AMENDED AND RESTATED

NI 43-101 PRELIMINARY ECONOMIC ASSESSMENT

OF THE

LA MINA

VMS PROJECT

COTOPAXI PROVINCE

ECUADOR

Respectfully submitted to:
Toachi Mining Inc.

Effective Date: March 30, 2019
Signing Date: July 17, 2019
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SGS Report # P2018-21

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Date and Signatures

This report has an effective date of March 30, 2019

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SGS North America Inc.

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1.0 Executive Summary

At the request of Mr. Alain Bureau, President and CEO of Toachi Mining Inc. (“Toachi”), SGS Geological Services (“SGS”) has prepared this technical report (“the Report”) conforming to the standards dictated by National Instrument 43-101 (“NI 43-101”), companion policy NI 43-101CP and Form 43-101F (Standards of Disclosure for Mineral Projects) in respect to the La Mina VMS Project, part of the La Plata Project (“the Project”) located in Cotopaxi Province, Ecuador. The following report presents the details of the Preliminary Economic Assessment that was announced in a press release dated April 30, 2019 for the Project. The Preliminary Economic Assessment has been prepared in compliance with the requirements of Canadian National Instrument (NI) 43-101 and in accordance with guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10, 2014.

The Property

The concession comprising the Project is situated within the Cordillera Occidental of central Ecuador, in the district of Toachi, Province of Cotopaxi. The project geographic coordinates are UTM 729 500 east; 9 957 300 north. By road, the Project is located approximately 100 kilometres to the west-south-west of Quito, the capital of Ecuador, and 25 kilometres to the south-east of the city of Santo Domingo de los Tsachilas. The property lies immediately north of the Iliniza Ecological Reserve, which covers 149,000 ha of the Andean Humid Forest. The southern portion of the La Plata concession is within the Toachi-Pilaton Protected Forest. Mining exploration activities are permitted, subject to Ministry approval, within protected forests.

Access, Local Resources and Infrastructure

La Mina VMS deposit is accessed by driving 85 km on paved road from the capital city of Quito to the village of Unión del Toachi. From the turn off point, a 13-km dirt road leads to the Project passing through the village of Palo Quemado where Compañia Minera la Plata’s field office and accommodation are located. The total driving time from Quito is approximately three hours using a four-wheel drive light truck.

Economically, the central part of Ecuador is characterized as rustic and rural with a local economy largely dominated by agriculture. Much of the area (60%) is occupied by small farms and slopes and incised valleys are covered with thick jungle. The climate conditions are good for farming activities such as bananas and sugar cane, and grazing cattle.

Local unskilled labour is readily available. Some exposure to modern exploration techniques has occurred in the past, and more recently Toachi has implemented modern training for best practice procedures.

Services such as environmental monitoring, assaying services, field equipment supply and drilling services are readily available from Quito. Other minor services are available from the nearby city of Santo Domingo de los Tsachilas.

The previously small-scale operating mine at La Mina VMS deposit had the basic infrastructure to support the operation, including an access road, power supply, water and ancillary facilities.

The access road is unpaved which departs from the main highway at the village of Unión del Toachi to the north. If the Project goes to commercial production, this road will require some improvement for mine construction and daily travelling in order to provide all necessary equipment and products for the development of the Project. The metal concentrates from La Mina could be shipped via the existing road network to the ports of Guayaquil or Esmeraldas on the coast.

An approximately 7 km long power line would be constructed from an existing substation on the National hydropower grid to a location adjacent to the proposed mill site. In 2018, Toachi representatives met with La Empresa Eléctrica Pública Estratégica Corporación Nacional de Electricidad (CNEL EP) Unidad de Negocio Santo Domingo and obtained preliminary cost information from CNEL EP for the construction of the power line and new substation.

The Toachi Pilaton Hydroelectric Power Plant Project (250MW) is under construction. This plant is located approximately 5 km from the proposed processing plant and mine site area clearly representing a potential energy source for future operations.

Water is available on site from Quebrada La Florida adjacent to the mine site, and from a number of different nearby water sources. The historic operation used water from this Quebrada La Florida source.

General and Regional Geology

The La Plata district is situated in the western flank of the Cordillera Occidental. It is hosted by the Paleocene to Eocene Macuchi Unit, defined as a submarine volcano-sedimentary sequence with pillow lavas and related intrusive bodies, derived from an oceanic island arc of basaltic to andesitic composition. This unit crops out for more than 500 kilometers along a NNE to SSW trend and shows thicknesses up to 50 km. The sector of interest in which all polymetallic volcanic-hosted massive sulphide (VHMS) deposit occurrences have been recognized is delimited between the 0° and 1°S parallels. The undifferentiated superficial Quaternary deposits of the Costa represent the western limit of the district. The eastern limit of the district has been defined by the Toachi and Pilaló-Sigchos faults.

All recorded VHMS deposits or occurrences of the Western Cordillera are found in this district. The most important ones are the La Mina deposit the Macuchi deposit located 65km south of La Mina and the El Domo deposit, 100km south of La Mina. These three deposits represent examples of Kuroko-type mineralization.

Updated Mineral Resource Estimate

The Mineral Resource for the La Mina VMS Deposit (previously referred to as the La Plata VMS Deposit) at the La Plata Project, Ecuador was previously estimated as at 13th September 2017. The current mineral resource update incorporates updated metal price assumptions determined during February 2019 and no new drilling or other technical data as relates to mineral resource estimation is available.

Mr. Brian Wolfe of International Resource Solutions Pty Ltd was requested by Toachi Mining Limited (Toachi) to generate a grade estimate using the available data. This data included historical exploration and grade control drilling collected prior to Toachi's involvement and recent drilling completed under the supervision of Toachi's technical staff. The grade estimate has been classified as Inferred in accordance with the CIM 2014 guidelines.

The updated Inferred Mineral Resource reported for the La Mina deposit is summarised in the table below. The preferred reporting cutoff grade is 4g/t AuEq.

La Mina VMS Deposit – Inferred Mineral Resource – Ordinary Kriging Estimate

Cutoff (AuEq – g/t)	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Aueq koz	Au koz	Ag koz	Cu kt	Pb kt	Zn t
2	2,373	10.6	3.4	42.4	2.7	0.5	3.8	812	260	3,238	64	13	89
3	2,055	11.9	3.8	46.9	3.0	0.6	4.2	787	251	3,097	62	12	87
4	**1,846**	**12.9**	**4.1**	**50.0**	**3.3**	**0.6**	**4.6**	**763**	**244**	**2,966**	**61**	**12**	**84**
5	1,655	13.8	4.4	53.7	3.5	0.7	4.9	736	236	2,860	59	11	80
6	1,461	14.9	4.8	58.5	3.8	0.7	5.2	701	225	2,747	56	11	75
7	1,320	15.8	5.1	62.4	4.1	0.8	5.5	672	217	2,647	54	10	72
8	1,195	16.7	5.4	66.0	4.3	0.8	5.8	642	208	2,535	51	10	69
9	1,111	17.3	5.7	68.6	4.4	0.9	5.9	619	202	2,449	49	10	66
10	1,019	18.0	5.9	71.4	4.6	0.9	6.1	591	194	2,340	47	9	63

Note: AuEq This Updated Mineral Resources Estimate as of March 30, 2019 was prepared in accordance with NI 43-101 and CIM Standards (2014).

1. The Updated Mineral Resource Estimate tonnages have been rounded to the nearest 1,000 and Au, Ag, Cu, Pb and Zn grades have been rounded to one decimal. Troy ounces have been rounded to kilo troy ounces (koz), and tonnes of Cu, Pb, and Zn have been rounded to kilo-tonnes (kt).
2. The Updated Mineral Resources Estimate for the La Mina VMS polymetallic deposit have been classified as Inferred Mineral Resources.
3. The Updated Mineral Resources has been reported at various cut-off grades to demonstrate the grade-tonnage relationship. The preferred reporting cut-off grade is 4 g AuEq / t (4 gold equivalent grams / tonne). The gold equivalent grade formula, which assumes 100% metallurgical recoveries, is:

   $$\text{AuEq ppm} = \text{Au ppm} + (\text{Cu \%} * 1.454) + (\text{Ag ppm} * 0.013) + (\text{Zn \%} * 0.654) + (\text{Pb \%} * 0.532)$$
4. The Updated Mineral Resources could be accessed by developing a mine ramp and are considered reasonable prospects for economic extraction in the foreseeable future.
5. The Updated Mineral Resources Estimate is based on the following three year trailing monthly average prices of metal, as of January 31st, 2019:

   $1,264/ troy oz Au $16.64/ troy oz Ag

   $2.68/lb Cu - $5,909 / t Cu $0.98/lb Pb – $2,162 / t Pb

   $1.21/lb Zn - $2,656 / t Zn
6. This Updated Mineral Resources Estimate was prepared by Brian R. Wolfe, BSc (Hons), MAIG.
7. Mineral Resources do not have demonstrated economic viability.
8. Mineral Resources are exclusive of material that has been mined.
9. This Updated Mineral Resources Estimate may be materially affected by environmental, permitting, legal title, taxation, socio-political, marketing or other relevant issues.

The following tables report the La Mina deposit, subdivided into South and North zones, based on the Quebrada Malo fault.

La Mina VMS Deposit – South Domain Inferred Mineral Resource Estimate

Cutoff (AuEq – g/t)	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Aueq koz	Au koz	Ag koz	Cu kt	Pb kt	Zn kt
2	981	13.0	4.7	64.7	2.7	0.8	4.8	411	147	2,041	27	8	47
3	889	14.1	5.0	68.7	3.0	0.8	5.2	404	144	1,962	27	7	46
4	**814**	**15.1**	**5.4**	**71.9**	**3.2**	**0.9**	**5.5**	**395**	**141**	**1,881**	**26**	**7**	**45**
5	738	16.2	5.8	77.0	3.5	0.9	5.9	384	138	1,827	25	7	43
6	681	17.1	6.2	81.3	3.7	1.0	6.1	374	136	1,779	25	7	42
7	630	18.0	6.6	85.5	3.8	1.0	6.4	364	133	1,732	24	7	40
8	593	18.6	6.8	88.8	3.9	1.1	6.6	355	130	1,695	23	6	39
9	562	19.2	7.0	91.2	4.1	1.1	6.8	346	127	1,648	23	6	38
10	518	20.0	7.4	95.0	4.2	1.1	7.0	333	123	1,581	22	6	36

1. Note: This Updated Mineral Resources Estimate as of March 30, 2019 was prepared in accordance with NI 43-101 and CIM Standards (2014).
2. The Updated Mineral Resource Estimate tonnages have been rounded to the nearest 1,000 and Au, Ag, Cu, Pb and Zn grades have been rounded to one decimal. Troy ounces have been rounded to kilo troy ounces (koz), and tonnes of Cu, Pb, and Zn have been rounded to kilo-tonnes (kt).
3. The Updated Mineral Resources Estimate for the La Mina VMS polymetallic deposit have been classified as Inferred Mineral Resources.
4. The Updated Mineral Resources has been reported at various cut-off grades to demonstrate the grade-tonnage relationship. The preferred reporting cut-off grade is 4 g AuEq / t (4 gold equivalent grams / tonne). The gold equivalent grade formula, which assumes 100% metallurgical recoveries, is:

   AuEq ppm = Au ppm + (Cu % * 1.454) + (Ag ppm * 0.013) + (Zn % * 0.654) + (Pb %*0.532)
5. The Updated Mineral Resources could be accessed by developing a mine ramp and are considered reasonable prospects for economic extraction in the foreseeable future.
6. The Updated Mineral Resources Estimate is based on the following three year trailing monthly average prices of metal, as of January 31st, 2019:

   $1,264/ troy oz Au $16.64/ troy oz Ag

   $2.68/lb Cu - $5,909 / t Cu $0.98/lb Pb – $2,162 / t Pb

   $1.21/lb Zn - $2,656 / t Zn
7. This Updated Mineral Resources Estimate was prepared by Brian R. Wolfe, BSc (Hons), MAIG.
8. Mineral Resources do not have demonstrated economic viability.
9. Mineral Resources are exclusive of material that has been mined.
10. This Updated Mineral Resources Estimate may be materially affected by environmental, permitting, legal title, taxation, socio-political, marketing or other relevant issues.

La Mina VMS Deposit – North Domains Inferred Mineral Resource Estimate

Cutoff (AuEq – g/t)	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Aueq koz	Au koz	Ag koz	Cu kt	Pb kt	Zn kt
2	1,392	9.0	2.5	26.8	2.7	0.4	3.0	401	113	1,198	37	5	42
3	1,166	10.2	2.9	30.3	3.1	0.4	3.5	383	107	1,135	36	5	40
4	**1,032**	**11.1**	**3.1**	**32.7**	**3.4**	**0.4**	**3.8**	**368**	**103**	**1,084**	**35**	**5**	**39**
5	917	11.9	3.3	35.1	3.6	0.5	4.1	352	98	1,033	33	4	37

6	780	13.0	3.6	38.6	4.0	0.5	4.3	327	89	969	31	4	34
7	690	13.9	3.8	41.3	4.3	0.6	4.6	308	84	915	30	4	32
8	602	14.8	4.1	43.4	4.6	0.6	4.9	287	79	840	28	4	30
9	549	15.4	4.3	45.4	4.8	0.6	5.1	273	75	801	26	3	28
10	501	16.0	4.4	47.1	5.0	0.7	5.2	258	71	759	25	3	26

1. Note: This Updated Mineral Resources Estimate as of March 30, 2019 was prepared in accordance with NI 43-101 and CIM Standards (2014).
2. The Updated Mineral Resource Estimate tonnages have been rounded to the nearest 1,000 and Au, Ag, Cu, Pb and Zn grades have been rounded to one decimal. Troy ounces have been rounded to kilo troy ounces (koz), and tonnes of Cu, Pb, and Zn have been rounded to kilo-tonnes (kt).
3. The Updated Mineral Resources Estimate for the La Mina VMS polymetallic deposit have been classified as Inferred Mineral Resources.
4. The Updated Mineral Resources has been reported at various cut-off grades to demonstrate the grade-tonnage relationship. The preferred reporting cut-off grade is 4 g AuEq / t (4 gold equivalent grams / tonne). The gold equivalent grade formula, which assumes 100% metallurgical recoveries, is:

   AuEq ppm = Au ppm + (Cu % * 1.454) + (Ag ppm * 0.013) + (Zn % * 0.654) + (Pb %*0.532)
5. The Updated Mineral Resources could be accessed by developing a mine ramp and are considered reasonable prospects for economic extraction in the foreseeable future.
6. The Updated Mineral Resources Estimate is based on the following three year trailing monthly average prices of metal, as of January 31st, 2019:

   $1,264/ troy oz Au — $16.64/ troy oz Ag

   $2.68/lb Cu - $5,909 / t Cu — $0.98/lb Pb – $2,162 / t Pb

   $1.21/lb Zn - $2,656 / t Zn
7. This Updated Mineral Resources Estimate was prepared by Brian R. Wolfe, BSc (Hons), MAIG.
8. Mineral Resources do not have demonstrated economic viability.
9. Mineral Resources are exclusive of material that has been mined.
10. This Updated Mineral Resources Estimate may be materially affected by environmental, permitting, legal title, taxation, socio-political, marketing or other relevant issues.

Mining Methods

Mine layout

The La Mina VMS deposit (formerly known as the La Plata VMS deposit) comprises the La Mina South block and the La Mina North block. For preliminary mine planning purposes, the La Mina North block is subdivided into two zones referred to as the “Upper North zone” and the “Lower North zone”; and the La Mina South block is referred to as the “South zone”. The mine development and production schedule is shown in Figure 1.B.

Mine Development and Production Schedule

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	LOM Total(1)
Upper North zone										
Development										
Production		240 kt	168 kt	91 kt						**499 kt**
Lower North zone										
Development										
Production			120 kt	177 kt	15 kt	22 kt		60 kt	11 kt	**405 kt**
South zone										
Development										
Production				20 kt	273 kt	266 kt	288 kt	228 kt	28 kt	**1,103 kt**
LOM Total (1)		**240 kt**	**288 kt**	**288 kt**	**288 kt**	**288 kt**	**288 kt**	**288 kt**	**39kt**	**2,007 kt**

(1) Potentially mineable mineralization

The Company would develop and operate the mine using its own labour and equipment. The mine is scheduled to operate for 7.1 years

Mechanized cut and fill mining methods

The PEA is based on mining the deposit using two variations of the widely-used and flexible mechanized cut and fill mining method:

1. A mechanized drift and slash mining with cemented rock fill (CRF) approach would be used in most stopes. Dilution and losses are estimated to be 15% and 5% respectively.

2. A mechanized room and pillar with CRF approach would be used in the flatter dip uppermost part of the South zone. Dilution and losses are estimated to be 15% and 5% respectively.

The mine plan is shown in the following table.

Table 1.F: Mine plan

Production Year	kt[A] Potentially mineable mineralization	Estimated Run-Of-Mine Grades				
		Au g/t	**Ag g/t**	**Cu%**	**Pb%**	**Zn%**
1	240	2.85	31	2.83	0.47	3.55
2	288	2.70	26	2.94	0.31	2.76
3	288	2.44	26	3.01	0.29	2.62
4	288	4.82	59	3.05	0.78	4.09
5	288	4.30	57	2.96	0.68	4.09
6	288	3.31	51	1.79	0.54	3.70
7	288	2.19	34	1.46	0.40	3.07
8	39	2.13	39	0.93	0.49	3.27
TOTAL	2 Mt	3.22	41	2.54	0.50	3.41

Mine equipment

The mine equipment fleet would include single and double boom drill jumbos, 7t and 10 t capacity LHD units, 30 t trucks, rock bolters and ancillary equipment. The mine services include a mine dewatering pump system, ventilation system, fuel storage and refuge stations.

Mine labour

The estimated numbers of mine personnel on the payroll in Year 2 (as an example) are summarized in the following table.

Estimated number of mine personnel on the payroll in Year 2

Area	Estimated number of persons on mine payroll in Year 2			
	G&A, Technical Services, Trainer	Mine Operations Department	Mine Maintenance Department	Total
General and Administration (G&A):				
Administration group	9			9
Social, environmental, nurse, security, kitchen	13			13
Technical Services staff	5			5
Trainer	1			1
Mine Operations Department:				
Mine supervisors		6		6
Jumbo drillers – mine development		6		6
Mechanical scaler operators – mine development		6		6
Rockbolter operators – mine development		6		6
Jumbo drillers - stoping		8		8

Manual bolters - stoping		16		16
Blasters		12		12
LHD operators		6		6
Mine truck drivers		12		12
Surface haul truck (to mill) drivers		4		4
CRF plant / placement equipment operators		12		12
Mine Maintenance Department:				
Mine maintenance supervisors			2	2
Maintenance planner			1	1
Lead mechanics			2	2
Mechanics			8	8
Welders			2	2
Lead electricians			2	2
Electricians			3	3
Instrumentation technician			1	1
Warehouse attendant			1	1
Total	**28**	**94[A]**	**22**	**144**

[A] The PEA assumes that mine workers would work on a three 8 hour shifts per day, 7 days per week basis with shift rotation.

Recovery Methods

This section describes the process design criteria applied to the crushing, grinding, flotation, and dewatering facilities for the proposed 800 tonnes per day copper, lead, zinc, gold and silver mineral processing plant at the La Mina Project is as follows:

In plant conceptual design:

1. The crushing plant would process the run-of-mine (ROM) material by using a primary jaw crusher to reduce the material from a nominal 30-45 cm (12-18 inches) to approximately 100% passing (P_{100}) of 249 mm (P_{80} of 114 mm).

2. The grinding circuit would be a semi-autogenous (SAG) mill - ball mill grinding circuit with subsequent processing in a flotation circuit. The SAG mill would operate in closed circuit with a vibrating screen. The ball mill would operate in closed circuit with hydrocyclones.

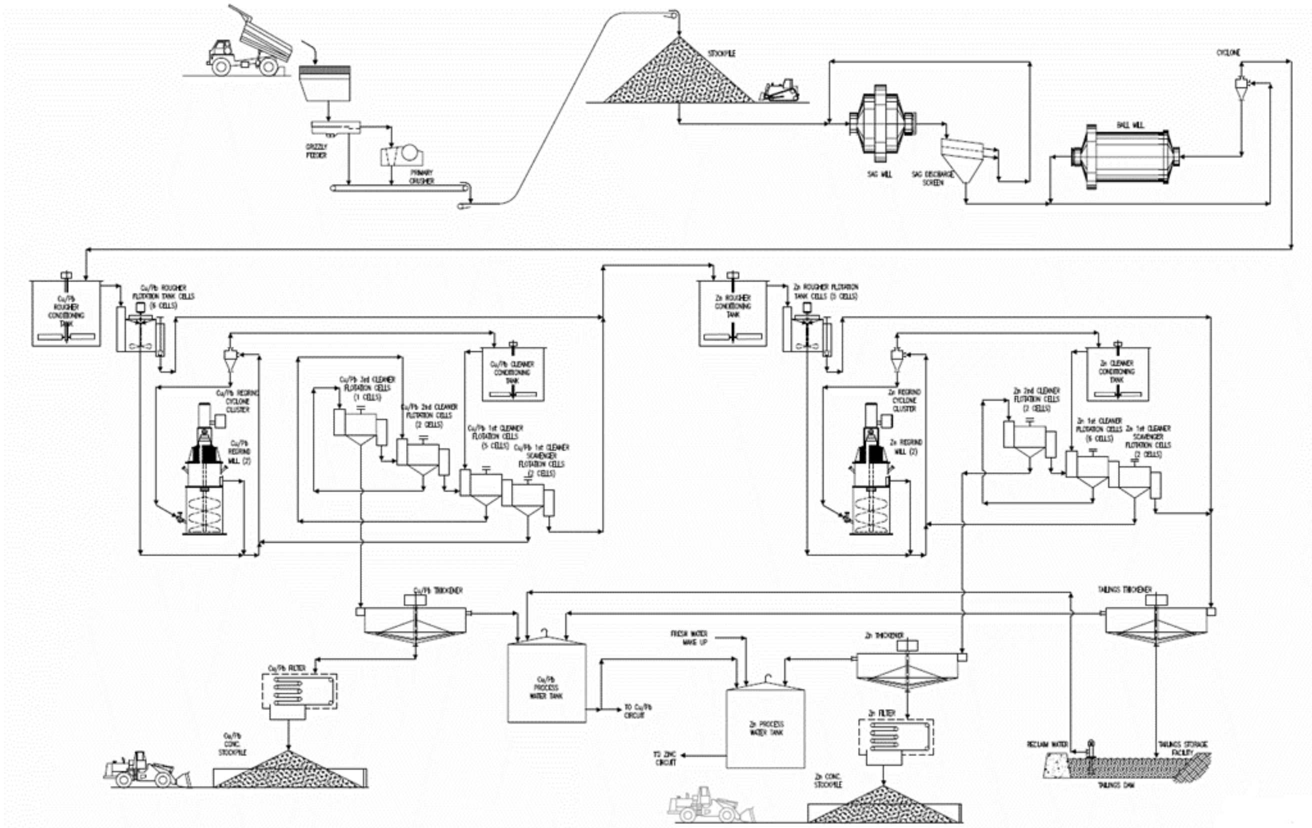
3. Cyclone overflow, the grinding circuit product, is fed to the flotation plant. The flotation plant would consist of copper/lead and zinc flotation circuits. The copper/lead flotation circuit would consist of rougher flotation and three-stage cleaner flotation. The zinc flotation circuit would consist of rougher flotation and two-stage cleaner flotation.

4. Both copper/lead and zinc concentrates are thickened, filtered, and stored in concentrate storage facilities prior to being bagged and loaded into containers for shipment to port.

5. Zinc rougher flotation tailing and zinc first cleaner scavenger tailing would be the final tailing. Tailing thickener underflow would be pumped to a tailings storage facility (TSF).

Plant water stream types include: copper/lead process water, zinc process water, fresh water, and potable water.

The overall flowsheet is shown in the following figure.



Infrastructure

The Project's infrastructure is summarized in the following table.

Project infrastructure

Item	Infrastructure
Access road	The PEA includes road improvements include such as adding culverts and road surfacing material to selected sections of road, truck pull-off areas and signage.
Power line & substation	An ~7 km long power line would be constructed from an existing substation on the National hydropower grid to a new substation location adjacent to the proposed mill site.
Main shop	The mine shop would have a concrete floor, lighting, work benches, tools, storage areas and an equipment wash bay, and office and lunch room space.

Item	Infrastructure
Main office	This office complex would also include a training classroom, mine rescue room, and separate male / female change rooms, and kitchen.
Mill infrastructure	The mill and mine would share the main shop, training and change house facilities. The mill would have a laydown yard and small parts and consumables storage containers. Process water would be sourced from wells and the effluent treatment plant pond.
Tailings Storage Facility	The PEA assumes that the mill tailings would be pumped approximately 1 km to a tailings storage facility (TSF) developed in a north-east trending valley. TSF drainage would be collected and treated.
Effluent treatment	A conventional lime-based final effluent treatment process including suspended particle settling and treated water pH adjustment would be used to produce a final treated effluent.
CRF plant	A cemented rock fill (CRF) plant would be constructed near the mine portal.
Fuel storage	Fuel would be stored in commercially available double walled steel tanks.
Social office	Toachi's social consultation group would work from its current office in the Palo Quemado village.
Other	The security gatehouse would be located at the main gate. Construction management and technical personnel would work from office trailers in Year -1. A 25 person capacity camp would be constructed in Year -1. The project would have phone, internet and radio services. Potable water would be obtained from a well. The Project would have a grey water / sewage treatment system.

Market Studies and Contracts

The PEA assumes that the Project would generate income from the sale of a copper/lead concentrate and a zinc concentrate to smelters in Asia.

The copper/lead concentrate would be transported to a copper smelter for treatment. For the purposes of this PEA, it is assumed that the copper concentrate would be bagged and loaded into containers and trucked to the Guayaquil port and ocean-transported to a copper smelter port in Asia. The zinc concentrate would be transported to a zinc smelter facility for pyro-metallurgical or hydro-metallurgical treatment. This PEA assumes that the zinc concentrate would be bagged and loaded into containers and trucked to the

Guayaquil port and then ocean-transported to a zinc smelter port in Asia. It is assumed that Toachi Mining would enter into an agreement with the smelters and receive payment in US dollars subject to smelter schedule treatment charges, minimum deductions, penalties, payables, concentrate transportation and handling costs, and other commercial terms.

The proposed Project is small on a global scale and its output would not affect metal prices. Toachi has not entered into any contracts for mine development, mining, concentrating, transportation, handling, sales and hedging, or forward sales contracts or arrangements.

Environmental Studies, Permitting and Social or Community Impact

Toachi already has the environmental license and authorizations needed for advance exploration activities on the La Plata mineral concession (concession) which is registered as a small scale (up to 300 tpd throughout) mining property. The PEA assumes that Toachi would follow the existing regulatory process and obtain a medium scale (301-1,000 tpd throughput) mining property registration for the concession. There is a possibility that after clarifying medium scale mining registration requirements Toachi may find it preferable to apply for large scale (e.g. >1,000 tpd mined material) registration.

The Company has successfully entered into surface land access / land use agreements its surface exploration and diamond drilling activities. Additional surface access / land use / land purchase agreements would be required for the proposed Project. SGS is of the option that the permitting and development of the Project will largely depend on the success of the company's community social consultation efforts with a latent risk of delays.

Capital and operating costs

The capital and operating costs were estimated using first principles cost estimates, preliminary budget quotes, factored cost estimates, reference data and cost allowances. Costs are estimated to ±35% accuracy.

Upfront capital cost

The estimated upfront capital cost is shown in the table below. The costs do not include Ecuadorian Value Added Tax, import duties or land purchase / land use costs.

Upfront capital costs

Item	Estimated Upfront CAPEX 15 month preproduction time line		
	Year -1	**Year 1 Q1**	**Total**
Mine capex			
• Mine equipment lease costs	$4.25M		$4.25M

• Mine pre-production development costs	$5.32M		$5.32M
Mill capex			
• Mill capex	$43.03M	$7.17M	$50.2M
Tailings storage facility (TSF)			
• TSF capex	$2.00M		$2.00M
• Effluent treatment capex	$0.95M		$0.95M
Infrastructure			
• Mine infrastructure	$3.52M		$3.52M
Indirect costs	$3.95M		$3.95M
Closure bond (Payment in Year -1)	$0.10M		$0.10M
Total pre-production CAPEX	**$63.1M**	**$7.17M**	**$70.27M**

Operating costs

The estimated life-of-mine (LOM) operating costs are shown in the following table.

Operating cost summary

Item	Estimated LOM operating cost ($/t milled)
Mining cost	$37.94/t
Processing costs	$26.62/t
Tailings disposal cost	$0.53/t
Effluent treatment cost	$0.70/t
Subtotal	$65.79/t

General and Administration (G&A)	$5.15/t
Subtotal	$70.94/t
Mine equipment lease / purchase costs	$8.21/t[A]
Total	$79.15/t

[A] Mine equipment lease / purchase costs are included as sustaining capex in the PEA cashflow. They are included here for reference by readers.

Economic analysis

The potential economic viability of the resource was assessed using a discounted cashflow model based on assumed 100% equity financing. The metal prices used in the cashflow are based on three year monthly trailing average prices as of January 31, 2019 and remain constant over the project life and are not escalated. Preproduction capital, sustaining capital, and operating costs are in reported in 2019 Q1 US dollars and are not escalated.

The PEA assumes the Project would be constructed and commissioned over a 15 month time line. Costs incurred prior to Year -1 such as but not limited to technical study costs, land use agreement costs, environmental assessment costs and Project permitting costs are assumed to be sunk costs not included in the cashflow.

The PEA cashflow makes 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 PEA will be realized.

Estimated Income

It is assumed the Project would produce and sell Cu/Pb and Zn concentrates to smelters in Asia and receive payment in US dollars – the Republic of Ecuador's adopted currency.

Annual revenues are based on projected recovered tonnages, diluted head grades, mill recoveries, NSR terms and Au, Ag, Cu and Zn prices. Readers are reminded that the PEA is preliminary in nature; the scheduled tonnes and grades do not represent a Mineral Reserve estimate; and that the Mineral Resources used in the PEA cashflow are not Mineral Reserves and do not have demonstrated economic viability.

The PEA cashflow is based on the LCT1 metallurgical testing results and includes the cost to bulk bag and transport Cu/Pb and Zn concentrates from the mill to smelters in Asia. The projected annual tonnages of concentrates are minor in comparison to global concentrate production levels, and developments affecting the Cu and Zn concentrate smelting marketplaces. The projected NSR factors are shown in the table below.

NSR factors used in the cashflow model

Item	Projected values		
Metal prices (3 year monthly trailing average prices to January 31, 2019)	Gold Price	$1,264 / troy oz Au	
	Silver Price	$16.64 / troy oz Ag	
	Copper Price	$2.68/ lb Cu ($5,909/ t Cu)	
	Lead Price	$0.98/ lb Pb ($2,162/ t Pb)	
	Zinc Price	$1.21/ lb Zn ($2,656/ t Zn)	
Cu/Pb concentrate mill recoveries (based on LCT1 test results)	Au % recovered	43.1%	
	Ag % recovered	47.9%	
	Cu % recovered	89.4%	
	Pb % recovered	71.6%	
	Zn % recovered	20.0%	
Zn con mill recoveries (based on LCT1 test results)	Au % recovered	26.6%	
	Ag % recovered	26.1%	
	Cu % recovered	4.5%	
	Pb % recovered	13.4%	
	Zn % recovered	70.5%	
Estimated concentrate (con) sales terms	Treatment	Cu/Pb con	Zn con
	Con grade	(20.9% Cu)	(51.2% Zn)
	Moisture content	8 wt%	8 wt%
	Treatment Charge	$132/ dt con	$188/ dt con
	Penalties	$13.10/ dt con	$9.45/ dt con
	Transportation	$95.45/wt con	$95.45/wt con
	Refining Charges (per unit of payable metal)	$0.135/ lb Cu $5/troy oz Au $2/troy oz/Ag	$5/troy oz Au $1/troy oz/Ag
	Payables	Cu/Pb con	Zn con
	Payment currency	USD	USD
	Au payable	96%	65%

Item	Projected values		
	Ag payable	90%	70%
	Cu payable	96.5%	0%
	Pb payable	0%	0%
	Zn payable	0%	85%

Preproduction CAPEX

The cashflow includes estimated Project pre-production costs: mine development costs; initial mine equipment lease costs; CRF backfill plant construction costs; mill construction costs; tailings storage facility (TSF) construction costs, effluent treatment plant construction costs; infrastructure costs including the power line and main substation costs; pre-development engineering, indirect operating and G&A costs; mill commissioning cost; and a mill cost contingency. The cashflow does not include working capital.

OPEX

The cashflow includes the estimated underground mining cost; surface haul cost; milling cost; TSF operating cost; effluent treatment cost; concentrate transportation and smelting and refining costs; indirect cost and G&A cost.

Sustaining CAPEX

The cashflow includes the following estimated sustaining capital expenditures for: sustaining mine development costs; mine equipment lease costs; mill sustaining capex; TSF expansion costs; closure surety bond costs; and closure work costs. It is assumed that $108k in closure surety bonding would be required in Year -1 and that the surety bonding coverage would be progressively increased over the operating life of the Project.

Salvage value

The estimate closure cost assumes there is no salvage value as there is no certainty that the Project materials and equipment can be salvaged and sold after production ceases.

Taxation

Corporate taxes were estimated after a general review of Ecuador's corporate tax requirements and with assistance from Toachi. It is projected that after taking initial CAPEX amortization into account, the Republic of Ecuador would receive at least 50% of the project's profit through receipt of 22% corporate taxes, Republic of Ecuador royalty, and its share of a mandated 15% share of profit including employee share of profit. The costs used in the cashflow do not include recoverable Value Added Tax.

Results of the economic analysis

The results of the economic analysis (i.e. the Internal Rate of Return "IRR, Net Present Value "NPV", and Payback) are shown in the following table.

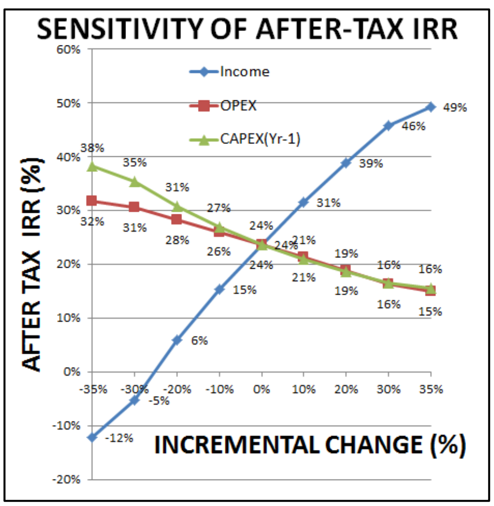
PEA cashflow results

	Before Tax	After Tax
IRR	38%	24%
NPV (5%)	$100M	$52M
NPV (8%)	$81M	$39M
NPV (10%)	$70M	$32M
Payback	2.8 years	3.5 years

Sensitivity Analysis

The sensitivity of the after tax Internal Rate of Return (IRR) to incremental changes in revenue, operating cost and upfront capital cost was assessed. The results of the assessment are graphically shown in the following figure.

After-Tax IRR sensitivity to changes in income, OPEX and CAPEX.



Interpretation and Conclusions

The Project warrants being carried forward to the Pre-Feasibility Study (PFS) stage (based on the proposed approach, the use of proven conventional technologies and equipment, and the positive results of the PEA) with a view to advancing the Project towards an eventual production decision.

Potential significant risks and uncertainties that could affect the updated mineral resources estimate and/or the projected economic viability of the Project are summarized in the following table.

Potential risks and uncertainties

Area	Potential Risks / Uncertainties
Mineral Resources	• The updated mineral resources estimate presented in the PEA does not include the 173kt historic mined tonnage which is based on available historic information. While the exact historic mine tonnage is uncertain, the potential for a significant impact to the updated mineral resources estimate and proposed mine plan is considered low.
Exploration	• Surface exploration diamond drilling aimed at intercepting potential addition mineralization below the La Mina VMS deposit South Block is warranted.
Mining	• The mine development and production plan is based on the use of conventional mining equipment and technologies. The underground workings would be accessed using ramps and would have required mine services. The proposed ground support requirements in the development headings and stopes are projections that are to be reviewed as part of the mine geotechnical study described in Section 26. There is a possibility that additional ground support requirements that may be introduced by the study and significantly increase mine operating costs. • The PEA assumes that the mine would hire experience lead miners and maintenance personnel and provide training to others. It is expected that most mine personnel would be hired from the Project region. There is a possibility that the Project could find it difficult to attract and retain people from the region. • The PEA assumes that the mine would be dewatered using conventional mine sumps and pumps with standby pumps and a spare dewatering pipe line to handle higher flows. There is a possibility that the results of the recommended hydrological/hydrogeological study described in Section 26 may identify a need for additional mine pumping capacity and add to the mine capital and operating costs. • There is a possibility that actual wages and salaries could be higher than estimated in the PEA. Possible reasons include as examples: higher wage levels for experienced lead hands; and additional social costs due to governmental factors.
Metallurgy	• Metallurgical testing on the 4 substantial sample composites provided (Composites 100, 200,300 and 400) has demonstrated that saleable Cu/Pb and Zn concentrates can be produced using the proposed process. • Gravity separation: A significant upgrading of the gold distribution present into a low weight (<0.2% of the feed) has been demonstrated – however further work will be required to quantify the benefits to the currently proposed circuit. For the current phase of project development, the flowsheet has not included a specific gravity circuit. However, as this has been important in the various previous processing activities on site, the final decision will be considered as part of the next phase of the project development. • Flotation: Significant development work was undertaken through the program to minimise the misreporting of Sphalerite into the Copper concentrate. The combination of high dosages of sphalerite/pyrite depressants (Na_2S, SMBS, and $ZnSO_4$) was found to produce optimal results. This has been confirmed by Locked Cycle Testing. • There is the likelihood that the additional metallurgical testing work described in Section 26 will provide a basis for metal recoveries and concentrate quality

Area	Potential Risks / Uncertainties
	improvements. Changes if any to the plant process may increase plant capital and operating costs.
Processing	• The proposed Plant would process material using conventional flotation recovery methods. The material would be crushed and conveyed to the processing plant. The material would be ground to 80% passing 70 microns in a SAG/Ball mill grinding circuit. The ground material would then be floated with the rougher concentrates being reground to 80% passing 20 microns prior to cleaning to produce high-value separate copper/lead (with gold and silver) and zinc (with gold and silver) concentrates. Concentrates would be bagged, containerized and trucked to port for ocean shipment to smelters. ▪ The following future evaluations may improve the project economics: perform a variability study to improve confidence in the metallurgical response of the deposit; and optimize flotation reagent addition rates to lower the process plant operating cost.
Infrastructure	• The powerline and a main substation near the plant would be permitted and constructed by CNEL EP. There is a risk that a significant delay in connecting the Project to the National hydropower grid could increase the estimated upfront capital cost.
NSR	• The PEA assumes that Toachi would produce and sell Cu/Pb and Zn concentrates to at least two different smelters in Asia at projected preliminary smelting, refining and payable / foreign exchange terms. The estimated net smelter return (NSR) terms used in the PEA are expected to change with time. There is a possibility that the estimated concentrate transportation costs, treatment costs and refining costs may increase. • The preliminary NSR terms used in the cashflow are based on projected smelter market conditions / NSR terms and it assumed that the Cu/Pb and Zn concentrates would be sold to smelters in Asia. There is a possibility that future market conditions / NSR terms may be significantly harsher or more favourable than projected, and/or that the Project would sell its concentrates to smelters in other parts of the world.
Tailings	• Exploration diamond drilling results could affect the waste rock/tailings storage program. If exploration drilling leads to an increase in the potentially mineable resources tonnage and tailings quantities, significant additional costs could be incurred to provide additional tailings storage capacity if available within a reasonable distance from the plant and /or include the use of tailings in the mine backfill program. The waste rock/tailings options study described in Section 26 would be expected to address such a scenario.
Environmental	• It is anticipated that the mine waste rock and tailings would be net acid generating and measures to inhibit acid drainage generation and conventionally lime-treat mine water and TSF drainage and seepage are included in the PEA. The mine geotechnical and mine waste rock/tailings disposal options studies described in Section 26 and the social and environmental assessment and permitting process may identify the need for additional engineered controls and add to the estimated capital and operating costs.
Permitting	• While the legal framework needed to develop and operate the Project is in place, it is expected that the permitting and development of the Project will

Area	Potential Risks / Uncertainties
	largely depend on the success of the company's community social consultation efforts. There is a latent risk of project permitting and development delays.
Projected potential economic viability	• The cashflow is based on projected revenues and costs. There is a risk that income from concentrate sales may be lower than projected due to higher concentrate transportation costs and harsher NSR terms.
Exclusion	• Land use agreement costs cannot be estimated with confidence at this time. These costs have been treated as sunk costs in the PEA.

Recommendations

Based on the positive results of the PEA, and in the absence of any currently identified material obstacle to advancing the Project towards an eventual production decision, it is recommended that the Project be advanced to the Pre-Feasibility Study (PFS) stage. The estimated cost of the FPS and supporting work / studies is shown in the following table.

Recommended budget

Area	Recommendation	Recommended work budget
PFS supporting work / studies:		
Geology	• Additional diamond drilling to convert Inferred Mineral Resources	$4.0M
Mine planning	• Mine geotechnical study • Waste rock/ tailings storage options study	$0.24M $0.4M
Mineral processing	• Metallurgical testwork to optimize metal recoveries/concentrate grades/mill process	$0.25M
Environmental	• Hydrological / hydrogeological water management study • Energy conservation	$0.25M
Social	• Expand socio-economic baseline	$0.25M
	Subtotal PFS supporting studies	**$5.39M**
Pre-Feasibility Study prepared in accordance with NI 43-101 requirements:		**$2.0M**

	Work budget	**$7.39M**

2.0 Introduction

2.1 Terms of Reference

At the request of Mr. Alain Bureau, President and CEO of Toachi Mining Inc. ("Toachi"), SGS Canada Inc.'s SGS Geological Services ("SGS") has prepared this technical report ("the Report") conforming to the standards dictated by National Instrument 43-101 ("NI 43-101"), companion policy NI 43-101CP and Form 43-101F (Standards of Disclosure for Mineral Projects) in respect to the La Mina VMS Project, part of the La Plata Project ("the Project") located in Cotopaxi Province, Ecuador. The following report presents the details of the Preliminary Economic Assessment that was announced in a press release dated April 30, 2019 for the Project. The Preliminary Economic Assessment has been prepared in compliance with the requirements of Canadian National Instrument (NI) 43-101 and in accordance with guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10, 2014.

Toachi is a Toronto based company trading on the TSX Venture exchange under the symbol "TIM.V" with its corporate office at:

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SGS Geological Services is SGS's geological and mining consulting division that was originally incorporated in the province of Quebec under the name of Systèmes Géostat. SGS provides a wide range of geological and mining consulting services to the international mining industry, including geological evaluation and valuation reports on mineral properties. The firm's services are provided through offices in Blainville and Quebec in Canada. SGS is not an insider, associate or affiliate of Toachi.

Neither SGS nor the authors of this report (nor family members or associates) have a business relationship with Toachi or associated company, nor with any company mentioned in this report that is likely to materially influence the impartiality or create a perception that the credibility of this Report could be compromised or biased in any way. The views expressed herein are genuinely held and deemed independent of Toachi.

This report presents a NI 43-101 compliant Preliminary Economic Assessment and updated Mineral Resources estimate for the La Mina VMS deposit (formerly known as the La Plata VMS Project). The PEA includes 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.

The effective date of this report is March 30, 2019

This report was prepared and co-authored by Messrs. Daniel Leroux, P.Geo., David Orava, P. Eng., Brian Wolfe, P.Geo., Simon Meik, CP and Quighua (Jason) Jin, P.E., all Qualified Persons ("QP") under the regulations of NI 43-101.

Toachi has accepted that the qualifications, expertise, experience, competence and professional reputation of SGS's Principles and Associate Geologists and Engineers are appropriate and relevant for the preparation of this report. Toachi has also accepted that SGS's principals and associates are members of professional bodies that are appropriate and relevant for the preparation of this report.

2.2 Site Inspections

During the course of completing the PEA technical Report of the La Mina VMS Project in 2018 and 2019, the following QPs visited the site to review the status of the Project, conduct audits, and discuss future plans with Toachi staff.

Site visits by the QPs for the Report were as follows:

David Orava visited the Project site on September 4 and 5, 2018.

A site visit by Mr Wolfe was undertaken from May 20th to May 23rd, 2017.

The purpose of the Report is to provide a NI 43-101 compliant updated Mineral Resources estimate and Preliminary Economic Assessment Technical Report on the La Mina VMS Project whereby the current 2019 Updated Mineral Resources Estimate was used as the basis for the study. SGS understands that the Company will use the Report internally for decision-making purposes and publicly in support of reporting obligations and possible corporate financing activities related to the Project. This report will be filed to conform with the requirements of NI 43-101.

2.3 Sources of Information

In preparing this report, SGS reviewed geological reports and maps, miscellaneous technical papers, company letters and memoranda, and other public and private information as listed in Section 27 "References" at the conclusion of this report. SGS has assumed that all of the information and technical documents reviewed and listed in the "References" are accurate and complete in all material aspects. While SGS carefully reviewed all of this information, it has not conducted an independent investigation to verify its accuracy and completeness.

In addition, SGS carried out discussions with Toachi corporate and local management, consultants and technical personnel, in particular, Jonathan Goodman, Chairman; Alain Bureau, President and CEO; Pierre Bureau, P.Eng., Project Manager; Andrés Ycaza, Toachi's legal advisor; Sinisa Glisic, Geologist; Dushan Navarro, Social Director; Andrea Chavez, Environmental Director; and Juan de Dios Saavedra, Office Manager. SGS also communicated with Barry Morocho, B.Eng., MBA, Toachi's taxation advisor.

Although copies of the licences, permits and work contracts were reviewed, SGS has not verified the legality of any underlying agreement(s) that may exist concerning the licences or other agreement(s) between third parties. SGS reserves the right, but will not be obligated to revise this Report and conclusions if additional information becomes known to SGS subsequent to the date of this Report.

Toachi has warranted that full disclosure of all material information in its possession or control at the time of writing has been made to SGS, and that it is complete, accurate, true and not misleading. The Company has also provided SGS with an indemnity in relation to the information provided by it, since SGS has relied on Toachi's information while preparing this report. The Company has agreed that neither it nor its associates or affiliates will make any claim against SGS to recover any loss or damage suffered as a result of SGS's reliance upon that information in the preparation of this report. Toachi has also indemnified SGS against any claim arising out of the assignment to prepare this report, except where the claim arises out of any proven willful misconduct or negligence on the part of SGS. This indemnity is also applied to any consequential extension of work through queries, questions, public hearings or additional work required arising out of the engagement.

A portion of the background information and technical data was obtained from the following Technical Reports previously filed by Toachi:

Wolfe, B.R. and Meik, S. (2017) Toachi Mining Inc. NI 43-101 Resource Report la Plata VHMS Project Cotopaxi Province Ecuador, South America. 322p.

2.4 Currency, Units, Abbreviations and Definitions

All units of measurement used in this report are SI metric unless otherwise stated. Where third party reports use units other than SI metric, then the original units have been preserved throughout.

Currency is expressed in US Dollars ('US$') unless otherwise stated.

