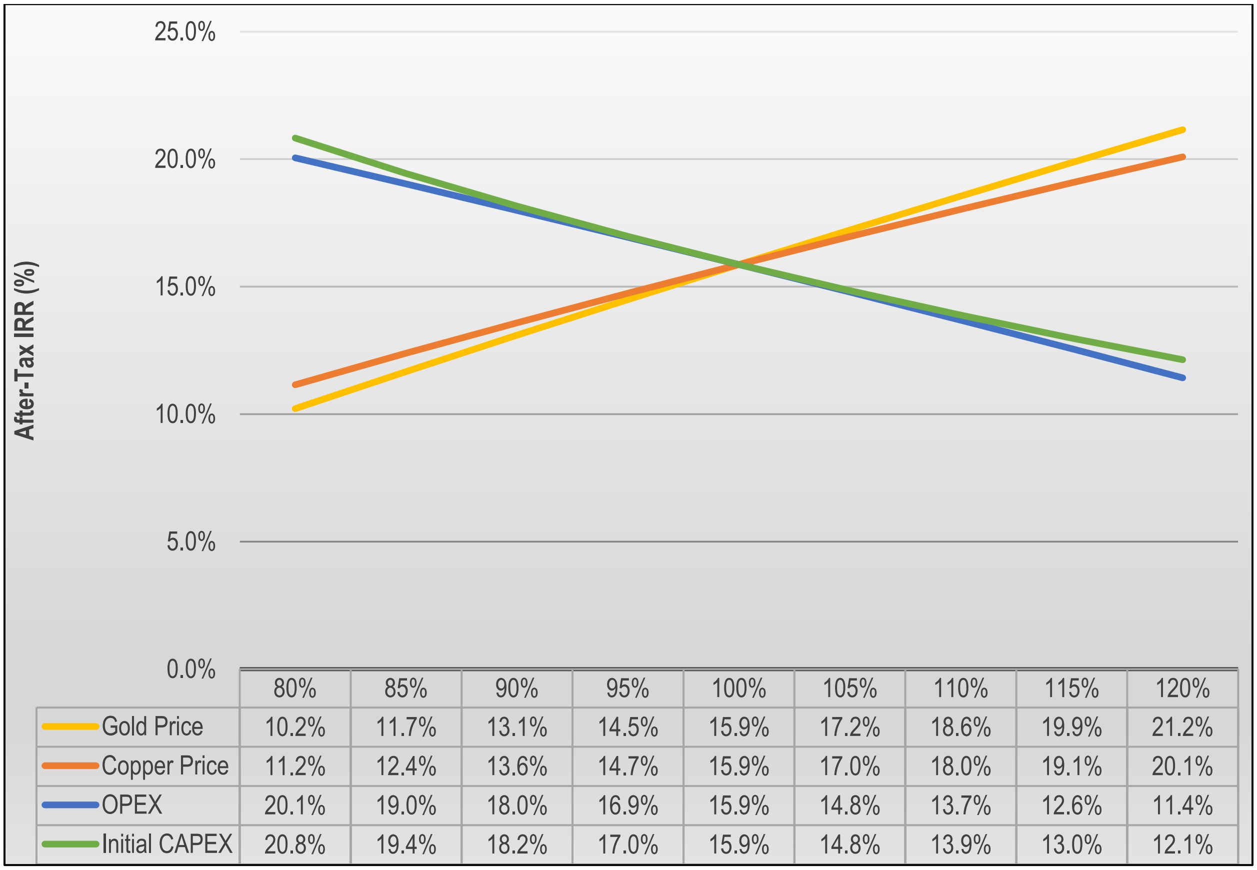
A sensitivity analysis was performed to test project value drivers on the project’s NPV using an 8% discount rate. The results of this univariate analysis are demonstrated in Figure 23-4 and Figure 23-5. Where a given variable was analyzed, all other inputs were held constant at their expected values. The Las Minas project proved to be most sensitive to changes in commodity prices, followed by operating costs, and is least sensitivity to initial capital costs. Changes in Gold pricing or head grades will have a higher impact on the project economics compared to the other commodities as Gold is the highest revenue generator.