Table 1. List of Abbreviations

	Description		Description
$	United States of America dollars	l/hr/m²	litres per hour per square metre
µ	microns	M	million
2D	two dimensional	m	metres
3D	three dimensional	Ma	Million years
AAS	atomic absorption spectrometer	MIK	Multiple Indicator Kriging
Au	gold	ml	millilitre
bcm	bank cubic metres	mm	millimetres
CC	correlation coefficient	MMI	mobile metal ion
cfm	cubic feet per minute	Moz	million ounces
CIC	carbon in column	Mtpa	million tonnes per annum
CIL	carbon-in-leach	Mt	Million tonnes
cm	centimetre	N (Y)	northing
cusum	cumulative sum of the deviations	NaCN	sodium cyanide
CV	coefficient of variation	NATA	National Association of Testing Authorities

DDH	diamond drillhole	NPV	net present value
DTM	digital terrain model	NQ_2	size of diamond drill rod/bit/core
E (X)	easting	ºC	degrees centigrade
EDM	electronic distance measuring	OK	Ordinary Kriging
EV	expected value	oz	troy ounce
g	gram	P80 -75µ	80% passing 75 microns
g/m³	grams per cubic metre	PAL	pulverise and leach
g/t	grams per tonne	ppb	parts per billion
HARD	half the absolute relative difference	ppm	parts per million
HDPE	high density poly ethylene	psi	pounds per square inch
HQ_2	size of diamond drill rod/bit/core	PVC	poly vinyl chloride
hr	hours	QC	quality control
HRD	half relative difference	Q-Q	quantile-quantile
ICP-MS	inductivity coupled plasma mass spectroscopy	RAB	rotary air blast
ID	Inverse Distance weighting	RC	reverse circulation
ID^2	Inverse Distance Squared	RL (Z)	reduced level
IPS	integrated pressure stripping	ROM	run of mine
IRR	internal rate of return	RQD	rock quality designation
ISO	International Standards Organisation	SD	standard deviation
ITS	Inchcape Testing Services	SGS	Société Générale de Surveillance
kg	kilogram	SMU	selective mining unit
kg/t	kilogram per tonne	t	tonnes
km	kilometres	t/m³	tonnes per cubic metre
km²	square kilometres	Y	year

3.0 Reliance on Other Experts

SGS has assumed, and relied on the fact, that all the information and existing technical documents listed in the references section of this report are accurate and complete in all material aspects. While all the available information presented to us has been carefully reviewed, we cannot guarantee its accuracy and completeness. SGS reserves the right, but will not be obligated, to revise our report and conclusions if additional information becomes known to us subsequent to the date of this report.

A draft copy of this report has been reviewed for factual errors by Toachi and SGS has relied on Toachi's historical and current knowledge of the property in this regard. Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this report.

3.1 Mineral Tenure

SGS has relied upon the following documents obtained by way of the Government of Ecuador, the Toachi land management expert, and legal opinions obtained by Toachi for the information included in Section 4.0 of this report.

3.2 Legal, political, social-environmental and taxation aspects

SGS relied upon the following experts in regards to legal, political, social-environmental and taxation:

- Andrés Ycaza, Toachi's legal advisor in regard to: clarifying SGS's understanding of legal requirements in regards to hours of work, pay rates, employment contracts and approvals, overtime payments, social cost components, environmental legislation and status, and employee share of profit / corporate taxation aspects.
- Dushan Navarro, Social Director and Andrea Chavez, Environmental Director in regards to social and environmental aspects respectively.
- Barry Morocho, B.Eng., MBA, Toachi's taxation advisor based in Ecuador. Mr. Morocho reviewed and finalized the corporate taxation section of the PEA cashflow model used to estimate after-tax cashflows and after-tax IRR, NPV and payback presented in this report.

4.0 Property Description and Location

The concession comprising the Project is situated within the Cordillera Occidental of central Ecuador, in the district of Toachi, Province of Cotopaxi (Figure 1). The project geographic coordinates are UTM 729 500 east; 9 957 300 north. By road, the Project is located approximately 100 kilometres to the west-south-west of Quito, the capital of Ecuador, and 25 kilometres to the south-east of the city of Santo Domingo de los Tsachilas. The property lies immediately north of the Iliniza Ecological Reserve, which covers 149,000 ha of the Andean Humid Forest (Figure 2). The southern portion of the La Plata concession is within the Toachi-Pilaton Protected Forest. Mining exploration activities are permitted, subject to Ministry approval, within protected forests.

Figure 1. Project Location

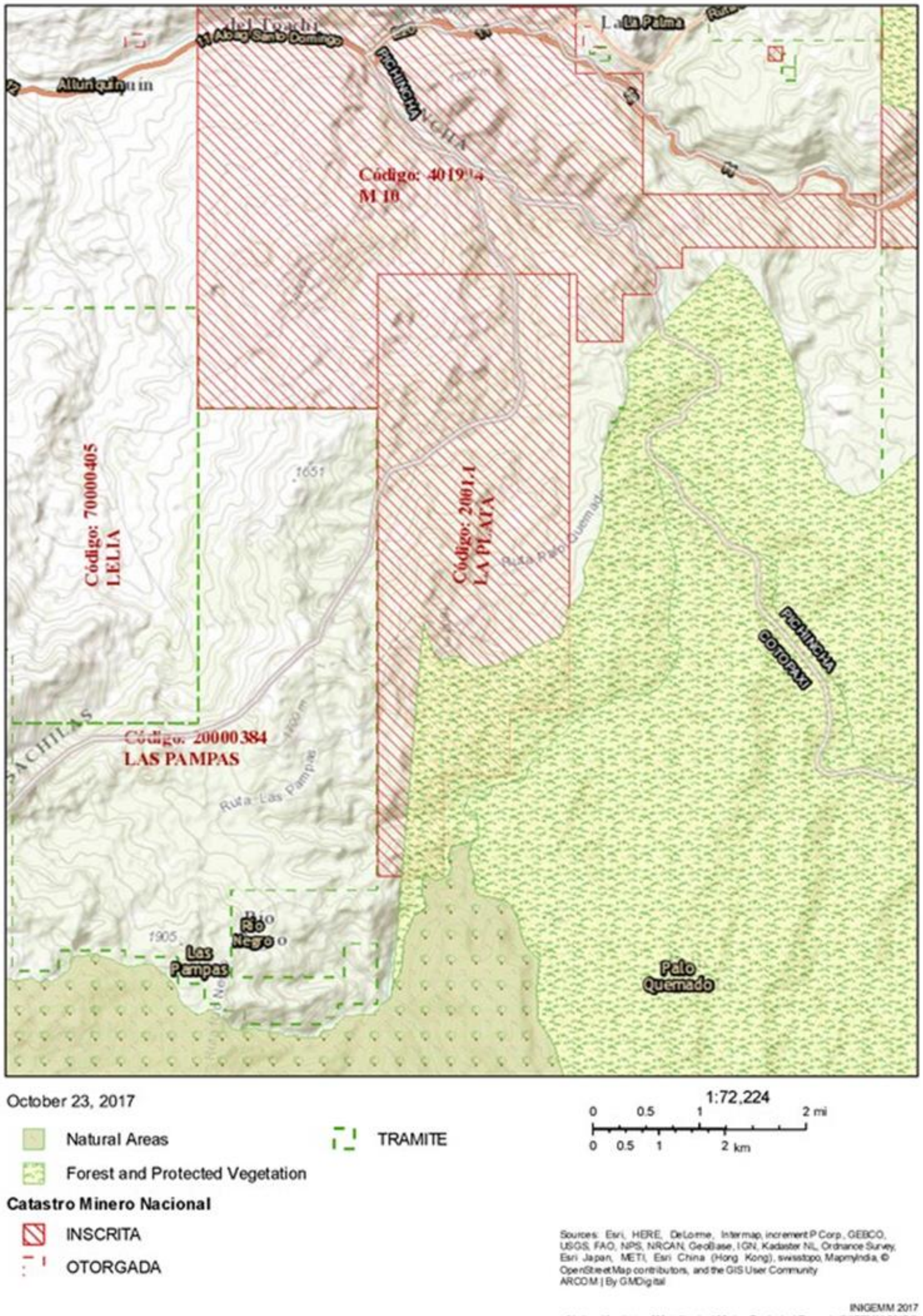

Figure 2. Ecological Reserve Location

4.1 Ownership, Mineral and Surface Rights, Tax and Royalties

4.1.1 Property Ownership

Originally, the mining concession "La Florida" was granted to the company Ecuatomin S.A., under resolution issued by the National Director of Mines and registered in the Book of Mining Concessions of the Property Registry of Latacunga on June 1, 1992.

Then, the concession "La Florida" was transferred to Promociones Mineras Prominex S.A., duly registered in the Book of Mining Concessions of the Property Registry of Sigchos on September 24, 2001. Table 2 summarizes the details of the original mining concessions of the La Plata project.

Sultana acquired the concession "La Florida" from Promociones Mineras Prominex S.A. by public deed under the Eleventh Public Notary of Quito, on June 25, 2002, and registered in the Book of Mining Concessions of the Property Registry of Sigchos on June 27, 2002.

Pursuant to the issue of the Mining Act of 2009, the Ministry of Non Renewable Natural Resources grants, on May 6, 2010 the Resolution of substitution of the mining title of "La Florida", which was notarized before the Eighteenth Notary of Quito on May 7, 2010, and registered in the Mining Registry on May 12, 2010.

In regard to the concession "Loma del Tigre", in June 28th, 2002, an assignment of mining rights was made between COMPAÑÍA PROMOCIONES MINERAS PROMINEX S.A. in favor of COMPAÑÍA SULTANA DEL CÓNDOR MINERA S.A. SULCOMI. Based on the record file, on May 6th, 2010, the substitution of the mining title, which was notarized on May 12th 2010 by the 18th Public Notary of the Quito District.

Table 2. La Mina Project Mining Concessions

Item	La Florida	Loma Del Tigre	La Plata
Reference Code	2001.1	2001.28	2001.1
Location	Cotopaxi Province	Cotopaxi Province	Cotopaxi Province
Surface Area	642 ha	1,658 ha	2,335 ha
Duration	30 years	30 years	25 years
Mine Title Granted	21 Aug 2001	10 Sept 2001	2 Aug 2016
Mine Title Subscribed	3 Sept 2001	13 Sept 2001	15 Nov 2016

Both concessions were transferred from Sultana del Condor to Compañía Minera La Plata ("CMLP") on August 2, 2016.

According to Resolution of the Regional Undersecretary of Mines from Riobamba of November 15, 2016, it was authorized the merger of the concessions "Loma del Tigre" and "La Florida", into a new one called "La Plata".

Finally, due to Resolution of September 20, 2017, the concession “La Plata” was qualified under the regime of small-scale mining, which was registered on September 25, 2017. Small scale mining is a regime to allow produce up to 300 tons per day at underground production, and up to 1,000 tons per day at open pit mining, also allows to conduct exploration activities with no restriction in time. Small scale mining does not need the signature an exploitation agreement with the Ecuadorian government, and has a fixed royalty of 3%, Income Tax rate is currently 22%, and VAT is 12%.

Ecuador´s mining regulation also provides the right to update to mid – scale mining, which allows production of up to 1000 tons per day at underground mining and up to 3,000 at open pit mining, with a fixed royalty of 4%.

“CMLP” holds the mineral rights for the La Plata concession but the area is covered by properties with surface rights held by more than two dozen individuals (Table 3). The area that is potentially relevant to infrastructure, site facilities, exploration, and the underlying mineralization constitutes approximately 80 ha and includes part or all of properties held by eight owners (Figure 3). CMLP has discussed with each property owner its interest in obtaining the surface rights to develop the project. The total cost for land acquisition is still unknown.

Ecuadorian Mining Law states that the title holder of a mining concession who needs to establish an easement on a piece of land or upon an adjacent concession and has not been able to arrive at an agreement with the owner or legal occupant, may apply to the Regional Director of Mining so that an easement may be obtained.

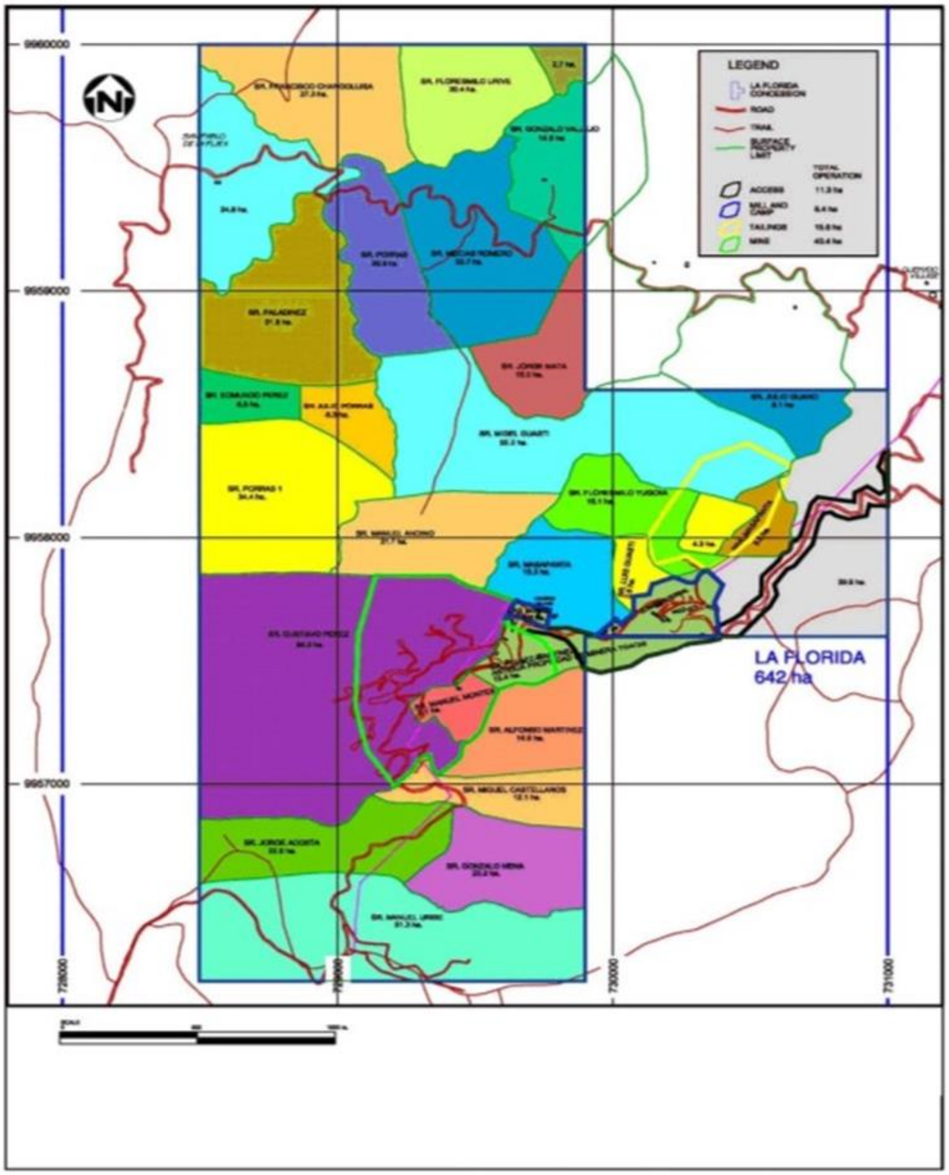

Figure 3. Landowners plan map location

Table 3. Surface Property Owners: La Florida and Loma del Tigre Concessions						
Owner	**Area (ha)**	**Mine Area (ha)**	**Mill & Camp Area (ha)**	**Tailings Area (ha)**	**Access (ha)**	**Total Operation (ha)**
LA FLORIDA CONCESSION						
Nerney Martinez (COPLASA)	15.7	4.9	7.0	-	3.8	15.7
Miguel Guasti	55.2	-	-	5.1	0	5.1
Jorge Mata	15.0	-	-	-	-	-
Julio Guano	8.1	-	-	-	-	-
Floresmilo Yugcha	15.1	-	-	3.7	-	3.7
Luis Guasti	4.3			4.3		4.3
Luis Guasti	3.9	-	-	-	-	-
Sra. Masapanta	5.5	-	-	2.7	-	2.7
Sr. Masapanta	15.3	-	-	-	-	-
Manuel Montes	5.1	5.1	-	-	-	5.1
Manuel Andino	21.7	-	-	-	-	-
Alfonso Martinez	14.9	-	-	-	-	-
Miguel Castellanos	12.1	-	-	-	-	-
Luis Gonzalo Orbe (Escuela)	0.9	-	0.9	-	-	-
Francisco Changoluisa	27.3	-	-	-	-	-
Floresmilo Uribe	20.4	-	-	-	-	-
Gonzalo Vallejo	14.8	-	-	-	-	-
Sr. Paladinez	31.6	-	-	-	-	-
Sr. Porras	20.9		-	-	-	-
Sr. Porras 1	34.4	-	-	-	-	-
Mecias Romero	33.7	-	-	-	-	-
Edmundo Perez	6.5	-	-	-	-	-
Julio Porras	8.3	-	-	-	-	-
Gustavo Perez	90.2	30.4	-	-	-	30.4
Jorge Acosta	22.6	-	-	-	-	-
Manuel Uribe	51.3	-	-	-	-	-
Gonzalo Mena	20.2	-	-	-	-	-
? (Northeast corner)	2.7	-	-	-	-	-
? (Northeast corner)	24.8	-	-	-	-	-
? (East edge)	39.6	-	-	-	8.0	8.0
Total La Florida	642.0	40.4	7.9	15.8	11.8	75.9

Owner	Area (ha)	Mine Area (ha)	Mill & Camp Area (ha)	Tailings Area (ha)	Access (ha)	Total Operation (ha)
LOMA DEL TIGRE CONCESSION						
Herney Martinez	4.1	-	-	-	4.1	4.1
TOTAL AREA	646.1	40.4	7.9	15.8	15.9	80.0

4.2.2 Ferrum-Toachi-Sultana Agreement

On October 28th, 2015, Ferrum ("The Company") announced the signing of a letter of intent with Sultana pursuant to which Ferrum has been granted the option to acquire a minimum 60% and a maximum 75% interest in the Project over a period of four years. To earn its 60% interest, Ferrum must make cash payments totalling US$2.0 million, including a US$75,000 exclusivity payment (paid on November 6th, 2015) and US$125,000 due on February 10th, 2016 and incur expenditures totalling US$4.0 million. If the Company earns the initial 60% interest, it has the right to finance the cost of a feasibility study over an additional two year period which will increase its interest to a total of 65% interest. If the Company arranges capital expenditure financing of less than US$60 million to build a mine, it will earn an additional 5% interest for a total 70% interest and if the capital expenditure financing exceeds US$60 million, it will earn an additional 10% interest for a total 75% interest.

The interest of Sultana will be carried through to commercial production. Upon the commencement of commercial production, project cash flow available for distribution will be divided by the owners as to their respective interests and 85% of Sultana's interest in its share of the project cash flow will be paid to the Company in repayment of any capital advanced by the Company in place of or supplementary to any contemplated project financing.

During the earn in period and thereafter, so long as the Company holds a minimum 50% interest, the Company will have the right to act as operator. A technical committee will be established and the Company's nominee will be appointed as Chairman, who will have final veto on matters pertaining to programs and budgets.

Sultana will have a buy back right which will take effect after the Company's acquisition of the 60% interest and which may be exercised by Sultana if, there is no substantial activity at the Project for a period of three consecutive years, and Ferrum has not made good faith best efforts to arrange for financing. The terms of the buy-back right will then be negotiated by the parties, in good faith.

On March 14, 2016, Ferrum Americas Mining Inc. "Ferrum" announced that the company has changed its name to Toachi Mining Inc. and has consolidated its common shares on the basis of five (5) existing common shares for one (1) new common share.

4.2 Environmental Issues

The environmental liabilities associated with past mining activities at La Plata include unsecured waste rock storage facilities and tailings disposal areas and abandoned and deteriorated buildings and infrastructure. Some water quality testing conducted in 2000 yielded anomalous metal concentrations in the La Plata Creek and relatively high coliform counts in the La Florida Creek and Toachi River. New water testing will be required to adequately characterize the quality of all water features around the Project area.

Agriculture practice such as the production of sugar cane, development of pasture land and harvesting in steeply-sloping terrain contribute to erosion and environmental degradation of soils.

According to a title opinion completed for Ferrum by the law firm of Bustamante & Bustamante, based in Quito, Ecuador on December 1, 2015, there are no environmental penalties or violations registered against the concessions or Sultana.

4.3 Legal Framework

Ecuador's environmental legislation is extensive and the requirements for early stage activities (i.e., exploration) are well defined. Ecuador is one of the few Latin American countries that have adopted an Environmental Impact Assessment process for exploration activities. Argentina, Chile, and Peru have adopted a similar process to conduct environmental assessments for early stage exploration.

Under the Ecuadorian Mining Law (Law 126), Title 5 (Obligations of Mining Concession Holders) Chapter 2 Environmental Preservation, Article 79 states that:

''Mining concession holders are required to complete environmental impact studies and environmental management plans to prevent, mitigate, rehabilitate, and compensate for environmental and social impacts as a result of their activities. These studies would be approved by the Sub secretary of the Environment within the Ministry of Energy and Mines''.

The environmental legal framework in Ecuador has been in effect since 1976, when general regulations for the prevention and control of pollution were established. Articles 79 to 87 of the Mining Law outline obligations and environmental considerations applicable to mining concessionaires, according to their activities. The Ministry of Energy and Mines issued the Environmental Regulation for Mining Activities (Executive Decree 625), to regulate environmental management in the mining industry and to assign the administrative control and responsibility of such to the Environmental Department of the Ministry of Energy and Mines.

The Ministry for the Environment of Ecuador is charged with the coordination and supervision of environmental protection work. The activities of the Ministry for the Environment are supported by the following laws:

- Prevention and Control of Pollution (general environmental regulation for water, soil, and air)
- Environmental Management

- Forestry and Natural Resources and Wildlife Conservation
- Conservation and Sustainable Use of the Biodiversity

The Environmental Department of the Ministry of Energy and Mines will evaluate and consider for approval the environmental studies of the Project. According to the Environmental Regulation for Mining Activities, the required environmental studies are:

- Environmental Impact Preliminary Evaluation ("EPIA"). The EPIA is a general environmental study, which describes the environment components, project activities, potential environmental effects, and planned prevention, correction, and/or mitigation measures
- Environmental Impact Assessment ("EIA"). The EIA is a detailed, multidisciplinary technical study which identifies and evaluates the potential negative environmental effects and details specific preventative or corrective measures for the effects
- Environmental Audit ("AA"). The AA provides a means of assessing and controlling the measures proposed in the EIA and legal framework

Prior to exploration, the mining company must complete and submit an EPIA. There have been several environmental studies submitted by Cambior, Cornerstone and Sultana to the government.

According to a due diligence review completed for Ferrum by the law firm of Bustamante & Bustamante on December 1, 2015, the Project was granted an environmental license from the Ministry of the Environment on November 7, 2011. Sultana's representatives also stated that all permitting is in place to carry out the next exploration phase on the Project (personal communication from Bolivar Reza, Sultana's senior geologist, December, 2015).

5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Access

La Mina VMS deposit is accessed by driving 85 km on paved road from the capital city of Quito to the village of Unión del Toachi. From the turn off point, a 13-km dirt road leads to the Project passing through the village of Palo Quemado where Compañia Minera la Plata's field office and accommodation is located. The total driving time from Quito is approximately three hours using a four-wheel drive light truck.

5.2 Climate

The climate in the region of the Project is sub-tropical and varies from cool and humid to semi-humid at higher elevations to a tropical climate at lower elevations. The average temperature in the Department of Cotopaxi ranges between 18° and 22°C. Rainfall is high, especially between December and May with annual precipitation reaching up to 2,657 mm.

5.3 Local Resources

Economically, the central part of Ecuador is characterized as rustic and rural with a local economy largely dominated by agriculture. Much of the area (60%) is occupied by small farms and slopes and incised valleys are covered with thick jungle. The climate conditions are good for farming activities such as bananas and sugar cane, and grazing cattle.

Local unskilled labour is readily available. Some exposure to modern exploration techniques has occurred in the past, and more recently Toachi has implemented modern training for best practice procedures.

Services such as environmental monitoring, assaying services, field equipment supply and drilling services are readily available from Quito. Other minor services are available from the nearby city of Santo Domingo de los Tsachilas.

5.4 Infrastructure

The previously small-scale operating mine at La Mina VMS deposit had the basic infrastructure to support the operation, including an access road, power supply, water and ancillary facilities.

The access road is unpaved which departs from the main highway at the village of Unión del Toachi to the north. If the Project goes to commercial production, this road will require some improvement for mine construction and daily travelling in order to provide all necessary equipment and products for the development of the Project. The metal concentrates from La Mina could be shipped via the existing road network to the port of Esmeraldas on the north coast of Ecuador.

An approximately 7 km long power line would be constructed from an existing substation on the National hydropower grid to a location adjacent to the proposed mill site. In 2018, Toachi representatives met with La Empresa Eléctrica Pública Estratégica Corporación Nacional de Electricidad (CNEL EP) Unidad de

Negocio Santo Domingo and obtained preliminary cost information from CNEL EP for the construction of the power line and new substation near at the mill based on a then-contemplated 12-15 MWA range power system.

The Toachi Pilaton Hydroelectric Power Plant Project (250MW) is under construction. This plant is located approximately 5 km from the proposed processing plant and mine site area clearly representing a potential energy source for future operations.

Water is available on site from Quebrada La Florida adjacent to the mine site, and from a number of different nearby water sources. The previous operation also used water from this Quebrada La Florida source.

Local cell phone service is also available in the area.

5.5 Physiography

La Plata lies on the western flank of the Cordillera Occidental of central Ecuador. The terrain is rugged and bisected by the Toachi and San Pablo Rivers and their tributaries, resulting in moderately steep terrain with local steep-walled canyons. Elevations range from 1,100 to 1,700 meters.

A summary of the physiography of Ecuador was presented in a thesis by David Tripodi, titled "The Early Tertiary Gold-Rich VHMS of La Plata, Western Cordillera, Ecuador, 2002" (Figure 4). The summary is presented as follows:

"Ecuador consists of three main physiographic regions corresponding to different geological provinces, the coastal plain or "Costa" to the west, the Amazon basin or "Oriente" to the east and the Andean region or "Sierra" in the middle.

The Costa encompasses a magmatic substratum (Goossens and Rose, 1973; Lebrat, 1985) representing an oceanic floor fragment of Early Cretaceous age, called the Piñón Formation (Reynaud et al., 1996; Reynaud et al., 1999), overlaid by Cretaceous to Paleogene volcanic and volcaniclastic arc series and finally by Tertiary forearc sedimentary series.

The Oriente is a Mesozoic to Cenozoic, hydrocarbon-rich sedimentary foreland basin that includes a platform carbonate sequence on top of a cratonic basement. Basements and cover sequences are intruded by large granitoid batholiths, mainly along the complex sub- Andean zone of folding and thrusting that lies along the boundary between the Oriente and the Andes (Hughes and Pilatasig, 2002).

Finally, the Sierra of Ecuador is an exception in the whole South American Andes, because of its two cordilleras, the Eastern Cordillera or Cordillera Real and the Western Cordillera or Cordillera Occidental, separated by a central graben called the inter-Andean graben, filled by Pliocene and Quaternary volcano-sedimentary rocks. On the basis of a gravimetric study, Feiniger and Seguin (1983) propose that the basement of the graben would be an extension of Cordillera Real rocks.

The La Plata district is hosted by the Paleocene to Eocene Macuchi Unit submarine volcano-sedimentary sequence, with pillow lavas and related intrusive bodies, derived from an oceanic island arc of basaltic to andesitic composition, the Macuchi terrane."

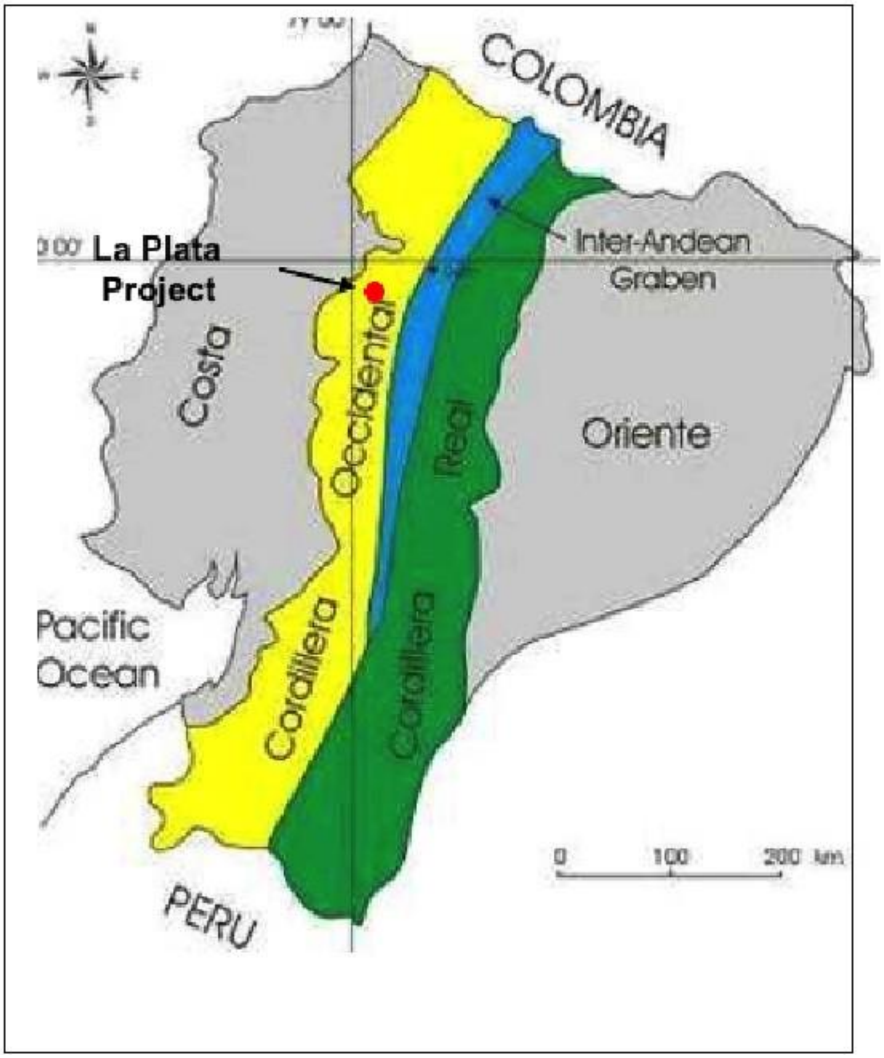

Figure 4. Location of the Three Distinct Physiographic Regions of Ecuador

6.0 History

The most important technical documents available for the preparation of this report are listed in Section 27 (References) of this report.

The history of works carried out on the La Mina VMS deposit (formerly known as the in La Plata deposit) extends possibly to before the eighteenth century; however, the chronicles began in 1740. Exact records of these works are unknown. Information is more concise from the 1940s onwards when the Cotopaxi Exploration Company began works, followed by several companies until the present day.

The main activities at the La Plata mining concession are summarised below:

During modern times, mineralization at La Plata was discovered around 1946 to 1947 by Cotopaxi Exploration. Three levels were opened and reserves of approximately 120,000 tonnes of polymetallic ore were defined.

From 1965 to 1967, Duncan-Derry Exploration of Toronto carried out a stream sediment sampling program over an area of 2,900 km^2 follow-up by soil sampling and drilling in the San Pablo area (south of Rio San Pablo and north of the La Mina deposit), the results of which have not been well documented. An intersection of 22 meters grading 1.8% Cu was reported in hole LP-24 as well as mineralization in several other holes. Records of the number of holes and how many meters drilled are not available.

In 1975, production was initiated at La Plata by Compañia Minera Toachi S.A., a company in which Outokumpu of Finland was the largest shareholder with approximately a 35% interest. From 1975 to 1981, Minera Toachi completed about 8,512 meters of diamond drilling (mostly executed in 1978 and 1979). The core is no longer available for review as core boxes have deteriorated and collapsed. There were three types of drill holes: MT series – 30 holes from surface, D series – 83 holes from underground and L series – 20 holes as cutting samples from jack leg drilling. During this period, 130,000 metric tonnes of ore grading 4.11 g/t Au, 65.1 g/t Ag, 3.62% Cu and 7.14% Zn was extracted. Mining rate was 80 tonnes per day and recoveries were 83% for Cu and 65% for Zn. All tonnage was mined above level 3. Minera Toachi posted mineral resources of 235,000 metric tonnes grading 3.87 g/t Au 43.37 g/t Ag, 4.35% Cu and 4.76% Zn; no cutoff grade was provided and the resources were referred to as "reserves". The mine was shut down in 1981 due to a number of problems including excessive government royalties, high administrative costs, poor management, high bank debt, high personnel demand and a strike.

From 1976 to 1977, Geominex S.A. conducted a stream sediment sampling program over an area of 500 km^2 centered on the La Plata mine. Results are reported to have been questionable. However, analysis of heavy mineral concentrates revealed that chalcopyrite was a frequent constituent of stream sediments in the San Pablo area, south of Rio San Pablo.

In 1977, Minera Toachi and Compañía Minera Buenaventura S.A. (''Buenaventura'') conducted exploration including self-potential, mise-à-la-masse, induced polarization and resistivity surveys in the mine area. They excavated approximately 300 meters of ramp from 1438.8 masl (surface) to 1310.0 masl.

In 1982, Metallgesellschaft of Germany prepared a feasibility study but the project was soon abandoned. Reserves below level 3 were estimated at 233,200 metric tonnes grading 3.87 g/t Au, 43.15 g/t Ag, 4.77% Cu and 4.35% Zn. A total of 13,500 metric tonnes, at similar grades to the historical production by Toachi, remained above level 3.

From 1989 to 1991, Ecuatomin S.A., a subsidiary of the BRGM of France, conducted geophysical (transient electromagnetic, magnetic, self-potential) surveys and geochemical (rock and soil) sampling using motorized auger over the Henri concession (more recently known as the Florida concession). This work identified a wide area of sericite alteration locally enriched in Cu, Ag, Au, Pb and Zn.

In 1991, Codigem conducted S.P. and magnetic surveys over portions of the Henri concession.

In 1993, Soria and Venegas completed their graduate thesis on the geology and mineralization at La Plata mine.

In 1993, Compañía Minera Oroecuador S.A. completed a compilation of the La Plata Mine and W. Bergy produced an evaluation report for Zappa Resources on Ecuadorian properties owned by Prominex which controlled Oroecuador.

From 1994 to 1995, Prominex carried out limited horizontal ground electromagnetic surveys and soil geochemistry in the Bella Vista and San Pablo areas; respectively in the north-east portion of the Loma del Tigre concession and the north-central portion of the La Florida concession.

In 1995, R. Laine produced a report on the ''La Plata Kuroko Massive Sulphide'' for Prominex.

In 1995, Zappa Resources Ltd. ("Zappa") acquired the property and a summary geological report was prepared by D. Allen. Zappa optioned the property to Cambior the following year.

From 1996 to 1998, Cambior reprocessed all Prominex geochemical data and completed an exploration program comprising of mapping, soil-stream-rock sampling (2,944 samples), geophysical surveys (magnetic, electromagnetic, induced polarization, surface and downhole pulse electromagnetic), surface and drill core whole rock geochemistry totalling more than 420 samples, 28 diamond drill holes totalling 8,626 meters, metallurgical testing and petrographic studies. In 1998, Cambior posted a mineral resource estimates of 840,459 tonnes grading 4.8 g/t Au, 54.4 g/t Ag, 4.1% Cu, 0.7% Pb and 4.2% Zn. Cambior dropped the option with Zappa in 1999 and the property returned to the hands of Prominex from Zappa in 2001.

In 2002 Sultana, a private Ecuadorian company acquired 100% ownership of La Plata from Prominex. They carried out metallurgical testing (Plengue, 2003) and updated the mineral resource estimates.

In 2004, Sultana hired AMEC to complete a review of the preliminary geological model of the La Plata deposit on cross sections and a vertical longitudinal section. At this time, the polygonal mineral resource estimates were 0.687 Mt at 6.3 g/t Au, 75.9 g/t Ag, 5.7% Cu, 0.6% Pb and 5.1% Zn (Sultana 2003, revised by AMEC and updated January, 2004). AMEC concluded that Sultana needed to complete a rebuild of the assay and geology database, undertake a comprehensive study of the mineralized composites and bulk density data and twin some of the drill holes completed by Minera Toachi to confirm the historical assay results before a new 3-D mineral resource block model could be constructed.

In December 2005, Sultana signed an agreement with Cornerstone, a Canadian mining and exploration company. In March 2006, Cornerstone commissioned AMEC to prepare a NI 43-101 technical report on La Plata. The two companies jointly conducted exploration on the Project from 2006 to 2009 including an environmental impact study, total field magnetometer and differential GPS surveys, 17 diamond drilling holes totalling 5,933 meters, down-hole transient electromagnetic surveys, whole rock geochemical sampling and mapping.

In 2009, the Cornerstone-Sultana JV was officially terminated and Sultana became the sole owner of the Project. AMEC amended its 2004 Scoping Study ("Sultana del Cóndor Minera S.A., La Plata Project Scoping Study, Ecuador, April 2004"). After incorporating the results of the 2006 to 2007 drilling campaign by Cornerstone in the database, Sultana updated its 2004 resource mineral estimates which totalled 0.914 Mt at 8.01 g/t Au, 88.29 g/t Ag, 5.01% Cu, 0.78% Pb and 6.71% Zn. In their 2009 study, AMEC also calculated a diluted mineable resource using a US$ 65 cut-off, based on August 2009 metal prices, of 0.861 Mt at 7.76 g/t Au, 86 g/t Ag, 4.95% Cu, 0.715% Pb and 6.38% Zn.

In September 2014 and March 2015 at the request of Sultana, AMEC updated the scoping study titled "La Plata Gold Rich VMS Project; Updated Cost and Financial Model" to produce the current AMEC Study. For its financial evaluation, the AMEC Study used as a base case scenario the same non-diluted mineral resource estimates as posted in the 2009 report of 0.914 Mt grading 8.01 g/t Au, 88.29 g/t Ag, 5.01% Cu, 0.78% Pb and 6.71% Zn). Following use of the O'Hara dilution factor, AMEC reported a subset mineable resource using a $US 86 cut-off, based on March 2015 metal prices, of 0.906 Mt at 7.54 g/t Au, 83 g/t Ag, 4.77% Cu, 0.68% Pb and 6.29% Zn.

From March 2015, Sultana had not reported any exploration or test work at La Plata.

On October 28th, 2015, Ferrum Americas Mining Inc. announced the signing of a letter of intent with Sultana pursuant to which Ferrum has been granted the option to acquire a minimum 60% and a maximum 75% interest in the Project over a period of four years.

In November 2015, Ferrum Americas Mining Inc. commissioned Alain Vachon P. Eng. Geologist to prepare an independent technical report for the La Plata project in compliance with the Canadian Securities National Instrument 43-101 Standards of Disclosure for Mineral Properties ("NI 43-101").

Toachi Mining Inc. was created from the restructuring of Ferrum Americas Mining Inc. The La Plata project was announced in November, 2015. Toachi Mining Inc. began trading on the Toronto Stock Exchange - Venture on March 14, 2016.

Compania Minera La Plata (CMLP) was formed from restructuring Sultana specifically for the La Plata project and the agreement with Toachi Mining Inc.

Physical works by Toachi (CMLP) began in August 2016, including diamond drilling, compilation, reinterpretation and validation of historical data, preliminary metallurgical test-work, and re-processing of historical geophysical data. Toachi have drilled 80 diamond drillholes for 13,747 metres and continues exploration diamond drilling works.

Based on the work completed. Toachi completed a National Instrument 43-101 resource calculation; The basis of this report.

All of the resource estimates listed above and dated prior to 2001, were generated prior to NI 43-101 and are considered historic estimates.

Resource estimates listed above and prepared after 2001 up to and including the 2009 resource calculation by the Cornerstone and Sultana JV are also deemed not current mineral resources or mineral reserves for the same reasons noted and should not be relied upon.

On September 2017, Toachi issued a NI 43-101 resources estimate. This Resources estimate was referring to the La Plata project. Since then, Toachi identified different exploration targets which are part of the La Plata mining concession. In order to clearly differentiate each deposit, Toachi will now name each project by the name of the deposit or the name of exploration target. Hence, this study presents the Preliminary Economical Assessment of the La Mina deposit, including La Mina South and La Mina North zones.

The following map (Figure 5) shows:

- La Plata Mining Concession borders;
- La Mina South (LMS)
- La Mina North (LMN)
- Others exploration targets

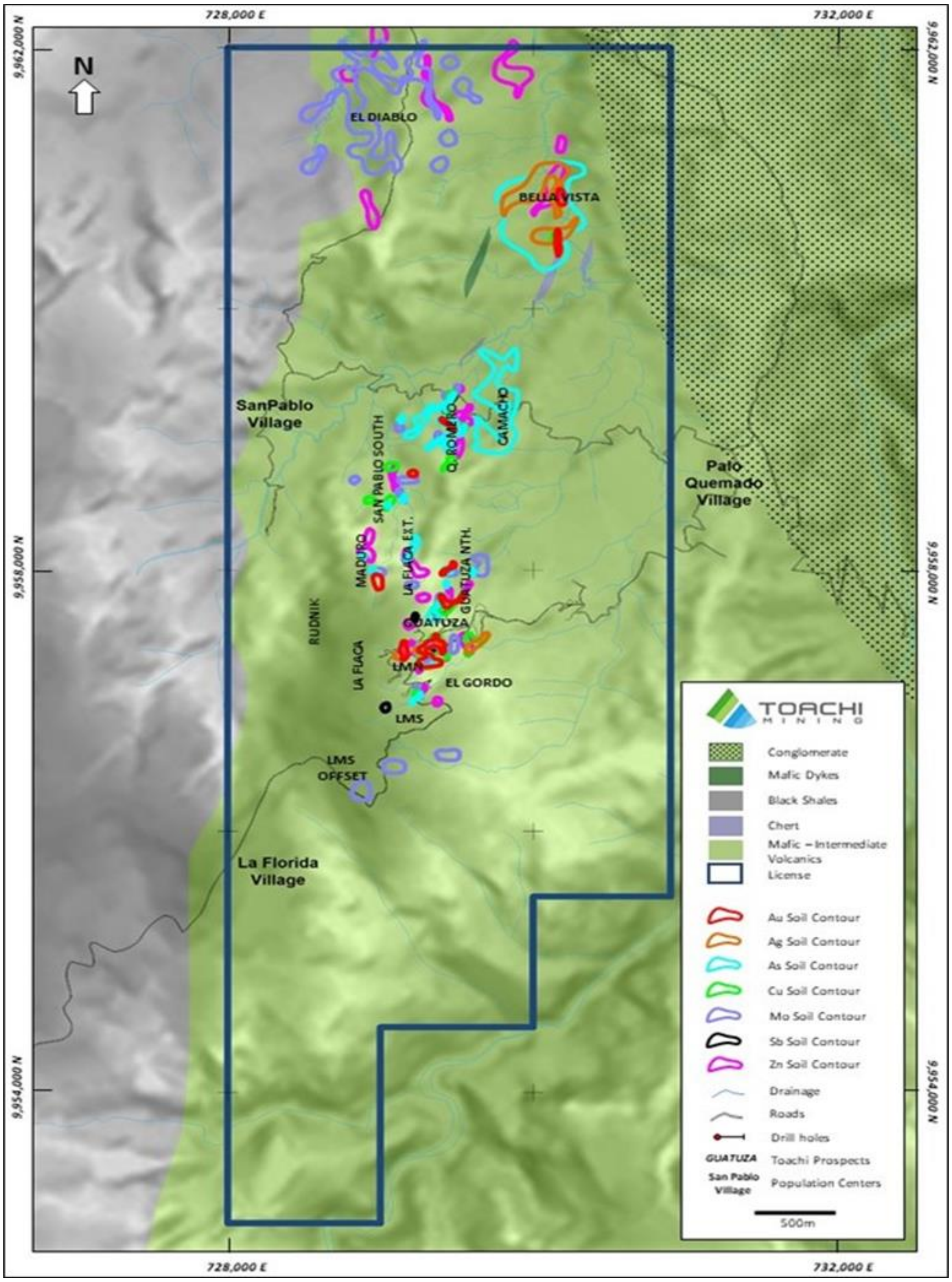

Figure 5. La Plata Mining Concession c/w LMS, LMN and Prospective Targets

7.0 Geological Setting and Mineralization

7.1 General

The following descriptions of the regional geology, the local geology and mineralization of the Project are drawn from several reports completed by O'Dowd (1999), Tripodi (2002), Cornerstone (2006), Franklin (2007), Charadia (2008), AMEC (2004-2006-2009-2014-2015) and Davis (2017), along with field observations, and core logging and review by Toachi staff.

7.2 Regional Geology

The La Plata district is situated in the western flank of the Cordillera Occidental (Figure 6). It is hosted by the Paleocene to Eocene Macuchi Unit, defined as a submarine volcano-sedimentary sequence with pillow lavas and related intrusive bodies, derived from an oceanic island arc of basaltic to andesitic composition. This unit crops out for more than 500 kilometers along a NNE to SSW trend and shows thicknesses up to 50 km. The sector of interest in which all polymetallic volcanic-hosted massive sulphide (VHMS) deposit occurrences have been recognized is delimited between the 0° and 1°S parallels. The undifferentiated superficial Quaternary deposits of the Costa represent the western limit of the district. The eastern limit of the district has been defined by the Toachi and Pilaló-Sigchos faults.

All recorded VHMS deposits or occurrences of the Western Cordillera are found in this district. The most important ones are the La Mina deposit which will be described in detail in the following sections, the Macuchi deposit located 65km south of La Mina (Figure 5) and the El Domo deposit, 100km south of La Mina. These three deposits represent examples of Kuroko-type mineralization.

According to historic information, from 1940 to 1946, the Mercedes mine, part of the Macuchi deposit, produced 435,000 t. at 11.7 g/t Au, 68 g/t Ag and 4.7% Cu with historical resources of 1.95 MT grading 3.8 g/t Au, 15 g/t Ag and 0.75% Cu. SGS has been unable to verify the information from the Macuchi deposit, and the information is not necessarily indicative of the mineralization on the "La Plata" Property As at La Mina, mineralization at Macuchi is hosted by a volcanic member of the Macuchi Unit. Mineralization occurs as semi-massive lenses of pyrite, chalcopyrite, sphalerite and galena within a strongly hydrothermally altered mafic volcanic rock (sericitization, silicification, pyritization and minor chloritization). No felsic volcanics have been reported at Macuchi which represents the main difference with La Mina.

The El Domo deposit, part of the Curipamba Project is similar to La Mina and Macuchi deposits. The geology of the area is dominated by a submarine mafic volcanic sequence (Macuchi Group, Paleogene–Eocene) with potential for VHMS deposits that is intruded by likely younger subvolcanic calc-alkaline plutonism also having potential for the formation of porphyry and epithermal styles of mineralization. The dominant mineralization consists of a large sub-horizontal lens of massive sulphides at least 800 m long (N-S) and 350–500 m wide that is part of a larger zone of mineralization hat has been partially dismembered by later intrusions. Estimated tonnages and grades are 6.080 Mt at 2.33% Cu, 3.06% Zn, 2.99 g/t Au, 55.81 g/t Ag, and 0.28% Pb for indicated resources; inferred resources are 3.882 Mt with 1.56% Cu, 2.19% Zn, 2.03 g/t Au, 42.92 g/t Ag, and 0.16% Pb. SGS has been unable to verify the information from the El Domo deposit, and the information is not necessarily indicative of the mineralization on the “La Plata” Property. El Domo is the largest VHMS deposit in the Central Andes of Ecuador.