Figure 23-4: After-Tax NPV 8% Sensitivity Results



	80%	85%	90%	95%	100%	105%	110%	115%	120%
Gold Price	9	16	22	29	35	41	48	54	61
Copper Price	13	19	24	30	35	41	46	52	57
OPEX	55	50	45	40	35	30	25	20	15
Initial CAPEX	49	46	42	39	35	32	28	24	21

Figure 23-5: After-Tax IRR Sensitivity Results



	80%	85%	90%	95%	100%	105%	110%	115%	120%
Gold Price	10.2%	11.7%	13.1%	14.5%	15.9%	17.2%	18.6%	19.9%	21.2%
Copper Price	11.2%	12.4%	13.6%	14.7%	15.9%	17.0%	18.0%	19.1%	20.1%
OPEX	20.1%	19.0%	18.0%	16.9%	15.9%	14.8%	13.7%	12.6%	11.4%
Initial CAPEX	20.8%	19.4%	18.2%	17.0%	15.9%	14.8%	13.9%	13.0%	12.1%

The After-tax economic results were also evaluated using various commodity pricing scenarios. Table 23-5 highlights recent spot prices as well as the approximate high and low commodity prices of the last three years.

Table 23-5: Sensitivity of After-Tax Economic Results to Changes in Commodity Prices

	Base Case	Spot Prices (July 29, 2021)	Upside	Downside
Gold (US$/oz)	1,625	1,830	2,000	1,200
Silver (US$/oz)	20.0	25.5	28.0	14.0
Copper (US$/lb)	3.25	4.45	4.75	2.25
Iron Concentrate (US$/dmt)	100	213.5	220	65
Cumulative Cash Flow (US$M)	99	237	276	-22
After-Tax NPV5% (US$M)	55	157	187	-37
After-Tax NPV8% (US$M)	35	122	148	-43
After-Tax IRR (%)	16	31	35	-5
Capex Payback (Years)	4.4	2.8	2.6	n/a
EBITDA for First Year of Full Production (US$M)	43	70	77	19

Notes:

Upside and Downside commodity price scenarios represent the approximate high and low prices for each individual commodity in the last 3 years.

24 ADJACENT PROPERTIES

There are no significant mineralized properties adjacent to the Las Minas project that are relevant to this Technical Report.

25 OTHER RELEVANT DATA AND INFORMATION

To the best of the authors’ knowledge there is no other relevant data, additional information or explanation necessary to make the Report understandable and not misleading.

26 INTERPRETATIONS AND CONCLUSIONS

26.1 Risks

The most significant project risks are summarized in Table 26-1:

Table 26-1: Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Metal Prices and Exchange Rates	The price of metals and US$ to Mexican peso exchange rate can have a dramatic impact of the economic viability of the project as shown in the economic sensitivities.	Hedging against metal price can limit downside risks but also impacts upside potential
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. 43% of the resources in the mine plan are Inferred.	Infill drilling may be recommended in order to provide a greater level of confidence in the resource.
Geological Complexity	The geological complexity of the Las Minas deposit could potentially lead to increased mining dilution and/or ore loss due to variability in mineralized domains. Grade control and proper mining execution will maintain minimal unplanned dilution, which would minimize potential impacts on grade, throughput, and operating costs.	A comprehensive, tight grade control program and geological monitoring will help minimize unplanned dilution and negative impacts during mining.
Data Quality	Data verification and data quality issues were encountered and addressed however, further issues could be discovered with ongoing data collection and exploration.	Check surveys and permanent collar monuments would mitigate data quality issues.
Drilling Uncertainty	There is no guarantee that further drilling will result in additional resources or increased classification. In addition, further work may disprove previous models and therefore result in condemnation of targets and potential negative economic outcomes.	Refinement and continuous improvement of drilling planning and models will continue to advance understanding and increase confidence.
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 testwork is needed at the next level of study.

Risk	Explanation/Potential Impact	Possible Risk Mitigation
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success. If OPEX increases then the NSR cut-off would increase and, all else being equal, the size of the mineable resource would reduce yielding fewer mineable tonnes.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
Permit Acquisition	The ability to secure all of the permits to build and operate the project is of paramount importance. Failure to secure the necessary permits could stop or delay the project.	The development of close relationships with the local communities and government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required. Maintain direct control with a clear solution.
Development Schedule	The project development could be delayed for a number of reasons and could impact project economics. A change in schedule would alter the project economics.	If an aggressive schedule is to be followed, PFS field work should begin as soon as possible.
Ability to Attract Experienced Professionals	The ability to attract and retain competent, experienced professionals 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.

26.2 Opportunities

The main opportunities identified for the project are listed in Table 26-2.