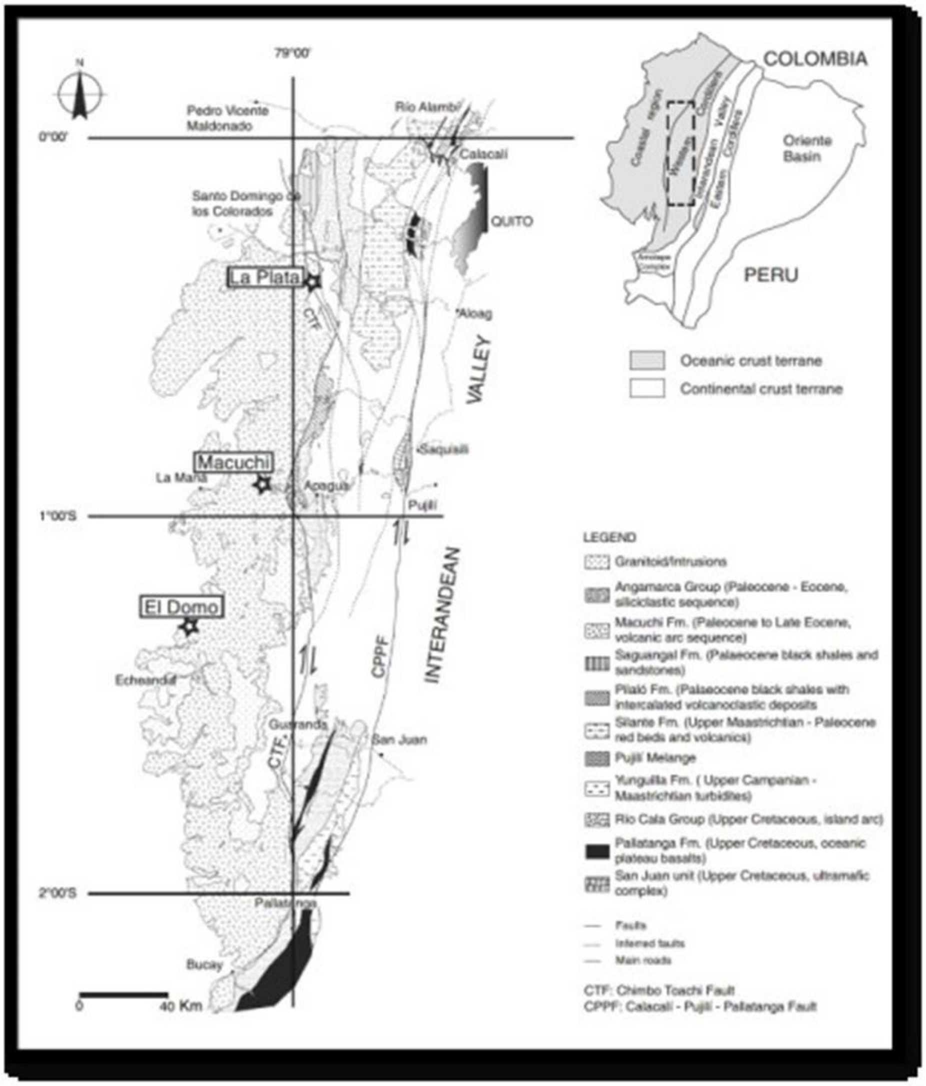

Figure 6. Regional Geology of the La Plata District

7.3 Property Geology, Mineralization and Alteration

1.1.1 7.3.1 Introduction

La Mina is mainly hosted by a sequence of predominantly basaltic to andesitic volcanics with a central felsic unit constituting the Eocene portion of the Macuchi Unit. The basalt-andesite volcanic sequence is in contact with a black shale unit to the west and a polymictic conglomerate to the east (Figure 7). The polymictic conglomerate is a much younger, unconformable unit covering in part the Macuchi Unit, and the black shales represent a conformable sedimentary sequence overlying the volcanic sequence.

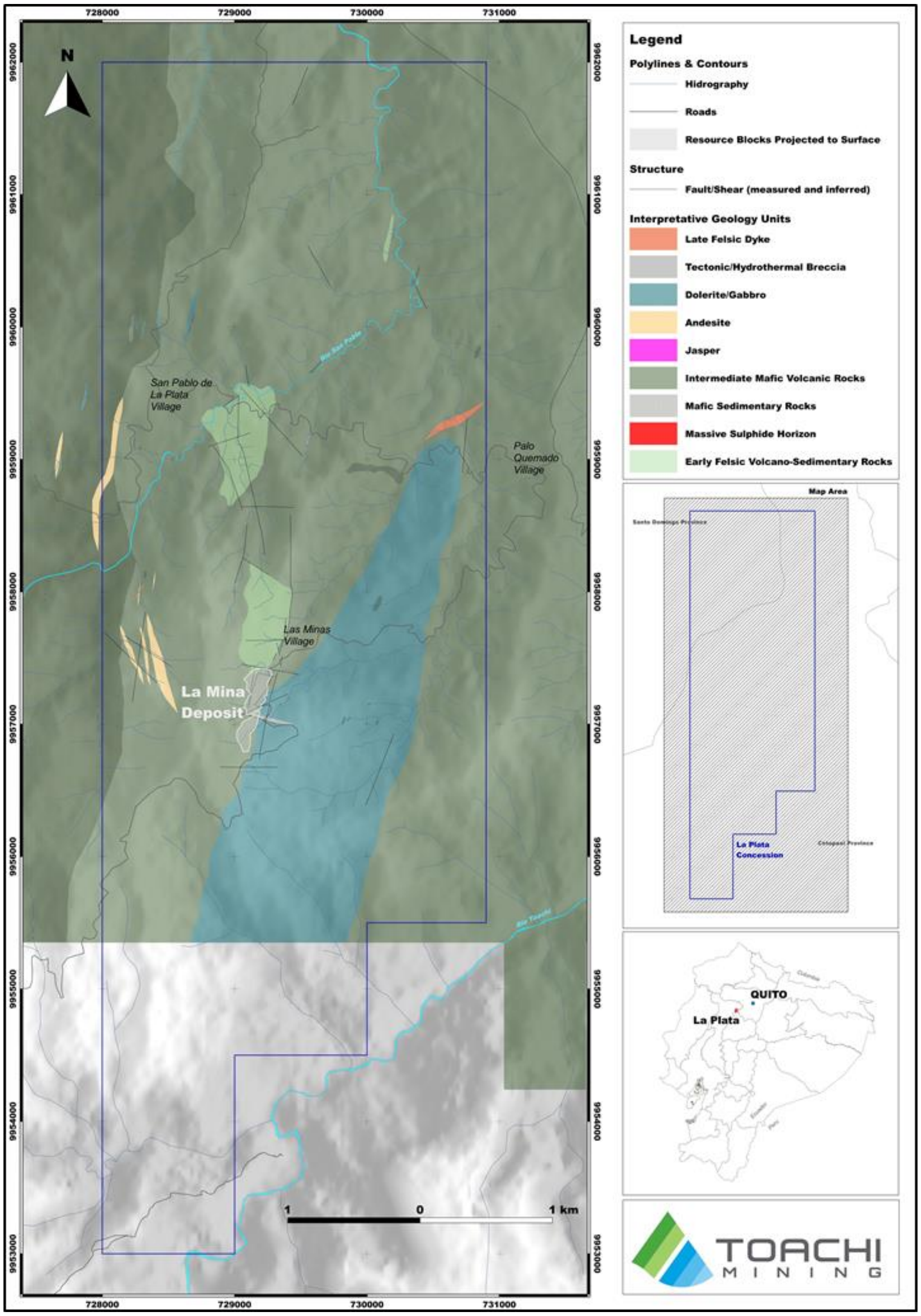

Figure 7. La Plata Project-scale Interpreted Geology

The volcanic and sedimentary sequence in the central and northern part of the La Mina deposit had been intruded by at least two different stages of late dykes, the coarse-grained rhyodacite and the fine to medium grained pyroxene-phyrric gabbro dykes. Toachi had additionally identified the thick mass of a syn- or post-depositional dolerite sill with a typical ophitic texture, intruding the upper parts of the La Mina deposit from the east to west. The stratiform sill is mainly affecting the hanging wall sequence but is also observed to be in a direct contact with the massive sulphide ore. The dolerite is uniformly stretching across the eastern part of the La Mina deposit in the SSW-NNE orientation, dipping at moderate to high angles to the east and is likely following the general geometry of the hanging – foot wall contact of the massive sulphide ore body. This sill has previously been interpreted as the part of the hanging wall volcanic sequence under the name of gabbro.

Apart from the series of later altered and weakly mineralized (in part) post mineralizing and post accretion shears and faults, the hanging wall rocks in general do not appear altered except for the incipient chlorite-hematite-epidote-quartz alteration that occurs in all rocks in the area. (refer to Section 7.3.2 – Structure for more details). At least two stages of epidote were identified in drill core. Hydrothermal alteration within the foot wall intermediate felsic unit is characterized by silicification, sericitization and pyritization. The foot wall of the massive sulphide ore body is characterized with the zones of sporadic but intense replacements of both matrix and clasts by a predominantly copper rich sulphides within the phyllic altered sedimentary breccias and competent volcanic flows. Several millimeter to centimeter massive sulphide veins occur in this zone along with disseminated sulphides.

The bulk of the Toachi drill holes intersected zones of moderate stockwork of pyrite-sericite veining with minor chalcopyrite along with the zones of disseminated sulphides within the foot wall rocks and occasionally even above the massive sulphides lenses. The intersected zone of intense gypsum replacement seems to be developed mainly in the zone of major faulting in the south part of the La Mina deposit North Block within the foot wall sequence. This is interpreted as a late event, related to the extensional, relaxation of the rock-pile. The intensity of the alteration makes it difficult to recognize original compositions, but it seems to be that footwall rocks are more of an intermediate composition in contrast to the previously reported central dacitic classification. It is interesting to note that the brecciated facies are more altered than the more massive ones even though the former are more distal to the center of the hydrothermal system.

7.3.2 Lithostratigraphic Sequence

The lithostratigraphic sequence of the mine area (La Mina Zone) has been summarized from diamond drilling, and surface exposures (Figure 8, modified from Tripodi, 2002)). The sequence below is derived from the recent Toachi's re-interpretation work (2017) including re-logging of Cambior's and Cornerstone's drilling program (1998 and 2006 respectively). The stratigraphic sequence presented below is the most up-to-date one. Not all these facies occurred in each drill holes or on surface exposures.

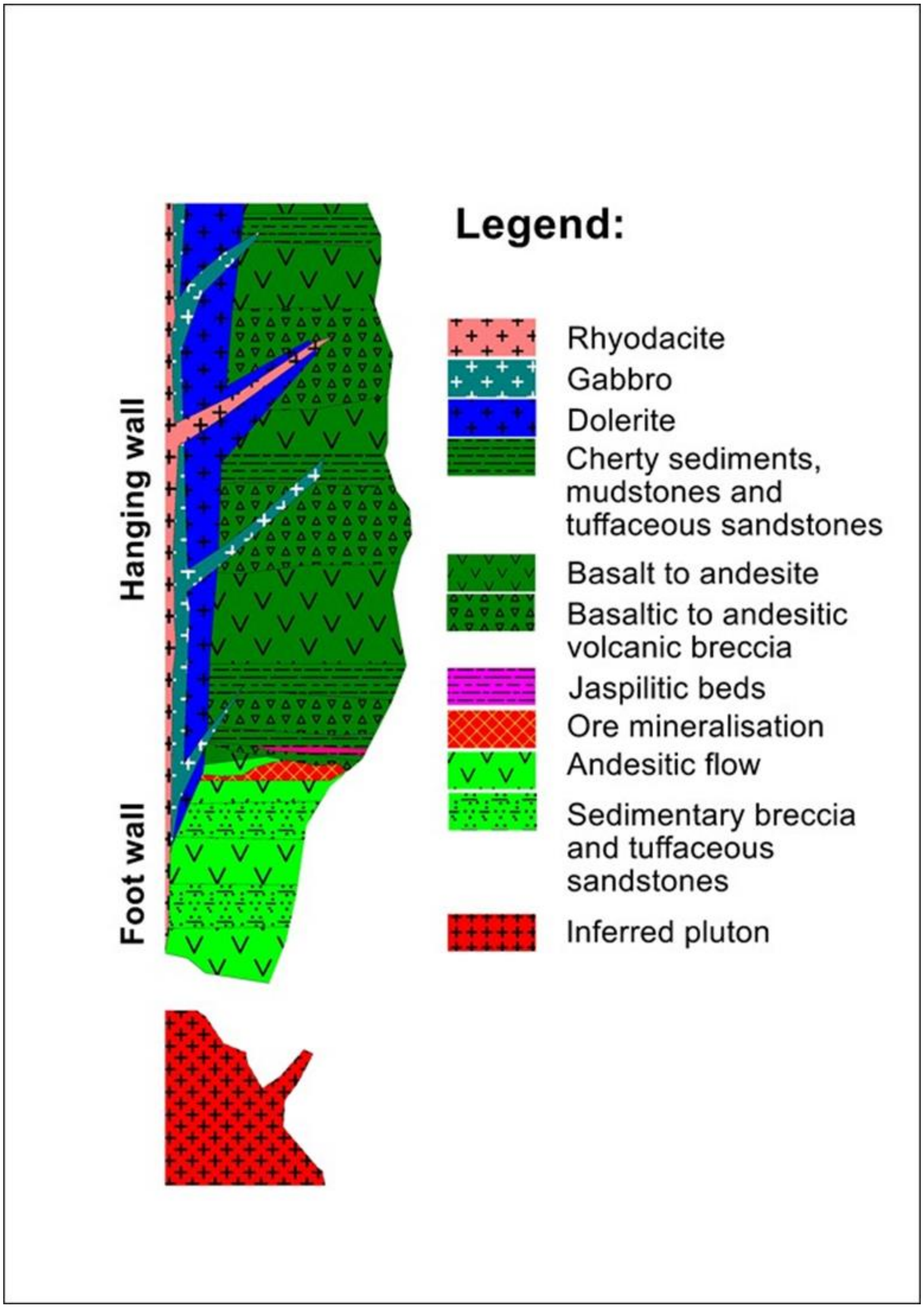

Figure 8. Lithostratigraphic Sequence of the La Plata Project

The sequence is comprised of basaltic, transitional andesite-basaltic and andesitic lavas and associated autoclastic breccias and sedimentary rocks. No pyroclastic rocks were logged or mapped. Sedimentary rocks are comprising repetitive chert-jaspillic beds or minor carbonaceous mudstones (pelagic sedimentation), tuffaceous sandstones and sedimentary breccias. Low grade metamorphism and lack of tectonism have resulted in excellent preservation of primary volcanic textures within the bulk of the rock-pile. However, due to the plasticity of the massive sulphide unit, intense deformation has occurred, and almost all primary textures have been obliterated. The following is a detailed summary of major rock sequences and cross-cutting dyke relationships from stratigraphic top to bottom:

Rhyodacite dyke

- Up to 20 meters thick, occurs on all drill sections from the 9,957,250 northing, northwards.
- Typically, very coarse grained, porphyritic to equigranular, generally fresh or weak propylitic altered with pyrite disseminations (Figure 9 top).
- The rhyodacite dykes are cutting through all the other identified lithologies in the deposit area and therefore are interpreted as the youngest feature.
- The contacts exhibit internal sericite rich chilled margins and are usually sharp and unconformable, unless faulted or within the gouge (when they are most probably using the pre-existing structures as a depositional pathways).

Gabbro dyke

- Usually thin (0.5-3m) but can be up to 10m thick in some intersections
- Mafic dark grey pyroxene-phyrric medium to fine grained porphyritic to massive rock
- Internal chilled margins on contacts may be present with addition of chlorite-epidote (Figure 9, left and right middle)
- Gabbro dykes were observed to be intruded into the hanging wall sequence as flat thin dykes
- Additionally, it was observed in at least one intersection that the gabbro unit was cutting through the mineralized foot wall rocks and stoping out part of the massive sulphide ore.
- These dykes have been interpreted to be older than the rhyodacite dykes but younger than the other interpreted lithologies in the La Mina zone.

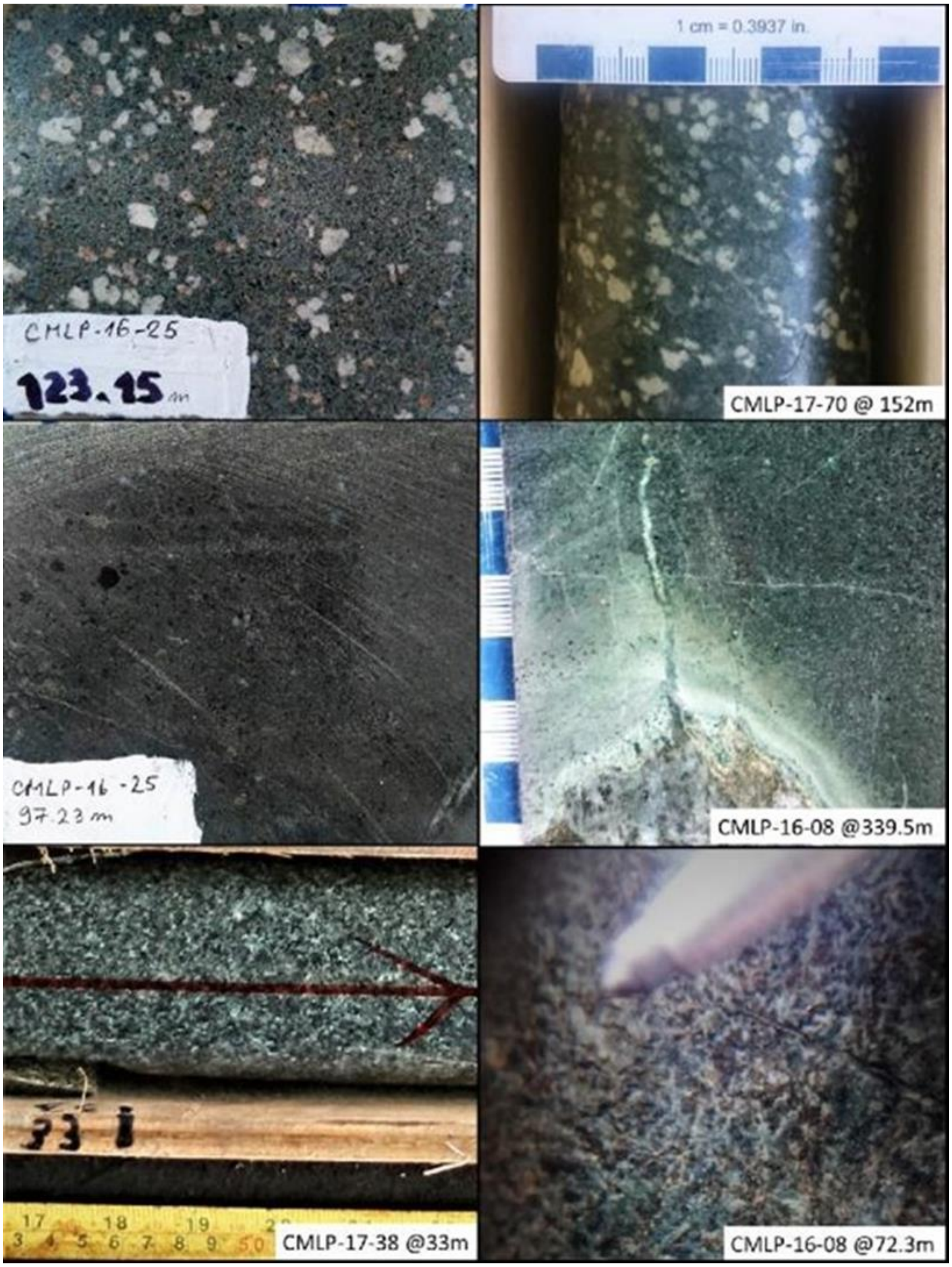

Figure 9. Representative examples of intrusive lithologies. Rhyodacite dyke is shown at the top. The gabbro dyke is shown in the middle with the chilled margin contact shown in the middle right. Dolerite sill with a typical ophitic texture is shown at the bottom. Field of view: 5cm, unless shown against scale or object.

Dolerite sill

- Up to 170 meters thick, with several intersections interpreted to have incorporated the detached blocks of hanging wall sequence as xenoliths
- Typically, equigranular, ophitic, medium to coarse grained and variably magnetic, generally fresh with weak incipient chlorite and epidote (Figure 9, left and right bottom).
- Rock appears semi-conformable and may be either post-mineral or volcanic arc-related or are syn-tectonic.
- The contacts are commonly sheared with the mudstone or basaltic volcanic sequence below or above.
- The sill was observed to be in a direct contact with the massive sulphide ore on at least one intersection.

The latest Toachi's interpretation is that the dolerite sill is older than the rhyodacite and gabbro dykes described above but later then foot wall and hanging wall sequences that it has intruded.

Cherty mudstone

- Up to 10 meters thick, but generally discontinuous and repetitive.
- Dark grey cherty mudstone and siltstone; appears to be of mafic derivation.
- Usually cut by late discrete or tensional quartz veins with or without epidote and sporadically pervasively hematite altered (Figure 10, top left).
- Part of the hanging wall sequence.

Basaltic Volcanic Sequence

- Minimum of 50 m thick, present on nearly all sections in the La Mina mineralized area as the hanging wall volcanic sequence. The rock colour varies from medium to dark green to maroon, feldspar porphyritic and generally and variably vesicular.
- Alteration assemblages characterized by chlorite-epidote-quartz-hematite+/-carbonate.
- Dominated by thick flows and associated autoclastic breccias, pillow lavas are not common but are present locally (Figure 10, top right).

Transitional Basalt-Andesitic Volcanic Sequence

- Up to 50m thick in the hanging wall; occurs in both hanging wall and footwall. Locally it may form the immediate hanging wall at the northernmost La Mina mineralized zones.
- The colour varies from light to medium grey-green, less maroon with alteration assemblages dominated by quartz-epidote and less chlorite-hematite.

- When intersected in the foot wall, it is phyllic altered.
- Typically, very fine grained to feldspar micro-porphyritic; variably vesicular and commonly with flow-foliated amygdales; perlitic cracks are present in vitric sections (Figure 10, middle left).
- Toachi interpreted this sequence as the transitional mafic to intermediate volcanics but as an integral part of the basaltic hanging wall sequence, although the similar rocks have been intersected in the deeper part of the system well below the massive sulphide ore presumably of a different age then those observed in the hanging wall.

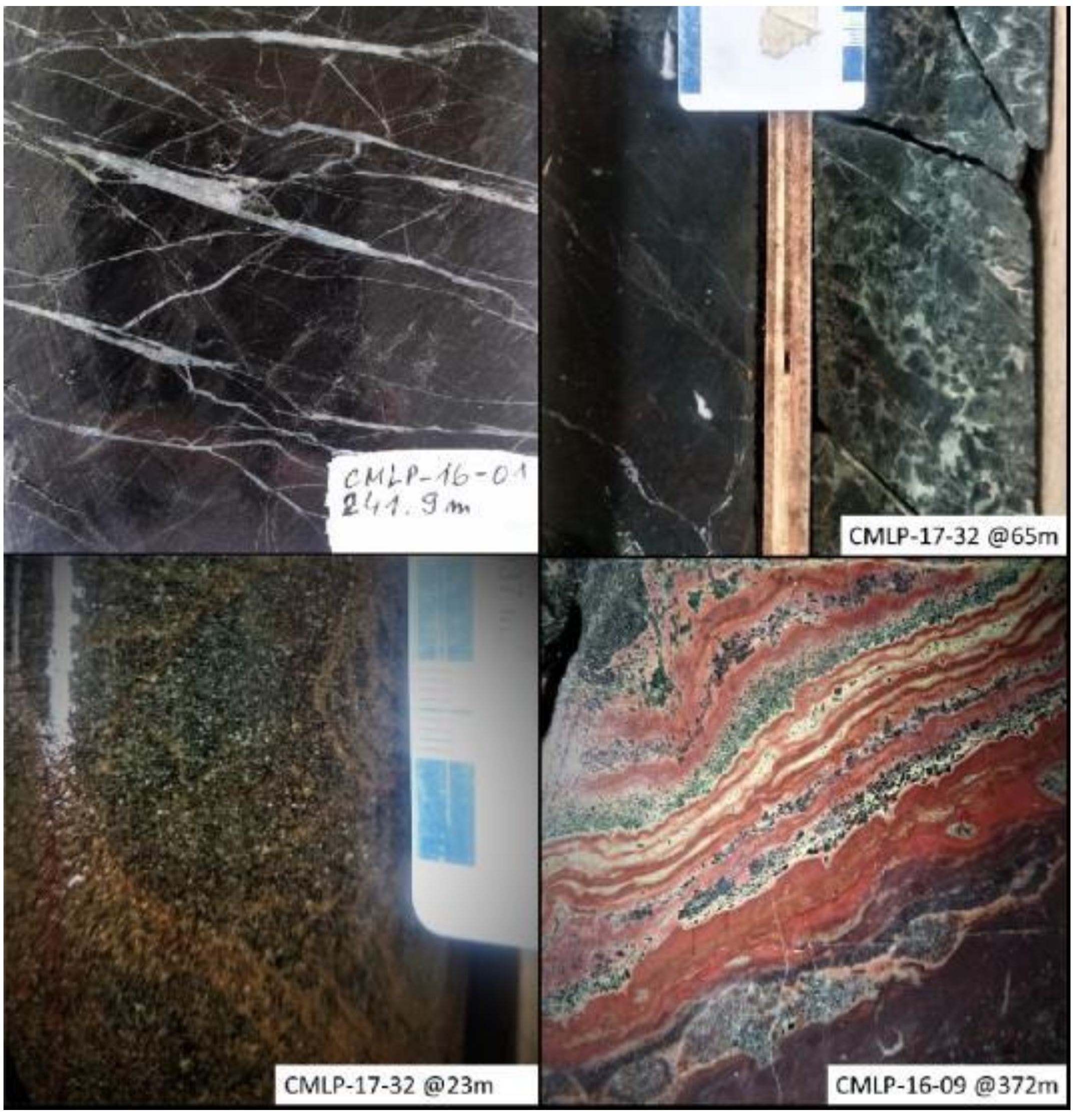

Figure 10. Representative examples of hangingwall volcano-sedimentary sequence lithologies. Cherty mudstone cut by tensional quartz veins (top left, field of view: 5cm), competent vesicular basalt and

mafic autoclastic breccia (top right), basaltic tuff (bottom left), jaspilitic beds / marker horizon (bottom right, field of view: 6cm).

Mafic Volcaniclastic/Sedimentary Sequence *(hanging wall 'marker horizon')*

- Typically, 1-3 m, up to ~ 5m thick; marks immediate hanging wall in most of the massive sulphides intersections when not obscured by late dyke, sill or fault gouge (Figure 10, bottom right).
- Mafic volcaniclastic sequence with variably cherty sedimentary rocks. Internally, comprised of multiple intervals of coarse volcaniclastic deposits dominated by poorly to well-sorted basaltic breccias with cobble to pebble size, angular to sub-angular clasts. Grades upwards to thin bedded sandstone-siltstone of mafic derivation and finely laminated grey, green and maroon chert/cherty mudstone.
- Some cherty intervals are jaspilitic.
- Basal contact is a sharp contact with underlying massive sulphides.
- The margins of the massive sulphide deposit tend to have thicker units of jaspilitic chert when compared with equivalent rocks above thick zones of massive sulphides.

Massive Sulphides

- Massive, typically fine grained, stratabound sulphide-barite lenses up to 10 meters thick (Figure 7.3.2_4, top). Upper contacts are generally knife sharp with the marker horizon mafic volcaniclastic or chert/jasper units (Figure 11, top).
- Mineralization consists of pyrite-chalcopyrite-sphalerite-galena-barite, with accessory bornite±covellite-chalcocite and sulphosalts (Tripodi, 2002).
- Breccia textures in massive sulphides zones occur but are not common; sulphide replacement and cross-cutting textures are more typical (Figure 11, second top). Lower contacts are usually gradational into stockwork zones or disseminated sulphides.
- Within the sulphide sequence, there is evidence of some vertical zonation, but is not well established.
- Gold has been observed in CMLP-17-68 as coarse grains in barite-quartz often associated with veinlets of bornite ± covellite and in CMLP-17-38 in late tensional quartz veins cutting silicified clast wrapped by ductile deformed massive sulphide ore (Figure 11, left and right centre).

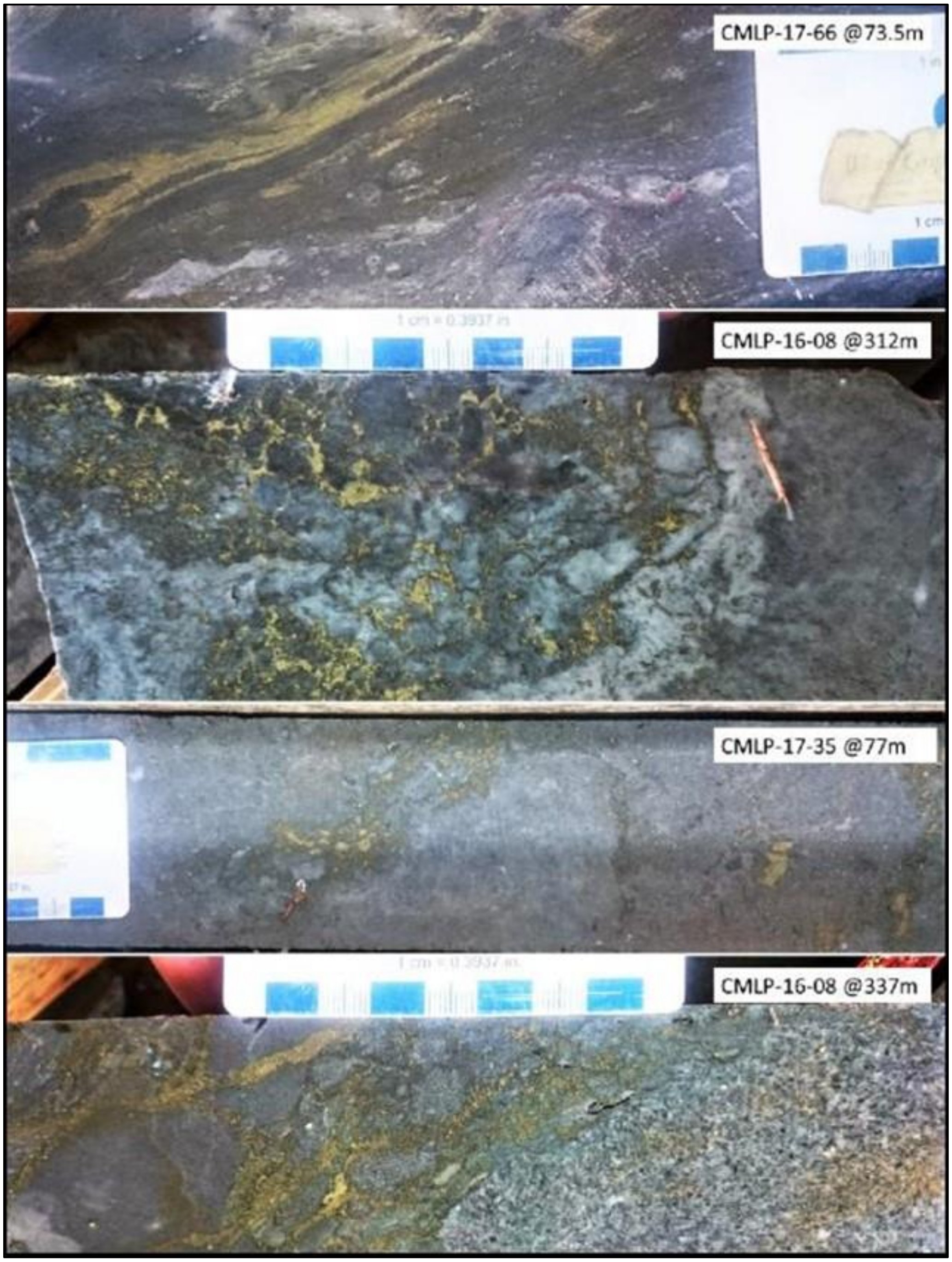

Figure 11. Sulphide Mineralisation at La Mina. From the top to bottom: Deformational layering of the massive sulphide horizon comprising banded fine sphalerite, chalcopyrite, galena and elongated deformed clasts of barite-quartz; Chalcopyrite stringers overprinting phyllic altered, silica flooded foot wall andesite; Chalcopyrite-pyrite replacement of phyllic altered footwall volcano-clastic rock; Chalcopyrite stockwork in the silica flooded brecciated volcanic rock terminating at the lower contact with the felsic tuffaceous sandstone hosting trace disseminated sulphide mineralization.

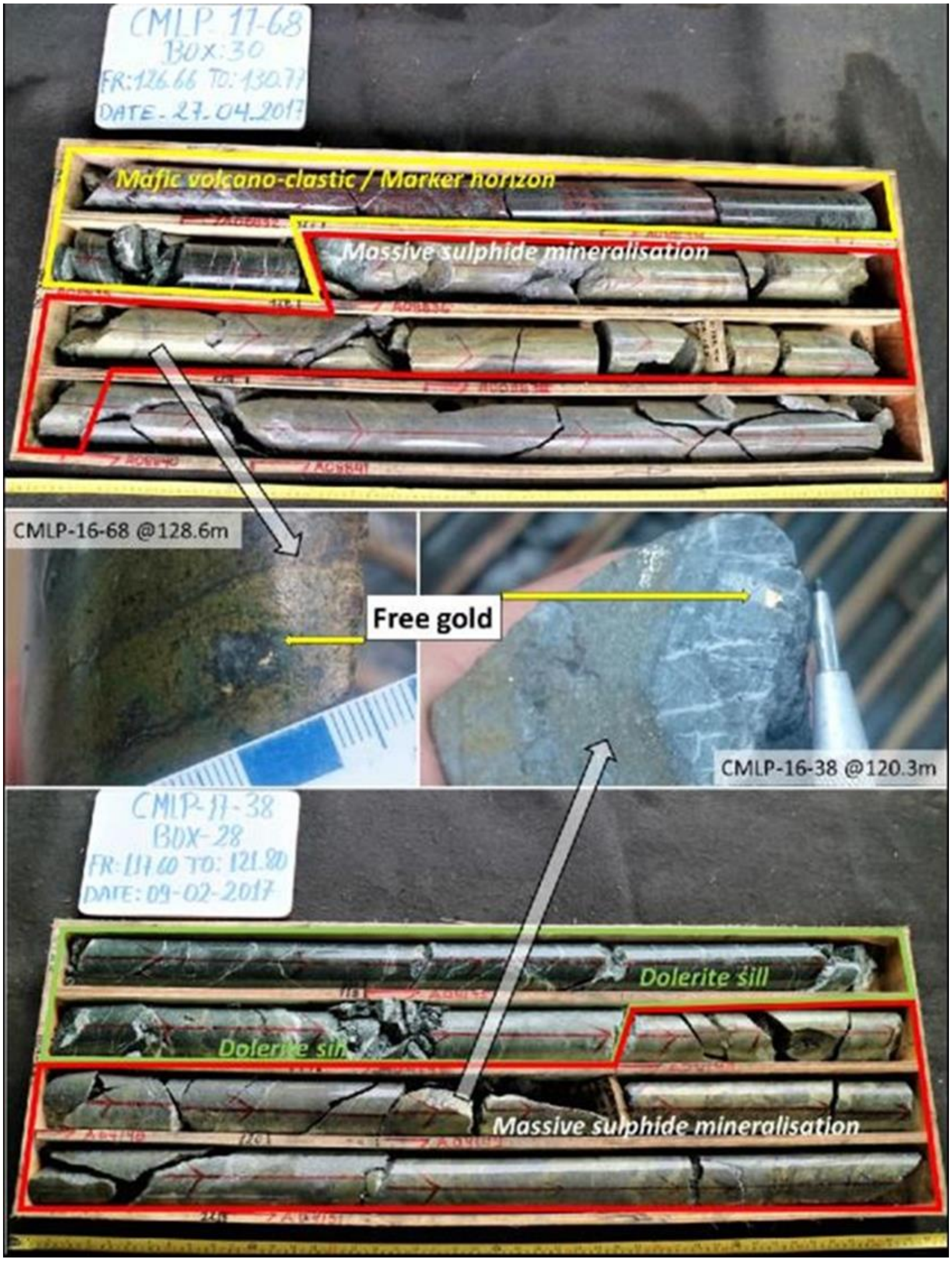

Figure 12. Example of free gold in drill core. Hanging wall mafic volcano-clastic rocks with jaspilitic beds in contact with chalcopyrite rich massive sulphide ore (top). Free coarse gold in barite-quartz (middle left). Free coarse gold in tension quartz veins cutting the silicified clast (middle right). Dolerite sill intrusive contact with massive sulphide mineralisation.

Intermediate Felsic / Andesitic Volcanic Sequence

- These rocks where intersected throughout the foot wall to the post-mineral mafic volcanic and volcaniclastic sequence
- The sequence is repetitive and is interrupted by the series of sedimentary breccias and sandstones.
- Hydrothermal alteration obliterates the protolith, making the identification of these rocks difficult, in terms of whether they are of an intermediate or more acidic composition. Toachi has not identified any dacitic rocks as the part of this sequence as it was stated in the previous reports (Figures 12 and 13).
- Unknown thickness/basal contact on most sections.
- Typically, aphyric to possible weakly feldspar micro-porphyritic. Very commonly perlitic in more vitric and vitri-clastic sections.
- Alteration varies from moderate to complete obliteration of protolith by quartz and sericite with disseminated and stockwork sulphides.
- Varies from massive lavas to autoclastic breccias; dominantly jig saw-fit breccias- in-situ hyaloclastite; granular massive to bedded hyaloclastite to very poorly sorted breccias, locally with massive sulphide clasts (DDH 177-97-11).

Figure 13. Representative examples of footwall volcano-clastic sequence. Phyllic altered competent volcanic rocks (top and middle left). Sericite-quartz-hematite-pyrite breccia (middle right). Partial chalcopyrite-pyrite replacement of the phyllic altered volcano-sedimentary breccia (bottom left). Intensely phyllic altered competent volcanic rock, cut by several stages of quartz-sericite-pyrite and chalcopyrite veins and stringers (bottom right). Field of view: 5cm.

7.3.3 Structure

On a regional scale, La Mina is hosted by a NNE-SSW trending belt of rocks that also contains other occurrences of similar-style deposits, including Macuchi and El Domo.

The volcanic rocks of the Macuchi Unit that host the VHMS mineralisation at La Mina initially underwent a shortening event (E-W) associated with the orogenic event, subduction and accretion of the volcanic pile onto the continent.

Deformation is inferred as being characteristic of thin-skinned tectonics/deformation, which typically involves the development of a shallow, gently-dipping décollement horizon composed of rheologically weak rocks. Thin-skinned tectonics cause rocks in the hangingwall to the décollement to accommodate significant strain by shearing along lithological layering (Davis, 2017).

Dislocation of layering sequences produces structural dislocation, ramping and the development of low-angle faults.

Rocks immediately beneath the basal décollement level, including the crystalline basement, remain undeformed and become shortened elsewhere, i.e., in the more internal parts of the orogen. The mafic dykes seen in outcrop and in drill core were most likely emplaced pre- or syn- this shortening event.

In addition to flatter dislocation structures, a series of reverse faults and shearing were set up throughout the property, with dominant orientations being N-S, NE-SW and NW-SE, all sub-vertical.

The rock-pile then underwent a relaxation, extensional event. These previously set up shear and fault systems were in part re-activated (now with a normal sense of movement). Plutonism probably occurred at this stage (evidence from the late, fresh and unaltered rhyodacite dykes that cross-cut all lithologies and alteration), with hydrothermal fluid introduction utilising these pre-existing pathways (exposed hangingwall rocks at the Tajo Abierto for example), (Davis, 2017).

This plutonic heat engine remobilised some of the existing sulphides associated with the primary VHMS mineralisation, and also introduced secondary pyrite, secondary Au (associated with Ba) and secondary Cu(?).

Interestingly, the dominant movement sense on the major N-S striking fault set was W-side-up, which is the opposite to that expected to produce the fold geometries of mineralised horizons. Thus, supporting the relaxation, extensional event with sulphide remobilisation.

The Quebrada Mala fault that separates the La Mina North and the La Mina South blocks is an example of this change of movement sense and is a very late brittle event.

This series of deformation events has resulted in the more ductile sulphide VHMS horizon being deformed intensely (mylonitic in part), whilst still remaining stratiform. There are no primary depositional features remaining within the VHMS horizon with total textural destruction of all the primary features (Davis, 2017).

7.4 Mineralization

With the exception of preferentially sulphide-replaced clasts, there are no primary sulphide textures at La Mina within the massive sulphide unit.

Well-developed planar layering in the sulphide bodies is tectonic in origin, with total textural destruction of primary features.

La Mina mineralization occurs at the contact between hanging wall basaltic to andesitic volcanics and the footwall andesitic rocks. The massive sulphide zone is usually conformably overlain by a narrow unit of jaspilitic exhalite which acts as a stratigraphic marker horizon.

Mineralization in the La Mina South and North Blocks occurs as massive to semi-massive lenses with thicknesses varying from several decimetres to a maximum of 12 meters, averaging 6 meters, with lateral extension that can reaches 100 meters (Lehne, 1990). The mineralization occurs mainly as bedded layers and, to a lesser extent, as massive bodies with nondescript textures. Within a single sulphide zone, no geometrical zoning and no barite cap have been observed.

The main minerals in the sulphide horizons are chalcopyrite, pyrite, sphalerite, bornite, and galena, with subordinate fahlore (tennantite), covellite, digenite, chalcocite, and native gold. Gangue minerals are barite and accessory amounts of quartz. Figure 14 shows a composite of various styles of mineralization found at the La Mina deposit and, their distribution is presented below by order of abundance (Tripodi, 2002):

Pyrite (FeS_2): Pyrite is ubiquitous in polished sections and the grains are xenomorphic, with the exception of a few hypidiomorphic crystals being replaced by galena and chalcopyrite. The pyrite shows textures that are observed in other minerals (i.e., chalcopyrite, bornite, sphalerite and galena) that are replacing pyrite.

Chalcopyrite ($CuFeS_2$): Chalcopyrite is abundant, often replacing pre-existing pyrite, but also filling spaces between adjacent grains of pyrite, sphalerite, galena and fahlore. The mutual mineral relations illustrate how chalcopyrite cross-cuts bornite and covellite, the latter being therefore a primary phase. Chalcopyrite also occurs as triangular grains up to 15 µm in size replacing sphalerite.

Sphalerite (ZnS): Sphalerite is a common phase, being partially replaced by chalcopyrite and fahlore. Small rounded grains are also frequent and associated with pyrite and chalcopyrite, exhibiting layered textures.

Bornite (Cu_5FeS_4): Bornite occurs in small grains and as large patches, replacing and penetrating pyrite grains. Textural observations, such as crystal penetration, suggest bornite is an older phase than chalcopyrite. Nevertheless, some other textural observations reveal that bornite is also cogenetic with chalcopyrite. Supergene covellite surrounds bornite grains or replaces bornite on fractures.

Galena (PbS): Galena is less common in the polished sections studied. It occurs as large grains, filling open spaces or replacing younger minerals, such as pyrite. Although galena is normally a late-formed mineral in the paragenesis, replacement by chalcopyrite has also been observed.

Tennantite ($(Cu, Ag, Fe, Zn)_{12}As_4S_{13}$ – $(Cu, Fe, Ag, Zn)_{12}Sb_4S_{13}$): This sulphide species is common, occurring as small anhedral grains, often associated with sphalerite and near chalcopyrite grain boundaries. It appears to postdate sphalerite. Electron microprobe analyses have revealed that the grains are close to the As-rich end member and are Zn-rich and can be therefore defined as Zn-rich tennantites.

Chalcocite (Cu_2S) and Digenite ($Cu_{1.8}S$): Chalcocite and digenite were observed within the barite gangue, in association with bornite, probably replacing it.

Covellite (CuS): Covellite occurs in association with bornite and as a fracture-filling mineral. Cross- cutting lamellae of chalcopyrite suggest the existence of primary covellite. Supergene covellite is also observed, and represents the more abundant phase, with respect to primary covellite.

Native Gold (Au): Typically, gold occurs as small xenomorphic grains ranging in size from 5 µm to 50 µm, but a large grain up to 350 µm was observed and has an Au:Ag ratio of about 85:15. The smaller grains often are round and occur within other sulphide phases. The larger occurrences fill open spaces that were available at the time of deposition. Gold occurs within early pyrite, but is also associated with bornite, chalcopyrite, galena, digenite and even as a single grain within barite gangue. Gold appears to be paragenetically related to chalcopyrite, bornite and (Fe-poor) sphalerite.

Two stages of gold are inferred from the current study and by Fox (2017). The first is associated with the primary deposition of massive sulphide mineralisation, and the second is a much younger overprinting phase associated with quartz-pyrite veining. Pervasive ductile deformation of massive sulphides has 'homogenised' grades. However, the second phase of gold mineralisation is associated with discrete zones of quartz veining.

Gangue: Barite ($BaSO_4$) and Quartz (SiO_2): Barite and, to a much lesser extent, quartz, are the main gangue minerals. Barite is very clearly a late phase, occurring as massive euhedral crystals, filling open spaces and fractures in pre-existing phases. Some disseminated crystals occur as microlaths and may be cogenetic to the main ore stage sulphides. Barite was ubiquitous in polished sections studied.

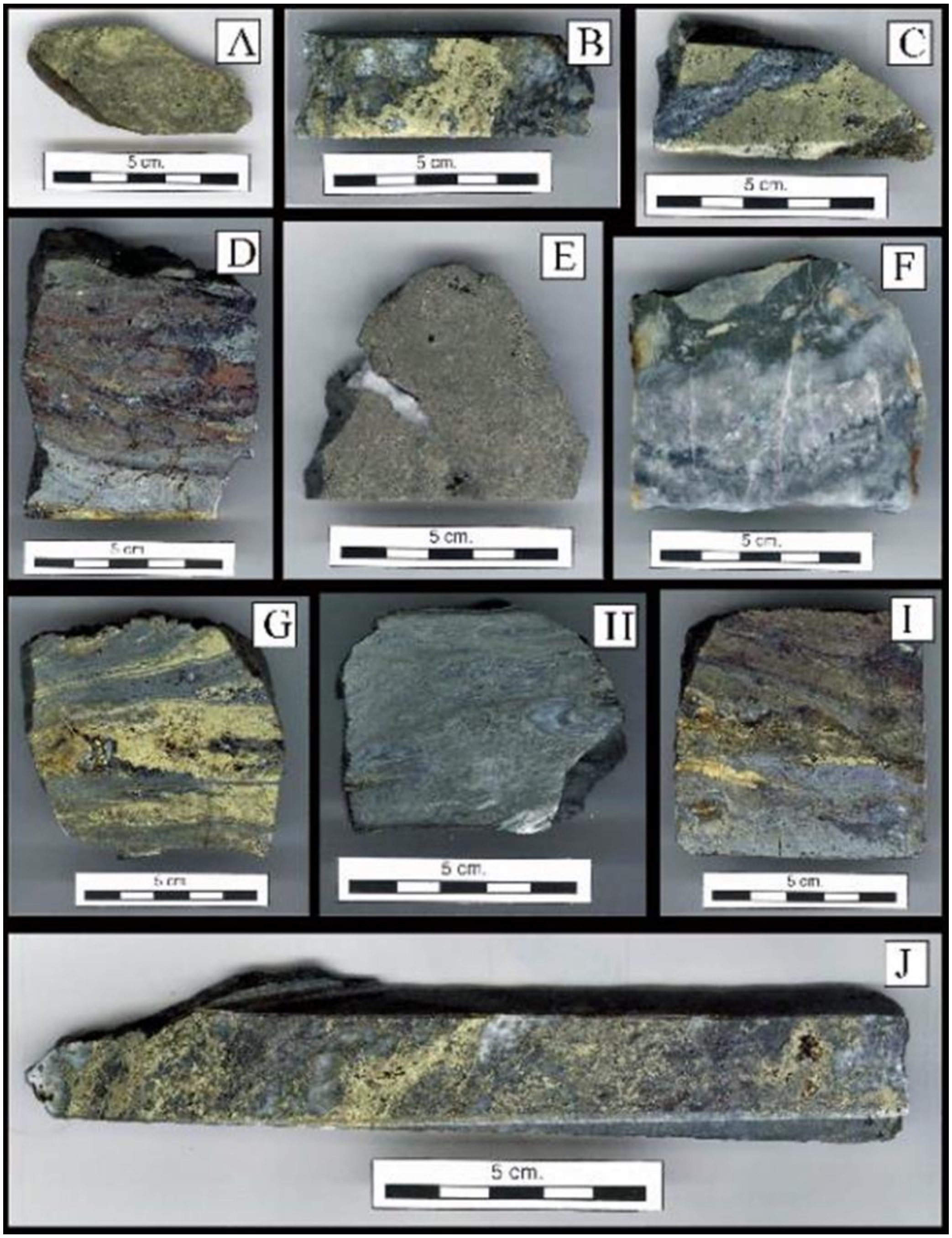

Figure 14. Mineralization at La Mina. Mineralization at La Mina Deposit (Source: Tripodi, 2002). A. massive pyrite and chalcopyrite. B. volcaniclastic with intercalations of massive chalcopyrite. C. barite vein in massive pyrite and chalcopyrite. D. sphalerite, bornite and chalcopyrite layers. E. massive pyrite with massive barite lamellae. F. massive barite. G. sphalerite and chalcopyrite layers. H. curved layers of sphalerite. I. bornite, chalcopyrite, sphalerite, covellite layers. J. chalcopyrite veins and disseminated sphalerite.