Table 26-2: Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Data Validation	Ongoing data verification and ground truthing may result in being able to re-introduce data that has been excluded.	This may result in an improved understanding of the deposit and grade distribution.
Mineral Resources	There is the potential for an increase in mineral resources with increased exploration drilling.	This may increase the mine life.
Exploration	There are many historic showings and discoveries that have been subject to limited exploration activities.	Exploration targets pose an excellent potential for expanding the project potential and resources.
Expansion of the Mine	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

27 RECOMMENDATIONS

It is recommended that the Las Minas Project proceed to the Pre-feasibility Study (PFS) stage in line with Mexican Gold's desire to advance the project. It is estimated that a PFS and supporting field work would cost approximately $2.2M.

27.1 Exploration, Geology & Resources

Additional drilling is recommended to increase drill density to potentially achieve a higher resource category in higher-grade areas. Additional drilling may increase resources, improve understanding and modelling of lithological units and better define the limits of the mineralization.

A review of QA/QC procedures is recommended to improve data quality and increase confidence in the dataset.

A comprehensive brownfields exploration program in the area is recommended to explore for additional targets.

Investigate and explore the historic mine workings and discoveries with the view of expanding resource base.

27.2 Metallurgy

A testwork program to prepare for a pre-feasibility study should include the following:

- Explore grind size vs recovery – the testwork conducted for the PEA used the upper limit of grind size tested (150 µm);
- Explore technologies such as hydrofloat which may allow a coarser grind than regular flotation;
- Variability testwork – understand the difference between the 2015 and 2021 testwork. Improve recovery;
- Variability testwork program:
    - Comminution parameters;
    - Gravity GRG test;
    - Flotation recoveries; and
    - Magnetite circuit.
- Settling/dewatering testwork.

The recommended budget for this program would be $250,000.

27.3 TSF Recommended Work Programs

Recommendations for the next phase of engineering for the TSF are summarized below:

- Collection of site-specific meteorological and hydrology data. This data will be used to confirm seasonal runoff values and design storms;
- Complete site investigations programs at the selected TSF location to support the next phase of design;
- Confirm the geotechnical characteristics of the tailings and construction materials;
- Complete geochemical characterization of tailings and construction materials to access potential acid rock drainage and other potential chemical releases (metal leaching); and
- Develop a full closure plan for the TSF based on the final design configuration.

28 REFERENCES

Barton, N. (2002). *Some new Q-value correlations to assist in site characterization and tunnel design. International Journal of Rock Mechanics & Mining Science* Vol. 39/2:185-216.

Chan, C., Sloan, R. (2015) *Metallurgical Testing of the Las Minas Deposit*. ALS Metallurgy Kamloops, Project No. 4508, June 24, 2015

CTA. 2020. *Las Minas Mining Project Environmental Baseline Study. Final Report*. Consultoría y Tecnología Ambiental México, S.A. De C.V. Project No. 277-001. August 20, 2020.

Deere, D.U., (1989). *Rock Quality Designation (RQD) after Twenty Years*. Report prepared for Department of the Army, US Army Corp of Engineers, 92 pages.

Hutchinson, D.J. and Diederichs, M.S. (1996) *Cablebolting in Underground Mines*. Bi Tech Publishers, Richmond, British Columbia, Canada, 406pp.

International Society for Rock Mechanics Commission on Standardization of Laboratory and Field Tests, Suggested Methods for the Quantitative Descriptions of Discontinuities in Rock Masses, International Journal of Rock Mechanics, Mining Sciences and Geomechanics Abstracts, Vol. 15, 1978. P. 319-368.

Liu, Y., Sloan, R. (2021) *Metallurgical Testing of Las Minas Samples*. ALS Metallurgy Kamloops, Project No. 6130, May 18, 2021

Martin, C.D., Maybee, W.G. (2000): *The strength of hard-rock pillars, International Journal of Rock Mechanics & Mining Sciences*; Volume 37; 1239-1246.

Mine Development Associates, Mine Engineering Services, Paul Tietz, C.P.G. (2019). *Technical Report and Estimated Resources for the Las Minas Project, Veracruz, Mexico*. October 9, 2019

Potvin, Y. and Hadjigeorgiou, J., (2001) *The Stability Graph for Open-Stope Design, Underground Mining Methods: Engineering Fundamentals and International Case Studies*, Ed. William Hustrulid, SME, pp 513 to 520.

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)
ac	acre
Acfm	actual cubic feet per minute
Ag	silver
amsl	above mean sea level
AN	ammonium nitrate
ARD	acid rock drainage
Au	gold
AWR	all-weather road
B	billion
BD	bulk density
Bt	billion tonnes
BTU	British thermal unit
BV/h	bed volumes per hour
bWi	bond work index
bya	billion years ago
C$	dollar (Canadian)
Ca	calcium
cfm	cubic feet per minute
CHP	combined heat and power plant
CIM	Canadian institute of mining and metallurgy
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
cP	centipoise
Cr	chromium
Cu	copper

Symbol / Abbreviation	Description
d	day
d/a	days per year (annum)
d/wk	days per week
dB	decibel
dBa	decibel adjusted
DGPS	differential global positioning system
DMS	dense media separation
dmt	dry metric ton
DWT	dead weight tonnes
EA	environmental assessment
ED	El Dorado
EIS	environmental impact statement
ELC	ecological land classification
Fe	iron
FEL	front-end loader
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
Ga	billion years
gal	gallon (us)
GJ	gigajoule
GPa	gigapascal
gpm	gallons per minute (us)
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
HLEM	horizontal loop electro-magnetic

Symbol / Abbreviation	Description
hp	horsepower
HPGR	high-pressure grinding rolls
HQ	drill core diameter of 63.5 mm
Hz	hertz
ICP-MS	inductively coupled plasma mass spectrometry
in	inch
in^2	square inch
in^3	cubic inch
IRR	internal rate of return
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
k	kilo (thousand)
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
kPa	kilopascal
kt	kilotonne
kV	kilovolt
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
LG	low grade
LOM	life of mine
m	metre
M	million
m/min	metres per minute
m/s	metres per second

Symbol / Abbreviation	Description
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
MAAT	mean annual air temperature
MAE	mean annual evaporation
MAGT	mean annual ground temperature
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
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum
mbs	metres below surface
mbsl	metres below sea level
mg	milligram
mg/L	milligrams per litre
min	minute (time)
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMER	metal mining effluent regulations
mo	month
MPa	megapascal
Mt	million metric tonnes
MVA	megavolt-ampere
MW	megawatt
NAD	North American datum
NG	normal grade
Ni	nickel
NI 43-101	National Instrument 43-101
Nm^3/h	normal cubic metres per hour
NQ	drill core diameter of 47.6 mm
OP	open pit
OSA	overall slope angles

Symbol / Abbreviation	Description
oz	troy ounce
P.Geo.	professional geoscientist
Pa	Pascal
PAG	potentially acid generating
PEA	preliminary economic assessment
PFS	preliminary feasibility study
PGE	platinum group elements
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
QA/QC	quality assurance/quality control
QP	qualified person
RC	reverse circulation
RMR	rock mass rating
ROM	run of mine
rpm	revolutions per minute
RQD	rock quality designation
s	second (time)
S.G.	specific gravity
Scfm	standard cubic feet per minute
SFD	size frequency distribution
SG	specific gravity
t	tonne (1,000 kg) (metric ton)
t	metric tonne
t/a	tonnes per year
t/d	tonnes per day
t/h	tonnes per hour
TCR	total core recovery
TFFE	target for further exploration
TMF	tailings management facility
tph	tonnes per hour
ts/hm^3	tonnes seconds per hour metre cubed
US	united states
US$	dollar (American)
UTM	universal transverse mercator
V	volt

Symbol / Abbreviation	Description
VEC	valued ecosystem components
VSEC	valued socio-economic components
w/w	weight/weight
wk	week
wmt	wet metric ton
WRSF	waste rock storage facility
µm	microns
µm	micrometre

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

30 CERTIFICATES

CERTIFICATE OF AUTHOR

GORDON EDWARD DOERKSEN, P.ENG.