7.5 La Mina Deposit Geometry, Deformation Style and Deposit Architecture

The La Mina VHMS deposit comprises of the La Mina South block, and the La Mina North block, both striking in a north-south orientation. The VHMS orebodies essentially dip between 80º and 45º towards the east with an overall strike length of 620m, and an average ‘flattened’ width of ~250m.

The La Mina South block and the La Mina North block are separated by a late northeast-southwest, sub-vertical rotational fault zone, which drops the La Mina South block down by ~120m, producing a ~35º plunging orebody towards the south.

The VHMS orebody itself is parallel to strata and sits on the boundary of the predominately andesitic footwall rocks, and the basalt-andesite hangingwall sequence.

The deposit is cross-cut by reverse faulting and shearing, with some late, re-activated normal faulting. Late felsic dyke and sill intrusives (rhyodacitic in composition) utilise some of these structures, particularly in the La Mina North block and the Guatuza prospect just to the north-east.

Outcrops at La Mina are scarce. However, those exposing primary layering in volcano-sedimentary rocks commonly show brittle and ductile features consistent with contractional deformation. Such features include folds (locally expressed as fault-propagation folds) and faults that have accommodated reverse movements (Davis, 2017).

The style of deformation is inferred as being characteristic of thin-skinned tectonics/deformation, which typically involves the development of a shallow, gently-dipping décollement horizon composed of rheologically weak rocks. Thin-skinned tectonics cause rocks in the hangingwall to the décollement to accommodate significant strain by shearing along lithological layering. Dislocation of layering sequences produces structural dislocation, ramping and the development of low-angle faults (Davis, 2017).

Rocks immediately beneath the basal décollement level, including the crystalline basement, remain undeformed and become shortened elsewhere, i.e. in the more internal parts of the orogen.

Creeks at the El Gordo prospect locally expose well-developed soft sediment structures, including convolute lamination and water escape structures.

These features indicate that the sequence youngs to the east, which is consistent with the interpreted stratigraphic younging of the sequence hosting VHMS mineralisation.

Current cross-sectional interpretations of the geometries of mineralised zones show them to be steeply dipping parallel to host lithologies. Structural ‘roll-overs’ are interpreted that produce open fold closures at the upper terminations of mineralisation, akin to fault propagation folds (Davis, 2017).

Consequently, any fold closures in the deposit are likely to be asymmetric, due either to formation as thrust horses and/or fault propagation folds. Beyond this, any folds would be expected to comprise broad, open flexures, or zones of strain accumulation adjacent to competent bodies (Davis, 2017).

Thrust repetition/dislocation of sulphide bodies during progressive, thin-skinned deformation could result in stacking of mineralised bodies. Ongoing shortening would produce further dislocation and/or open folding of the thrust-repeated mineralisation.

The complex geo-tectonic history of the Macuchi Belt, hosting the La Mina Deposit and at least two more known similar VMS deposits, is reflected onto the micro scale. Deciphering the local structural regime is the key component in defining the present-day morphology of the inferred yet to be discovered high-grade VMS horizons in La Plata property.

The La Mina deposit is hosting at least three known mineralized tectonic blocks separated by a complexly arranged zones of sheared rocks. The two blocks, La Mina North and La Mina South are hosting an inferred resource of high-grade polymetallic ore. The third block immediately to the north is in a Guatuza prospect and it was not included into the resource estimate calculations up to date. All three blocks are affected by a prominent syn-volcanic dextral fault cutting through the district. This structure is parallel to the regional Chimbo-Toachi Fault, the crustal scale suture, bounding the Macuchi belt to the east which is inferred to be situated close to the eastern margin of the La Plata concession.

Prior to and during the field reconnaissance and stream prospecting programs north of the La Mina deposit, the Toachi team considered the evidence that the extensive shear zones mapped across this area may have been re-activated several times throughout their geological history. These shear zone structures might have acted as conduits allowing hydrothermal fluids to recurrently circulate leading to alteration of the adjacent mafic volcano-sedimentary hanging wall sequence and possibly remobilizing the primary sulphide mineralization giving the appearance of a phyllic altered footwall or a VMS stringer zone. These weakened zones of intense faulting and shearing exhibit numerous deformation styles and varying geometries and are inferred to be formed along the susceptible mafic-felsic contact zones due to their different rheologic properties.

The Toachi team has identified these deformed contact zones as an additional robust vectoring tool, that when properly interpreted, will help in the understanding of the relationship between present-day geomorphology and mineralization within the area, and ultimately may lead to the discovery of new VMS horizons.

8.0 Deposit Types

Mineralization at La Mina is defined as a gold-rich polymetallic volcanic-hosted massive sulphide (VHMS) deposit which shows similarities to Noranda / La Ronde / Kuroko type Cu–Pb–Zn (±Au, Ag) massive sulphide deposits. Mineralization generally occurs as massive to semi-massive sulphide lenses within a calc-alkaline bimodal arc succession.

At La Mina, the VHMS mineralized horizons are stratiform and internally highly deformed with no discernible sulphide zonation. There is often a low-grade stockwork immediately below, and sometimes above this horizon, and is a result of secondary shearing and sulphide and metal remobilization. Barite generally occurs within and peripheral to the VHMS and stockwork mineralisation. A laminar chert (siliceous and hematite-rich layer) generally occurs several metres above the VHMS horizon. This can be used as a marker horizon.

Typically, individual deposits are 1 to 2 Mt in size and there is potential for other deposits or other mineralized horizons within the same stratigraphic package. Several similarities of La Mina deposit to the Noranda-Kuroko style mineralization are:

- Occurrence within a bimodal volcanic sequence in a sub-aqueous environment with potential for more prospective stratigraphic levels on the property;
- Lenses of massive sulphide occur at a contact between andesite and dacite which represents a key exploration guide on the property; although the rocks at La Mina are more andesitic and basaltic, rather than dacitic; and,
- Barite as a gangue mineral within sulphide zones and red banded jasper marker horizon assist in determining location in stratigraphy relative to potential mineralization.

One aspect of the La Mina deposit is that it is relatively Au-rich, compared to typical Noranda / La Ronde / Kuroko Cu–Pb–Zn massive sulphide deposits. This may be due to a secondary porphyritic mineralizing event, partially overprinting the deposit. Further exploration and investigation are needed to confirm this hypothesis.

9.0 Exploration

9.1 Introduction

In addition to the mineralization known as the North and South Blocks of the La Mina area, at least 15 other key exploration targets have been identified along a 6-km trend of the La Plata project in favourable stratigraphy. These targets were outlined following two comprehensive exploration programs carried out by Cambior from 1996 to 1998 and Cornerstone-Sultana from 2006 to 2009, and by Toachi Mining from 2016 to 2017.

Cambior completed 28 diamond drill holes totalling 8,628 m and collected a total of 2,944 soil-rock-core samples through an integrated exploration including mapping, trenching and auger drilling, reconnaissance sediment and rock sampling, litho-geochemistry and reprocessing of Geominex geochemistry data. In addition, Cambior completed geophysics including magnetic, electromagnetic, induced polarization and surface and downhole pulse electromagnetic. The exploration efforts by Cambior were successful in delineating mineralization and identifying several targets for further exploration

Cornerstone executed an exploration program on the Project as part of a joint venture with Sultana. The program included prospecting, geophysics, diamond drilling and a systematic whole rock geochemical sampling and bedrock mapping program under the direction of James Franklin, a principal consultant to Cornerstone. Cornerstone contracted Val d'Or Geophisica of Peru to complete a ground geophysical campaign including induced polarization and resistivity, differential GPS and total field magnetic surveys. The survey covered approximately 100 km over the known favourable volcanogenic sulphide system in the central portion of the property identifying broad chargeability anomalies over La Mina, Guatuza, San Pablo, Bella Vista and El Diablo areas.

During 2006 and 2007, Cornerstone completed two diamond drill programs (LP-06-01 to 06-08 and LP-07-09 to 07-17) totalling 5,933 meters. They tested some geophysical and litho-geochemical anomalies as well as the extensions of the mineralization in the North and South Blocks. A transient electromagnetic survey campaign was also carried out by Val d'Or Geophisica from August to December 2006. During the TEM campaign, 10 drill holes and 12 km of grid were surveyed.

Toachi Mining, in a JV agreement with Compania Minera La Plata initiated exploration and resource definition activities in 2016 and is currently continuing exploration activities at La Plata Project. Between 2016 and 2019, 13,747m were drilled, the majority of which were designed to define the La mina deposit, ready for a NI 43-101 resource calculation. Peripheral exploration drilling also took place as has surface geochemical investigation and trenching activity and sampling.

Toachi also completely reviewed all relevant historic data, reinterpreted mapping, soil geochemistry, previous geological interpretations, and reprocessed the existing gravity survey over the La Mina deposit, and the property-wide ground magnetic data collected by Cornerstone. Based on this, Toachi has identified a number of additional targets on the La Mina property.

9.2 Exploration Target Areas Adjacent to Known Mineralization

A good understanding of the La Mina deposit, architecture, alteration and mineralizing controls is well under way. This understanding, through resource drilling and reinterpretation of the existing historic data has identified scope for additional mineral resources within the existing La Mina mine area.

Given the structural setting of the La Mina deposit, the best opportunities for exploration in the immediate area of the known mineralization are outlined as follows (Figure 15).

Eastern flank depth extensions in the LMN and LMS Blocks: Drilling has yet to close off the known mineralization on the eastern flanks of the La Mina deposit, and the potential for additional resources exists.

Deep drilling below the existing resource area: Parallel lens repetitions are common in this style of VHMS deposit. Deeper drilling to date below the Tajo Abierto has confirmed that suites of prospective volcanic rocks and alteration still exist at depth. Geophysical imagery also indicates potential mineralized orebodies at depth.

9.3 Property Exploration

1.1.2 **9.3.1 Introduction**

Based on the current geological understanding of the La Mina deposit, this understanding, along with geophysics, geochemistry and field reconnaissance mapping offer good property scale exploration tools to assist in identification of additional VHMS deposits, and potentially epithermal and porphyry mineralization.

The most interesting prospects that have been outlined from all exploration works are situated in a NNE-trending corridor from the southern extension of the La Mina deposit, through to Bella Vista in the north. There appear to be a second, more westerly corridor, centred on El Diablo and San Pablo North in the north-western part of the property and Aguas Calientes in the south-west as well as the third, more easterly corridor centred on Lucho Lisica and Lisica South prospects (Figure 16).

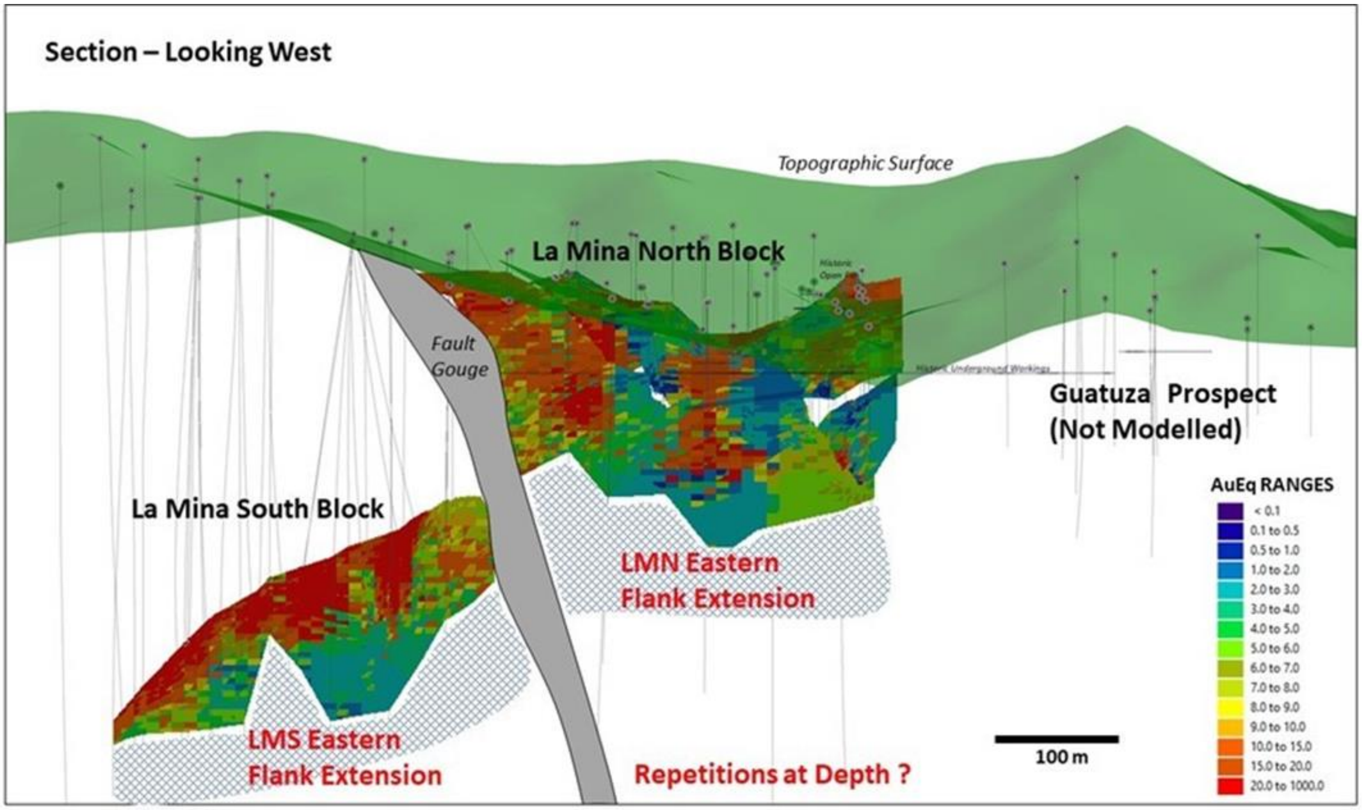

Figure 15. Exploration Areas Close to La Mina Mine

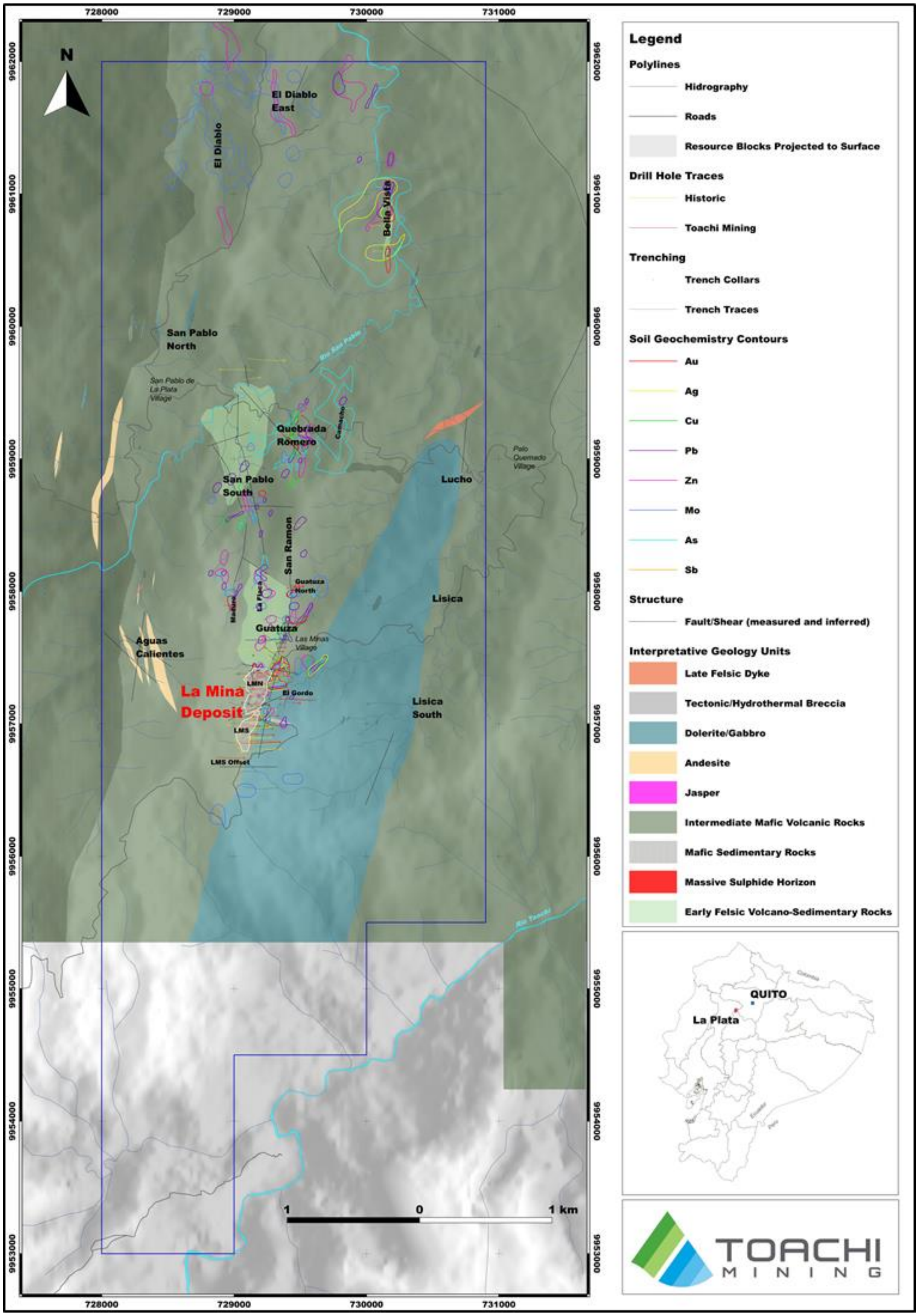

Figure 16. Prospective Exploration Targets

The following sub-sections summarize the highlights of each prospect.

1.1.3 9.3.2 Southern Extension of the South Block (LMS Offset)

- Location: 200 m southwest of the southern extent of the LMS deposit.
- Geophysics: subtle gravity anomaly, consistent with the offset seen between Guatuza and the Tajo Abierto at the northern part of the La Mina deposit.

9.3.3 Guatuza

- Location: 100 to 450m NE of the Tajo Abierto.
- NNE trending westerly dipping dextral strike-slip fault dismembering the high-grade VMS horizon identified in trenches extending for at least 250m along strike.
- Geology: hydrothermally altered volcanic sequence; pyrite stockwork with chalcopyrite veins and VHMS fragments in drilling. Sheared VMS horizon up to 5m thick discovered in trenches.

Geochemical sampling in trenches:

14 m at 1.53 g/t Au, 16.43 g/t Ag, 0.04% Cu and 0.08% Zn;

16 m at 0.99 g/t Au, 19.23 g/t Ag, 0.37% Cu and 0.99% Zn;

15 m at 0.98 g/t Au, 0.03% Cu and 0.18% Zn.

2.5 m at 5.36 g/t Au, 71.18 Ag, 2.14% Cu, 10.86% Zn & 3.22% Pb (new intercept)

2.7 m at 2.36 g/t Au, 45.41 Ag, 0.65% Cu, 8.87% Zn & 1.99% Pb (new intercept)

6 m at 2.02 g/t Au, 32.50 g/t Ag, 2.34 % Cu, 0.79 % Pb, 2.59 % Zn (new intercept)

1m @ 1.08 g/t Au, 44.40 g/t Ag, 0.48 % Cu, 1.92 % Pb, 7.94 % Zn (new intercept)

3.4m @ 1.15 g/t Au, 24.42 g/t Ag, 0.23 % Cu, 1.24 % Pb, 4.39 % Zn (new intercept)

- Soil anomalies: Au, Cu, Zn, Mo, Sb and As.
- Intense Au, Ag, Cu, Pb, Zn, As, Sb and Ba stream sediment anomalies in drainage.
- Geophysics: combined self-potential, mise-à-la-masse, chargeability and gravity anomalies.

9.3.4 Guatuza North Target - San Ramon Prospect

- Location: 650-1200 m NNE of the Tajo Abierto.
- Geochemistry: soil anomalies of Au, Cu, Zn, Mo and As (Guatuza North Target).
- Stream sediment anomalies of Ag, Cu, Zn, Pb, As, Sb and Ba defined over the prominent NNE trending ridge 550m along strike.
  Xenolith rock chip sample assaying 4.5 g/t Au, 51.3 g/t Ag, 16.55% Cu and 5.83% Zn.
- Geophysics: Gravity high and EM conductor defined in Guatuza North Target.

9.3.5 La Flaca

- Location: 300 m W of Guatuza North.
- Geochemistry: soil anomalies of Zn, Mo and As.
- Stream sediment anomalies of Cu, Zn, Pb and Ba.
- Anomalous Au, Zn and Ba in rock chip sample.
- Geophysics: Gravity high.

9.3.6 Maduro

- Location: 250 m W of La Flaca Extended.

  Geochemistry: soil anomalies of Au, Zn, Mo and As.

- Stream sediment anomalies of Cu, Zn, Pb and Ba.
- Anomalous Au, Ba and Pb in rock chip sample.
- Geophysics: Gravity high.

9.3.7 Aguas Calientes

- Location: 650 m W-NW of La Mina.
- Geophysics: Three distinctive parallel gravity highs coinciding with favourable stratigraphy with NNW-SSE trending vertical geometry of the contacts.
- Jaspillitic replacement was seen in the outcropping mafic volcanic rocks.
- Discrete parallel quartz comb veins up to 7cm thick returned anomalous gold values in rock chip samples (0.2ppm – 1.14ppm).
- Gold, copper and barium stream sediment anomalies.

9.3.8 El Gordo

- Location: 250 m E of La Mina North.
- Geochemistry: soil anomalies of Cu, Mo and Zn.
- Geophysics: Gravity high.

9.3.9 Quebrada Romero

- Location: 1.8 km north of Tajo Abierto.
- Geology: Strongly altered volcanics, outcropping disseminated sphalerite mineralization, blocks of massive pyrite in trenches.
- Rock chip samples assaying up to 2.38 g/t Au, 7.6 g/t Ag, 1.72% Cu and 0.84 % Ba.
- Geochemical sampling in trenches:
- 1 m at 0.45 g/t Au and 0.14% Cu;
- 15 m at 0.43 g/t Au, 0.24% Cu and 0.07% Zn;
- 2 m at 0.43 g/t Au, 0.64% Cu, and 0.06% Zn;
- 1 m at 0.47 g/t Au, 0.22% Cu and 0.16% Zn.
- Soil anomalies of Au, Cu, Zn, Mo, Sb and As, 500 m in length.
- Geophysics: I.P. anomaly, 2 EM conductors.

9.3.10 Camacho

- Location: 280 m E of Quebrada Romero.
- Geology: disseminated sphalerite and laminar jasper fragments found in drainage.
- Geochemistry: Large As soil anomaly.
- Geophysics: 2 EM conductors.

9.3.11 San Pablo South

- Location: 950 m NW of Guatuza and 400-500 m WSW of Quebrada Romero.
- Geochemistry: Au, Cu, Zn, Mo and As, soil anomaly – linear, discontinuous 650 m in length.
- Stream sediment anomalies of Cu, Pb, Zn, Sb, Ag, Mo and Ba.
- Rock chip samples assaying up to 0.58 - 1.62 g/t Au, 1.22 % Zn and 0.42 % Cu.
- Geochemical sampling in trench (Quebrada Bermeo):
    - m at 4 g/t Au, 6.47 g/t Ag and 0.31% Cu (new intercept)
- Geophysics: EM conductor.
- Typical footwall phyllic altered rocks with sulphide stringers and disseminations confirmed.

9.3.12 Bella Vista

- Location: 3.5 km NNE of the Tajo Abierto.
- Geology: Intermediate and mafic volcanics, stockwork of pyrite-chalcopyrite-sphalerite.
- Geochemical sampling in trenches:
- 32 m at 0.50 g/t Au, 5.27 g/t Ag, 0.13% Cu, 0.37% Zn;
- 14 m at 0.67 g/t Au, 8.26 g/t Ag, 0.20% Cu, 0.60% Zn;
- 7 m at 0.67 g/t Au, 15.19 g/t Ag, 1.01% Cu, 0.50% Zn;
- 3.7 m at 0.58 g/t Au, 0.13% Cu, 1.52% Zn;
- Sample H-1105 : 6.20 g/t Au, 30.10 g/t Ag, 0.40 % Cu, 0.17% Zn.
- Geochemical Soil anomaly: Au, Cu, Zn and As, 900 m linear anomaly.
- Geophysics: EM conductor, I.P. anomaly, strong linear gravity high.

9.3.13 El Diablo

- Location: 4.3 km north of the Tajo Abierto.
- Geology: veins and disseminations of pyrite, chalcopyrite and sphalerite.
- Geochemical sampling in trenches: 31.03 m at 0.44% Zn.
- Large soil anomaly: Zn and Mo.

9.3.14 El Diablo East

- Location: 650 m east of El Diablo.
- Geochemistry: Soil anomalies of Zn and Mo.
- Favourable stratigraphy confirmed.
- Rock chip sample in faulted basalts assaying 2.19 g/t Au, 7.3 g/t Ag, 538 ppm Ba and 47 ppm Mo
- Geophysics: strong gravity high 280 x 180 m.

9.3.15 San Pablo North

- Location:1 km south of El Diablo.
- Geophysics: strong gravity high 500 x 250 m.
- Anomalous Au, Ba and As in rock chip sample.

9.3.16 Lucho – Lisica - Lisica South

Location: NNE trending corridor 1.2 km east of the La Mina North, Guatuza North and Quebrada Romero. Very strong 2 km long linear gravity high anomaly.

- Favourable lithology and sub-economic Cu-Zn mineralization in outcrops confirmed.

9.4 Geophysics

Geophysical exploration has been responsible for many major discoveries of massive sulphide deposits in mining camp such as Noranda, Matagami, Kidd Creed, Brunswick and Eskay Creek in Canada. These massive sulphide and stringer mineralization systems contain significant quantities of chalcopyrite, pyrite and pyrrhotite which are highly conductive and can be detected using various airborne and ground geophysical techniques such as airborne electromagnetic, ground based horizontal-loop electromagnetic or time domain electromagnetic, resistivity and induced polarization, magnetometer and gravity surveys.

At La Plata, geophysics offer good property scale exploration tools to assist in the identification of new additional deposits, as favourable geology occurs along a 6 km trend north of the La Mina mine area, and also south. Since the 1970's, HLEM, TEM, IP and TFM surveys have been carried out at the Project. In addition to the above geophysical methods, mise-à-la-masse, self-potential and gravity surveys have also been undertaken. From 2006 to 2007, Cornerstone contracted Val d'Or Geophisica to complete a ground geophysical campaign that consisted of induced polarization and resistivity surveys, differential GPS surveying, total field magnetic and surface and down hole transient electromagnetic surveys.

The results obtained for the resistivity and induced polarization surveys were:

- The chargeability results consistently outlined, over more than 5 km of strike.
- The surveys defined five anomalies (IP-1, -2, -3, -4 and IP-5).
- The known massive sulphide lenses were detected from surface to a nominal depth of 300 m below the surface by means of chargeability and resistivity measurements. The geophysical response outlined weak to moderate chargeability values associated with low resistivity values. Anomaly IP-1 corresponds to the historic open pit.
- Three additional IP anomalies were outlined along the favourable corridor hosting the La Mina mineralization, namely anomalies IP-2 (San Pablo prospect), IP-3 (located in Quebrada San Pablo, at approximately 500 m NW of IP-2), and IP-4 (El Diablo prospect). One anomaly, IP-5, was outlined east of the inferred anomalous corridor (Bella Vista prospect).
- This IP survey would benefit from modern re-processing, as the identified anomalies are highlighting large areas of phyllic alteration, associated with post VHMS hydrothermal alteration (associated with post-accretionary structures and plutonism (?)). Reprocessing should narrow the size of the anomalies, and this combined with techniques such as gravity and soil geochemistry will be a good set of exploration tools.

Surface and downhole transient electromagnetic ("TEM") surveys were conducted to test the electromagnetic response over known massive sulphide lenses, and to detect the presence of other conductive lenses. The aim of the downhole TEM survey was to identify the presence of new targets within 100 m from the following drill holes; LP-06-01, 06-02, 06-03, 06-05, 06-07, 06-08, 07-09, 07-10, LP-07-11 and 177-96-09.

The surface TEM survey aimed to test for the presence of the same type of mineralization (semi-massive to massive sulphides) based on surface traverses over the known favourable stratigraphic corridor. The survey can be used to detect electromagnetic conductors from the surface to a nominal depth of 200 m below the surface, in optimum conditions. The results obtained during the surface and downhole transient electromagnetic surveys campaign were:

- The massive sulphide lens intersected in LP-06-01 does not respond to electromagnetic methods. It appears that the high zinc content and pervasive jasper development impedes electromagnetic induction, as commonly observed with zinc-rich VHMS mineralization.
- The only electromagnetic response obtained in the borehole survey is the interpretation of a narrow VHMS lens detected at a depth of 105 m along the profiles of hole LP-06-07.
- The surface surveys did not detect any valid conductors. The only responses were caused by the presence of power lines. The operators whilst conducting this survey experienced multiple equipment failures, which most likely would have also contributed to the poor results.

9.5 Lithogeochemistry and Hydrothermal Alteration

Whole rock data have been interpreted by a number of workers including James Franklin and Lawrence Winter. On the basis of immobile element plots, Winter (2006) defined a sub-alkaline chemical affinity for the basalt, andesite and dacite. The three rock types are characterized by Zr/Ti ratios of 260 (dacite), 140 (andesite) and 60 (basalt).

These lithogeochemistry studies proved to be of little value, as the secondary structure and alteration (and controls), now understood on the property was not taken into account when collecting, analyzing or interpreting the results. In reality, there are no occurrences of dacite at La Plata. Lithogeochemistry studies did not discern the difference between hydrothermally altered andesite and dacite.

To date this survey method has not resulted in any additional VHMS discoveries on the property.

10.0 Drilling

10.1 Introduction

There have been five drilling campaigns completed on the La Plata Property focussed on the La Mina mineralisation. These five drilling campaigns accounted for a total 258 holes, 103 from underground and 155 from surface, for a total length of 36,818 meters (Table 3). The first two campaigns were executed before 1980 and little data is available for review. This work has been discussed in Section 6 - History as the data is not relevant to the current mineral resource estimate. This data is presented in Table 3 for the sake of completeness and is not considered further. For the three last campaigns, all pertaining data including logs, assay sampling and results and, core were available for detailed review.

Table 3. Drilling Campaigns at the La Mina Project

Operator	Year	Type	Holes	Hole Numbers	Length
Duncan-Derry	1965–1967	Winkie Drill	N/A	N/A	Unknown
Minera Toachi	1978–1979	U/G D-type diamond EX	83	D-01 to D-074	2,842 m
				D-85 to D-95	162 m
				D-105 to D-106	52.5 m
		U/G L-type Jack Leg Drill cuttings	20	L-42 to L-220	262 m
		M T Surface diamond BQ diameter	30	MT-01 to MT-30	5,193 m
				Sub-Total	**8,512m**
Cambior	1996–1998	Diamond NQ	28	177-96-01 to 177-96-10	2,041 m
				177-97-11 to 177-97-22	3,619 m
				177-98-23 to 177-98-28	2,966 m
				Sub-Total	8,626m
Cornerstone	2006–2007	Diamond NQ	17	LP-06-01 to LP-06-08	2,518m
				LP-07-09 to LP-07-17	3,415m
				Sub-Total	**5,933m**
Toachi Mining	2016-2017	Diamond NTW-HTW	80	CMLP-16-01 to 27	6,845m
				CMLP-17-28 to 80	6,903m
				Sub-Total	**13,747m**
				OVERALL TOTAL	**36,918m**

10.2 Survey Coordinate System

The coordinate system used for all data collection and surveying on the La Plata property is the Universal Transverse Mercator (UTM) projection, Zone 17 South using the Provisional South American 1956 datum (PSAD56).

10.3 Cambior

Cambior completed 8,628 m of NQ size (47.6 mm diameter) diamond drilling in 28 holes between 1996 and 1998 (Table 3). Most of Cambior's drilling targeted the La Mina South Block and its immediate extensions. Several holes were drilled in the North Block to define known mineralization, reproduce past intersections and verify true widths of the mineralization (177-96-02 and 177-96-03, 177-97-13, 177-97-16, 177-97-17, 177-97-18 and 177-97-19). Two drill holes (177-96-04 and 177-96-10) were drilled directly to the north of the open pit but intersected only distal pyritic mineralization. Cambior tested a few exploration targets to the north of the La Mina deposit along the altered andesitic breccia (177-96-05 to 177-96-08). Drill hole 177-96-05 intersected 6 meters of semi-massive pyrite within a highly fractured and altered intermediate volcanic breccia in the Quebrada Romero target. Holes 177-96-06 and 177-96-07 cut thick sequences of altered intermediate breccia with up to 20% disseminated pyrite in the Bella Vista target. Drill hole 177-96-08 also cut the andesitic breccia; it appears more distal and less altered with disseminated pyrite but yielded some gold values in the order of 1 g/t Au in the San Pablo South target.

Core is stored at Toachi’s core shack facility located in the village of Palo Quemado.

10.4 Cornerstone

In 2006 and 2007, Cornerstone drilled 17 holes totalling 5,933 meters after a review of all existing Cambior drill core and a detailed compilation outlining the most important geophysical and geological anomalies (Table 3). Drill holes LP-06-01, -02, -03, -12 and LP-07-13 were drilled to define the southern end of the South Block while hole LP-06-05 was drilled to extend the north end of the North Block. Holes LP-06-04, -06, -07 and LP-07-14, -15 and -16 were drilled to test the Guatuza zone, 250m northeast of the La Mina deposit. Holes LP-06-08, -09, -10 and -11 were drilled to test the strongly silica and sericite altered andesite at the Quebrada Romero and San Pablo West targets, approximately 2 km north of the La Mina Mine area.

Results of this drilling program and the most spectacular results were from hole LP-06-01 which returned 8.9 m of massive sulphide mineralisation, assaying 31.73 g/t Au, 289.39 g/t Ag, 5.86% Cu, 1.61% Pb and 18.16% Zn and, from hole LP-06-02 that yielded 9.5 m assaying 8.2 g/t Au, 62.5 g/t Ag, 1.65% Cu, 2.32% Pb and 16.99% Zn.

Hole LP-06-06 drilled in the Guatuza zone, 200 meters north of the La Mina deposit, cut two sulphide zones: the upper zone returned 6.84 m assaying 0.99 g/t Au, 9.54 g/t Ag and 6.97% Cu while the lower zone yielded 6.55 m assaying 0.98 g/t Au, 12.06 g/t Ag, 3.59% Cu and 3.19% Zn. This mineralized zone represents a new, near surface massive sulphide lens and is interpreted to be the extension of the North Block mineralization. All Cornerstone’s core is stored at Toachi’s core shack facility in the village of Palo Quemado.

10.5 Toachi Mining Inc

In 2016 and 2017, Toachi drilled 80 drill holes totalling 13,747m. This is the only drilling undertaken on the property on behalf of the issuer. Drill holes CMLP-16-19, CMLP-16--23 and CMLP-16-26 were drilled on the peripheral El Gordo gravity anomaly, just to the east of the la Mina North zone, and drill holes CMLP-16-21, CMLP-16-22, CMLP-16-24 and CMLP-17-46, CMLP-17-48 and CMLP-17--50 were drilled at the Guatuza zone.

The remainder of the drill holes were designed to define the known extents of the La Mina deposit, and to test and expand on this.

Drilling was also designed to verify the surface drill data from the Minera Toachi, Cambior and the Cornerstone campaigns. Drill hole CMLP-16-01 twinned Cornerstone's LP-06-01 drill hole at La mina South. Drill hole CMLP-16-02 twinned Cambior's drill hole 177-97-17, and drill holes CMLP-16-03 and CMLP-16-12 twinned Minera Toachi's drill hole MT-26. Data for the cornerstone and Cambior drill hole twins validated well, but the Minera Toachi drill hole did not.

All Toachi drill holes were surveyed downhole for dip and azimuth deviations, mostly on 30m intervals using the Reflex EZ-TRAC digital surveying tool.

The bottom-of-hole measurements were also taken on as many drill holes as possible using the Reflex ACT II tool.

All Toachi drill collar locations were surveyed by a licensed survey using a differential DGPS, along with as many of the Cambior and Cornerstone drill hole collar locations.

11.0 Sample Preparation, Analyses and Security

11.1 Introduction

The drilling sampling methods of Cambior and Cornerstone have been reviewed, as most of the data and core were available on site. AMEC did an exhaustive review of all drilling sampling methods and the following paragraphs summarize their conclusions. No review of the sampling methods for other sample types including stream sediment, soil, rock and channel samples were completed, either by AMEC or by other parties. No information is available for these samples.

11.2 Duncan-Derry Exploration

No data or core was available for review. TAs no information is available concerning this drill program, the data could not be verified. This data was not used for the 2017 and 2019 updated NI 43-101 resource calculation by Toachi Mining Inc.

11.3 Minera Toachi

Drilling data pertaining to the L series are the result of sampling of jack leg drill hole cuttings. This sampling method is generally used to detect the presence of mineralization instead of quantifying it. It cannot be used in any potential resource estimation as bias of grades (high grading) often occurs using this type of sampling. No assay certificates were available to confirm the assay results in the project database.

Drilling data for the D series are poorly supported due its small hole diameter. In general, assay values resulting from EX diameter drill core can be positively biased in areas of high-grade mineralization and negatively biased in areas of low-grade mineralization. Many of the underground D series holes are missing analyses for gold and silver; these holes are D-15, 17, 18, 22, 30, 35, 37, 38, 45, 47, 48, 105, and 106.

For the M series holes, core could not be reviewed and assay database is incomplete.

Although the Minera Toachi drill hole results have been utilized in the historical 1998 Cambior resource estimate and subsequently by Sultana-AMEC in the various versions of the reports completed by AMEC, none of the Minera Toachi drilling data is deemed to be reliable and as a result cannot be used in a resource estimate without verifying the drill hole database. Missing details of the sample collection, preparation and assay methods, lack of QA/QC program and missing assay results in several holes rend the overall results from this drilling program unreliable.

Therefore, this data was not used for the 2017 and updated 2019 NI 43-101 resource calculation by Toachi Mining Inc.

11.4 Cambior

11.4.1 Introduction

Cambior completed 8,628 m of NQ size diamond drilling in 28 holes between 1996 and 1998 under the supervision of the onsite geologist. The following is a summary of the drill core handling and core cutting procedures

- From the drill site, wooden, numbered boxes containing drill core were transported to Cambior core shack.
- The boxes were then cleaned and core recovery was measured.
- Core was logged and sample intervals were delimited with respect to the geological and mineralization boundaries. The core was sawed in half; sample lengths vary from 0.10 m to 6 m averaging 1.3 m in the sulphide zones.
- Samples were taken both as solid fragments and of the clay-grinded portions along the same side of the core. Approximately 50% of the volume was taken as a sample with the other 50% remaining in the box. Samples tags were pinned in the box at the beginning of each sample interval.
- The samples were placed in plastic bags along with a sample tag for each sample. The sample bags were then placed in large polyethylene bags in groups of 10 which were carefully sealed and sent by pick-up truck to Bondar Clegg's sample preparation laboratory in Quito.
- Then, the samples were dried, crushed, a 2 kg split was crushed to -10 mesh and a 250 grams sample was pulverized to -150 mesh.
- The analyses were completed by Bondar Clegg Laboratories in Vancouver (now ALS Chemex Laboratories). The core was analysed by inductively coupled plasma (''ICP — 34 elements''), initially using an aqua regia digestion; later the method was changed to a total digestion. All of the samples were assayed for gold and silver using a 30 g fire assay method with an atomic absorption finish.

11.4.2 QAQC Data

Cambior's drilling database including collars, logs, sampling results, surveying, RQD, maps and cross sections are all available.

The Cambior drill hole data has some associated QA/QC data that was possible to review. In 2006, AMEC completed a comprehensive review of the assay results and geology data outlining some data errors that were ultimately corrected in the database. AMEC re-assayed, as part of a check assay program, 161 pulp samples representing approximately 10% of all core samples of the Cambior drilling program. These samples were chosen randomly throughout the database, regardless of grade or lithology. AMEC compiled all the results found on the assay certificates for laboratory standard and duplicate data. There were 71 gold and 82 base metal duplicate pairs. The duplicates are laboratory pulp duplicates that were prepared by Bondar Clegg and analysed concurrent with the reported core assay results. The QA/QC analysis for gold show good correlation below 2 g/t Au but two samples with higher results show poor reproducibility. The duplicate pair results for silver, copper, lead, and zinc show good reproducibility.

The results from the check assay study correlate well with the original assay results in the Cambior drill hole database. Below the 1 g/t Au grade level, there were no anomalous results. No bias was also observed: the results from ALS Chemex are neither consistently higher nor lower than the original Bondar Clegg results. Between 1 and 10 g/t Au, the samples from ALS Chemex were consistently higher than the original results from Bondar Clegg. Above 10 g/t Au, the results of four samples from Bondar Clegg were consistently higher than the ALS Chemex results. The results from all grade ranges indicate that for future sampling programs, sample size and assay method should be reviewed in order to increase the precision of the gold assays. The re-assay results for silver, copper, lead, and zinc show good correlation between ALS Chemex and Bondar Clegg.

Toachi have additionally performed checks on the Cambior drill data. A selection of ~ 4% of the pulps and coarse rejects from the Cambior drilling was collected by Toachi for re-analysis. And where Lab Assay Certificates were unavailable, these intervals were also re-assayed.

The statistical analysis of the re-assaying of the representative suite of samples from the Cambior drilling is summarised below in Section 11.6. Acceptable levels of precision and accuracy were noted.

11.5 Cornerstone

11.5.1 Introduction

Cornerstone completed 5,933 m of NQ size diamond drilling in 17 holes between 2006 and 2007 under the supervision of the project manager for Cornerstone. Of note is that, before Cornerstone, no quality assurance work (QA/QC) was done on drill programs. The following is a summary of the drill core handling and core cutting procedures:

- From the drill site, wooden, numbered boxes containing drill core were transported to the Cornerstone core shack.
- The boxes were then cleaned and core recovery was measured.
- Core was logged and sample intervals were delimited with respect to the geological and mineralizes boundaries. The core was cut in half with a core saw; sample lengths vary from 0.25 m to a maximum length of 2.58 m.
- Samples were taken both as solid fragments and of the clay-grinded portions along the same side of the core. 50% of the volume was taken as a sample with the other 50% remaining in the box. Samples tags were pinned in the box at the beginning of each sample interval.
- The samples were placed in plastic bags along with a sample tag for each sample. The sample bags were then placed in large polyethylene bags in groups of 10 which were carefully sealed and sent by pick-up truck to ALS Chemex sample preparation laboratory in Quito.

- Then, the samples were dried, crushed, a 2 kg split was crushed to -10 mesh and a 250 grams sample was pulverized to -150 mesh.
- Assays were performed at ALS Chemex Laboratories in Vancouver. The analysis was done by ICP — 34 elements using the total digestion method. All of the samples were assayed for gold and silver using a 30 g fire assay method with an atomic absorption finish.

11.5.2 QAQC Data

Cornerstone's drilling database including collars, logs, sampling results, surveying, RQD measurements, maps and cross sections are available as a series of Excel and PDF files. AMEC completed a comprehensive review of the assay results and geology data outlining no errors in 1,420 distinct data entries in the database. The Cornerstone drill hole assay data had a limited QA/QC program, which includes insertion of certified material by Cornerstone and by the laboratory. A total of 69 standards were analysed during the program. The standards were supplied by CDN Laboratory of Vancouver ("CDN"), Canada.

In their 2009 report, AMEC did an exhaustive compilation and comparison of the results for the standards inserted by Cornerstone and the duplicate-blank-standard from ALS Chemex. They concluded that assay results of standards and blanks indicate the possibility of sample switches, sample handling issues and, the possibility of low precision in silver assays over 50 g/t Ag. They do not suggest excessive bias with the gold and silver assays for the 2006 and 2007 drill campaign. Copper and zinc standards were not analyzed during the Cornerstone drill campaign.

Core twinning was not performed and neither coarse crushed reject nor pulp duplicates were re-analyzed as part of a QA/QC program. Nevertheless, a reliable indication of precision can be made for the assays from the 2006 and 2007 drill campaign.

Toachi Mining conducted its own QAQC checks on the Cambior drill data. A selection of ~ 4% of the pulps and coarse rejects from the Cornerstone drilling was collected by Toachi for re-analysis. All original assay certificates were available. The statistical analysis of the re-assaying of the representative suite of samples from the Cornerstone drilling is summarised below in Section 11.6. Acceptable levels of precision and accuracy were noted

11.6 Toachi Mining Inc

11.6.1 Introduction

Toachi implemented a concise QA/QC program for all their drill hole assay data. Toachi completed a total of 80 diamond drill holes for a total of 13,747 m between August 2016 and July 2017 under the supervision of the project manager for Toachi. The drill holes were drilled using diamond drill technique, with 12,557m

HTW diameter core (71.0mm), with 1,137m of NTW (56.2mm) and 53m of BTW (42.1mm) diameter. Toachi implemented a strict QA/QC regiment related to all aspect of core handling, sampling and logging.

11.6.2 Toachi Mining Inc standard procedures – drill core handling, core cutting and sampling

The following is a summary of the drill core handling and core cutting procedures:

- Upon arrival at the drill machine, both Toachi’s field staff and the driller signed off on the completed core boxes before they were transported.
- From the drill site, wooden, numbered boxes containing drill core were transported to the Toachi core shack.
- The boxes were then cleaned, labelled correctly, metre marks added, and core recovery was measured.
- Geotechnical measurements were then conducted; Recovery, RQD, orientation marks, core diameter and hardness were recorded.
- Technicians then oriented the drill core based on the orientation marks provided by the drilling company. Drawing a centreline, representing bottom of hole. No orientation = full red line, Low confidence = dashed black line and high confidence = full black line.
- Geological logging and structural measurements were taken.
- Sample intervals are marked. Samples are selected based on logged geological features, such as rock type, mineralization, alteration, veining etc. Sample length does not exceed 1.2 m nor is smaller than 20 cm. In areas of similar geological characteristics, sample length is, in general, 1 m.
- Each sample interval is assigned a sample number. This is marked on the core box with a red marker, recorded in the Company’s sample book, and a ticket allocated to a marked calico bag (which also contains sample details). Blanks, standards and duplicates are also added in at this time.
- The core boxes are then photographed, with details of each core box clearly visible. Photographs are taken dry and wet.
- Marked sample intervals are then cut. The core is cut and sampled, always sampling the right-hand side of the drill core. Half core is sampled, with the exception of field duplicates, which is quarter core.
- Samples are then bagged in calico bags, and blanks and standards inserted at this time. They are bundled into larger, sealed poly-weave bags, ready for submission to the laboratory.
- The samples are either driven by Toachi staff, or shipped to the prep labs via a contracted 3 tonne truck.
- After sampling, geologists relog each sample interval, and add to the geological logging table. And all data is entered into the validated Excel geological database.

11.6.3 Toachi Mining Inc standard procedures – sample dispatch and assaying

The following procedures summarise sample despatch and assaying:

- Samples are sent either to ALS Chemex sample preparation laboratory in Quito, or to LAC y Asociados sample preparation laboratory in Cuenca.
- For both labs, samples are dried, entire sample crushed to 2mm, then a split of 250g is taken, and pulverized to 85% -75 microns.
- For assays performed at ALS Chemex Laboratories in Lima, Peru. The analysis was done by ICP — 33 elements, 4 acid digest ICP-AES. Au was performed using a 50g fire assay with AAS finish. Over-limit samples were assayed using a 4 acid digest with an ICPAES finish. All samples within the ore zones were also assayed for total S% by LECO.
- For assays performed at MSA Laboratories in Vancouver, Canada. The analysis was done by ICP — 33 elements, 4 acid digest ICP-AES. Au was performed using a 50g fire assay with AAS finish. Over-limit samples were assayed using a 4 acid digest with an ICPAES finish. All samples within the ore zones were also assayed for total S% by LECO.

11.6.4 Toachi Mining Inc standard procedures – density measurements

Bulk density measurements were completed in-house on all available assayed sample intervals within the mineralized zones returned from drilling at La Plata (for the Toachi Mining, Cambior and Cornerstone drilling programs). Two procedures were used as follows, depending on the porosity of the sample:

- Water displacement method 1 (non-porous samples):
  - This method is used to determine the dry bulk density of reasonably competent rock samples that do not have any vughs or other large voids. The sample is assumed to displace a volume of water equal to its own bulk volume minus the volume of water absorbed into sample.
  - The samples are sun-dried.
  - Weigh the sample to determine the dry mass using electronic scales (Figure 17).
  - Weigh the dry measuring vessel.
  - Fill the measuring vessel with water and weigh using electronic scales.
  - Slowly submerge the sample in water-filled vessel and weigh (Figure 18)
  - Measure the water displaced by the sample.
  - Retrieve the sample from the water vessel and weigh it, to determine the mass of the sample plus absorbed water.
  - Determine the volume of the sample.
  - The Bulk Density of sample is calculated: mass of sample divided by volume of sample.
  - Each sample is measured a minimum of three times, then averaged to obtain a final value.
  - The data is compiled in a spreadsheet.

- Water displacement method 1 (porous samples):
    - For highly weathered and clay rich samples water displacement method 1 is not suitable because samples tend to slake or disintegrate when immersed in water. To overcome this problem samples are coated with wax after drying. This method is summarised as follows:
    - Sun dry the sample.
    - Weigh the sample to determine the dry mass using electronic scales.
    - Determine the density of wax.
    - Completely seal the sample with hot wax
    - Allow the coated sample to cool to room temperature, then weigh.
    - Fill the measuring vessel with water and weigh.
    - Slowly submerge the sample in water-filled vessel and weigh.
    - Measure the water displaced by the sample.
    - The Bulk Density is calculated as the mass of sample divided by volume of the sample, after accounting for the volume of wax.
    - Each sample is measured a minimum of three times, then averaged to obtain a final value.
    - The data is compiled in a spreadsheet.

Check bulk density measurements were also carried out by submitting a representative suite of samples to ALS Chemex Laboratories in Lima, Peru.

Figure 17. Density measurement electronic scales and measuring vessel

Figure 18. Water immersion density measuring procedure

11.6.5 Toachi Mining Inc standard procedures – standards and duplicates

A total of 10% of the samples submitted are certified (and uncertified) blanks and certified standards and field duplicates with, as a minimum, one blank submitted at the beginning of each sample batch. Certified standards are submitted at an average of 6% of the samples submitted. Field duplicates are taken at a rate of 1 in 20 of the samples taken.

Blanks consisted of certified blanks from CDN Laboratory of Vancouver ("CDN"), Canada, barren river sand, and barren limestone. The standards were also supplied by CDN Laboratory. 9 different standard sets were used, which covered a range of suitable multi-element values for the deposit type.

Field duplicates were prepared by quartering the drill core, and submitting both halves to the laboratory.

Both labs (ALS and MSA) also completed their own internal checks. This data, along with the Toachi Mining checks was validated using QCAssure software.

Toachi Mining also conducted additional QA/QC checks on all their data by submitting approximately 7% of their drilling database to an independent laboratory. Samples selected covered the top 200 assay values, then a mix of different assay values throughout each original lab job returned. Refer to Section 12 for details.

11.6.6 QAQC Data Analysis

A suite of nine CRMs has been routinely submitted to both the MSA and ALS laboratories as part of the standard sample sequence. Table 4 summarises the expected values for the five-element CRMs that have been specially prepared from volcanogenic massive sulphide material by CDN Resource Laboratories Ltd of Langley, British Columbia (CDN). The mean grade, the two-standard deviation value and the percentage range of the upper and lower limit for each standard were used in statistical analysis of the CRMs.

Table 4. Summary of certified values of CDN Resource Laboratories reference standards

Standard	Au g/t	Ag g/t	Cu %	Pb %	Zn %	Approximate S%
CDN-FCM-2	1.37	73.9	0.756	0.479	1.739	35
CDN-FCM-3	0.40	23.6	0.291	0.152	0.543	
CDN-HLHC	1.97	111.0	5.07	0.17	2.35	26
CDN-HLLC	0.83	65.1	1.49	0.29	3.01	33
CDN-ME-1204	0.975	58.0	0.519	0.443	2.36	20.1
CDN-ME-1402	13.90	131.0	2.90	2.48	15.23	12.5

CDN-ME-1405	1.295	88.8	0.685	0638	3.02	29.0
CDN-ME-1410	0.542	69.0	3.80	0.248	3.682	27.2
CDN-ME-1414	0.284	18.2	0.219	0.102	0.732	7.0

The CDN CRM data has been assessed, grouped by the primary assay laboratory of either MSA or ALS. Table 5 summarises the mean percentage bias data and the percentage of standards within the certified tolerance levels for the analyses carried out by MSA laboratories. Acceptable and low levels of bias from the certified values have been returned along with extremely high percentages within the certified tolerance levels. In summary, MSA slightly under-calls the certified values.

Table 5. CDN Resource Laboratories reference standards: MSA performance

Standard	**No.**	**Au g/t**		**Ag g/t**		**Cu %**		**Pb %**		**Zn %**	
		% Bias	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**
CDN-FCM-2	10	-3.43	100	-3.76	100	-0.39	100	-3.21	100	-0.91	100
CDN-FCM-3	8	-3.66	100	-1.43	100	0.96	100	-0.70	100	-1.80	100
CDN-HLHC	6	-0.41	100	-0.30	100	-1.80	100	-4.28	100	-1.31	100
CDN-HLLC	15	-1.93	93.75	-0.08	100	-0.77	100	-4.85	100	-1.71	100
CDN-ME-1204	62	-0.16	100	1.70	100	1.53	96.8	-1.74	98.4	-1.70	100
CDN-ME-1402	65	-0.66	100	-1.32	100	-2.27	98.5	-0.92	98.5	-0.18	100

CDN-ME-1405	61	0.25	100	-1.08	100	0.58	98.4	-4.26	100	-0.09	100
CDN-ME-1410	61	1.00	100	-0.76	100	0.12	100	-3.23	87.1	-0.03	95.2
CDN-ME-1414	68	-0.60	100	1.19	100	-0.12	100	-1.42	100	-1.39	92.7

Table 6 summarises the same data for the analyses carried out by ALS laboratories. ALS returned higher ranges of relative bias compared to MSA along relatively lower percentages within the certified tolerance levels compared to MSA. Acceptable levels of accuracy were returned for the CDN-FCM-2 and CDN-FCM-3 standards but the CDN-HLHC and CDN-HLLC standards proved more problematic for ALS. In all cases an under-call was noted. As such, the assay results from ALS might be considered slightly conservative.

Table 6. CDN Resource Laboratories reference standards: ALS performance

Standard	No.	Au g/t		Ag g/t		Cu %		Pb %		Zn %	
		% Bias	% Pass	% Bias	% Pass	% Bias	% Pass	% Bias	% Pass	% Bias	% Pass
CDN-FCM-2	40	-2.24	97.5	-2.05	100	-5.97	50	-3.94	95	-0.26	100
CDN-FCM-3	40	-0.05	97.5	-1.45	100	-1.63	95	-3.43	100	-2.48	90
CDN-HLHC	40	-5.97	83.3	-4.24	87.8	-4.69	66.7	-8.30	19.1	-0.52	81
CDN-HLLC	23	-3.43	87.0	-0.43	85.7	-8.32	0.0	-7.73	87.0	-1.52	95.7

The same standards were routinely submitted within the samples sent to SGS Bor for referee analysis of the samples originally analysed by ALS and MSA. Table 7 summarises the performance of SGS Bor on the inserted CDN CRMs. SGS Bor had issues with CRM CDN-ME-1204 for Au, Pb and Zn. Otherwise acceptable levels of relative bias were returned for the other CDN CRMS.

Table 7. CDN Resource Laboratories reference standards: SGS Bor performance											
Standard	**No.**	**Au g/t**		**Ag g/t**		**Cu %**		**Pb %**		**Zn %**	
		% Bias	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**
CDN-ME-1204	8	-7.87	87.5	2.13	100	-6.23	25	-25.2	0	-12.8	0
CDN-ME-1402	8	0.81	100	0.76	100	-0.34	100	-0.20	100	-0.22	100
CDN-ME-1405	7	-2.42	85.7	1.77	100	0.58	100	1.07	100	-0.33	85.7
CDN-ME-1410	7	-0.18	100	-0.21	100	0.94	100	0.95	57.1	-0.40	85.7
CDN-ME-1414	5	1.48	100	0.26	100	2.99	60	1.33	100	-0.25	100

A suite of representative pulps from the Cornerstone and Cambior drilling were sent to MSA for referee analysis (see further discussion below for the comparative analytical assessment). The same suite of independent reference standards was included in the sample sequence for analysis by MSA. Table 8 summarises the performance of MSA on this set of CDN CRMs. Acceptable levels of relative bias were returned for the all the CDN CRMS.

Table 8. CDN Resource Laboratories reference standards: MSA performance during referee analysis of Cornerstone and Cambior sample pulps											
Standard	**No.**	**Au g/t**		**Ag g/t**		**Cu %**		**Pb %**		**Zn %**	
		% Bias	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**	**% Bias**	**% Pass**

CDN-ME-1204	4	0.10	100	1.59	100	-0.05	100	-1.60	100	-0.23	100
CDN-ME-1402	3	1.92	100	-3.31	100	-1.77	100	-2.38	100	0.32	100
CDN-ME-1405	3	1.62	100	-3.49	100	-1.63	100	-5.28	100	0.54	100
CDN-ME-1410	3	2.83	100	2.08	100	-0.52	100	-3.05	100	0.07	100
CDN-ME-1414	3	1.64	100	0.92	100	-1.58	100	-0.70	100	-0.28	100

Comparative referee analysis of Cornerstone and Cambior sample pulps by MSA was carried out for a set of 248 sample intervals. Table 9 summarises the relative performance statistics once low level assays are omitted (which can often return step-like, order of magnitude assays differences, for example 0.01 and 0.02). Inclusion of the samples at or near the detection limit for each element can negatively distort the relative analytical comparative statistics. It is evident that excellent comparative statistics were returned for all five elements, with similar means, normal levels of precision (Half Absolute Relative Difference or HARD) and bias (Half Real Difference or HRD) and very high correlation statistics. As such, referee analysis of this representative selection of pulps from the Cambior and Cornerstone drilling has confirmed both the tenor and distribution of grades for the five elements over their grade ranges. The Cambior and Cornerstone drilling data can thus be used in conjunction with the Toachi-managed drilling in resource estimation. Figure 19 displays an example of the statistical output from QAQC analysis of the MSA referee analysis of the Cambior and Cornerstone sample pulps.

Similarly, a representative suite of 583 samples from the Toachi-managed drilling, which was analyses by MSA and ALS has been sent to SGS Bor for referee analysis, along with CDN CRMs and blanks. Table 10 summarises the results of the comparative analyses, once very low-grade assays are omitted in order to prevent statistical distortion due to analyses close to or at the detection limits. Very similar means, coefficients of variation (CV) were returned, along with acceptable levels of precision (low HARD values) and low levels of bias (low HRD Values). Very high levels of correlation between the two data sets were also returned. In summary the comparative and referee analysis, along with the inserted CRMs has demonstrated that the Cornerstone, Cambior and Toachi drilling data sets can be combined for resource estimation purposes.

Summary
(Zn >0.05)

	Orig-Zn-pcnt	MSA-Zn-pcnt	Units			Result
No. Pairs:	147	147			Pearson CC:	0.99
Minimum:	0.05	0.05	g/t		Spearman CC:	0.98
Maximum:	51.23	51.50	g/t		Mean HARD:	7.31
Mean:	4.99	5.01	g/t		Median HARD:	3.92
Median	1.56	1.76	g/t			
Std. Deviation:	7.98	7.85	g/t		Mean HRD:	-1.99
Coefficient of Variation:	1.60	1.57			Median HRD	-1.43

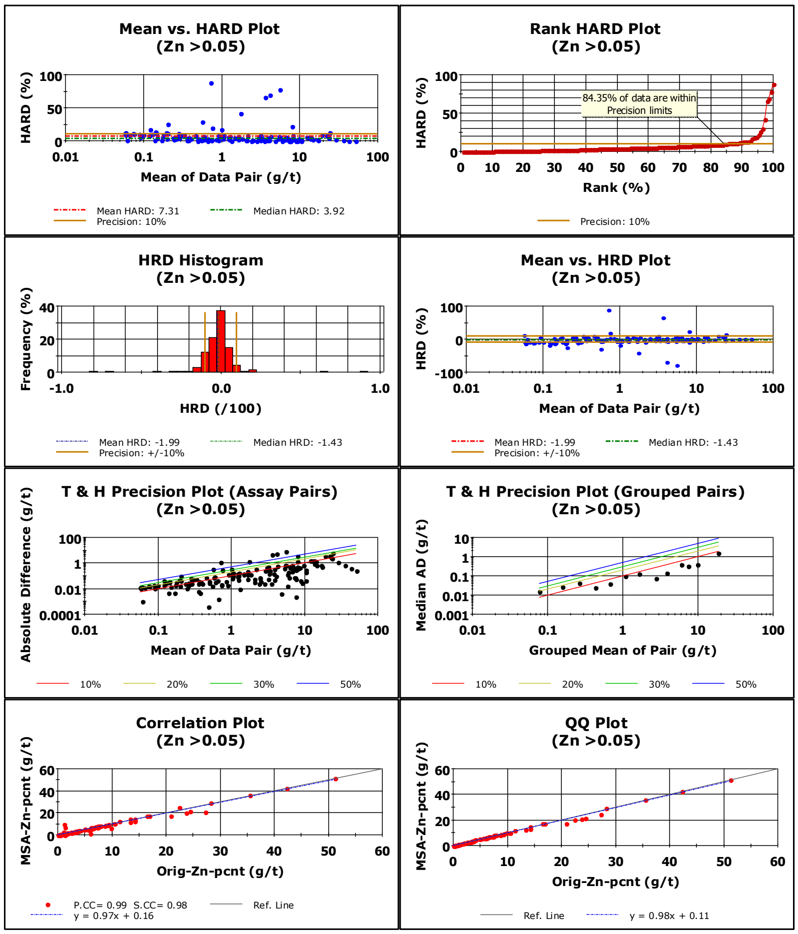

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Figure 19. QAQC analysis of MSA referee assaying of Cambior and Cornerstone pulps: Zn%

Table 9. MSA referee analysis of Cornerstone and Cambior sample pulps										
Element	**Limit**	**No.**	**Original Data**		**MSA**		**Mean HARD (%)**	**Mean HRD (%)**	**Correlation Coefficients**	
			Mean	**CV**	**Mean**	**CV**			**Pearson**	**Spearman**
Au	>0.1	190	4.11	2.42	3.73	2.06	6.64	-2.23	0.92	0.96
Ag	>1.0	167	51.4	1.64	50.5	1.54	6.67	-4.17	0.94	0.98
Cu	>0.1%	161	3.30	1.51	3.11	1.47	4.42	-0.81	1.00	1.00
Pb	>0.01%	152	0.64	1.72	0.64	1.72	6.67	-3.34	0.96	0.98
Zn	>0.05%	147	4.99	1.60	5.01	1.57	3.92	-1.99	0.99	0.98

Table 10. SGS Bor referee analysis of Toachi sample pulps previously analysed by MSA and ALS										
Element	**Limit**	**No.**	**Original Data**		**MSA**		**Mean HARD (%)**	**Mean HRD (%)**	**Correlation Coefficients**	
			Mean	**CV**	**Mean**	**CV**			**Pearson**	**Spearman**
Au	>0.1	394	2.94	2.07	2.81	2.07	4.21	2.84	0.94	0.96
Ag	>1.0	373	31.4	1.73	31.6	1.90	4.21	-3.57	0.86	0.98
Cu	>0.1%	336	3.42	1.23	3.32	1.24	2.34	0.15	0.97	0.99
Pb	>0.01%	315	0.55	2.13	0.58	2.08	5.26	-0.13	0.96	0.99
Zn	>0.05%	359	4.09	1.56	4.25	1.74	3.52	-1.23	0.97	0.96

Mr. Wolfe visited the project between May 20th and May 23rd, 2017. Drilling, core handling, logging, sampling and data collection procedures were reviewed in detail with Toachi technical personnel. It was concluded that the procedures being employed meet international best practice.

During the site visit by the principal author, a suite of 66 representative samples from mineralisation were collected from the historical reject drill sample material stored on site. The samples were transported to ALS Quito for sample preparation and subsequently shipped to ALS Lima, Peru for analysis. Analysis was conducted via 50g Fire Assay for gold with an AAS finish and acid digest/AAS finish for Cu, Pb and Zn. Ag was analysed by an ICP – AES method (Table 11). Review of the assay results indicates no bias between the original and the referee sampling.

Table 11. ALS Lima referee analysis of historical Toachi sample pulps

Element	Limit	No.	Original Data		MSA		Mean HARD (%)	Mean HRD (%)	Correlation Coefficients	
			Mean	CV	Mean	CV			Pearson	Spearman
Au	>0.003	66	0.67	0.91	0.67	0.91	5.67	-0.95	0.99	0.99
Ag	>0.14	66	10.01	1.19	11.50	1.23	11.76	-8.63	0.97	0.99
Cu	>0.01%	66	0.24	1.43	0.25	1.44	5.06	-1.41	0.98	0.99
Pb	>0.001%	64	0.12	1.66	0.15	2.07	5.69	-3.38	0..97	1.00
Zn	>0.007%	66	0.85	1.56	0.87	1.52	5.61	-2.93	0.99	1.00

12.0 Data Verification

12.1 Independent Qualified Person Review and Verification

Mr Brian Wolfe visited the Project between May 20th and May 23rd, 2017. Steps undertaken to verify the integrity of data used in this report include:

- Field visits to the areas outlined in this report.
- During the site visit by the principal author, a suite of 66 representative samples from mineralisation were collected from the historical reject drill sample material stored on site (Section 11.6.6).
- Inspection of drill core.
- Inspection of DC drilling sites, sampling and logging procedures.
- Review of data collection, database management and data validation procedures.
- Review of the previous NI 43-101 reports for the project.

Mr. Wolfe also completed the updated resource estimate for the La Mina Deposit Additional data verification steps undertaken during this estimate process included the following:

- Validation of drilling, geology and assay database (including checks overlapping intervals, samples beyond hole depth and other data irregularities).
- Review of QAQC charts for standards, blanks and duplicates (Section 11).
- Visual and statistical analysis of resource estimate model outputs versus primary data.
- Random cross checks of assay hardcopy reports against the database.

Based on this review work, Mr. Wolfe is of the opinion that the dataset provided is of an appropriate standard to use for resource estimation work.

12.2 Conclusion

As no information is available pertaining the phase of drilling undertaken by Duncan Derry this data could not be verified. This data was therefore not used for the current NI 43-101 resource calculation by Toachi Mining Inc.

As the available Minera Toachi drillhole database is incomplete and of poor quality the data could not be verified. Additionally, the database is not supported by any drill core, assay certificates or quality control data. This data was therefore not used for the current NI 43-101 resource calculation by Toachi Mining Inc.

The remaining available drillhole database relating to drilling undertaken by Cambior, Cornerstone and Toachi Mining Inc has been reviewed and verified by the QP as being accurate to the extent possible and to the extent possible all geological information was reviewed and confirmed. There were no material errors or issues identified with the drillhole database that relates to work undertaken by Cambior, Cornerstone and Toachi Mining Inc. Mr. Wolfe is of the opinion that the portion of the database that relates to work undertaken by Cambior, Cornerstone and Toachi Mining Inc is of sufficient quality to be used for the current / updated Mineral Resource estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Historical Metallurgical Testing, 1997 – 2016

Several metallurgical laboratory studies have been conducted for the La Plata project now known as the La Mina VMS Project since 1997. The initial objective of the programmes concentrated primarily with the recovery of gold, and then to recover the other contained metals. The laboratory studies are summarized in Table 12, and included: gravity (centrifugal and tabling) concentration, cyanide leaching, and bulk and selective flotation of copper and zinc concentrates.

Table 12. 1997 – 2016 Testwork Summary. (Vachon 2016)

Date	Company	Test Work Executed	Results/Conclusions
Jul-97	Cambior	Gravity (Tabling); Bottle Rolls (Cyanidation); Flotation	Low grade gold conc.; Cyanide Residues; low grade Au and Cu concentrates
Feb-98	Cambior	Gravity and Flotation test work; Flotation at fine (P80 ~ 44 microns)	Amenability of the gold to gravity concentration; Produced separate Cu/Zn concentrates; Potential to leach flotation tails
Sep-02	Buenaventura	Gravity and Flotation (2 stage Cl.)	Low grade gold and copper concentrate. Max Recovery of 61% for Au, and 41% for Cu predicted
May-03	Sultana	Centrifugal Gravity testing Flotation - various grind sizes and reagents:	Good Gold Recovery by gravity achieved
2006	Cornerstone Capital Resources	Technical Report (AMEC) (Not NI 43-101 compliant)	Flotation flowsheet and mass balance developed, based on information available to date
2011	Sultana	Centrifugal Gravity confirmation: Sequential flotation - Cu, Pb, and Zn Concentrates	Gravity Au recoveries confirmed. Poor Cu, Pb, and Zn grades and recoveries to individual Concentrates
2014	Sultana	AMEC - Updated Costs & Financial Model (Not NI 43-101 compliant)	300 t/d (upgradable to 500t/d) Gravity/Flotation plant proposed

The 2011 study confirmed the requirements for a gravity circuit, and pursued various metallurgical processes to treat the mill feed material, resulting in a confirmation of the previous conceptual process flowsheet. AMEC (2014) consolidated the previous work and the results and recoveries to produce an upgraded metallurgical balance (Table 13) on the basis of average LOM mill feed at the time.

The proposed process plant would recover gold, silver, copper and zinc metals into gold, copper and zinc concentrates. A conventional crushing and grinding circuit involving gravity recovery of a gold concentrate after the grinding circuit, followed by differential flotation and dewatering systems was the proposed flowsheet.

Table 13. Proposed Metallurgical Balance (AMEC, 2014)

		Assays				Metal Distribution			
Product	**Wt %**	**Au, g/t**	**Ag, g/t**	**%Cu**	**%Zn**	**%Au**	**%Ag**	**%Cu**	**%Zn**
Mill Feed	100.0	7.5	83.1	4.8	6.3	100.0	100.0	100.0	100.0
Gravity Conc.	4.0	98.0	729.8	5.4	2.2	51.4	34.7	4.5	1.4
Cu Conc.	8.8	31.6	376.4	32.7	1.7	36.9	40.0	60.4	2.3
Zn Conc.	14.4	1.5	53.1	1.4	30.6	2.9	9.2	4.1	70.1
Tailings	72.8	0.9	18.5	2.0	2.3	8.9	16.2	31.1	26.3

AMEC recommended the plant would commence at a 300 t/d treatment rate, which would be upgraded to treat 500 t/d, and concluded the precious metal recoveries to gravity and flotation concentrates were reasonable. However, those for both Cu and Zn to their concentrates were considered average, and required further development to be optimised. The study also discussed the potential to produce a lead concentrate, which would represent an alternative to possible lead penalties levied on the copper concentrate, should these become excessive.

13.2 Metallurgical Testing (SGS – May, September 2017[1])

13.2.1 Samples Tested

A scoping level program was completed at SGS Lakefield on four samples of the dominant rock types available, with the main aim of confirming, and improving if feasible, the potential of the recommended flowsheet. Two samples each from the North Block (including previously mined areas), and South Block (recently drilled areas) zones were selected, comprising of "VMS" and "Footwall" rock types for each section. The selected intersections were subsampled from the coarse assay rejects from which the individual assay samples are taken. While the samples were representative of the required overall intersections, the relative 'fineness' precluded any 'gravity' gold, and standard Bond Rod and Ball Mill grinding circuit parameter tests.

The initial characterisation tests included size assaying and mineralogical analysis (by QEMSCAN) to determine the individual mineral associations and develop the flotation testing program. The 'as-received' size distributions (assay reject samples), head assays, and individual mineral liberation sizes are presented in Tables 14, 15 and 16 respectively.

Table 14. Test Sample 'As Received' Size Distributions (SGS 2017)

Size - % Retained	South VMS	South Footwall	North VMS	North Footwall
28	37.1	34.0	36.2	34.5
65	19.8	21.9	19.9	19.4
200	15.6	13.7	14.6	15.6
635	9.8	10.2	10.8	13.9
Pan	17.8	20.2	18.5	16.7
Total	100.0	100.0	100.0	100.0
K_{80}. µm	1,073	969	1,044	998

[1] SGS refers to the samples as Met #s 1, 2, 3, 4, referring to South VMS, South Footwall, North Footwall and North VMS respectively

Table 15. Test Sample Head Assays

Assays	South VMS	South Footwall	North VMS	North Footwall
%Cu	1.83	0.19	7.99	2.98
%Pb	1.73	1.11	0.84	0.41
%Zn	8.03	3.61	7.08	6.12
Au g/t	5.81	0.70	2.41	2.65
Ag g/t	40.7	7.6	60.1	30.6
%S (Total)	18.0	8.5	34.6	16.4
%Sulphide S	12.8	5.9	33.4	14.7
% Sulphate S (SO_4)	6.20	4.30	2.00	2.30

The minerals present, individual particle size and liberation data were extrapolated from the QEMSCAN tables. As indicated from the head assays shown, the presence relatively high proportions of sulphide minerals are confirmed in each of the samples. In addition, the non-sulphide minerals include varying, and potentially significant quantities of Barite ($BaSO_4$).

Table 16. Distribution of Minerals in Samples

Sample	South VMS	South Footwall	North VMS	North Footwall
Pyrite/Marcasite	17.3	5.0	40.4	5.7
Sphalerite	10.5	6.7	11.2	12.8
Chalcopyrite	9.1	0.6	20.5	5.2
Galena	0.6	1.6	1.2	2.2

Sample	South VMS	South Footwall	North VMS	North Footwall
Quartz	28.7	32.2	5.8	2.9
Sericite/Muscovite	12.4	13.4	3.1	2.5
Barite	13.5	32.7	13.1	64.7
Other	7.9	7.7	4.7	4.0
Total	100.0	100.0	100.0	100.0

Table 17 shows the mean diameter of the major minerals present in the samples tested, and confirm the requirements of an ultra-fine grind size P_{80} of < 20 microns to achieve liberation for final separation into their respective concentrates.

Table 17. Average Mineral Particle Size, Microns

Sample	South VMS	South Footwall	North VMS	North Footwall
Pyrite/Marcasite	24	20	23	25
Sphalerite	17	19	14	20
Chalcopyrite	13	13	11	15
Galena	13	13	11	11
Quartz	24	20	19	13
Barite	10	11	9	13

13.2.2 Flotation Testing

The testwork was carried out in 2 phases, with the first series of 8 tests on the North Footwall Sample, which carried the highest combined base metal grades (~16% Cu + Zn). This followed the standard SGS base metal (Cu/Pb/Zn) test program, which is typically up to 3 or 4 stages of flotation in series. Three of the four samples received had relatively low Pb levels, and both the Cu and Pb minerals were collected as a combined concentrate as the first stage, and Zinc as the second. A third concentrate was taken to collect the remaining sulphides (mainly pyrite). Each test was an evolution of the previous, with the aim of developing the reagent regime – maximizing recovery of Copper[2], and more particularly to depress the misplaced Zinc in the Cu/Pb floats, while recovering the Zinc into the second concentrate. In addition, the grind size was sequentially reduced to improve the liberation of each of the major minerals as the tests progressed, as more liberation and mineralogical information became available. The grind size used in the various tests decreased from the original P_{80} value of 65 down to 25 microns for the first 8 tests.

The second phase on the remaining samples (South VMS, South Footwall and North VMS) was completed once the reagent conditions had been established for the North Footwall sample (tests F1 through F8). Two tests each for South Footwall (tests F11 and F13) and North VMS (tests F9 and F12), and one for South VMS (F10) were completed. The second tests (F12 and F13) were at a slightly finer grind than the first tests, aimed at improving liberation.

13.2.3 Flotation Results

Recognising the relatively fine particle sizes of both the sulphide and gangue minerals, the program concentrated on optimising the potential mineral recovery in the most straightforward Rougher/Scavenger flowsheet – at the expense of a relatively fine primary grind stage. The best overall results produced for each process material type are shown in Table 18, with those for the VMS samples shown first followed by the assays/metal distributions highlighted for each test.

Table 18. 2017 Test Program Results – Summary of be Results Achieved

Sample/ Test Number	Grind Size P_{80}	Product	Wt %	Assay, %, g/t						Distribution, %					
				Cu	Pb	Zn	Au	Ag	S	Cu	Pb	Zn	Au	Ag	S
South VMS - F10	15 µm 60 min	Cu/Pb Conc.	14.2	12.7	11.8	12.0	32.3	126	25.4	91.2	88.0	19.7	78.9	44.5	19.9
		Zn Conc.	13.9	0.59	0.8	48.1	2.0	102	29.0	4.1	5.8	77.4	4.8	35.1	22.3
		Py Conc.	9.7	0.64	0.44	1.7	7.61	67	30.6	3.2	2.2	1.9	12.7	16.2	16.4
		Py Tail	62.3	0.05	0.12	0.14	0.34	2.7	12.0	1.5	3.9	1.0	3.7	4.2	41.4
		Head (Calc.)	100	1.97	1.9	8.63	5.8	40.1	18.1	100	100	100	100	100	100

[2] And, by inference, most of any liberated/associated Au

North VMS - F12	20 µm 80 min	Cu/Pb Conc.	30.9	8.77	1.12	8.2	6.47	59.8	21.6	92.0	85.9	42.2	70.6	62.5	41.3
		Zn Conc.	20.0	0.62	0.15	17.0	1.96	39.2	19.5	4.2	7.3	56.7	13.8	26.5	24.1
		Py Conc.	12.1	0.55	0.094	0.29	1.92	14.5	31.8	2.3	2.8	0.6	8.2	5.9	23.8
		Py Tail	37.1	0.12	0.043	0.09	0.56	3.97	4.68	1.5	3.9	0.5	7.3	5.1	10.8
		Head (Calc.)	100	2.94	0.4	6.00	2.83	29.5	16.1	100	100	100	100	100	100
South Footwall - F13	19 µm 70 min	Cu/Pb Conc.	25.9	0.7	3.91	6.17	2.3	17.0	11.6	78.4	92.6	45.1	64.5	59.2	34.4
		Zn Conc.	22.5	0.05	0.22	8.5	0.6	9.2	10.4	5.2	4.5	54.1	15.2	27.9	26.8
		Py Conc.	7.4	0.075	0.18	0.15	2.19	10.0	17.3	2.5	1.2	0.3	17.8	10	14.6
		Py Tail	44.2	0.07	0.041	0.04	0.05	0.5	4.8	13.9	1.7	0.5	2.4	3	24.3
		Head (Calc.)	100	0.23	1.09	3.55	0.9	7.4	8.7	100	100	100	100	100	100
North Footwall - F7	26 µm 45 min	Cu/Pb Conc.	45.0	16.4	1.69	8.11	2.75	85.7	35.8	87.2	86.0	55.8	51.9	62.0	48.4
		Zn Conc.	32.1	3.05	0.33	8.86	3.34	63.5	42.4	11.6	11.9	43.6	45	32.8	41.0
		Py Conc.	3.1	1.23	0.14	0.32	1.36	29	42.9	0.5	0.5	0.2	1.8	1.5	4.0
		Py Tail	19.8	0.33	0.071	0.13	0.16	11.8	10.9	0.8	1.6	0.4	1.3	3.7	6.5
		Head (Calc.)	100	8.48	0.88	6.53	2.4	62.2	33.2	100	100	100	100	100	100

Of particular note are the results for the South VMS sample (Test F10), which are highlighted in the as follows:

a) Cu/Pb[3] Conc.: Cu - 91% recovery at 13% Cu, containing Au – 79% at 32 g/t and Ag – 44% at 126 g/t.

b) Zinc Conc.: Zn - 77% to 48% Zn; Au – 5% to 2 g/t; Ag – 35% to 102 g/t.

c) Pyrite Conc.: Au - 12% to 8 g/t and Ag – 16% to 67 g/t. The Sulphur grade was 31%S, which would need to be upgraded in a cleaner circuit to > 45%S.

d) The tailings product was 62% of the weight with less than 5% of the minerals being lost. The remaining Sulphur in the product is substantially included in Barite ($BaSO_4$).

13.2.4 Conclusions

The very fine primary grind show that final grades of concentrate can be produced directly from the rougher circuit.

These results confirm the relative complexity of the mineral assemblages, and the potential difficulties in the standard approach to base metal flotation. The South VMS test result (Test F10), however, was exceptional, and showed that relatively high recoveries of the important minerals could be achieved into their respective concentrates, once sufficient inter-sulphide size liberation was achieved. In this case the primary grind (80% passing 15 microns) had no apparent detrimental effect on flotation performance, as acceptable individual Copper and Zinc concentrates were produced. It should be noted however, that the head grades of each of the metals of interest were relatively high, as shown by the lead content produced in the concentrate, which was very similar to that of copper. This is due to the higher head grade of this particular sample.

13.3 Metallurgical Testing (SGS – March 2019)

13.3.1 Composites Tested

Following the 2017 test program on the four individual samples, a comprehensive sampling program was undertaken based on the core library of the deposit to produce zones that were most representative of the deposit for further testing. The full details of the samples delivered to SGS are included in Table 19.

[3] Lead - 88% recovery at 11%Pb, reflects the particularly high grade in this sample.

Table 19. Sample Details

Sample	Hole Numbers	Nos of Samples	Total Weight (kg)
Composite 100	CMLP: 16 - 01, 03, 07, 08, 12	51	98.2
Composite 200	CMLP: 17 - 28, 32, 36, 38, 40, 42, 47, 49, 55, 65, 77	187	97.5
Composite 300	CMLP: 16 - 02, 04, 17, 18, 20, 25; 17 - 37, 41, 56, 57, 63, 66, 73	174	97.4
Composite 400	CMLP: 16 - 13, 14	47	47.6
South 100	Sample # A015001	1	25.6
North 200-300-400	Sample # A015002	1	28.3

The four individual composite samples (100, 200, 300 and 400) were all subsamples of the core library assay rejects. The South 100, and the North 200-300-400 were quarter core samples from the then recent drilling campaign, which were provided for the comminution testing requirements. Head samples of each of the individual zone composites were taken for analysis, presented in Table 20.

13.3.2 Mineralogy – Assay and Analysis

A head sample from each composite was submitted for assay analysis, and the results for the metals of interest are given in Table 20.

Table 20. Sample Head Assays

Sample	Composite 100	Composite 200	Composite 300	Composite 400	Comp. North 200/300/400
Cu%	1.55	3.74	2.69	1.90	3.68
Pb%	0.95	0.22	0.69	0.63	0.74
Zn%	6.98	2.11	5.15	6.80	4.84

Sample	Composite 100	Composite 200	Composite 300	Composite 400	Comp. North 200/300/400
Au, g/t	3.12	1.71	3.08	3.83	3.12
Ag, g/t	47.60	29.00	35.00	34.90	40.50
S%	14.40	19.00	17.70	13.50	18.00
SG	3.37	3.34	3.32	3.19	3.40

Of note are the comparatively high specific gravity (SG) of each of the head samples, reflecting the high sulphide mineralised content.

A semi-quantitative mineral analysis was completed by QEMSCAN[4], where each sample was ground to 100% passing 150 µm and analysed as a single fraction. The mineralogical analysis indicated that copper was present predominantly as chalcopyrite, with lesser amounts of covellite and bornite, while lead and zinc were present as galena and sphalerite, respectively. Dilution minerals consist of pyrite, quartz, sericite/muscovite, and bornite. The association results indicate moderate to good liberation of the sulphide minerals for the samples, as tested. The study included modal analysis and liberation data for each of the sulphides present, and also for all the various gangue minerals. Table 21 shows the distributions of the sulphide minerals present in each of the composites, together with the dominant gangue minerals.

Table 21. % Distribution of Sulphide Minerals in Composites

Sample	Composite 100	Composite 200	Composite 300	Composite 400	Comp. North 200/300/400
Pyrite/Marcasite	11	26	22	13	21
Sphalerite	15	4	11	14	11
Chalcopyrite	5	10	8	6	11
Galena	2	0.4	1	1	2

[4] Covered in detail in the 2019 SGS test Report

Sample	Composite 100	Composite 200	Composite 300	Composite 400	Comp. North 200/300/400
Barite	20	8	9	9	12

The individual particle sizes of the component minerals present are presented in Table 22. These are very similar to those developed from the 2017 program (Table 21), and further confirms the ultimate grind size for the La Mina mineralization for the production of separate sulphide concentrates by flotation will be less than 20 microns.

Table 22. Average Mineral Particle Size, Microns

Sample	Composite 100	Composite 200	Composite 300	Composite 400	Comp. North 200/300/400
Pyrite/Marcasite	21	22	22	17	23
Sphalerite	16	16	19	19	22
Chalcopyrite	16	18	19	17	19
Galena	10	12	13	13	15
Quartz	18	22	22	22	25
Barite	11	13	14	16	15

13.3.3 Comminution

The two samples for comminution were delivered as ¼ core, and were subjected to two standard SGS tests[5], the SAG Power Index (SPI), and the Bond Ball Work Index (BBWI). The results are combined in Table 23.

[5] These tests form part of the overall Ore Hardness Tests available at the testing facility

Table 23. Comminution Test Results

Sample Tested	SAG Power Index, minutes	Bond Ball Mill Work Index, kWh/t
South Block, Ore Zone 100	32.3	14.6
North Block, Ore Zone 200-300-400	38.8	13.3

The SAG POWER INDEX test was specifically developed to measure the power requirement of the autogenous (AG) or semi-autogenous (SAG) components of any comminution circuit, in much the same way as the standard Bond Rod and Ball Mill Indexes are used for Rod and Ball Mills. The developers found that each ore body has a specific 'hardness' profile, and provided sufficient samples are tested then, then the power requirements for the primary milling part of the circuit can be confidently developed. The methodology of how the Index is used for AG/SAG mill sizing has been described in the literature and in a recent textbook on Mineral Processing[6]. This is often used to supplement more traditional mill sizing methodologies, and all engineering companies, mill suppliers, as well as testing facilities and consultants provide such services[7].

The results produced for both measurements are relatively 'Soft' to 'Moderate' when compared to the SGS testing database, which placed both samples at, or less than, the 20th percentile for the SPI, and ~ the 50th percentile for the Ball Mill Work Index. These are on the 'Soft' to 'Medium' classification respectively, with the consequence that no circuit difficulties are expected to be encountered in any of the selected unit process stages, especially for the ultimate grind sizes required for the final production of concentrates.

13.3.4 Gravity Testing

The standard SGS bulk gravity separation test was utilized to determine the potential application of this unit process. The test comprises of a first pass through a laboratory centrifugal 'Knelson' separator, followed by upgrading the concentrate produced by a laboratory 'Mozley' separator[8]. This is used to produce a low weight product which contains any 'free' gold available. The test is completed on a simulated primary cyclone underflow product, in this case produced in a rod mill, with a feed size (P80) of between 60 and 80 microns for each of the samples. Two programs, the first using 2 kg of feed material, the second 10kg,

[6] Wills, 2006, pp 112/3

[7] AG and SAG mill sizing, and Comminution circuit design and mine/mill throughput forecasting is also provided by SGS (Toronto)

[8] This is a laboratory 'Shaking Table' which is used to upgrade produce a low weight 'tip' containing any 'free' gold available from the original feed..

were undertaken on each of the main composites. The second series was completed to provide sufficient sample masses for the subsequent flotation program, originally envisaged to follow the previous flowsheet as described in Section 13.2.

The results from the second series are shown in Table 24.

Table 24. Distribution of Gold recorded in 'Gravity' Test products

Parameter	Composite 100	Composite 200	Composite 300	Composite 400	Comp. North 200/300/400
Feed P80, µm	81	81	73	77	90
%Wt. of original feed	0.086	0.164	0.128	0.155	0.090
Assay, g/t Au	1,127	276	987	1,034	1,693
Au Dist., % Feed	28.20	24.50	40.10	41.70	34.80
Gravity Tail, Au - g/t	2.05	1.88	1.86	2.22	2.86
Calc. Head, Au - g/t	3.46	2.55	3.16	3.84	4.39

There is a significant upgrading for all the composites into a small proportion of the original feed. This does not, however, indicate a significant proportion of 'free gold' as it would normally be classified. However, the grade of the concentrate produced indicates any 'free' gold present will be very fine, and most likely associated with some, or all of the sulphide minerals present in the feed. Confirmation of the application of a gravity stage within the future milling complex will be need to be completed in the next phase of testwork.

13.3.5 Flotation Testing

The results of the 2017 program[9] confirmed the feasibility of separate production of low grade individual copper and zinc concentrates by conventional sequential flotation at primary grind P80 sizes of less than, or around, 20 microns. The bulk gravity tails produced were then prepared for the next series of testing by size reduction to a P80 of ~15 microns, the size that produced the optimum performance (South VMS

[9] Summarized in Table 13.2_5

sample - test F10). One important issue with the initial program on the three other samples was that of the 'misplaced' zinc to the Copper/Lead concentrate (40 – 60 %). Reducing this to levels below 20% (as for test F10) was the main focus of the current development phase, and more than 50 Rougher/Cleaner tests were completed to develop the conditions suitable to achieve the project objectives.

A secondary, but substantial effort was made to determine the effect and the metal losses to tailings at a more conventional[10] primary grind size, typically of between 60 and 75 microns. Once provisionally acceptable sulphide performance was completed (this was set at less than 5% losses of Cu and Zn to the tails fraction), extensive testing of all applicable and relevant flotation parameters, particularly the reagent combinations and slurry chemistry were then developed to achieve the required base metal distributions (Table 25).

Table 25. Summary of conditions tested through the Test Program

SGS Test #	Feed P80, microns	Composites Tested	Summary of Test Aims and Outcomes
F1 - F5	15	100, 200,300, 400, North 200/300/400	2017 reagent suite: tested on the gravity tails at the fine grind for each composite to produce separate Cu/Pb, Zn, and pyrite concentrates. Significant 'misplaced' Sphalerite (40-63%) in the Cu/Pb concentrate, with low Au/Ag into the pyrite conc.
F6- F16	15	200	Reagent development with Composite 200 - aimed at reducing the misplaced Sphalerite in the Cu/Pb conc.
F17- F28	65	300	Reagent development continued at reducing the misplaced Sphalerite. Tests F25-28 showed the beneficial effects of SMBS/ZnSO4 compared to NaCN
F29 - F34	65	300	Developed the high addition rates of SMBS/ZnSO4 as Sphalerite depressant combined with high additions of Cu/Pb collector -3418A
F36 - F40	65	300	Additional Zn depression tests with reagent variations – test 40 (38%) compared with tests 36 – 39 (63 to 84%) showed a step reduction in misplaced Sphalerite
F42 - F45	70, 82, 95, 131	300	Primary Grind Size - testing Sulphide mineral separation from gangue. Similar performance seen up to 95 microns, reduces at the coarsest size
F46 - F54	74; 20 – 25 for regrind	300	The first series of Cu/Pb and Zn Cleaner tests after regrinding

[10] Particularly for base metal flotation circuits

SGS Test #	Feed P80, microns	Composites Tested	Summary of Test Aims and Outcomes
F55 - F56	74; 20 – 25 for regrind	300	Cleaner circuit optimization tests.; F55/56 were preliminary tests to separate Pb from the Cu/Pb concentrate
F57	70	300	A two-cycle test to confirm the conditions as a precursor to the Locked Cycle Tests
LCT #s 1 and 2	70; ~22 for conc. regrinds	300	2 off, 6 - cycle tests. The first under optimised conditions, the second with some reagent variations – overall results substantially the same.

13.3.6 Results

Following the results of the Phase 1 (2017) program, the flowsheet development focused initially on the fine (P80 ~15 microns) reground gravity tailings, with the first five being 'sighter' tests for the new composites under the original conditions. These tests produced separate low grade rougher Cu/Pb and Zn concentrates - the former containing between 40 and 60% of the sphalerite. However, the combined 'gravity' and Cu/Pb concentrates contained on average 70% of the gold. In addition, the tests showed the separate pyrite concentrate contained minimal (<1.5g/t) levels of Au.

Tests F 6 – F16 continued to attempt to reduce the sphalerite entrainment into the Cu/Pb concentrate, with on average ~60% of the zinc misreporting to the Cu/Pb concentrate. The test procedure was then switched to the samples being prepared directly prior to each test, specifically to avoid any additional surface oxidation that may have been caused by the test procedure. This procedure enabled the pulp chemistry required to minimise the quantity of misplaced sphalerite and pyrite in the primary copper/lead concentrate to be developed.

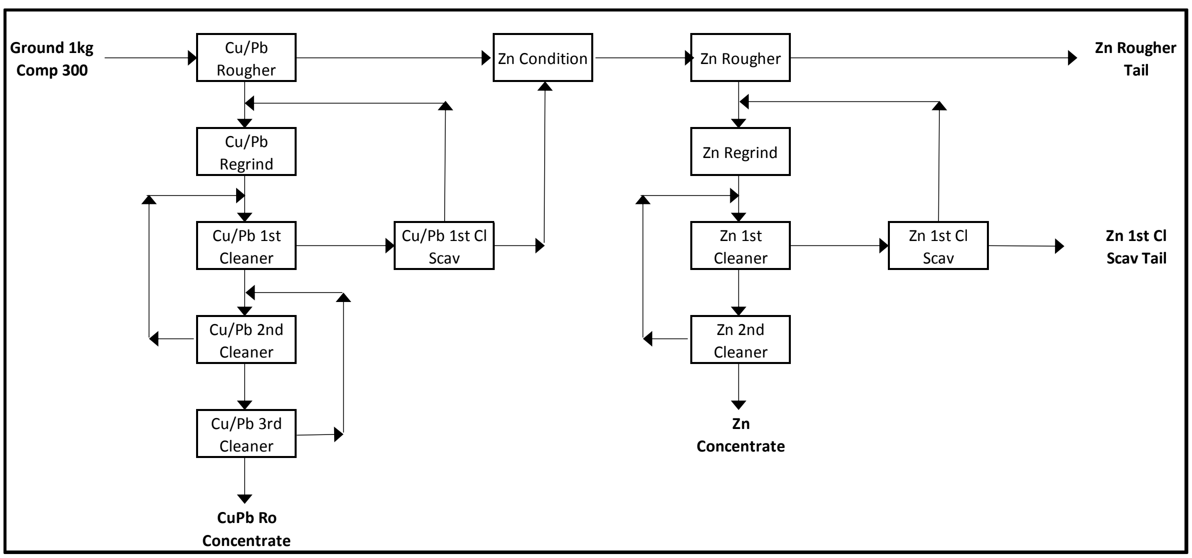
Tests F17 – F40 were undertaken at a coarser primary grind size (P80 ~65 microns,) and the development culminated in a robust chemical combination for the Cu/Pb rougher stage. From all the potential sphalerite/pyrite 'depressant' chemicals tested, the most effective reagent combinations were found to be staged additions of sodium sulphide (Na_2S), sodium metabisulfite ($Na_2S_2O_5$), and zinc sulphate ($ZnSO_4$) which succeeded in reducing the misplaced minerals to acceptable levels.