I, Gordon Edward Doerksen, P.Eng., do hereby certify that:

1. I am President – Engineering Division for JDS Energy & Mining Inc. with an office at 900-999 W. Hastings St., Vancouver, B.C.
2. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico", with an effective date of September 18, 2021, (the "Technical Report") prepared for Mexican Gold Mining Corp.;
3. I am a graduate of the Montana College of Mineral Science and Technology (Montana Tech) in 1990 with a B. Sc. in Mining Engineering. I have continuously practiced my profession since 1990. I worked for over 20 years in precious, base metal and other mining operations and have 15 years of international mining consulting experience. I have authored many PEAs and Technical Reports including numerous base metal and Mexican studies.
4. I am a member in good standing of the Engineers and Geoscientists of British Columbia (Lic. No. 32273).
5. I visited the Project property on January 16-19, 2021.
6. In the independent report entitled "Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico" with effective date September 18, 2021, I am responsible for Sections 1.1 to 1.3 inclusive, 1.11, 1.15 to 1.17 inclusive, 2, 3, 4, 5, 18, 19 and 24 to 29 inclusive.
7. I have not had prior involvement with the company nor the property that is the subject of this Technical Report.
8. I am independent of Mexican Gold Mining Corp. as defined in Section 1.5 of National Instrument 43-101.
9. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I fulfil the requirements of a Qualified Person as defined in National Instrument 43-101.
10. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that 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 National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Effective Date: September 18, 2021
Signing Date: September 20, 2021

(original signed and sealed) "Gordon Edward Doerksen, P.Eng."
Gordon Edward Doerksen, P.Eng.

CERTIFICATE OF AUTHOR

MICHAEL MAKARENKO, P. ENG.

I, Michael Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico”, with an effective date of September 18, 2021, (the “Technical Report”) prepared for Mexican Gold Mining Corp.;
2. I am currently employed as Vice President, Engineering 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. 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 fifteen years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and technical report writing for mining projects worldwide;
4. I am a Registered Professional Mining Engineer in Alberta (#48091), British Columbia (#49223) and the Northwest Territories (#1359);
5. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I fulfil the requirements of a Qualified Person as defined in National Instrument 43-101;
6. I have not visited the Las Minas Project;
7. I am responsible for Sections 1.4, 1.9, 12.4, 15, 16 (except 16.2) of this Technical Report;
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have had no prior involvement with the property that is the subject of this Technical Report;
10. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and
11. I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Effective Date: September 18, 2021
Signed Date: September 20, 2021

(Original signed and sealed) “Michael Makarenko, P. Eng.”

Michael Makarenko, P. Eng.

CERTIFICATE OF AUTHOR

MICHAEL LEVY, P. ENG.

I, Michael Levy, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico”, with an effective date of September 18, 2021, (the “Technical Report”) prepared for Mexican Gold Mining Corp.;
2. I am currently employed as Geotechnical Manager with JDS Energy & Mining Inc. with an office at Suite 100 – 14143 Denver West Parkway, Golden, Colorado, 80401;
3. I am a Professional Civil Engineer (P.Eng. #2692) registered with the Association of Professional Engineers Yukon and Colorado (P.E. #40268). I am a current member of the International Society for Rock Mechanics (ISRM) and the American Society of Civil Engineers (ASCE).

   I hold a bachelor’s degree (B.Sc.) in Geology from the University of Iowa in 1998 and a Master of Science degree (M.Sc.) in Civil-Geotechnical Engineering from the University of Colorado in 2004. I have practiced my profession continuously since 1999 and have been involved in numerous mining and civil geotechnical projects across the Americas;
4. I have not visited the Las Minas Project;
5. I have had no prior involvement with the property that is the subject of this Technical Report;
6. I am responsible for section 16.2 of the Technical Report;
7. I have read the definition of “Qualified Person” set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association, and past relevant experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101;
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1; and,
10. As of the effective date of the 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.

Effective Date: September 18, 2021
Signing Date: September 20, 2021

(Original signed and sealed) “Michael Levy, P. Eng.”

Michael Levy, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

SHANE TAD CROWIE, P. ENG.

I, Shane Tad Crowie, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico”, with an effective date of September 18, 2021, (the “Technical Report”) prepared for Mexican Gold Mining Corp.;

2. I am currently employed as Sr. Metallurgist 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 British Columbia with a B.A.Sc. in Mining and Mineral Process Engineering, 2001. I have practiced my profession continuously since 2001. I have worked in technical, operations and management positions at mines in Canada. I have been responsible for recovery optimization projects, capital improvement projects, budgeting, planning, and pilot plant operations;

4. I am a Registered Professional Mining Engineer in British Columbia (#34052);

5. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I fulfil the requirements of a Qualified Person as defined in National Instrument 43-101;

6. I have not visited the Las Minas Project;

7. I am responsible for Sections 1.6, 1.10, 12.3, 13, 17, 22.4, and 27.2 of this Technical Report;

8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

10. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and

11. I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Effective Date: September 18, 2021
Signed Date: September 20, 2021

(Original signed and sealed) “Shane Tad Crowie, P. Eng.”