The second circuit followed normal Sphalerite flotation procedures, achieved by conditioning the pulp with copper sulphate ($CuSO_4$) for reactivation, and flotation with a xanthate collector (sodium isopropyl xanthate - SIPX) at a high pH (~11.5 – for pyrite rejection).

The rougher/scavenger testing phase demonstrated high (~95%) recoveries of the Cu/Pb and Zn minerals were possible at the development phase grind size (P80 of ~65[11] microns), and this continued with tests F46 – F56 included regrinding to the target grind size of ~20 microns, and the subsequent cleaner stages required for both the Cu/Pb and Zn Rougher concentrates.

The program culminated in a series of two Locked Cycle Tests (LCT)[12] which has confirmed the concentrate, tailings and metals deportments achievable following these developments. The circuit used for the test is shown in Figure 20, the detailed balance of the selected test (LCT #1)[13] is summarized in Table 26.

Figure 20. The Flotation flowsheet utilized for the Locked Cycle Tests (SGS – March 2019)



[11] A relatively coarse primary grind size (compared to the individual mineral particle sizes - Table 13.3_4)

[12] This is the industry standard for predicting actual plant performance of sequential flotation stages in flotation circuit design

[13] The reagent scheme used for LCT#2 is a variant of #1,

Table 26. Locked Cycle Test (LCT) #1

Product	%Wt.	Assay, % or g/t						Metal Distribution, %					
		Cu	Pb	Zn	Au	Ag	S	Cu	Pb	Zn	Au	Ag	S
Cu/Pb Concentrate	11.5	20.9	4.1	9.0	13	141	35.7	89.4	71.8	20	43.1	48	23.6
Zn Concentrate	7.1	1.71	1.3	51.2	13	125	31.3	4.5	13.5	70.5	26.5	26.2	12.7
Final Tailings	81.5	0.2	0.1	0.6	1.3	10.7	13.6	6.1	14.8	9.5	30.4	25.8	63.7
Zn 1st Cl Tails	14.7	0.31	0.18	1.63	1.42	15.9	14.4	1.7	3.9	4.7	6.0	6.9	12.2
Zn Rougher Tails	66.8	0.18	0.11	0.37	1.26	9.6	13.4	4.4	11.1	4.9	24.3	19	51.6
Calculated Head	100	2.68	0.66	5.13	3.46	33.73	17.39	100	100	100	100	100	100
Assayed Head		2.69	0.69	5.15	3.08	35	17.7						

The following summarize the results - for the Copper/Lead concentrate:

- Copper - 89% recovery to 21% Cu, containing 43% and 48% of the Au and Ag @ 13 and 141 g/t respectively into 12% for the feed weight.
- Lead - 72% recovery to 4.1% Pb
- Zinc - 20% of the distribution @ 9% Zn

For the Zinc concentrate:

- Zinc – 71% recovery to 51% Zn, containing 27% and 26% of the Au and Ag @ 13 and 125g/t respectively, into 7% of the feed weight.

The combined tailings streams:

- Contain 81% of the feed weight, and 30 and 26% of the Au and Ag overall. 67% of the weight is from the rougher tails, containing 24 and 19% of the Au and Ag @ 1.3 and 10.7 g/t respectively.
- Contain 63% of the Sulphur @ 13.6% S.

These overall results have been used as the base case for the flowsheet design and preliminary costings for the circuit engineered in the subsequent sections.

In parallel, a separate analysis on the various metal deportments was undertaken to more fully understand the distributions, particularly of Gold throughout the circuit. The distributions into the two concentrates are clear from the LCT results. The two tailings streams contain the remainder of the of the metals, with ~80% of the mass, and 70 – 80% of the metals (except Zinc at ~50%) contained in the Rougher tailings. The assays of all the metals are very similar for both the zinc rougher and the zinc first cleaner tails, and measuring ~14% total sulphur. Samples from both streams were assayed for 'sulphide' and 'sulphate' S, to determine the species present. A pyrite concentrate was produced from one of the LCT rougher tails streams (Cycle D) to confirm both the Au and Ag content, AND the final tailings sulphur grade should a separate pyrite concentrate be produced. The updated balance is presented in Table 27.

Table 27. Predicted Final Balance for Locked Cycle Test (LCT) #1

Product	% Wt.	Assay, % or g/t						Metal Distribution, %					
		Cu	Pb	Zn	Au	Ag	S	Cu	Pb	Zn	Au	Ag	S
Cu/Pb Concentrate	11.5	20.9	4.1	9.0	13.0	141	35.7	89.4	71.8	20	43.1	47.9	23.6
Zn Concentrate	7.1	1.7	1.3	51.2	13.0	125	31.3	4.5	13.5	70.5	26.6	26.1	12.7
Pyrite Concentrate	21.2	0.35	0.14	0.78	1.41	18.43	34.4	4.0	5.9	5.3	10.0	16.4	51.4
Final Tailings	60.3	0.05	0.06	0.16	0.74	2.77	2.2	2.1	9.0	4.2	20.3	9.5	12.3
Calculated Head	100	2.68	0.66	5.13	3.46	33.73	17.4	100	100	100	100	100	100
Assayed Head		2.69	0.69	5.15	3.08	35.00	17.7						

The pyrite concentrate is at rougher grade and would need to be reground for cleaning to produce a saleable concentrate, which is normally @48+%S. While it does contain 10% of the gold distribution, the Au grade (~1.4 g/t) in all likelihood would not be payable. However, the residual S content of the tails will likely be completely inert, as the remaining S containing mineral is substantially Barite ($BaSO_4$).

13.3.7 Concentrate Quality

The Copper/Lead and Zinc concentrates produced from the LCT were assayed for minor metals and standard minor element impurities, of which the most important of which are listed in Table 28.

Table 28. Concentrate Analysis

Element	Cu/Pb - 3rd Cl Conc.	Zn - 2nd Cl Conc.
Cu %	21.6	2.05
Pb %	4.23	1.61
Zn %	9.22	50.1
Fe %	26.9	5.72
Au g/t	10.7	14.1
Ag g/t	143	131
S %	35.6	31.3
As %	0.22	0.11
Hg ppm	60	250
Cd %	0.034	0.202

Following preliminary indications from a potential purchaser of concentrates, no significant problems with the marketing with these levels of impurities are envisaged. This will need to be further confirmed in the next stage of project development.

14.0 Mineral Resource Estimates

14.1 Introduction

The Mineral Resource for the La Mina Deposit at the La Mina VMS Project, Ecuador has been previously estimated as at 13th September 2017. The current mineral resource update incorporates updated metal price assumptions determined during February 2019 and no new drilling or other technical data as relates to mineral resource estimation is available.

Mr. Brian Wolfe of International Resource Solutions Pty Ltd was requested by Toachi Mining Limited (Toachi) to generate a grade estimate using the available data. This data included historical exploration and grade control drilling collected prior to Toachi's involvement and recent drilling completed under the supervision of Toachi's technical staff. The grade estimate has been classified as Inferred in accordance with the CIM 2014 guidelines.

14.2 Study Approach

The grade estimation study included the following aspects:

- The loading and review of the drill hole database (historic and Toachi), and correction of any data identified as erroneous. This data was loaded into the Vulcan mine planning software package for interpretation and modelling.
- The coding of the interpreted mineralisation and drill type to the database.
- An assessment of an appropriate mineralisation and geological wireframes and adjustment of wireframes.
- The mineralisation wireframes were coded to the drill hole database and 1m composites generated and used for the ordinary kriged grade estimation.
- The completion of a statistical and geostatistical investigation.
- The estimation of Cu, Au, Ag, Pb and Zn via ordinary kriging (OK) into an appropriate block model. Ancillary elements estimated are As, Ba, Fe, S, Sb and also density.
- Appropriate validation and checks of the block model.

14.3 Database Validation

The resource estimation was based on the available exploration drill hole database which was compiled in-house by Toachi technical personnel. The database has been reviewed and validated prior to commencing the resource estimation study.

The database includes samples from diamond drilling that have been included for the purposes of modelling and grade estimation. A total of 126 drillholes were selected at La Mina. The resultant database was validated and the checks made to the database prior to use included:

- Check for overlapping intervals.
- Downhole surveys at 0m depth.
- Consistency of depths between different data tables.
- Check gaps in the data.
- Replacing less than detection samples with half detection.
- Replacing intervals with no sample with -999.
- Replacing intervals with assays not received with -999.

The database, as supplied, was used for resource estimation after a review was completed to correct identified errors. The database consists of surface and some underground diamond drilling. The underground diamond drilling has not been used due to positional uncertainty. Database statistics are provided below as Table 29.

Table 29. Summary of the Available Drillhole Database

Company	Holes	Metres	% of Total
Toachi	80	13,747	49
Cambior	28	8,628	30
Cornerstone	18	5,933	21

Several mineral resource estimates have been completed in the past, however, none of these estimates meet the requirements of, nor are compliant with, the standards of NI 43-101 and cannot be relied upon. One issue is the lack of any verifiable data for the initial drilling campaigns carried out by Cornerstone and Cambior (which has been addressed by submitting a representative suite of sample pulps from these early drilling campaigns for independent check analysis, as discussed in Section 6. The check analysis of analytical results from the earlier drilling campaigns has shown acceptable levels of accuracy and precision. In addition, there is currently significantly more drilling data available (Toachi, 2016 and 2017) and therefore any concerns relating to the quality of some of the historical data have been ameliorated by the quality of the newer data. All surface diamond drilling has, therefore, been included in the resource estimation study.

A gold equivalent variable was calculated to allow resource investigations. The formula was calculated based on the following United States (US) dollar metal prices supplied by Toachi and assumes 100% metallurgical recoveries for the purposes of the calculation:

- Cu $2.68/lb
- Au $1264/oz
- Ag $16.64/oz
- Pb $0.98/lb
- Zn $1.21/lb

The following gold equivalent formula was applied:

Aueq ppm = Au ppm + (Cu % * 1.454) + (Ag ppm * 0.013) + (Zn % * 0.654) + (Pb%*0.532)

The gold equivalent variable (AuEq) was used to assist in defining mineralised zones and as an aid to resource reporting.

14.4 Raw Sample Statistics

Summary statistics for the elements of interest have been generated on the raw sample intervals on a company-by-company basis prior to compositing. It should be noted that the historic databases have been more selectively sampled than Toachi and this has affected the resultant sample statistics. Notwithstanding this observation, sufficient positive correlation exists between the statistics derived from the different exploration phases to conclude that all surface drilling data is fit for the purpose of resource estimation. Table 30 summarises the key statistics of the different data sets and elements, subdivided by different drilling groups.

Table 30. Raw Sample Statistics

Element	Domain	Count	Min	Max	Mean	Std. Dev	Variance	CV
Au ppm	Toachi	6,868	0.003	25.32	**0.27**	1.14	1.30	4.18
Au ppm	Cornerstone	1,276	0.001	76.96	**0.40**	2.52	6.35	6.34
Au ppm	Cambior	2,876	0.001	52.18	**0.32**	1.97	3.88	6.11
Ag ppm	Toachi	6,868	0.25	857.14	**3.22**	15.89	252.35	4.94
Ag ppm	Cornerstone	1,276	0.009	467.00	**5.56**	28.20	795.49	5.07
Ag ppm	Cambior	2,876	0.25	390.00	**4.04**	20.99	440.67	5.20
Cu pct	Toachi	6,868	0	20.02	**0.24**	1.06	1.13	4.53
Cu pct	Cornerstone	1,276	0.005	15.46	**0.24**	0.98	0.96	4.13
Cu pct	Cambior	2,876	0.005	20.70	**0.22**	1.32	1.75	6.05
Pb pct	Toachi	6,868	0	7.59	**0.04**	0.24	0.06	5.92
Pb pct	Cornerstone	1,276	0.005	4.70	**0.07**	0.31	0.10	4.23
Pb pct	Cambior	2,876	0.005	4.45	**0.04**	0.23	0.055	5.46
Zn pct	Toachi	6,868	0	30.00	**0.32**	1.57	2.46	4.87
Zn pct	Cornerstone	1,276	0.005	49.72	**0.46**	2.68	7.17	5.77
Zn pct	Cambior	2,876	0.005	21.59	**0.29**	1.21	1.46	4.20
As ppm	Toachi	6,868	2.5	5,532	**112**	255	64,837	2.27

Element	Domain	Count	Min	Max	Mean	Std. Dev	Variance	CV
As ppm	Cornerstone	1,276	1.4	5,824	**171**	353	124,924	2.07
As ppm	Cambior	2,043	2.5	4,363	**73**	186	34,465	2.54
Ba ppm	Toachi	6,868	5	20,000	**1036**	2141	4,585,676	2.07
Ba ppm	Cornerstone	1,276	5	3,560	**142**	318	101,224	2.24
Ba ppm	Cambior	2,043	2.5	2,000	**430**	507	256,555	1.18
Fe pct	Toachi	6,868	0	39.31	**6.30**	3.81	14.54	0.61
Fe pct	Cornerstone	1,276	0.6	27.90	**5.29**	3.47	12.04	0.66
Fe pct	Cambior	2,043	0.01	10.00	**5.67**	1.86	3.45	0.33
S pct	Toachi	6,868	0.005	46.03	**4.40**	6.04	36.45	1.37
S pct	Cornerstone	1,276	0.01	39.94	**5.20**	5.03	25.30	0.97
S pct	Cambior	560	0.005	47.00	**9.44**	8.44	71.29	0.90
Sb ppm	Toachi	6,868	2.5	1,301	**12**	42	1,760	3.54
Sb ppm	Cornerstone	1,276	0.038	1,370	**12**	74	5,525	6.41
Sb ppm	Cambior	2,043	2.5	1,530	**10**	49	2,396	5.05

In general, the mean analytical grades are higher for the Cornerstone drilling and this is particularly evident for gold, zinc and silver. Copper grades are equivalent. Upon examination of the 3D distribution of the drilling, it is evident that, in addition to being more selectively sampled than the remainder of the drilling, the Cornerstone drilling has targeted the higher-grade portions of the deposit. It may therefore be considered unlikely that any material bias would be introduced by use of this data due to the declustering effect of ordinary kriging.

14.5 Mineralisation Constraints

14.5.1 Geological interpretation

In addition to the mineralisation constraints, site personnel have supplied the interpreted contact between the 'hanging wall' sequence of lithologies and the 'footwall' sequence of lithologies (which was used in mineralised zone interpretation), barren, post-mineralisation felsic intrusive and dolerite models, major fault structures and the base of weathering. A geological model was developed using the interpreted geological wireframes and structures and has been subsequently coded into the block model. East west sections are presented in Figures 21 (North Block) and 22 (South Block) demonstrating the respective lithological and mineralisation relationships.

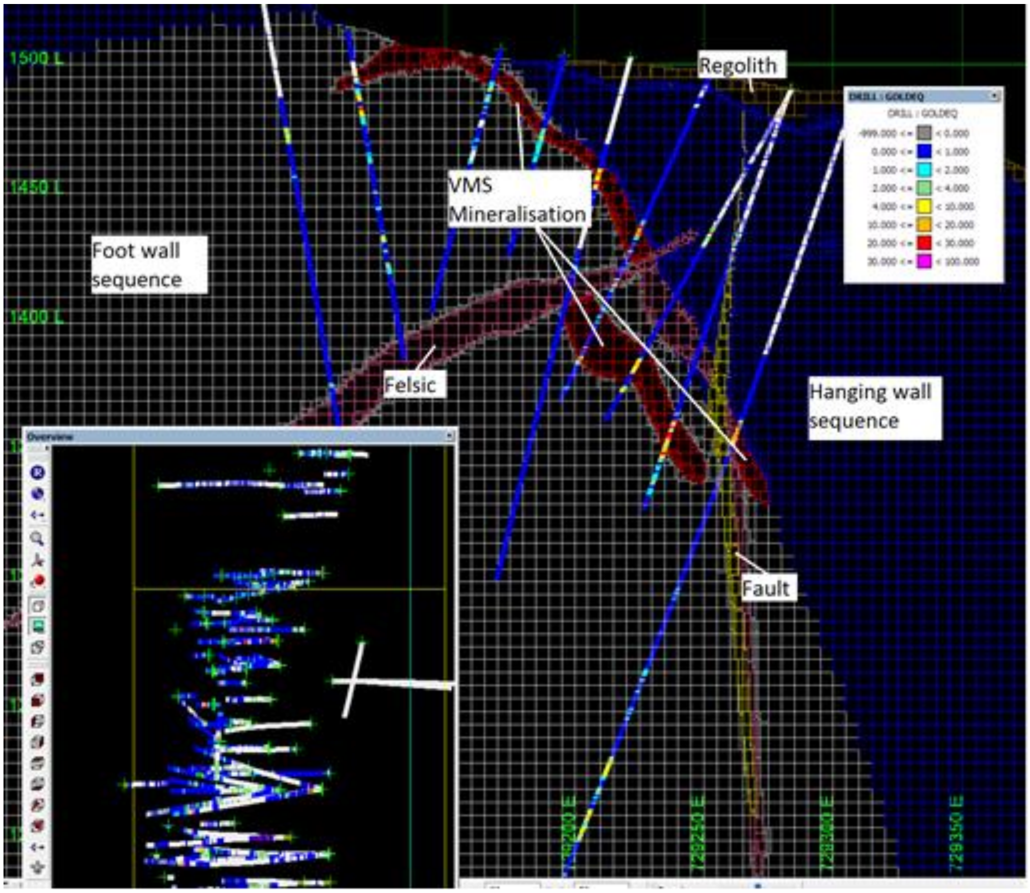


Figure 21. Section 9,957,360mN - Lithological sequence with interpreted mineralisation and drilling

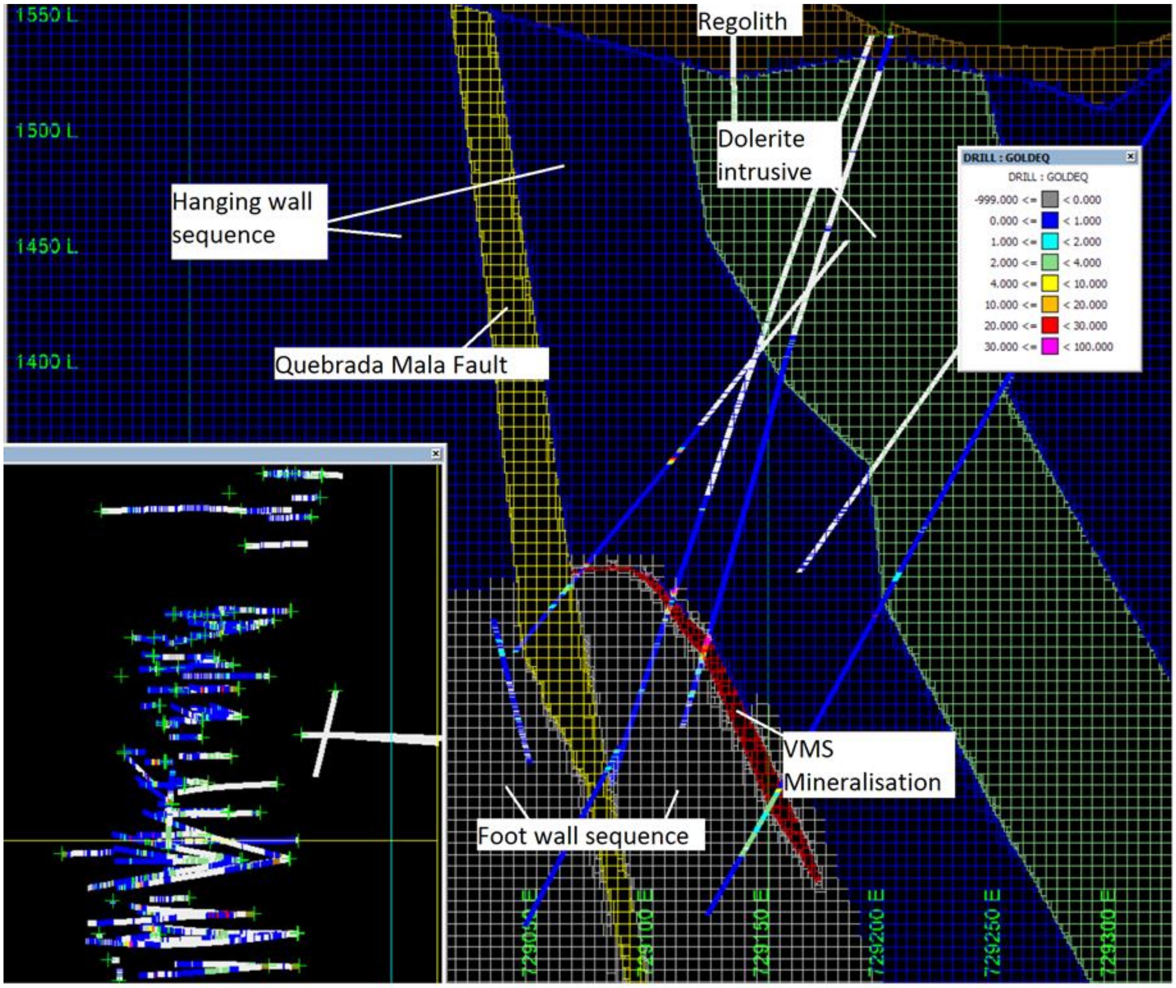


Figure 22. Section 9,957,015mN - Lithological sequence with interpreted mineralisation and drilling

14.5.2 Mineralisation Interpretation

The grade estimation was constrained based on wireframes generated on the basis of geology, mineralisation, multi-element analysis and a gold equivalent grade threshold. The gold equivalent (AuEq) grade that was chosen as a guide in mineralised zone interpretation is 4 g/t AuEq, which is approximately US$165 in situ gross value, as a rough approximation of possible underground mining and subsequent processing costs. For the purposes of overall geological continuity and in order to produce coherent mineralised zone wireframed morphology, the grade threshold has not been strictly adhered to and portions of the wireframes therefore contain significant portions of drilling below this threshold, particularly in the northern sector of the deposit.

The constraining wireframe models were then coded to the drill hole database and also used in the construction of the estimation block model. Individual numerical codes were assigned as follows:-

- South Block Coded Domain 100
- North-east Block Coded Domain 200
- North-west Block Coded Domain 300
- North-east Block Lower Coded Domain 400

An example view of the mineralization model is provided as Figure 23 below.

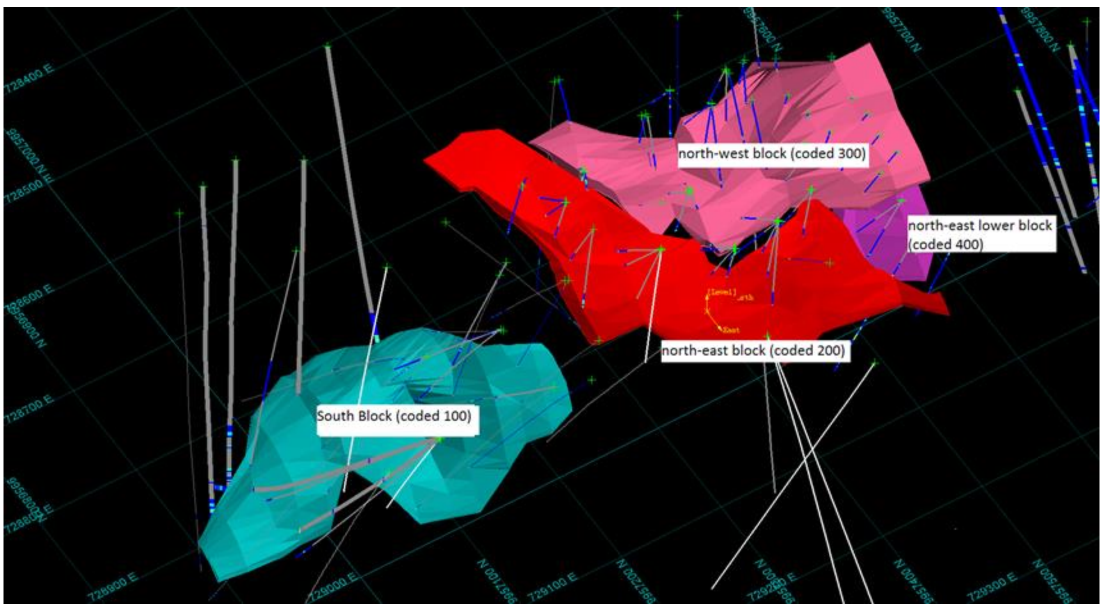


Figure 23. Oblique SE view displaying composites and mineralization wireframe models

14.5.3 Data Quality and Comparisons

Toachi has followed an extensive and robust suite of protocols to manage and document data acquisition and quality control (QAQC). Little quality control data is available for the different historic data sets (i.e. Cornerstone and Cambior). A full discussion of the QAQC procedures employed is included in Section 12.

14.5.4 Data Flagging and Compositing

Drillhole samples were flagged with the relevant mineralisation wireframes described in previous sections. Coding was undertaken on the basis that if the individual sample centroid fell within the mineralisation wireframe boundary it was coded. Each domain has been assigned a unique numerical code to allow the application of hard boundary domaining if required during grade estimation.

The drillhole database coded within each mineralisation wireframe was then composited as a means of achieving a uniform sample support. It should be noted, however, that equalising sample length is not the only criteria for standardising sample support. Factors such as angle of intersection of the sampling to mineralisation, sample type and diameters, drilling conditions, recovery, sampling/sub-sampling practices and laboratory practices all affect the 'support' of a sample. Exploration/mining databases which contain multiple sample types and/or sources of data provide challenges in generating composite data with equalised sample support, and uniform support is frequently difficult to achieve.

After consideration of relevant factors relating to geological setting and mining, including likely mining selectivity and bench/flitch height, a regular 1m run length (down hole) composite was selected as the most appropriate composite interval to equalise the sample support at La Mina.

14.5.5 Uncut Descriptive Statistics

Summary descriptive statistics (1m composites) for each element of interest were generated for all domains and are summarised in Table 31 for the main elements and density. The statistics for the uncut auxillary variables are listed in Table 32.

Table 31. Summary Statistics Uncut 1m Run Length Composites – Main Variables

Domain		100	200	300	400	combined
Gold g/t	Number	206	200	226	67	699
	Minimum	0.14	0.12	0.066	0.003	0.003
	Maximum	77.00	24.98	32.72	25.32	76.96
	Mean	**4.48**	**1.88**	**2.52**	**2.32**	**2.90**
	Std Dev	8.30	2.85	4.28	4.57	5.63
	Variance	68.93	8.11	18.29	20.86	31.65
	Coeff Var	1.85	1.51	1.70	1.96	1.94
Silver g/t	Number	206	200	226	67	699
	Minimum	0.25	0.341	0.331	0.25	0.25
	Maximum	467	193.0	221.9	131.8	467.0
	Mean	**60.4**	**21.0**	**27.5**	**18.0**	**34.4**
	Std Dev	85.24	32.85	39.5	24.65	57.48
	Variance	7265.28	1079.08	1556.4	607.70	3304.32
	Coeff Var	1.41	1.57	1.44	1.37	1.67
Copper %	Number	206	200	226	67	699

Domain		100	200	300	400	combined
	Minimum	0.005	0.023	0.006	0.006	0.005
	Maximum	20.70	20.02	18.51	6.51	20.70
	Mean	**2.33**	**2.76**	**2.15**	**1.14**	**2.28**
	Std Dev	4.14	3.99	3.32	1.43	3.68
	Variance	17.12	15.89	11.02	2.03	13.54
	Coeff Var	1.77	1.45	1.55	1.25	1.61
Lead %	Number	206	200	226	67	699
	Minimum	0.004	0.004	0.002	0.001	0.001
	Maximum	4.784	3.16	7.59	3.12	7.59
	Mean	**0.75**	**0.16**	**0.42**	**0.30**	**0.43**
	Std Dev	1.02	0.32	0.93	0.60	0.84
	Variance	1.03	0.10	0.87	0.36	0.70
	Coeff Var	1.35	1.97	2.21	1.98	1.93
Zinc %	Number	206	200	226	67	699
	Minimum	0.02	0.01	0.01	0.008	0.008
	Maximum	49.72	11.97	25.81	26.01	49.72
	Mean	**5.00**	**1.63**	**3.33**	**3.08**	**3.31**
	Std Dev	7.10	2.27	5.06	6.02	5.45
	Variance	50.46	5.14	25.58	36.27	29.75
	Coeff Var	1.42	1.39	1.52	1.95	1.65
Density	Number	206	189	215	63	673
	Minimum	2.17	2.48	2.39	2.73	2.17
	Maximum	4.58	4.39	4.44	4.17	4.58
	Mean	3.37	3.27	3.23	3.03	3.26
	Std Dev	0.65	0.48	0.52	0.33	0.55
	Variance	0.42	0.23	0.27	0.11	0.30
	Coeff Var	0.19	0.15	0.16	0.11	0.17

Table 32. Summary Statistics Uncut1m Run Length Composites – Auxillary Variables

Domain		100	200	300	400	combined
Arsenic ppm	Number	152	200	202	67	621
	Minimum	27.00	8.16	35.00	8.00	8.00
	Maximum	5,824.49	5,531.62	4,181.00	2,083.17	5,824.49
	Mean	694.02	610.02	516.38	225.49	558.63
	Std Dev	938.40	837.44	669.54	324.92	785.18
	Variance	880,603.79	701,300.66	448,286.84	105,574.14	616,505.83
	Coeff Var	1.35	1.37	1.30	1.44	1.41
Barium ppm	Number	152	200	202	67	152
	Minimum	5.00	23.72	5.00	10.30	5.00
	Maximum	20,000.00	20,000.00	20,000.00	3,982.67	20,000.00
	Mean	1,132.79	3,052.11	1,885.78	303.78	1,132.79
	Std Dev	3,780.27	5,690.21	4,025.36	568.26	3,780.27
	Variance	14,290,413	32,378,479	16,203,548	322,917	20,077,706
	Coeff Var	3.34	1.86	2.13	1.87	3.34
Iron %	Number	152	200	202	67	621

Domain		100	200	300	400	combined
	Minimum	0.74	2.00	0.00	2.00	0.00
	Maximum	25.09	36.00	39.31	21.23	39.31
	Mean	5.81	13.82	10.70	7.30	10.14
	Std Dev	4.27	7.88	7.82	3.54	7.46
	Variance	18.20	62.09	61.18	12.55	55.65
	Coeff Var	0.73	0.57	0.73	0.49	0.74
Sulphur %	Number	172	194	222	67	655
	Minimum	1.54	1.24	0.95	0.01	0.01
	Maximum	40.85	46.03	43.04	38.72	46.03
	Mean	14.56	18.11	14.99	10.36	15.33
	Std Dev	9.85	9.94	9.57	7.67	9.84
	Variance	96.95	98.84	91.49	58.80	96.76
	Coeff Var	0.68	0.55	0.64	0.74	0.64
Antimony ppm	Number	152	200	202	67	621
	Minimum	2.50	2.50	2.50	2.25	2.25
	Maximum	1,529.83	839.50	1,300.69	850.16	1,529.83
	Mean	131.89	67.19	89.47	77.46	91.38
	Std Dev	253.98	105.68	157.42	145.45	174.19
	Variance	64,505.23	11,168.98	24,781.01	21,156.64	30,341.45
	Coeff Var	1.93	1.57	1.76	1.88	1.91

The coefficient of variation (CV - calculated by dividing the standard deviation by the mean grade) is moderate, with Zinc, Gold, Silver, Copper and Lead returning very similar CVs for both the individual mineralized wireframes and for the combined data set. As a result, there is a limited requirement for upper cuts prior to grade estimation.

14.5.6 High Grade Outlier Analysis

Based on a statistical review, high-grade cuts were determined, although limited cutting of the composites was required. The high-grade cuts applied are detailed in Table 33. In all instances where top-cutting was required, the cuts are light and only affecting the top one or two highest grade composites in the datasets. As such, mean grades are generally only slightly affected.

Table 33. High Grade Cuts

Domain	100	200	300	400
Au g/t	30	12	20	12
Ag g/t	-	160	150	65
Cu %	-	-	-	5.5
Pb %	-	1.8	-	-
Zn %	36	-	-	-
Ba ppm	15,000	15,000	15,000	-
Sb ppm	1,200	-	1,200	-

14.5.7 Cut and Declustered Composite Descriptive Statistics

The sample dataset has also been declustered to examine the possibility of spatial bias present in the dataset where for example higher-grade areas may have been preferentially drilled. There is no finding of any such bias and the converse may be true as it has been determined that in some instances the mean grade of many of the elements of interest increase over a wide range of cell declustering sizes. Cut run-length summary statistics for the main variables and density are displayed in Table 34, top cut and declustered sample statistics are presented for the main variables and density in Table 35 and top cut and declustered sample statistics are presented for the auxillary variables in Table 36.

Multivariate correlation coefficients for all estimated elements have been determined on a domain by domain and global (all mineralised domains combined) basis. The global coefficients are detailed in Table 37.

Figures 24, 25, 26, 27, 28 and 29 display the histograms and log histogram plots for Au, Ag, Cu, Pb, Zn and density respectively. Note the statistical plots and statistical summarises are based on the combined composite data prior to high-grade cuts being applied. All elements display relatively well behaved, positively skewed, lognormal distributions with the exception of density which displays a marked bimodal mode of distribution which may potentially be attributed to different styles of mineralisation (massive/semi massive etc).

Table 34. Summary Statistics - Top-cut1m Run Length Composites – Main Variables

		100	200	300	400	combined
Gold g/t	Number	206	200	226	67	699
	Minimum	0.137	0.12	0.066	0.003	0.003
	Maximum	30.00	12.00	20.00	12.00	30.00
	Mean	**4.07**	**1.76**	**2.42**	**1.96**	**2.67**
	Std Dev	5.79	2.04	3.71	3.00	4.16
	Variance	33.47	4.18	13.78	8.99	17.32
	Coeff Var	1.42	1.16	1.53	1.53	1.56
Silver g/t	Number	206	200	226	67	699
	Minimum	0.25	0.341	0.331	0.25	0.25
	Maximum	467.0	160.0	150.0	65.0	467.0
	Mean	**60.4**	**20.8**	**26.7**	**16.7**	**34.0**
	Std Dev	85.2	32.1	36.4	20.5	56.6
	Variance	7265.3	1027.3	1321.5	419.9	3204.4
	Coeff Var	1.41	1.54	1.36	1.22	1.67
Copper %	Number	206	200	226	67	699
	Minimum	0.005	0.023	0.006	0.006	0.005
	Maximum	20.70	20.02	18.51	5.50	20.70
	Mean	**2.33**	**2.76**	**2.15**	**1.13**	**2.279**
	Std Dev	4.14	3.99	3.32	1.37	3.68
	Variance	17.12	15.89	11.02	1.89	13.53

		100	200	300	400	combined
	Coeff Var	1.77	1.45	1.55	1.22	1.61
	Number	206	200	226	67	699
	Minimum	0.004	0.004	0.002	0.001	0.001
	Maximum	4.78	1.80	7.59	3.12	7.59
Lead %	Mean	**0.75**	**0.16**	**0.42**	**0.30**	**0.43**
	Std Dev	1.02	0.27	0.94	0.60	0.83
	Variance	1.03	0.07	0.87	0.36	0.70
	Coeff Var	1.36	1.72	2.21	1.98	1.93
	Number	206	200	226	67	699
	Minimum	0.02	0.01	0.01	0.008	0.008
	Maximum	49.72	11.97	25.81	26.01	49.72
Zinc %	Mean	**5.00**	**1.63**	**3.33**	**3.09**	**3.31**
	Std Dev	7.10	2.27	5.06	6.02	5.45
	Variance	50.46	5.14	25.58	36.27	29.75
	Coeff Var	1.42	1.39	1.52	1.95	1.65

Table 35. Summary Statistics – Topcut and Declustered1m Run Length Composites – Main Variables

		100	200	300	400	combined
	Number	206	200	226	67	699
	Minimum	0.137	0.12	0.066	0.003	0.003
	Maximum	30.00	12.00	20.00	12.00	30.00
Gold g/t	Mean	**4.18**	**2.02**	**2.26**	**1.65**	**2.83**
	Std Dev	5.79	2.31	3.46	2.68	4.31
	Variance	33.47	5.32	11.96	7.16	18.59
	Coeff Var	1.39	1.14	1.53	1.62	1.52
	Number	206	200	226	67	699
	Minimum	0.25	0.341	0.331	0.25	0.25
	Maximum	467.0	160.0	150.0	65.0	467.0
Silver g/t	Mean	**63.9**	**26.0**	**26.3**	**14.9**	**38.4**
	Std Dev	83.6	36.8	34.3	19.3	59.2
	Variance	6989.7	1350.9	1178.5	372.8	3501.2
	Coeff Var	1.31	1.41	1.31	1.30	1.54
	Number	206	200	226	67	699
	Minimum	0.005	0.023	0.006	0.006	0.005
	Maximum	20.70	20.02	18.51	5.50	2070
Copper %	Mean	**2.12**	**3.27**	**1.64**	**0.96**	**2.11**
	Std Dev	3.56	4.12	2.66	1.27	3.35
	Variance	12.67	16.95	7.09	1.62	11.20
	Coeff Var	1.68	1.26	1.62	1.32	1.59
	Number	206	200	226	67	699
	Minimum	0.004	0.004	0.002	0.001	0.001
Lead %	Maximum	4.78	1.8	7.59	3.12	7.59
	Mean	**0.81**	**0.19**	**0.47**	**0.26**	**0.51**
	Std Dev	1.11	0.34	1.02	0.54	0.96

		100	200	300	400	combined
	Variance	1.23	0.13	1.03	0.29	0.92
	Coeff Var	1.38	1.73	2.18	2.09	1.87
Zinc %	Number	206	200	226	67	699
	Minimum	0.02	0.01	0.01	0.008	0.008
	Maximum	49.72	11.97	25.81	26.01	49.72
	Mean	**5.19**	**1.79**	**3.12**	**2.52**	**3.51**
	Std Dev	7.77	2.55	4.81	5.43	5.87
	Variance	60.30	6.50	23.12	29.43	34.43
	Coeff Var	1.50	1.42	1.54	2.15	1.67
Density	Number	206	189	215	63	673
	Minimum	2.17	2.48	2.39	2.73	2.17
	Maximum	4.58	4.39	4.44	4.17	4.58
	Mean	**3.37**	**3.35**	**3.14**	**3.00**	**3.26**
	Std Dev	0.65	0.47	0.48	0.30	0.55
	Variance	0.42	0.22	0.23	0.09	0.31
	Coeff Var	0.19	0.14	0.15	0.10	0.17

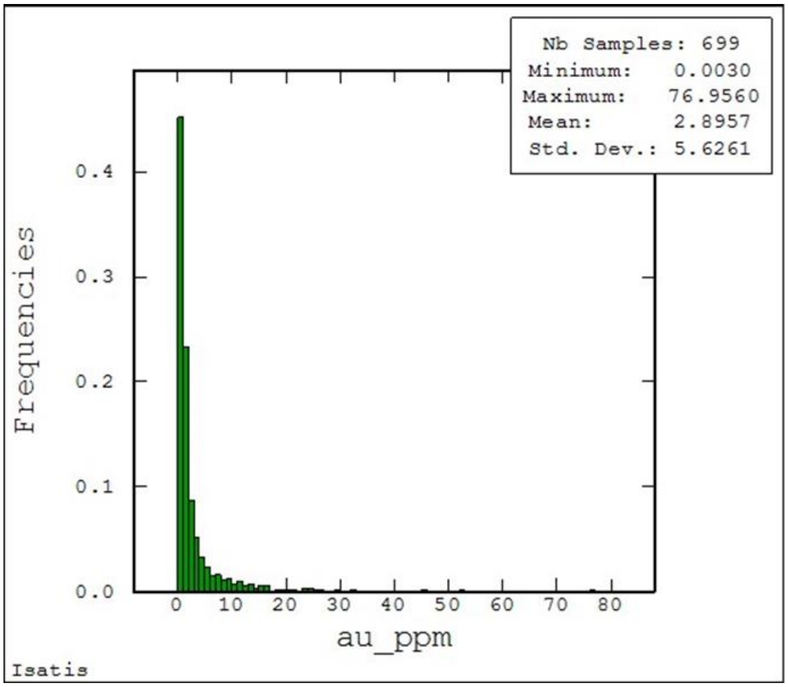
Table 36. Summary Statistics – Topcut and Declustered1m Run Length Composites – Auxillary Variables

		100	200	300	400	combined
Arsenic ppm	Number	152.00	200.00	202.00	67.00	621.00
	Minimum	27.00	8.16	35.00	8.00	8.00
	Maximum	5,824.49	5,531.62	4,181.00	2,083.17	5,824.49
	Mean	**677.29**	**703.48**	**488.06**	**194.25**	**567.22**
	Std Dev	897.34	870.49	616.72	284.37	760.45
	Variance	805,220	757,752	380,346	80,866	578,287
	Coeff Var	1.32	1.24	1.26	1.46	1.34
Barium ppm	Number	152.00	200.00	202.00	67.00	621.00
	Minimum	5.00	23.72	5.00	10.30	5.00
	Maximum	15,000.00	15,000.00	15,000.00	3,982.67	15,000.00
	Mean	**729.83**	**3,748.71**	**2,181.99**	**263.01**	**2,068.14**
	Std Dev	2,603.59	5,171.66	3,889.63	510.65	4,028.37
	Variance	6,778,660	26,746,052	15,129,244	260,766	16,227,775
	Coeff Var	3.57	1.38	1.78	1.94	1.95
Iron %	Number	152.00	200.00	202.00	67.00	621.00
	Minimum	0.74	2.00	-	2.00	-
	Maximum	25.09	36.00	39.31	21.23	39.31
	Mean	**5.96**	**14.91**	**9.90**	**7.03**	**9.95**
	Std Dev	4.21	8.14	6.51	3.41	7.11
	Variance	18	66	42	12	51
	Coeff Var	0.71	0.55	0.66	0.49	0.71
Sulphur %	Number	172.00	194.00	222.00	67.00	655.00
	Minimum	1.54	1.24	0.95	0.01	0.01
	Maximum	40.85	46.03	43.04	38.72	46.03
	Mean	**14.73**	**19.29**	**13.98**	**9.41**	**15.09**

		100	200	300	400	combined
	Std Dev	9.54	9.83	8.38	7.25	9.38
	Variance	91.05	96.70	70.20	52.58	87.99
	Coeff Var	0.65	0.51	0.60	0.77	0.62
Antimony ppm	Number	152.00	200.00	202.00	67.00	621.00
	Minimum	2.50	2.50	2.50	2.25	2.25
	Maximum	1,200.00	839.50	1,200.00	850.16	1,200.00
	Mean	**148.60**	**79.65**	**85.13**	**62.27**	**97.95**
	Std Dev	265.10	113.46	145.63	129.82	178.76
	Variance	70,280	12,874	21,208	16,852	31,955
	Coeff Var	1.78	1.42	1.71	2.08	1.83

Table 37. Global Multivariate Correlation Coefficients – Topcut and Declustered1m Run Length Composites

	Au	Ag	As	Ba	Cu	Fe	Pb	S	Sb	Zn	Density
Au	1.00	**0.56**	0.43	-0.02	0.22	-0.11	**0.73**	0.25	**0.65**	**0.63**	**0.52**
Ag	**0.56**	1.00	**0.62**	0.03	0.37	-0.02	0.47	0.31	**0.63**	**0.65**	**0.50**
As	0.43	**0.62**	1.00	0.00	**0.59**	0.29	0.25	0.49	**0.63**	0.37	**0.52**
Ba	-0.02	0.03	0.00	1.00	0.02	0.03	0.02	-0.02	-0.04	-0.06	-0.05
Cu	0.22	0.37	**0.59**	0.02	1.00	0.48	0.02	**0.62**	0.29	0.17	**0.59**
Fe	-0.11	-0.02	0.29	0.03	0.48	1.00	-0.23	**0.82**	-0.02	-0.13	0.46
Pb	**0.73**	0.47	0.25	0.02	0.02	-0.23	1.00	0.11	**0.61**	**0.59**	0.37
S	0.25	0.31	0.46	-0.02	**0.62**	**0.82**	0.11	1.00	0.32	0.33	**0.82**
Sb	**0.65**	**0.63**	**0.63**	-0.04	0.29	-0.02	**0.61**	0.32	1.00	**0.77**	0.49
Zn	**0.63**	**0.65**	0.37	-0.06	0.17	-0.13	**0.59**	0.33	**0.77**	1.00	**0.51**
Density	**0.52**	**0.50**	0.52	-0.05	**0.59**	0.46	0.37	**0.82**	0.49	0.51	1.00



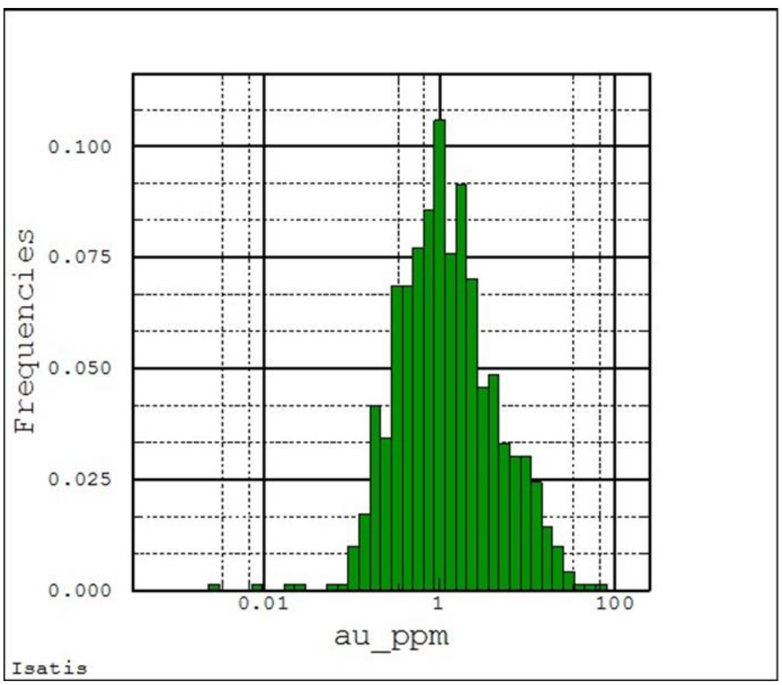


Figure 24. Distribution Statistics – Au ppm (1m Composites)

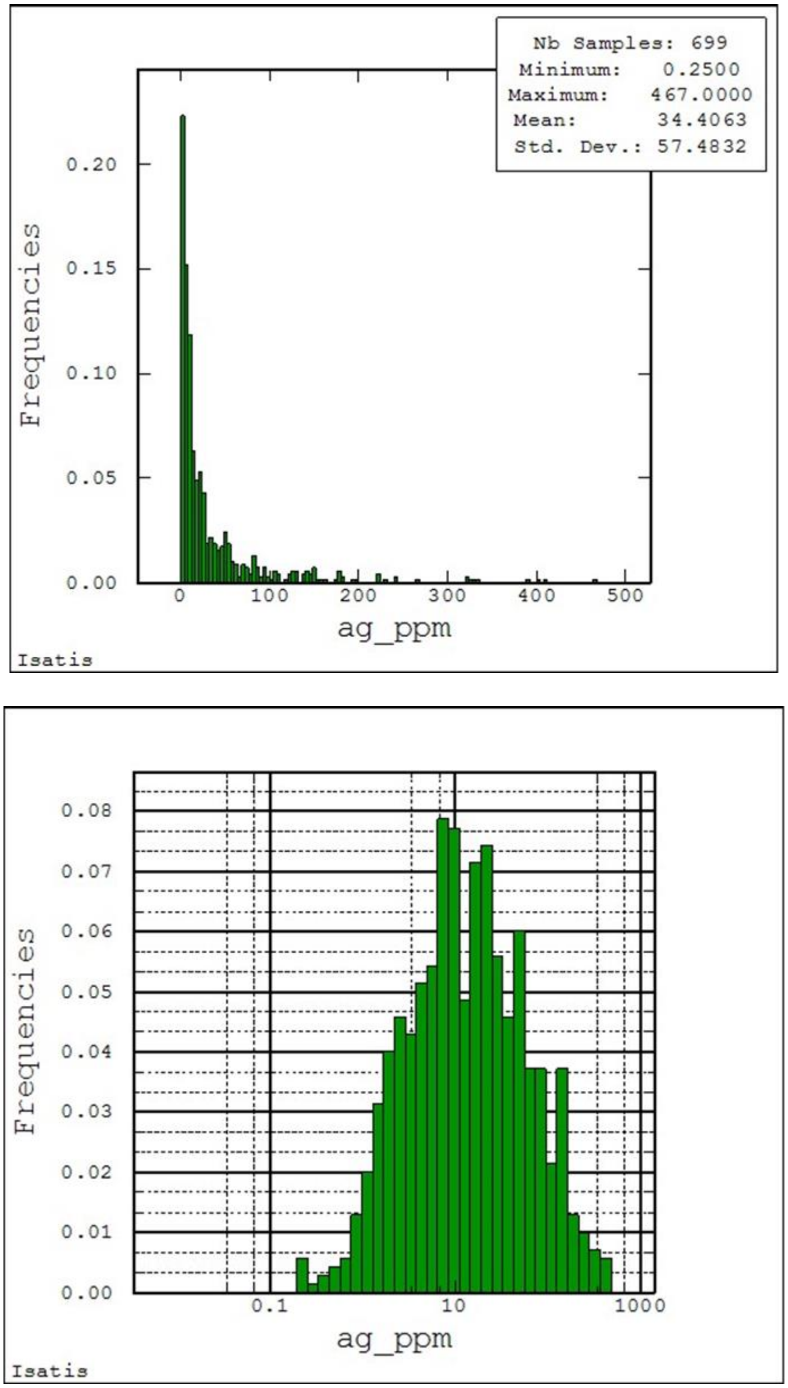


Figure 25. Distribution Statistics – Ag g/t (1m Composites)

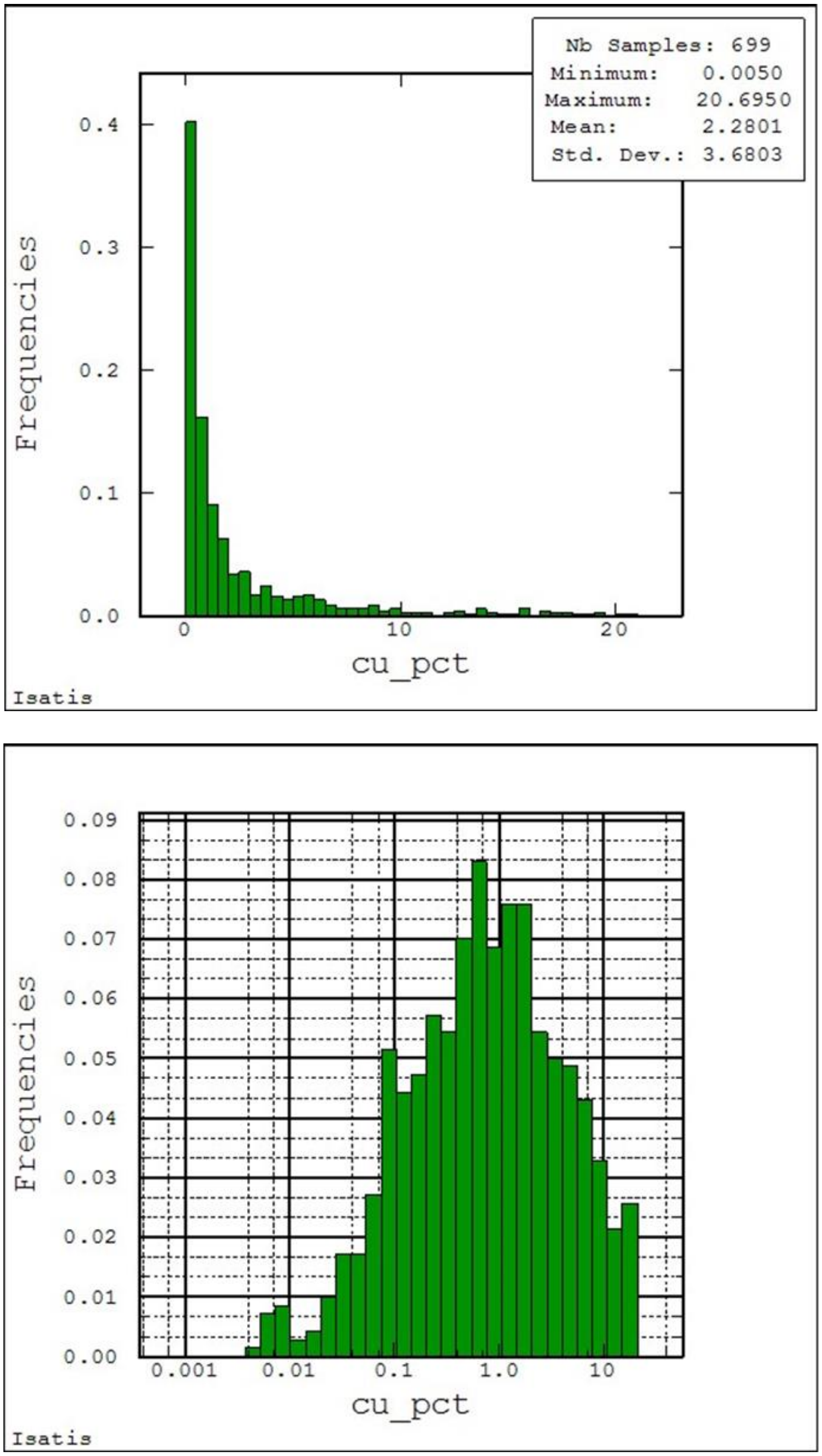


Figure 26. Distribution Statistics – Cu % (1m Composites)

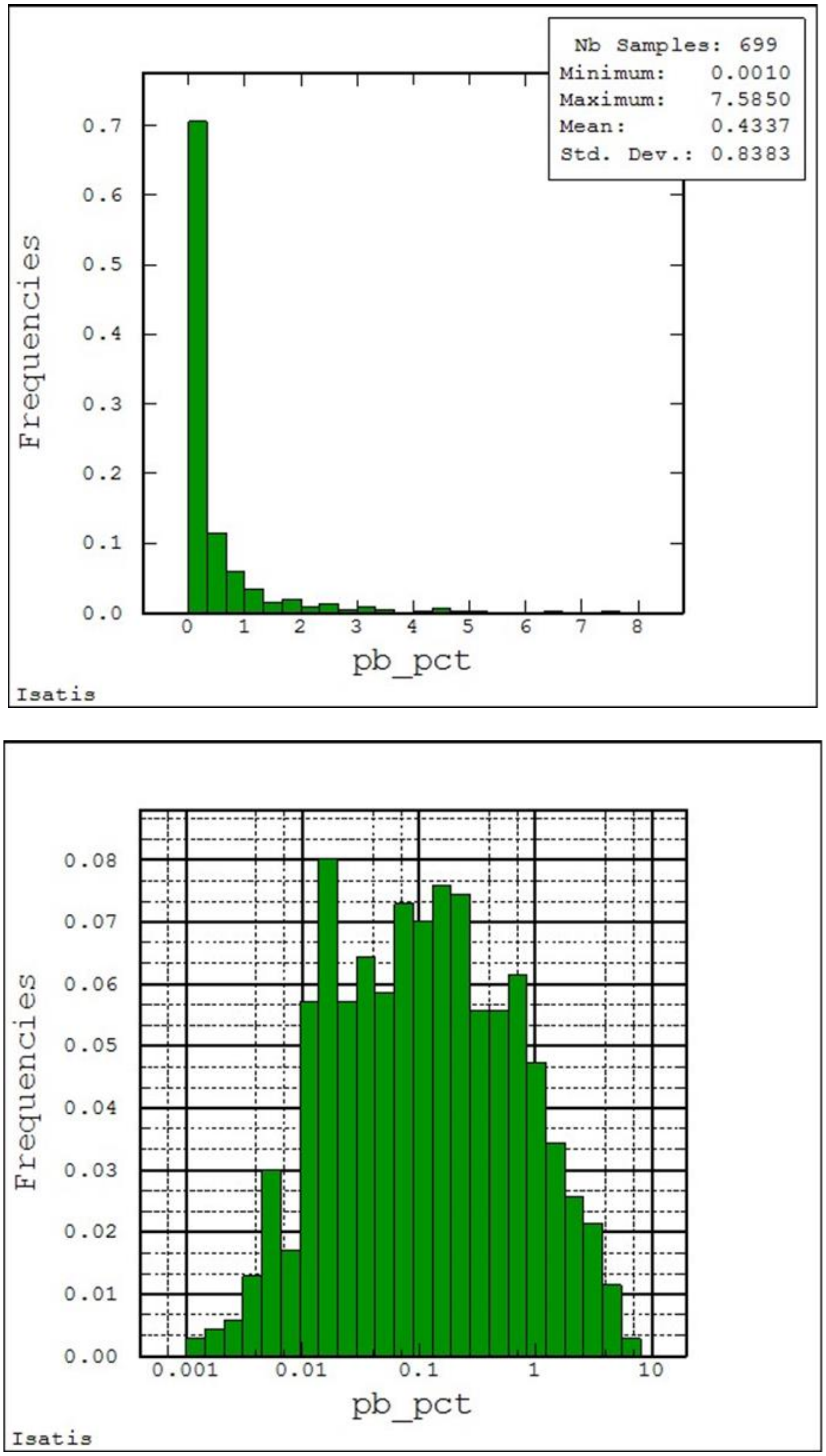


Figure 27. Distribution Statistics – Pb % (1m Composites0

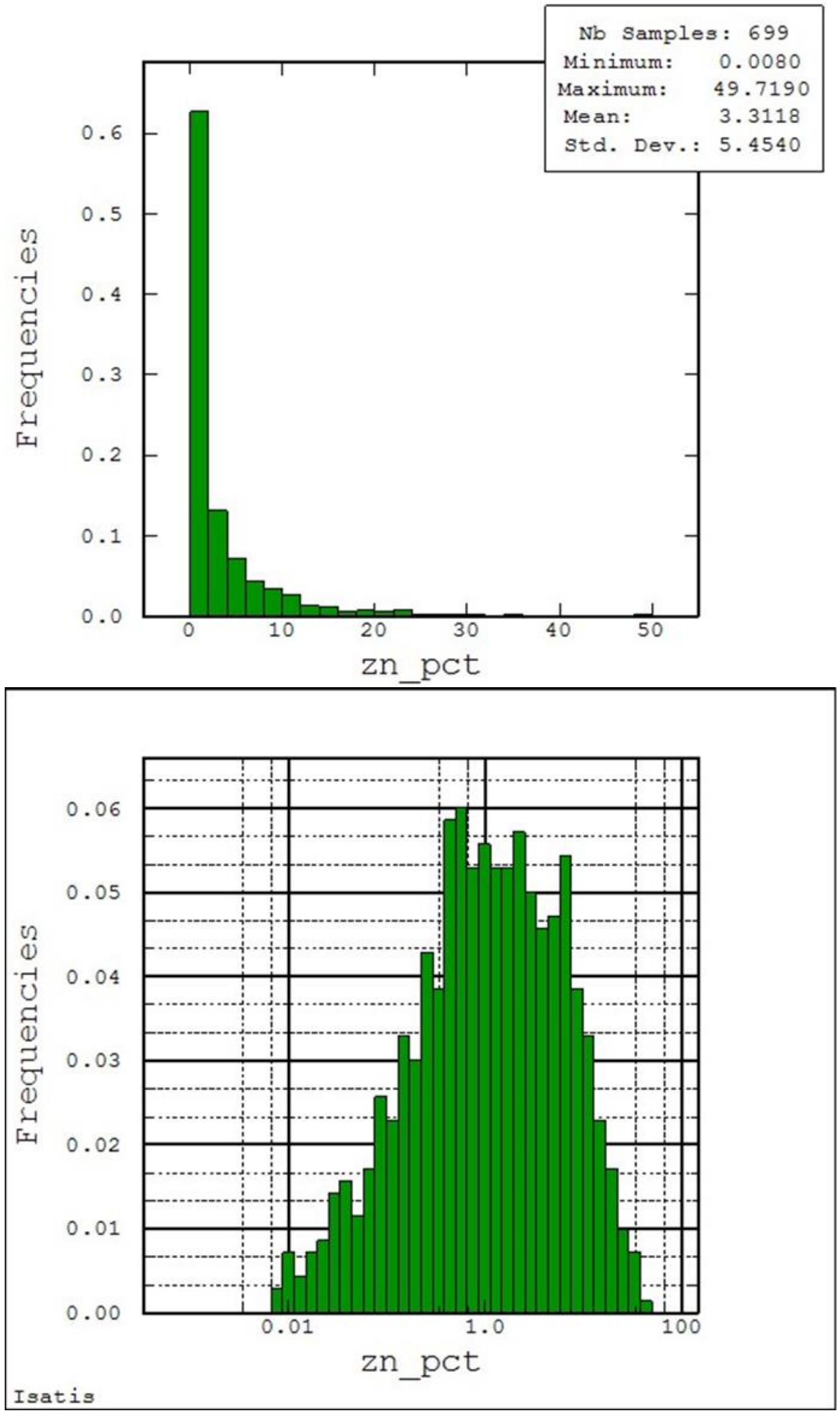
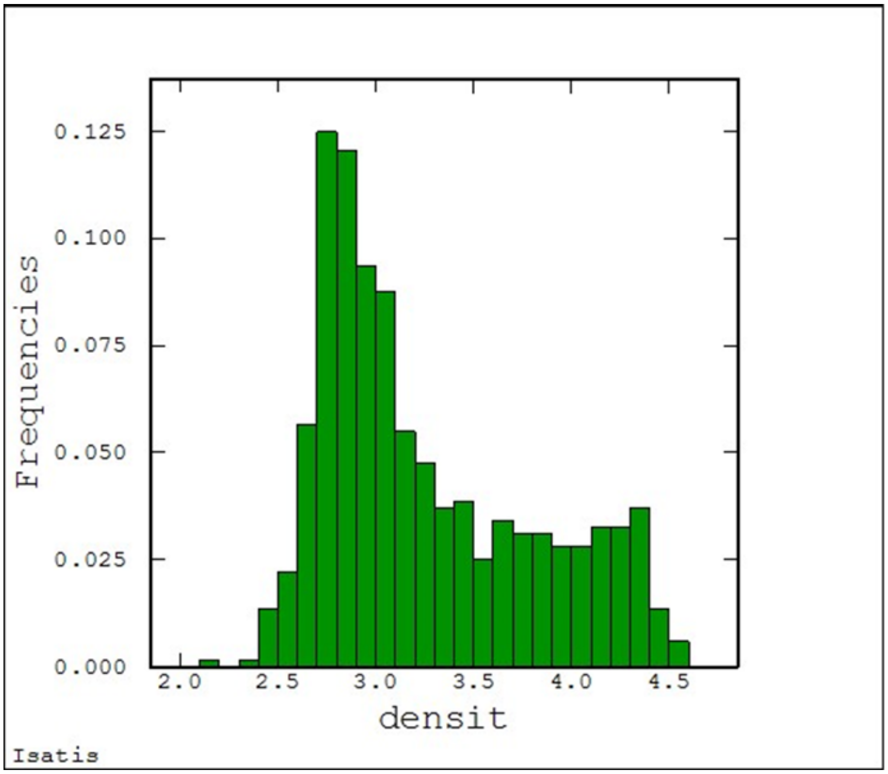


Figure 28. Distribution Statistics – Zn % (1m Composites



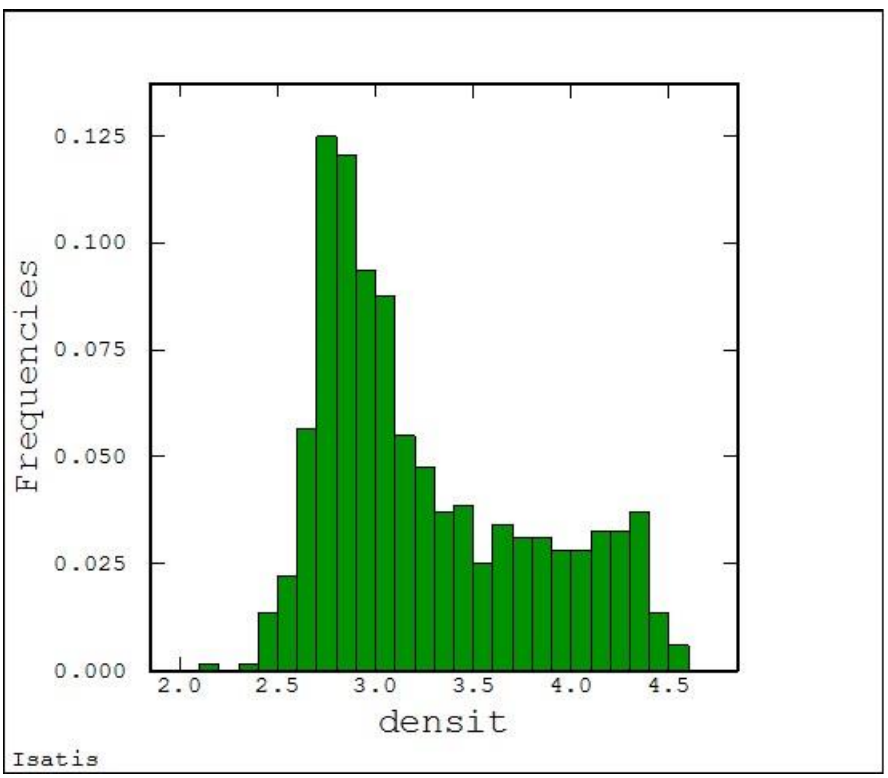


Figure 29. Distribution Statistics – Density (1m composites)

14.6 Variography

Variography is used to describe the spatial variability or correlation of an attribute (gold, silver etc.). The spatial variability is traditionally measured by means of a variogram, which is generated by determining the averaged squared difference of data points at a nominated distance (h), or lag (Srivastava and Isaacs, 1989). The averaged squared difference (variogram or $\gamma(h)$) for each lag distance is plotted on a bivariate plot, where the X-axis is the lag distance and the Y-axis represents the average squared differences ($\gamma(h)$) for the nominated lag distance.

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

- Traditional variograms are calculated from the raw assay values.
- Log-transformed variography involves a logarithmic transformation of the assay data.
- Gaussian variograms are based on the results after declustering and a transformation to a Normal distribution.
- Pairwise-relative variograms attempt to 'normalise' the variogram by dividing the variogram value for each pair by their squared mean value.
- Correlograms are 'standardized' by the variance calculated from the sample values that contribute to each lag.

Fan variography involves the graphical representation of spatial trends by calculating a range of variograms in a selected plane and contouring the variogram values. The result is a contour map of the grade continuity within the domain.

The variography was calculated and modelled in the geostatistical software, Isatis. The rotations are tabulated as dip and dip direction of major, semi-major and minor axes of continuity.

A variographic analysis was completed on the coded composite data for the resource estimate study. Composite data from all domains has been combined for variographic analysis. Note that, as the samples have been 'flattened' or 'unfolded' for the purposes of grade estimation, the variograms are presented in the direction of optimal continuity, as determined from fan variography calculated in the flattened plane. A principal orientation of 15° has been determined with the orthogonal directions to this being 105° and vertical. Assumptions and parameters are presented in the following paragraphs.

Traditional semi-variograms were selected as the appropriate spatial measure given the spatial distribution of the data and the sample statistics.

Semi-variograms were generated in the direction of interpreted continuity and are presented as Figures 30 through to 35 for Au, Ag, Cu, Pb, Zn and density respectively. Models are presented in Table 38.

Table 38. Summary Variography Models – 1m Composites

Var	Nugget (C0)	orientation			C1	Range (a1)			C2	Range (a2)		
		Major	Semi major	Minor		Major	Semi	Minor		Major	Semi	Minor
Au	6	0°→15°	0°→105°	90°→0°	4	40	30	10	8.6	90	50	18
Ag	650	0°→15°	0°→105°	90°→0	1,350	40	20	10	1,500	100	50	18
Cu	1.1	0°→15°	0°→105°	90°→0	5.05	50	30	10	5.05	100	60	18
Pb	0.16	0°→15°	0°→105°	90°→0	0.25	50	40	8	0.50	110	70	15
Zn	4	0°→15°	0°→105°	90°→0	13.5	40	40	8	12.2	100	60	20
Density	0.05	0°→15°	0°→105°	90°→0	0.1	55	40	12	0.15	110	80	20
As	40,000	0°→15°	0°→105°	90°→0	200,000	50	40	12	335,000	110	70	20
Ba	2,007,770	0°→15°	0°→105°	90°→0	5,746,700	20	16	10	8,479,800	110	110	20
Sb	3,000	0°→15°	0°→105°	90°→0	12,100	40	30	10	12,500	100	60	20
S	10	0°→15°	0°→105°	90°→0	38	40	30	14	40	100	70	20
Fe	6	0°→15°	0°→105°	90°→0	21	55	40	12	12	110	80	20

The variograms are only moderately well-structured and therefore considered moderate confidence. This is a reflection of the data spatial distribution and the sometimes-extreme changes in grade between adjacent drill holes.

Nugget effects have been modelled as low to moderate and are in keeping with the style of mineralisation at La Plata. Extended ranges of 90m to 110m have been fitted in the major axis. Shorter ranges of between 50m and 70m have been fitted for the semi-major and the minor axis has been modelled with 15m to 20m ranges. Short-range variability (nugget variance and the close-range structure) dominates the variogram. The following relative nugget values (nugget divided by the total sill) have been returned:

- Au: 33%
- Ag: 19%
- Cu: 10%
- Pb: 18%
- Zn: 13%

As is typical, Au shows the highest level of close range variability (relative nugget of 33%), whilst the base metals and silver returned materially lower relative nugget levels.

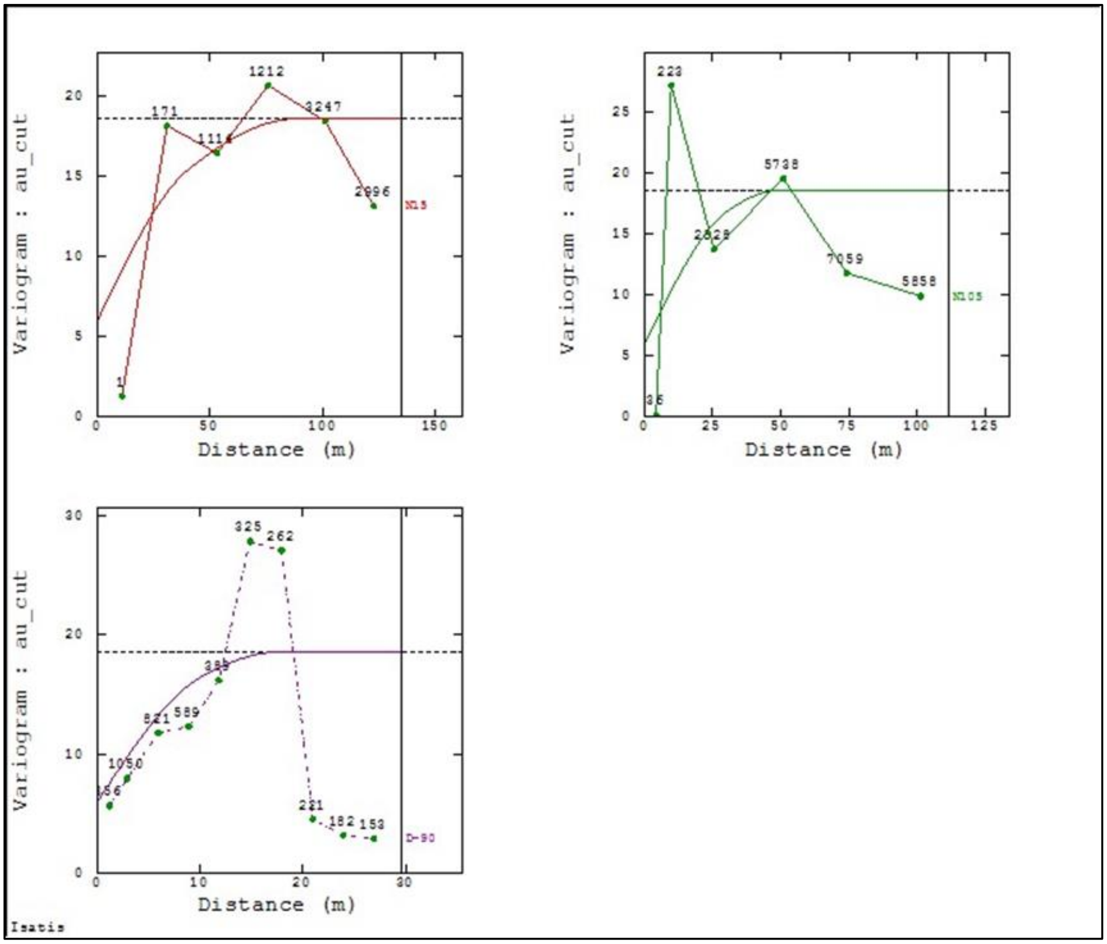


Figure 30. Au variogram

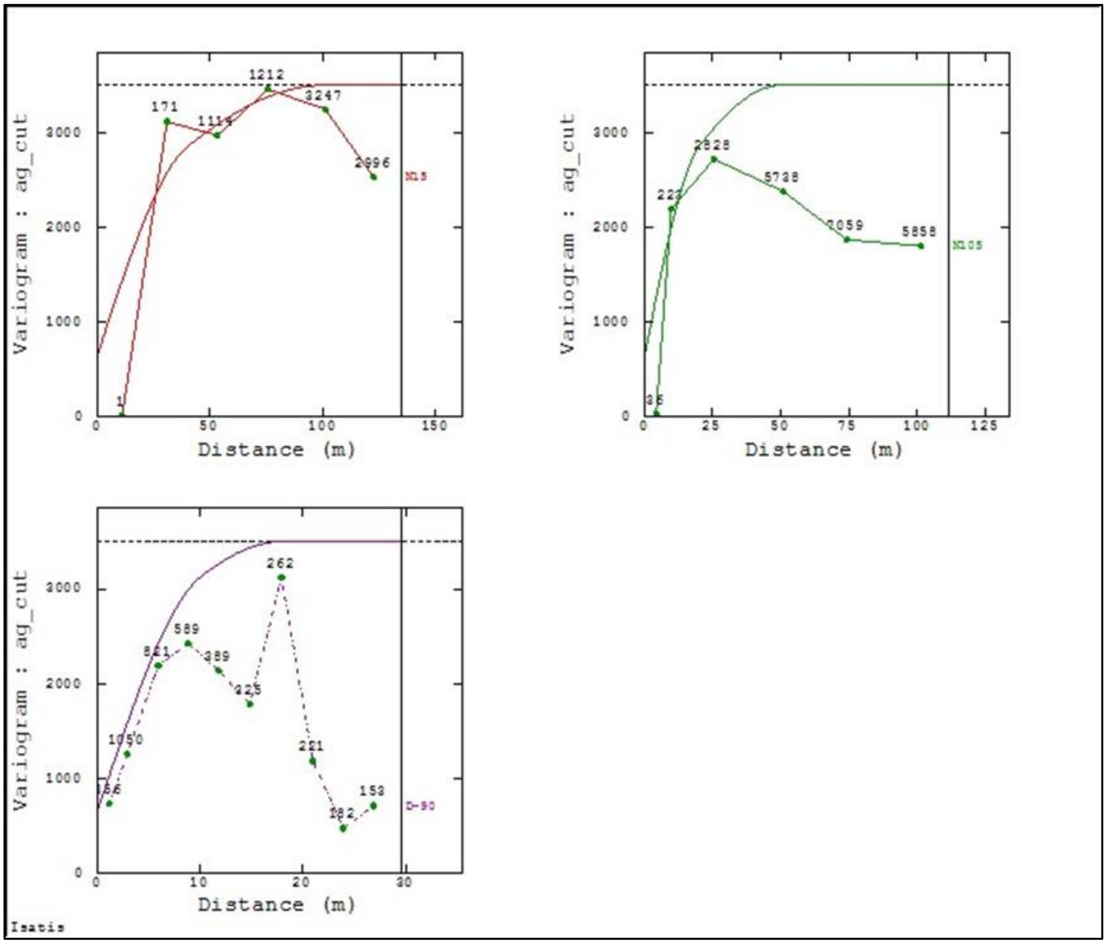


Figure 31. Ag variogram

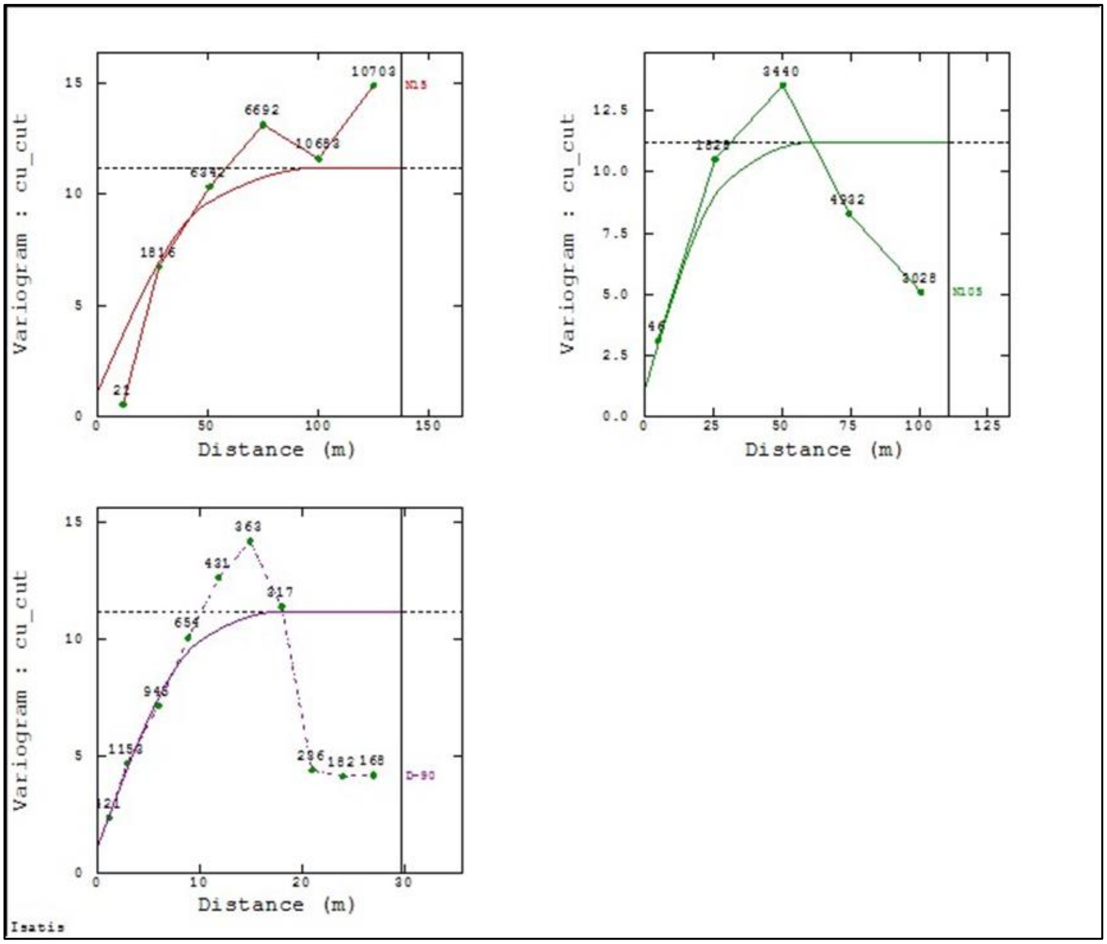


Figure 32. Cu variogram

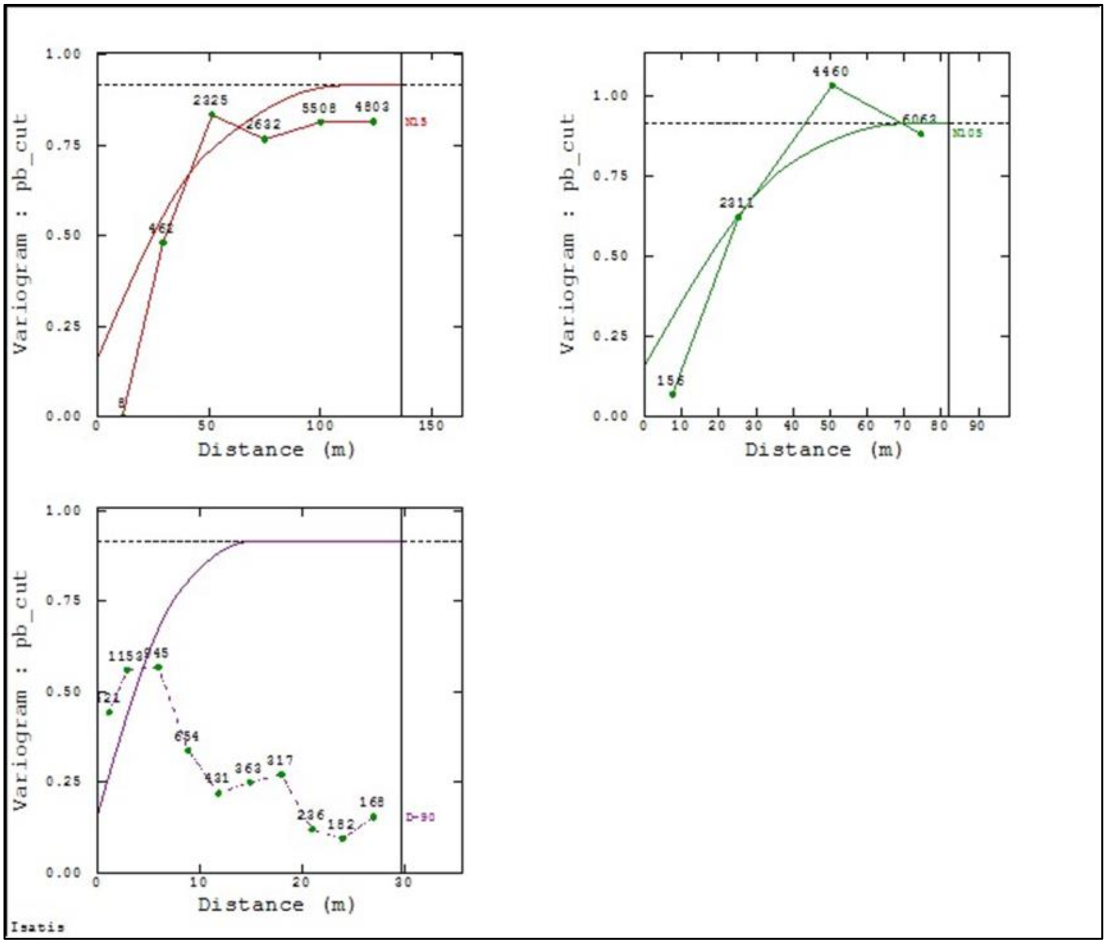


Figure 33. Pb variogram

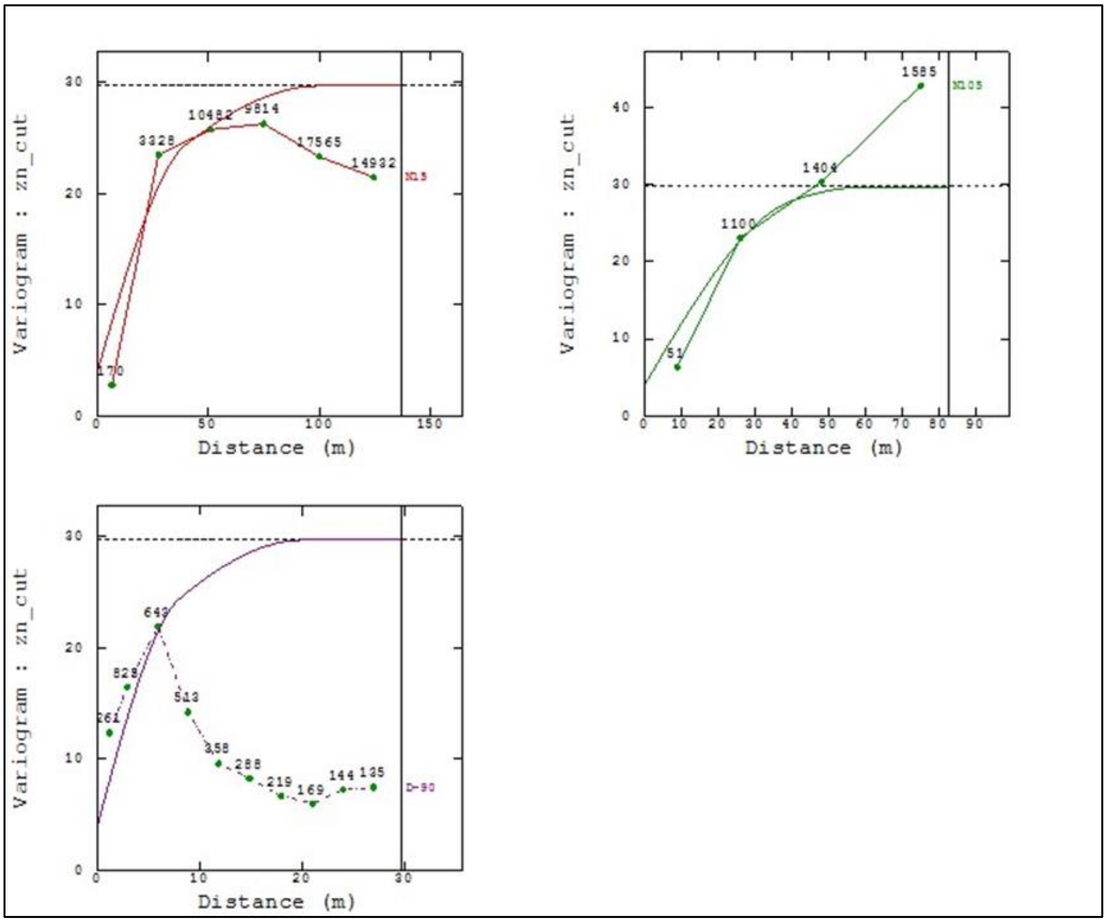


Figure 34. Zn variogram

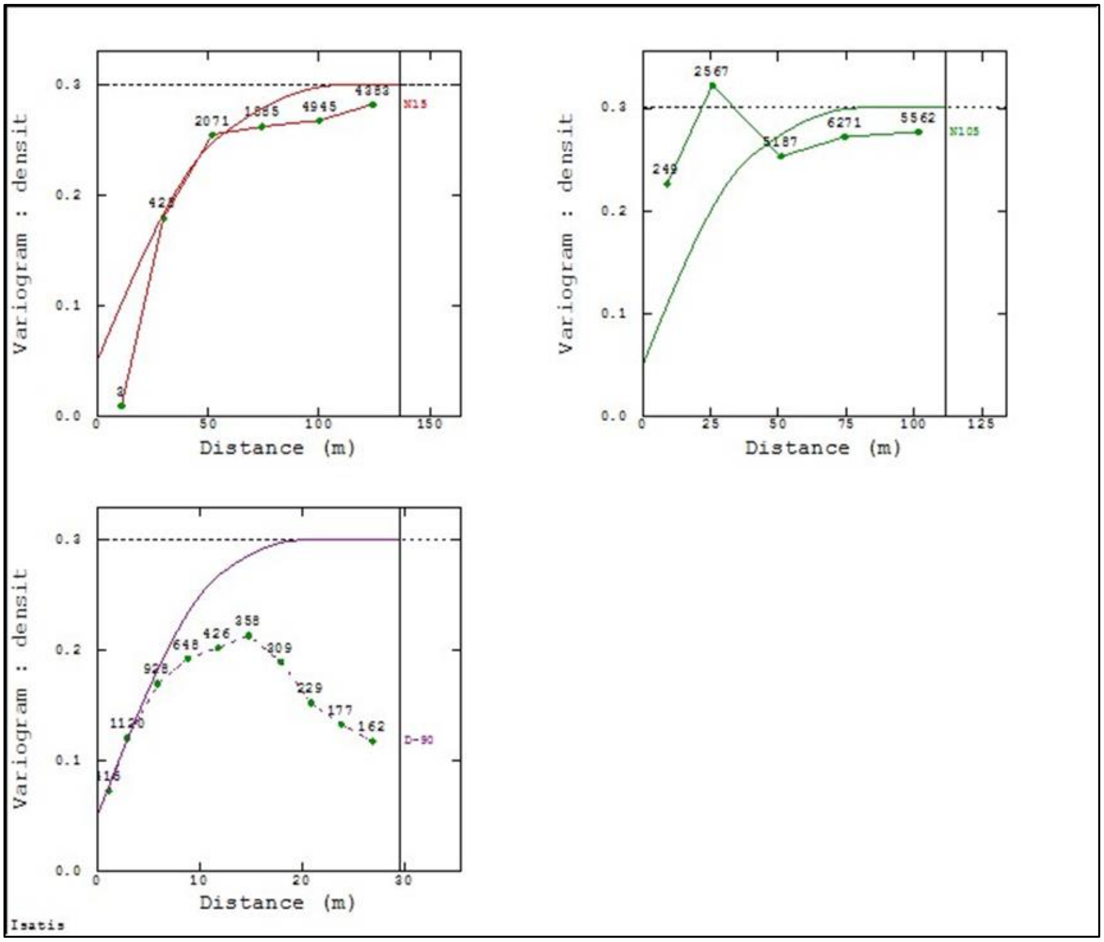


Figure 35. Density variogram

14.7 Block Modelling

The grade estimation was completed into a sub-blocked block models with a 5m x 10m x 5m cell size sub-blocked to 1m in all 3 directions as summarised in Table 39. No rotation has been applied. Block model variables were added to enable OK estimates of all variable of interest including auxiliary variables and density. Service variables were also added to record various estimation parameters to review the quality of the estimates.

The model has been at least partially depleted to account for historic production by the void and topo wireframes received from Toachi. Based on plans provided by Toachi, the likely depletion is moderately significant versus the total model however the capture and modelling of all development and stoping is considered critical to high confidence mine planning.

Table 39. Block Model Construction Parameters

	Min (m)	Max (m)	Extent (m)	Parent/subcell Block Size
Easting	728,800	729,500	700	5/1
Northing	9,956,600	9,957,800	1,200	10/1
Elevation	1,100	1,650	550	5/1

14.8 Bulk Density Data

A dry bulk density database has been supplied containing a total of 2,638 data. The database can be subdivided on the basis of flagging by the various mineralisation wireframes and a total of 621 density determinations correspond to flagged mineralised intervals. Descriptive statistics are discussed in Section 14.10 and variograms have been calculated and modelled (Section 14.11). Densities have been assigned to the block model via Ordinary Kriging (OK) for tonnage reporting purposes and this is discussed in the following section.

14.9 Grade Estimation

An ordinary kriging (OK) grade estimation plan was devised which utilised hard boundaries throughout. For each metal the domain coded material was estimated first followed by material within a halo to the VMS domains in which the metal distribution and model of occurrence is less well understood.

For the purposes of the grade estimate, the coded domains and corresponding composites were unfolded to form a flat surface via the tetra modelling function in the Vulcan software package. Both a hangingwall and a footwall surface to the mineralisation were constructed prior to the estimation. The unfolding algorithm then adjusts the RLs so that a transformed thickness is assumed to be 1. In this manner, the search neighbourhood ellipse is 'folded' or 'bent' in cartesian space to follow the contours of the mineralisation domain and thereby correctly estimates the grades both locally and spatially. Once the estimate is complete, the unfolded coordinates are transformed back into the original coordinates. In the case of La Mina, where extreme grade fluctuations can exist both between adjacent drill holes and within individual drill holes, this approach is considered necessary to preserve the spatial distribution of grades within the various domains.

A number of grade estimate scenarios were performed, with the accepted grade estimate based on using a minimum of 2 drill holes and 6 input composite samples. Three estimation passes were made for each variable estimated, with the sample search ranges expanded for each successive pass not estimated by the previous estimates and reducing the minimum number of input composites. The estimation parameters are summarised in Table 40. Note for the purposes of the unfolding implemented during the estimation the minor search radii was set to 0.3 for the first pass and 5 for the second and this relates to the unfolded coordinate system and not the cartesian coordinate system. The third estimate pass was executed in cartesian space as there is a significant time penalty inherent in the unfolded estimates as the search neighbourhoods increase in size due to the complexity of the mathematics involved. Additionally, the first and second estimation passes informed more than 95% of the blocks and the third pass can be considered negligible in terms of the total grade tonnage distribution and was only carried out to ensure that all blocks within the wireframes were estimated.

Table 40. Estimation Parameters

Estimation Pass	Sample Search					Disc (X x Y x Z)
	Orientation (X Y' Z'')	Search Radii (X Y' Z'')	Min	Max	Max Data per Hole	
1	15º/0º/0º	60m x 60m x 0.3	6	6	3	3 x 3 x 2
2	15º/0º/0º	250m x 250m x 5	4	6	-	3 x 3 x 2
3	15º/0º/0º	20m x 250m x 40m	3	6	-	3 x 3 x 2

The input 1m composites had high-grade cuts applied prior to grade estimation with the high-grade cuts based on the statistical analysis described in Section 14. In addition, a high-grade distance restriction was applied to grade estimation of copper in Domain 100 (South Zone). The distance restriction was based on a review of the distribution plots, variography and visual inspection of the composite data in 3D. Grades above the nominated threshold of 15% Cu were not collected for estimation when located in excess of a sample search radii of 20m x 20m x 10m. The distance restriction was applied to limit extrapolation of high-grade copper data into lower grade regions, which was originally evident in block statistics calculated after initial grade estimation runs without distance restrictions.

The OK grade estimates were generated with sample search neighbourhoods designed to minimise the level of smoothing given the degree of grade variation present and the current drill spacing. While this approach is generally not recommended, it is possible to produce an acceptable global estimate that is consistent with the likely global grade-tonnage distribution. However, when applying the restricted kriging approach, significant local variation between the block estimates and the actual block grades is likely. It is considered that, as further drill data is collected, a much greater level of confidence in the grade estimates will be generated.

14.10 Depletion for mining activity

Mining has been undertaken both from open pit operations and underground. Records exist; however, the completeness of these records cannot be verified. Underground workings have been digitally transferred from existing records and the model has been depleted using these via insertion of a block model 'flag'. Additionally, open voids have been encountered in surface drilling and the existence of underground stopes has been inferred from this. These voids have been modelled into likely stope positions and volumes and the model has been additionally depleted. Total depletion of mineralised zones from underground workings is thus 173kt and this tonnage has been excluded from any subsequent grade tonnage reporting in Section 14.18. Underground depletion solids are depicted in Figure 36with stopes as the grey shapes within the mineralised bodies. Note only the north blocks are depleted, as no mining has taken place from the South block.

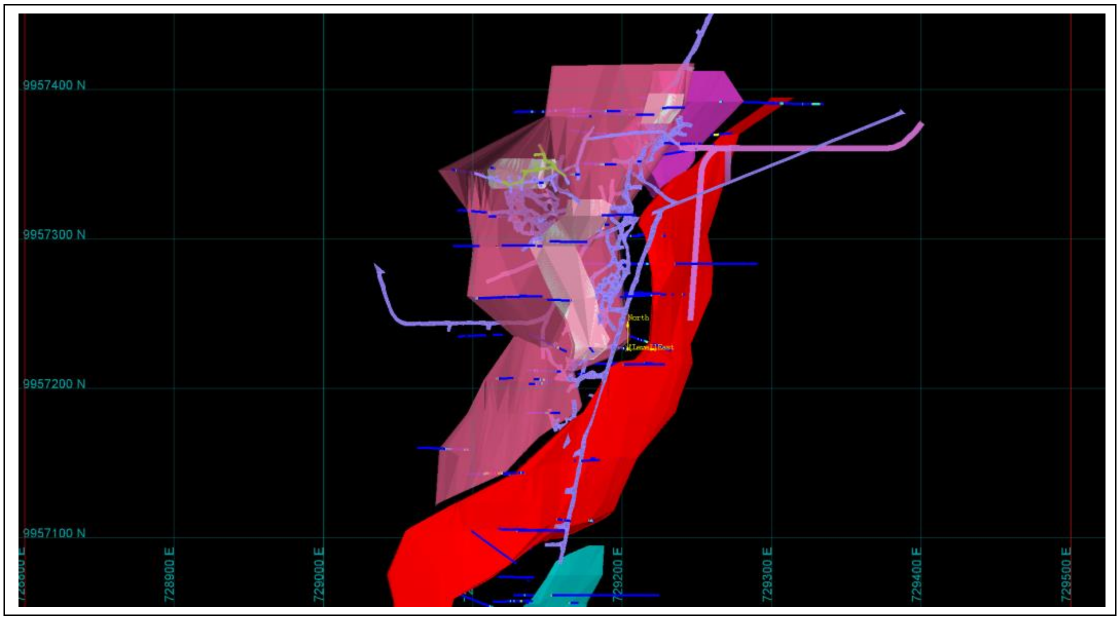


Figure 36. Underground Depletion

14.11 Model Validation

Validation was completed via the following approaches:

- Comparison of the input composite mean grades versus the block grades (as summarised in Table 41). The block model grades are generally very similar to the composite mean grade for the mineralisation interpretation and the block model grades are within 6% of the declustered input composite means. The differences in mean grades are at an acceptable level for the data spatial configuration, the overall number of data points and the proposed resource classification.