Shane Tad Crowie, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

TYSEN HANTELMANN, P. ENG.

I, Tysen Hantelmann, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico”, with an effective date of September 18, 2021, (the “Technical Report”) prepared for Mexican Gold Mining Corp.;
2. I am currently employed as a Mining Engineer 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, 2001 and with a M.Eng. in Mining Engineering, 2003. I have practiced my profession continuously since 2002. I have been involved with mining operations, mine engineering and consulting for over 100 projects covering a variety of commodities at locations in North America, Latin America, Africa, and Europe;
4. I am a Registered Professional Mining Engineer in Alberta (#71697), and Yukon (#2631);
5. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I fulfil the requirements of a Qualified Person as defined in National Instrument 43-101;
6. I have not visited the Las Minas Project;
7. I am responsible for Sections 1.13, 1.14, 21 22 and 23 of this Technical Report;
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have had no prior involvement with the property that is the subject of this Technical Report;
10. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and
11. I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Effective Date: September 18, 2021
Signed Date: September 20, 2021

(Original signed and sealed) “Tysen Hantelmann, P. Eng.”

Tysen Hantelmann, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

KENNETH EMBREE, P. ENG.

I, Kenneth Embree, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico", with an effective date of September 18, 2021, (the "Technical Report") prepared for Mexican Gold Mining Corp.

2. I am employed as President of Knight Piésold Ltd. with an office at Suite 1400 - 750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.

3. I am a graduate of the University of Saskatchewan with a B.Sc. in Geological Engineering, 1986. I have practiced my profession continuously since 1986. My experience includes tailings, waste and water management for mine developments in Canada, USA, Mexico, and South America

4. I am a Professional Engineer in good standing with Engineers and Geoscientists of British Columbia in the area of geological engineering (No. 17439). I am also registered as a Professional Engineer in Ontario (No. 100040332), Yukon (No. 2694) and the Northwest Territories and Nunavut (No. L3766).

5. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I fulfil the requirements of a Qualified Person as defined in National Instrument 43-101.

6. I have not visited the Las Minas Project site.

7. I am responsible for Sections 1.12, 18.6, 18.7, and 20 of this Technical Report.

8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

9. I have had no involvement with the property that is the subject of this Technical Report.

10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report and that 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 National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Effective Date: September 18, 2021
Signed Date: September 20, 2021

(Original signed and sealed) "Kenneth Embree, P. Eng."

Kenneth Embree, P. Eng., Knight Piésold Ltd.

CERTIFICATE OF AUTHOR

GARTH DAVID KIRKHAM, P.GEO.

I, Garth David Kirkham, P.Geo., do hereby certify that:

1. I am a consulting geoscientist and Principal of Kirkham Geosystems Ltd. since 1987 with an office at 6331 Palace Place, Burnaby, British Columbia.
2. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico”, with an effective date of September 18, 2021, (the “Technical Report”) prepared for Mexican Gold Mining Corp.;
3. I am a graduate of the University of Alberta in 1983 with a B. Sc. I have continuously practiced my profession since 1988. I have authored many resource estimations and NI43-101 technical reports including Esperanza Au-Ag skarn, Morelos State, and Cerro Las Minitas Ag-Zn-Pb-Au-Cu skarn, Durango State, Mexico, along with Cerro Blanco, Debarwa, and Kutcho Creek poly-metallic deposits.
4. I am a member in good standing of the Engineers and Geoscientists of British Columbia.
5. I have visited the property on January 16-19, 2021.
6. In the independent report entitled “Preliminary Economic Assessment Technical Report Las Minas Project, Veracruz, Mexico” with effective date September 18, 2021, I am responsible for Sections for Sections 1.4, 1.5, 1.7, 6, 7, 8, 9, 10, 11, 12.1, 12.2, 14 and 25.1.
7. I have not had prior involvement with the company nor the property that is the subject of this Technical Report.
8. I am independent of Mexican Gold Mining Corp. as defined in Section 1.5 of National Instrument 43-101.
9. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I fulfil the requirements of a Qualified Person as defined in National Instrument 43-101.
10. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that 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 National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Effective Date: September 18, 2021
Signing Date: September 18, 2021

(original signed and sealed) “Garth Kirkham, P.Geo.”
Garth Kirkham, P.Geo., Kirkham Geosystems Ltd.