- Sections (20m) through the block model comparing block grades and composite grades (swath plots). Good reproduction of the input data was noted with the usual local fluctuation, as is typical.

The grade estimate is considered to be globally robust and reproduces the input data to an acceptable level. The grade estimate is considered acceptable for resource classification and reporting.

Table 41. Comparison of Block Grades with Composite Mean Grades – All Data Used

Variable	Zone	Composites (declustered, capped)	Block Model Grades (Capped and Distance Restriction Estimate)	% Diff Block Model versus Mean
Au (g/t)	100	4.18	4.39	5.0%
	200	2.02	1.90	-6.0%
	300	2.26	2.65	17.3%
	400	1.65	1.98	19.5%
	All	**2.83**	**3.01**	**6.3%**
Ag (g/t)	100	63.9	63.2	-1.2%
	200	26.0	22.3	-14.1%
	300	26.3	28.3	8.0%
	400	14.9	18.3	23.3%
	All	**38.4**	**38.4**	**0.0%**
Cu (%)	100	2.12	2.23	5.6%
	200	3.27	2.83	-13.4%
	300	1.64	1.95	18.8%
	400	0.96	0.90	-7.0%
	All	**2.11**	**2.23**	**5.9%**
Pb (%)	100	0.81	0.77	-4.8%
	200	0.19	0.19	-1.0%
	300	0.47	0.48	1.7%
	400	0.26	0.34	30.4%
	All	**0.51**	**0.49**	**-4.9%**
Zn (%)	100	5.19	4.85	-6.5%
	200	1.79	1.78	-0.5%
	300	3.12	3.27	4.9%
	400	2.52	2.98	18.3%
	All	**3.51**	**3.39**	**-3.5%**

14.12 Resource Classification

The grade estimate was classified in accordance with the current CIM guidelines as an Inferred Mineral Resource based on the confidence levels of the key criteria as presented in Table 42. The resource categorisation was based on the robustness of the various data sources available, including:

- Geological knowledge and interpretation.
- Variogram models and the ranges of the first structure in multi-structure models.
- Drilling density and orientation.
- Estimation quality statistics.

Table 42. Confidence Lavels by Key Criteria

Items	Discussion	Confidence
Drilling Techniques	Diamond - Industry Standard approach. Less certain for previous operators.	Moderate/High
Logging	Standard nomenclature has been adopted. Logging electronically recorded and observed procedures to a high technical standard.	Moderate/High
Drill Sample Recovery	Recoveries are not recorded in entire database. Review of current drilling suggests recoveries are of acceptable standard.	Moderate/High
Sub-sampling Techniques and Sample Preparation	Diamond sampling conducted by industry standard techniques.	Moderate/High
Quality of Assay Data	Appropriate quality control procedures are available. They were reviewed on site and considered to be of industry standard. Previous operators QAQC limited however the available data does not indicate the likelihood of bias occurring.	Moderate/High
Verification of Sampling and Assaying	Sampling and assaying procedures have been assessed and are considered of appropriate industry standards.	Moderate
Location of Sampling Points	Survey of all collars conducted with accurate survey equipment. Investigation of downhole survey indicates appropriate behaviours.	Moderate/High
Data Density and Distribution	Majority of regions defined on a notional 50mE x 50mN drill spacing or better however significant local variation in geological continuity and grade continuity is evident.	low/moderate
Audits or Reviews	no external reviews completed	N/A
Database Integrity	Assay certificates have been verified and no issues were identified.	Moderate
Geological Interpretation	Mineralisation controls are moderately well understood. The mineralisation constraints are robust but relatively broad and therefore of moderate confidence.	Moderate
Estimation and Modelling Techniques	Ordinary Kriging is considered to be appropriate given the geological setting and grade distribution.	High
Cutoff Grades	A lower cutoff grade of 4g/t gold equivalent has been applied to the mineralisation constraints and grade reporting which approximately equates to an NSR value per tonne of US$165 in situ gross value, as a rough approximation of possible underground mining and subsequent processing costs.	Moderate
Mining Factors or Assumptions	A 5mE x 10mN x 5mRL block size has been selected assuming an underground mining scenario. Block model size is relatively small compared to data density and distribution leading to poor local estimates.	Moderate
Metallurgical Factors or Assumptions	Not applied to resource estimate	N/A
Tonnage Factors (In-situ Bulk Densities)	Sufficient data exists to enable moderate quality estimates of density via ordinary kriging	moderate

14.13 Resource Reporting

The grade estimate was classified in accordance with the current 2014 CIM guidelines as an Inferred Mineral Resource. The Inferred Mineral Resource reported for the La Mina deposit is summarised below as Table 43. The preferred reporting cutoff grade is 4g/t AuEq.

Tables 44 and 45 report the La Mina deposit, subdivided into South and North zones, based on the Quebrada Malo fault.

Table 43. La Mina Deposit – Inferred Mineral Resource – Ordinary Kriking Estimate

Cutoff (AuEq – g/t)	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Aueq koz	Au koz	Ag koz	Cu kt	Pb kt	Zn t
2	2,373	10.6	3.4	42.4	2.7	0.5	3.8	812	260	3,238	64	13	89
3	2,055	11.9	3.8	46.9	3.0	0.6	4.2	787	251	3,097	62	12	87
4	**1,846**	**12.9**	**4.1**	**50.0**	**3.3**	**0.6**	**4.6**	**763**	**244**	**2,966**	**61**	**12**	**84**
5	1,655	13.8	4.4	53.7	3.5	0.7	4.9	736	236	2,860	59	11	80
6	1,461	14.9	4.8	58.5	3.8	0.7	5.2	701	225	2,747	56	11	75
7	1,320	15.8	5.1	62.4	4.1	0.8	5.5	672	217	2,647	54	10	72
8	1,195	16.7	5.4	66.0	4.3	0.8	5.8	642	208	2,535	51	10	69
9	1,111	17.3	5.7	68.6	4.4	0.9	5.9	619	202	2,449	49	10	66
10	1,019	18.0	5.9	71.4	4.6	0.9	6.1	591	194	2,340	47	9	63

Note: AuEq This Updated Mineral Resources Estimate as of March 30, 2019 was prepared in accordance with NI 43-101 and CIM Standards (2014).

10. The Updated Mineral Resource Estimate tonnages have been rounded to the nearest 1,000 and Au, Ag, Cu, Pb and Zn grades have been rounded to one decimal. Troy ounces have been rounded to kilo troy ounces (koz), and tonnes of Cu, Pb, and Zn have been rounded to kilo-tonnes (kt).
11. The Updated Mineral Resources Estimate for the La Mina VMS polymetallic deposit have been classified as Inferred Mineral Resources.
12. The Updated Mineral Resources has been reported at various cut-off grades to demonstrate the grade-tonnage relationship. The preferred reporting cut-off grade is 4 g AuEq / t (4 gold equivalent grams / tonne). The gold equivalent grade formula, which assumes 100% metallurgical recoveries, is:

    AuEq ppm = Au ppm + (Cu % * 1.454) + (Ag ppm * 0.013) + (Zn % * 0.654) + (Pb %*0.532)
13. The Updated Mineral Resources could be accessed by developing a mine ramp and are considered reasonable prospects for economic extraction in the foreseeable future.
14. The Updated Mineral Resources Estimate is based on the following three year trailing monthly average prices of metal, as of January 31st, 2019:

    $1,264/ troy oz Au $16.64/ troy oz Ag

    $2.68/lb Cu - $5,909 / t Cu $0.98/lb Pb – $2,162 / t Pb

    $1.21/lb Zn - $2,656 / t Zn
15. This Updated Mineral Resources Estimate was prepared by Brian R. Wolfe, BSc (Hons), MAIG.
16. Mineral Resources do not have demonstrated economic viability.
17. Mineral Resources are exclusive of material that has been mined.
18. This Updated Mineral Resources Estimate may be materially affected by environmental, permitting, legal title, taxation, socio-political, marketing or other relevant issues.

Table 44. La Mina Deposit – South Domain Inferred Mineral Resource – Ordinary Kriging Estimate

Cutoff (AuEq – g/t)	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Aueq koz	Au koz	Ag koz	Cu kt	Pb kt	Zn kt
2	981	13.0	4.7	64.7	2.7	0.8	4.8	411	147	2,041	27	8	47
3	889	14.1	5.0	68.7	3.0	0.8	5.2	404	144	1,962	27	7	46
4	**814**	**15.1**	**5.4**	**71.9**	**3.2**	**0.9**	**5.5**	**395**	**141**	**1,881**	**26**	**7**	**45**
5	738	16.2	5.8	77.0	3.5	0.9	5.9	384	138	1,827	25	7	43
6	681	17.1	6.2	81.3	3.7	1.0	6.1	374	136	1,779	25	7	42
7	630	18.0	6.6	85.5	3.8	1.0	6.4	364	133	1,732	24	7	40
8	593	18.6	6.8	88.8	3.9	1.1	6.6	355	130	1,695	23	6	39
9	562	19.2	7.0	91.2	4.1	1.1	6.8	346	127	1,648	23	6	38
10	518	20.0	7.4	95.0	4.2	1.1	7.0	333	123	1,581	22	6	36

11. Note: This Updated Mineral Resources Estimate as of March 30, 2019 was prepared in accordance with NI 43-101 and CIM Standards (2014).
12. The Updated Mineral Resource Estimate tonnages have been rounded to the nearest 1,000 and Au, Ag, Cu, Pb and Zn grades have been rounded to one decimal. Troy ounces have been rounded to kilo troy ounces (koz), and tonnes of Cu, Pb, and Zn have been rounded to kilo-tonnes (kt).
13. The Updated Mineral Resources Estimate for the La Mina VMS polymetallic deposit have been classified as Inferred Mineral Resources.
14. The Updated Mineral Resources has been reported at various cut-off grades to demonstrate the grade-tonnage relationship. The preferred reporting cut-off grade is 4 g AuEq / t (4 gold equivalent grams / tonne). The gold equivalent grade formula, which assumes 100% metallurgical recoveries, is:

    AuEq ppm = Au ppm + (Cu % * 1.454) + (Ag ppm * 0.013) + (Zn % * 0.654) + (Pb %*0.532)
15. The Updated Mineral Resources could be accessed by developing a mine ramp and are considered reasonable prospects for economic extraction in the foreseeable future.
16. The Updated Mineral Resources Estimate is based on the following three year trailing monthly average prices of metal, as of January 31st, 2019:

    $1,264/ troy oz Au $16.64/ troy oz Ag

    $2.68/lb Cu - $5,909 / t Cu $0.98/lb Pb – $2,162 / t Pb

    $1.21/lb Zn - $2,656 / t Zn
17. This Updated Mineral Resources Estimate was prepared by Brian R. Wolfe, BSc (Hons), MAIG.
18. Mineral Resources do not have demonstrated economic viability.
19. Mineral Resources are exclusive of material that has been mined.
20. This Updated Mineral Resources Estimate may be materially affected by environmental, permitting, legal title, taxation, socio-political, marketing or other relevant issues.

Table 45. La Mina Deposit – North Domains Inferred Mineral Resource – Ordinary Kriging Estimate

Cutoff (AuEq – g/t)	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Aueq koz	Au koz	Ag koz	Cu kt	Pb kt	Zn kt
2	1,392	9.0	2.5	26.8	2.7	0.4	3.0	401	113	1,198	37	5	42
3	1,166	10.2	2.9	30.3	3.1	0.4	3.5	383	107	1,135	36	5	40
4	**1,032**	**11.1**	**3.1**	**32.7**	**3.4**	**0.4**	**3.8**	**368**	**103**	**1,084**	**35**	**5**	**39**
5	917	11.9	3.3	35.1	3.6	0.5	4.1	352	98	1,033	33	4	37
6	780	13.0	3.6	38.6	4.0	0.5	4.3	327	89	969	31	4	34
7	690	13.9	3.8	41.3	4.3	0.6	4.6	308	84	915	30	4	32
8	602	14.8	4.1	43.4	4.6	0.6	4.9	287	79	840	28	4	30
9	549	15.4	4.3	45.4	4.8	0.6	5.1	273	75	801	26	3	28
10	501	16.0	4.4	47.1	5.0	0.7	5.2	258	71	759	25	3	26

11. Note: This Updated Mineral Resources Estimate as of March 30, 2019 was prepared in accordance with NI 43-101 and CIM Standards (2014).

12. The Updated Mineral Resource Estimate tonnages have been rounded to the nearest 1,000 and Au, Ag, Cu, Pb and Zn grades have been rounded to one decimal. Troy ounces have been rounded to kilo troy ounces (koz), and tonnes of Cu, Pb, and Zn have been rounded to kilo-tonnes (kt).
13. The Updated Mineral Resources Estimate for the La Mina VMS polymetallic deposit have been classified as Inferred Mineral Resources.
14. The Updated Mineral Resources has been reported at various cut-off grades to demonstrate the grade-tonnage relationship. The preferred reporting cut-off grade is 4 g AuEq / t (4 gold equivalent grams / tonne). The gold equivalent grade formula, which assumes 100% metallurgical recoveries, is:

    AuEq ppm = Au ppm + (Cu % * 1.454) + (Ag ppm * 0.013) + (Zn % * 0.654) + (Pb %*0.532)
15. The Updated Mineral Resources could be accessed by developing a mine ramp and are considered reasonable prospects for economic extraction in the foreseeable future.
16. The Updated Mineral Resources Estimate is based on the following three year trailing monthly average prices of metal, as of January 31st, 2019:

$1,264/ troy oz Au	$16.64/ troy oz Ag
$2.68/lb Cu - $5,909 / t Cu	$0.98/lb Pb – $2,162 / t Pb
$1.21/lb Zn - $2,656 / t Zn	
17. This Updated Mineral Resources Estimate was prepared by Brian R. Wolfe, BSc (Hons), MAIG.
18. Mineral Resources do not have demonstrated economic viability.
19. Mineral Resources are exclusive of material that has been mined.
20. This Updated Mineral Resources Estimate may be materially affected by environmental, permitting, legal title, taxation, socio-political, marketing or other relevant issues.

15.0 Mineral Reserve Estimates

A Mineral Reserve has not been estimated for the Project as part of this PEA.

Several mineral resources estimates have been made for the La Mina VMS deposit over the preceding decades as previously described in this PEA. SGS has identified references to historical mineral reserve estimates (for portions of the La Mina VMS deposit known at those times) that were made circa 1950 and 1980 well before the introduction of NI 43-101 Standards of Disclosure for Mineral Projects. As such, SGS is not aware of any existing mineral reserve estimates for the La Mina VMS deposit that were developed based on NI 43-101 requirements.

The PEA includes 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.

16.0 Mining Methods

The Inferred Mineral Resources used in this PEA 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 PEA will be realized. The words "production", "mineable" and mine plan" are used in the PEA in an operational sense and not in an accounting sense.

16.1 Overview

Toachi would operate the proposed underground mine using its own labour and conventional technologies and equipment. Mine consumables such as fuel and rock bolts would be purchased from suppliers.

The La Mina VMS deposit comprises the La Mina South block and the La Mina North block. For present preliminary mine planning purposes, the La Mina North block is subdivided into two zones referred to as the "Upper North zone" and the "Lower North zone"; and the La Mina South block is referred to as the "South zone". The proposed mine plan includes production from each of these three zones. The PEA assumes that Toachi would:

- Commence underground mine development in Year -1 by developing a new portal and a ramp to the Upper North zone. The Upper North zone is scheduled to be mined in Years 1 to 3. It is understood that a previous mine operator mined approximately 173 kt from the Upper North zone in years 1975 to 1981. The PEA assumes that some historic mine access headings would be rehabilitated and used as mine ventilation / escapeway headings.
- Develop a second new portal and ramp to access the Lower North zone and the South zone which are scheduled to be mined in Years 2 to 8 and Years 3 to 8 respectively.
- The proposed 800 tpd mill is scheduled to ramp-up and be commissioned in the first quarter of Year 1. The mine is scheduled to deliver 240 kt of material to the mill for processing in Year 1 increasing to 288 kt/yr in Year 2.

The ramps and stope access headings would be developed 5 m high x 4.5 m wide with gradients ranging from -15% to +15%.

16.2 Mine Layout

The conceptual mine layout is shown in Figures 37, 38 and 39.

Figure 37. Conceptual mine layout

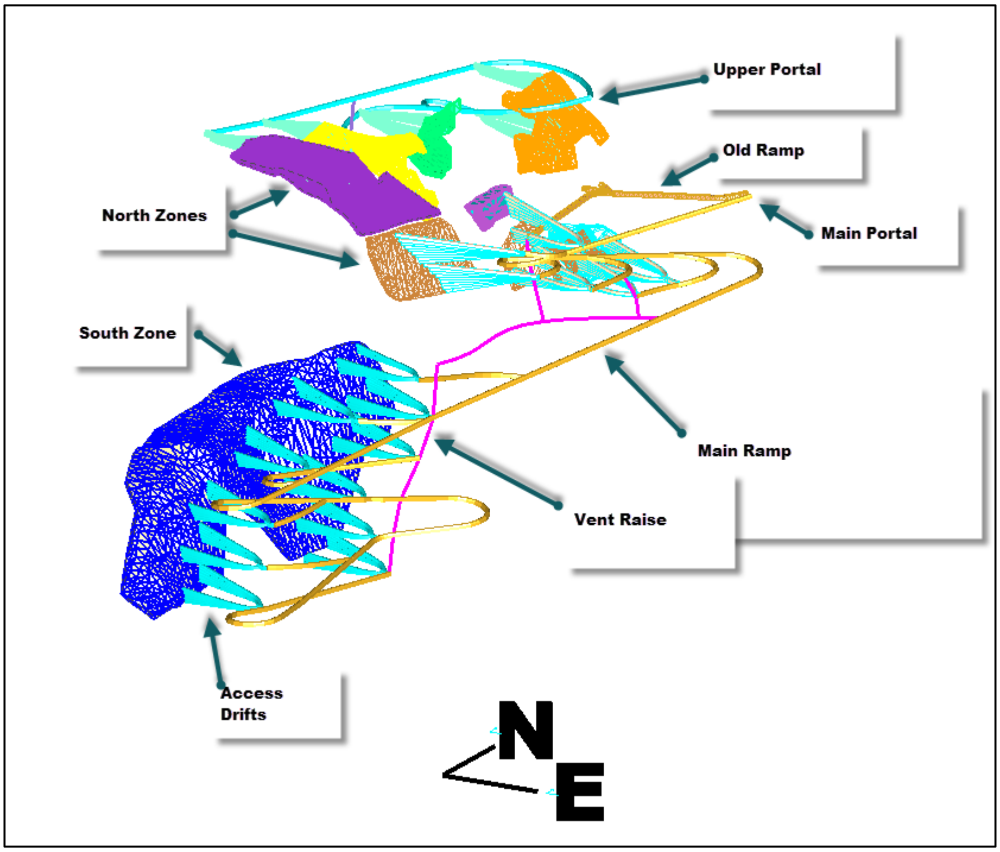


Figure 38. Longitudinal projection looking West

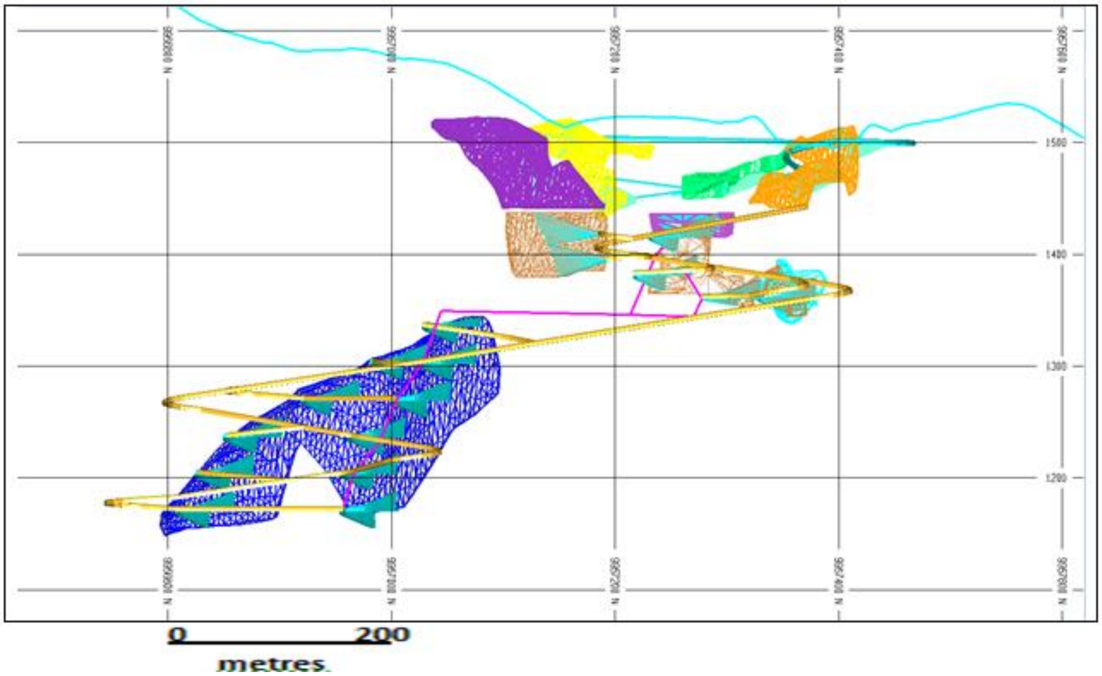
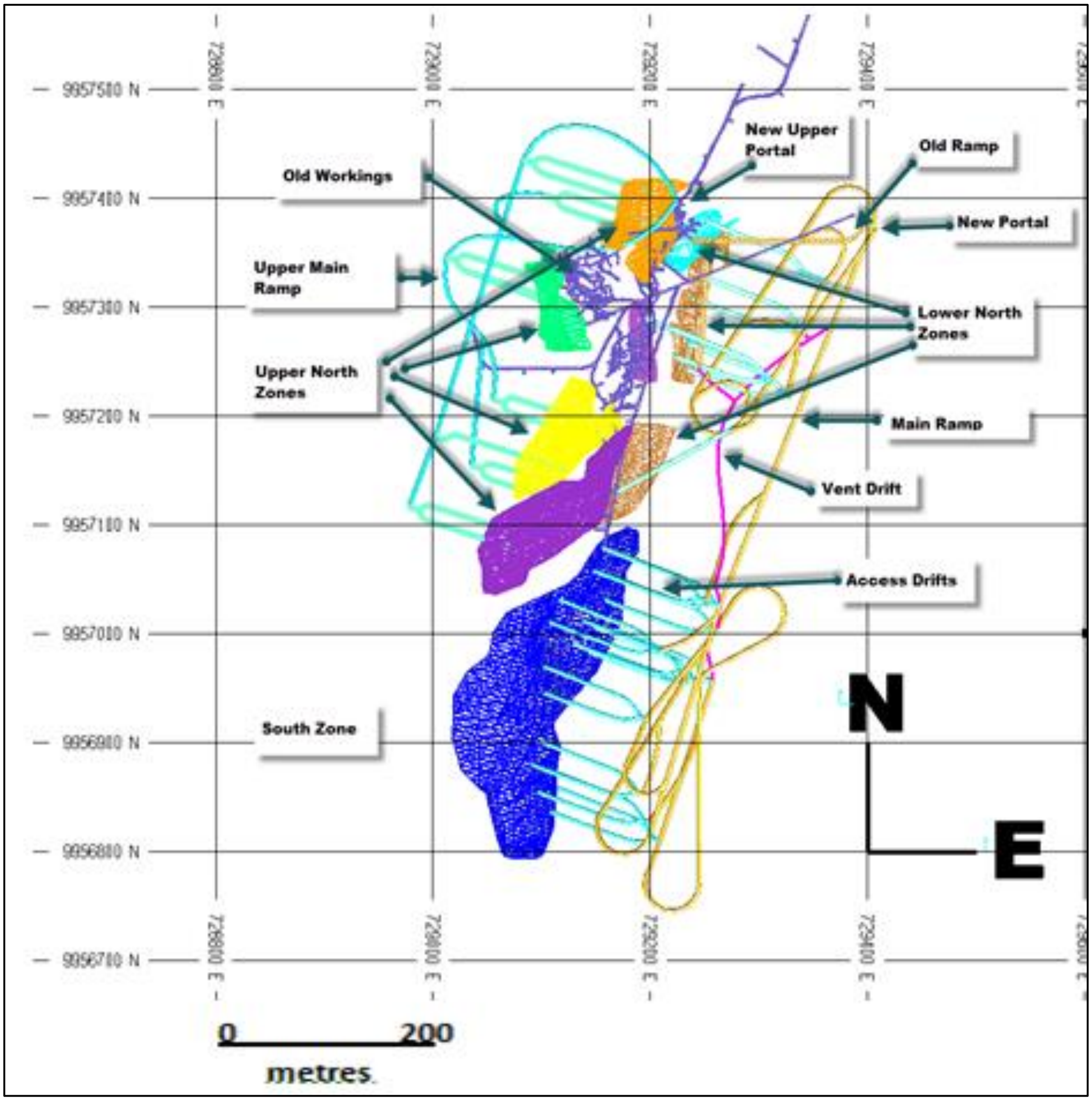


Figure 39. General plan view of proposed mine



16.3 Mechanized Cut and Fill Mining

The PEA is based on mining the deposit using two variations of the widely-used and flexible mechanized cut and fill mining method:

- A mechanized drift and slash mining with cemented rock fill (CRF) approach would be used in most stopes.

- A mechanized room and pillar with CRF approach would be used to mine stopes in the flatter dip uppermost part of the South zone.

The proposed mining methods were selected following the review and assessment of: the geology; the mineral deposit characteristics including faulting; the local topography; geotechnical, hydrogeological and hydrological aspects that affect stope design and mining method selection; climate data; local settlements and land and water uses; available information on the historic mine; the resource tonnage and grade distributions; projected metallurgical recoveries; health and safety and ecological protection aspects; estimated mine development and infrastructure requirements and development time lines; projected stoping cycles; projected training requirements; projected internal and external dilution factors and production losses; projected cut-off grade; relevant Republic of Ecuador legislation; mine planning guidelines and best management practices; and mining method and backfilling options. As part of this process, SGS reviewed the ranges of the footwall and hangingwall dips and resource mineralization widths in the Upper North, Lower North and South zones and applied this and other factors to select the proposed mining methods.

Sections through the deposit are shown in Figures 40 and 41. The orientation of these sections on plan is shown in Figure 42.

Figure 40. North Zone Section 531 looking North 19 East

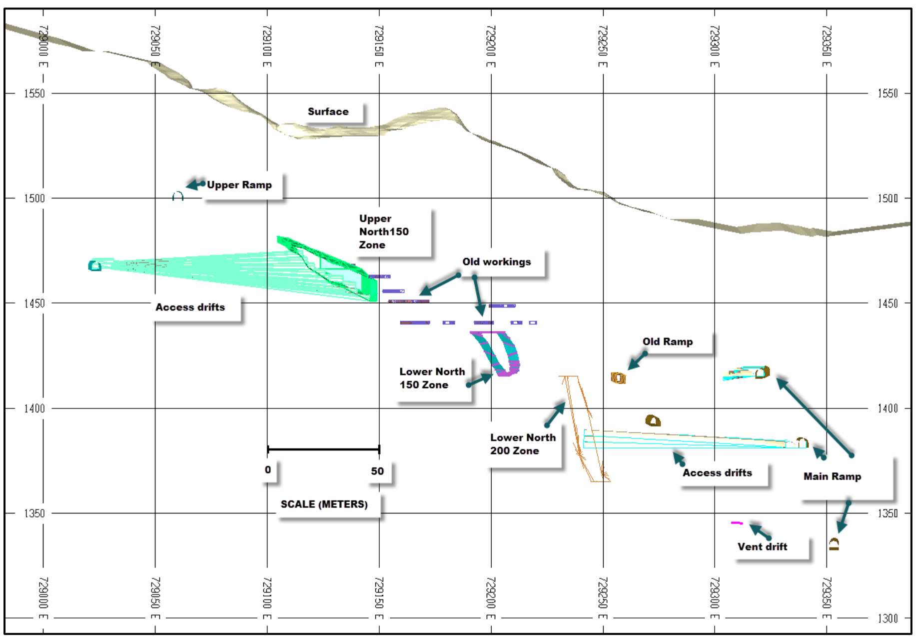


Figure 41. South Zone Section 311 looking North 19 East

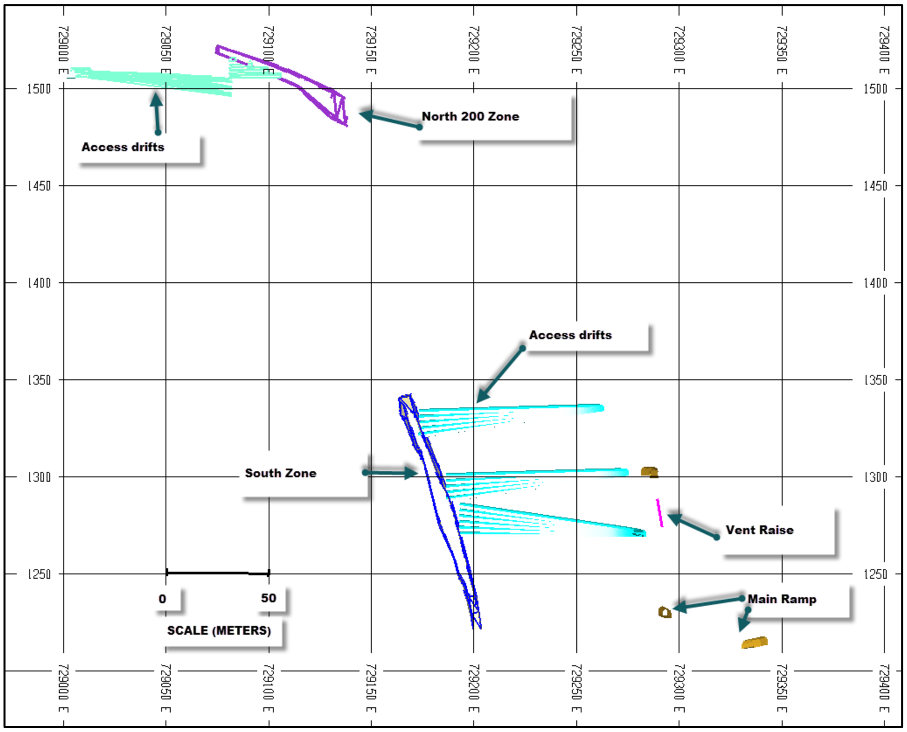
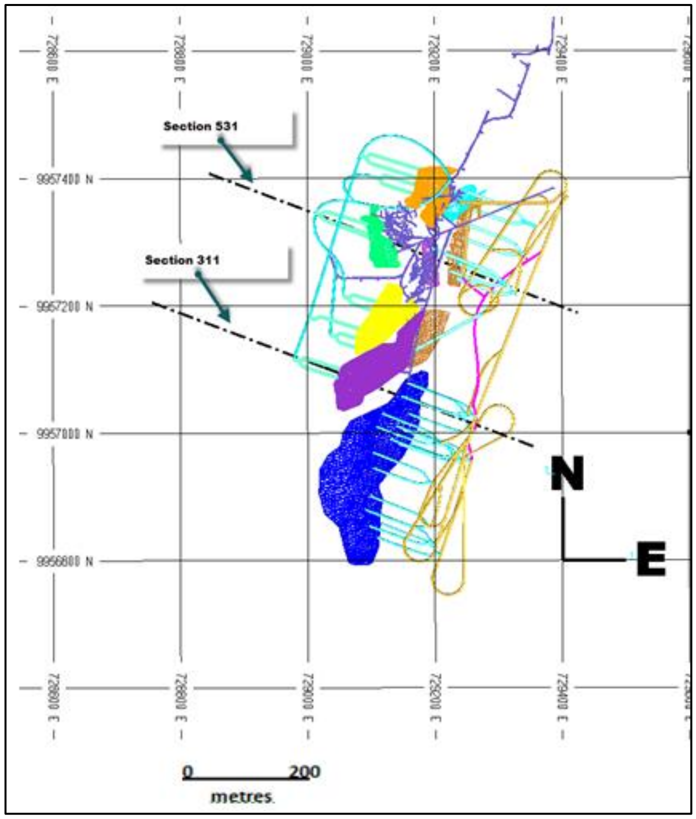


Figure 42. Orientation of Section lines 531 and 311 on plan view

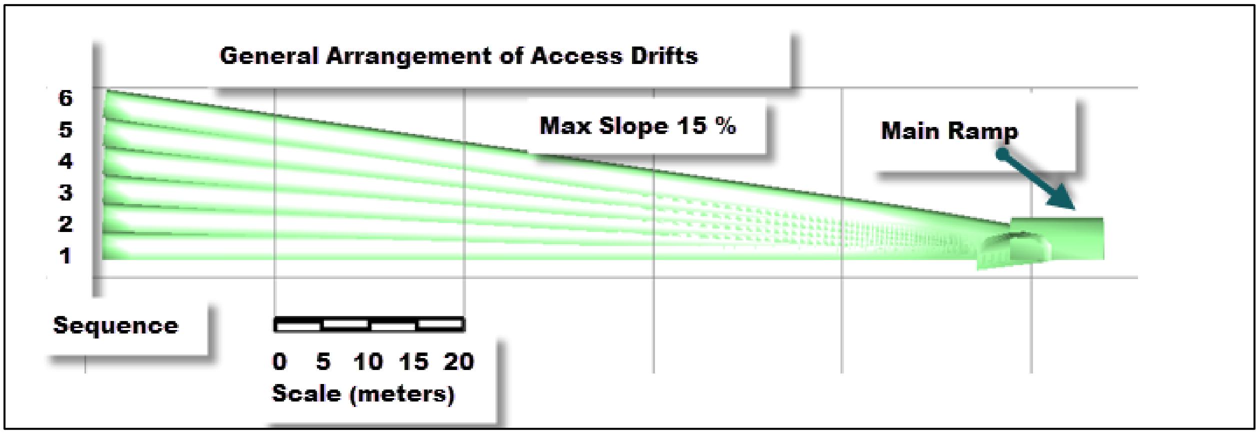


16.3.1 Drift and slash mining with CRF

In the drift and slash mining with CRF approach:

- Stopes would be accessed from the main ramp and mined in 3 m high lifts. The access heading would include: a decline portion to access the lifts in the lower section of the stope; and another ramp to access the lifts in the upper section of the stope – see Figure 43. Once a lift is mined and CRF backfilled, the back of the access ramp would be slashed as required to provide access to the next lift. Some lifts would be developed under a CRF back. The PEA assumes that an 8 wt% cement CRF mixture would be used to backfill lifts that would be undermined later.
- Each stope would be developed between the footwall and hangingwall and extend over a strike distance of about 10 to 80 metres. In many cases a single lift would be used to mine adjacent lenses along strike.

Figure 43. Access ramp to upper lifts.



The lifts in the stopes would be mined using conventional mining equipment where:

- A 3 m H x 4.1 m W pilot drift would be developed in mineralization along strike. After a pilot drift round has been drilled, blasted and excavated the drift back and walls would be manually bolted and screened.
- The mineralization between the pilot drift and the footwall would be drilled-off, loaded and blasted (slashed) to create a shanty footwall. The back and footwall would be manually bolted and screened.
- The mineralization between the pilot drift and the hangingwall would be drilled-off, loaded and blasted (slashed) to create a shanty hangingwall. The hangingwall would be manually bolted and screened. Figures 44, 45 and 46 show conceptual sections of lifts in the Upper North, Lower North and South zones.

Figure 44. Upper North zone – Conceptual section of a lift

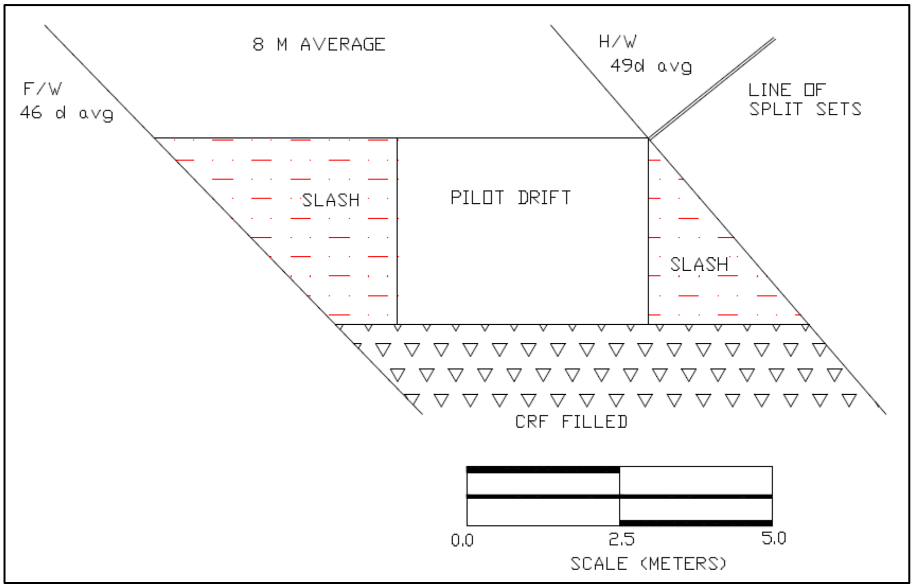


Figure 45. Lower North zone - Conceptual section of a lift

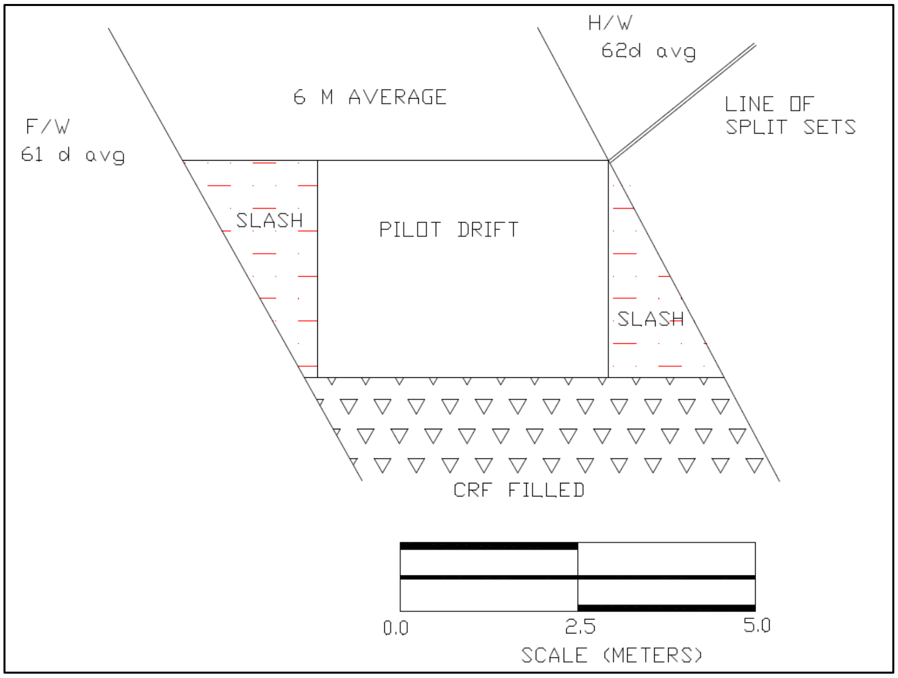
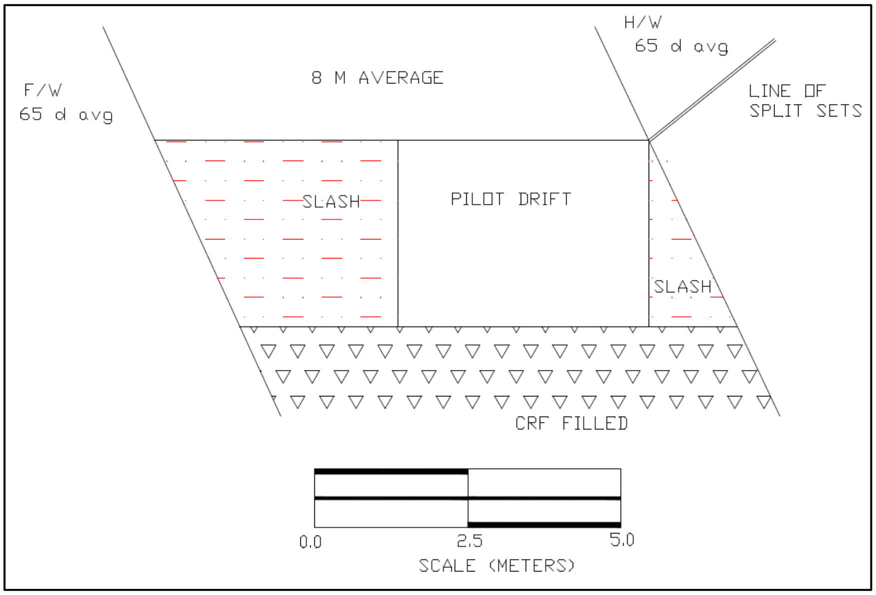


Figure 46. South zone – Conceptual section of a lift



The key criterion used to develop the proposed stoping concept are to: selectively mine; restrict spans; minimize disturbance to stope backs and the footwall and hangingwall; have miners work under secured ground; make use of conventional mining equipment and materials; and help reduce dilution and losses. Features incorporated into the proposed drift and slash mining with CRF mining method are summarized below.

- This mining method allows the selected lenses to be mined.
- Stopes are developed in successive backfilled lifts. Lift heights and the stope development sequence are designed to limit spans and help limit footwall and hangingwall exposure. The use of pre-shear perimeter blast holes would be expected to reduce disturbances to the stope back and hangingwall and footwall. Ground support would be installed using conventional stoper and jackleg drills.
- The mining sequence can be modified in wide mineralization areas to allow an initial pilot draft and a slashed area to be CRF backfilled before a second pilot drift is driven beside it.
- The preliminary dilution and loss estimates consider: variable mineralization widths; anticipated footwall and hangingwall conditions; the loss of blasted mineralization particles to an underlying CRF-backfilled lift; waste that would be mined in order to maintain a minimum 4 m mining width; the possible excavation of CRF along with blasted mineralization; mineralization losses during transportation to the surface stockpile / during stockpile reclamation or due to being inappropriately treated as waste; and dilution from adjacent exposed CRF walls (when present).

The estimated powder factors are shown in Table 46. Blasts would be loaded using pre-shear explosives on the final back and side walls, ANFO, packaged explosives in lifter blast holes, and non-electric type detonators. The estimated tonnage excavated per metre of advance in a lift is shown in Table 47. Dilution and losses are estimated to average 15% and 5% respectively.

Table 46. Estimated powder factors

Estimated powder factors	Upper North Zone	Lower North Zone	South Zone
Pilot drift	0.9 kg/t	0.9 kg/t	0.9 kg/t
Slash	0.6 kg/t	0.6 kg/t	0.6 kg/t

Table 47. Drift and slash and CRF stopes – estimated tonnage per metre of advance in a lift

Item	Upper North Zone	Lower North Zone	South Zone
Estimated average tonnes broken per metre advance along strike in a lift.	86 t/m	73 t/m	91 t/m

16.3.2 Mechanized room and pillar mining with CRF

A mechanized room and pillar with CRF backfilling approach would be used to mine the flatter-dip uppermost part of the South zone. In this approach a 4.5 m H x 4.5 m W drift would be developed transversely in mineralization using conventional equipment with the blasted stope material trucked to a surface stockpile. The drift would then be backfilled using CRF, and the back of the stope access ramp would be slashed to allow access to the next lift. Following an assumed CRF cure time of about 10 days, the next lift would be developed above the backfilled lift. Lifts would be successively developed up to the hangingwall.

The primary transverse drifts (e.g. primary stopes) would be separated by pillars. Once the primary stopes are mined, the secondary stopes (e.g. the mineralized pillars) would be mined using a similar mechanized approach. This approach would allow waste zones between lenses to be left in place as permanent pillars. Dilution and losses are estimated to average 15% and 5% respectively.

When a primary or secondary stope is advanced up to the hangingwall contact, the PEA assumes that the exposed hangingwall would be supported using a combination of cable bolts, welded wire mesh, pattern bolting, and shotcrete.

16.4 Break-even cut-off

16.4.1 Break-even cut-off estimate

The mine would produce broken and diluted polymetallic rock that would be processed at the proposed on-site 800 tpd capacity mill in order to produce a Cu/Pb concentrate and a Zn concentrate for sale.

The estimated break-even cut-off value is shown in Table 48. It represents the value of a tonne of mined diluted polymetallic rock delivered to the mill, where the revenue generated by mining and processing that tonne of material equals the costs incurred in producing the revenue.

Table 48. Break-Even Cut-Off estimate

	Average cost per tonne milled in Years 1 to 8	**Total ($/t milled)**
Mine direct costs:		
Stoping direct costs	$18.39/t	
CRF backfilling direct costs	$7.75/t	
Total	$26.14/t	$26.14/t
Mine operations support costs:		
Mine equipment lease costs	$8.21/t[A]	
Mine direct indirect operating costs	$7.57/t	

	Average cost per tonne milled in Years 1 to 8	**Total ($/t milled)**
Mine indirect operating costs	$4.23/t	
Total	$20.01/t	$20.01/t
Processing costs:		
Mill operating cost	$26.62/t	
TSF operating cost	$0.53/t	
Effluent treatment cost	$0.70/t	
	$27.85/t	$27.85/t
	Subtotal (marginal cut-off value)	**$74.00/t**[B]
Sustaining cost expenditures:		$21.50/t[C]
General & Administration cost:		$5.15/t
Total (break-even cut-off value)		**$100.65/t**[D]
		$101/t (rounded)[E]

[A] Mine equipment lease costs are included in mine operations support costs in this table. Mine equipment lease costs are treated as a sustaining capital cost item in the cashflow model.

[B] Marginal cut-off value = Mine direct costs + mine operations support costs + processing costs.

[C] Based on high sustaining cost expenditure years 1 to 4. Excludes mine equipment lease costs.

[D] Break-even cut-off value = Mine direct costs + mine operations support costs + processing costs + sustaining cost expenditures + General & Administration costs = All-In Sustaining Cost.

[E] This is equal to the NSR value of run-of-mine material grading 4.24 g Au Eq. / t .

16.4.2 Net Smelter Return

The Net Smelter Return (NSR) approach was applied to assign values to blocks considered in the proposed mine plan. The NSR represents the revenue that would be obtained from the sale of mineral products (i.e. the Cu/Pb concentrate and the Zn concentrate) after deducting concentrate transportation costs, treatment costs (TC's), refining costs (RC's) and other related off-site costs. The NSR was estimated using projected recoveries; projected metal prices; estimated concentrate transportation costs, TC's, RC's and related off-site costs not limited to smelter deductions and penalties. It is assumed that the Project would receive payment from smelters in US dollar currency. The factors used to estimate the NSR are shown in Table 49. The estimated NSR's per tonne milled in Years 1 to 8 are shown in Table 50.

The NSR approach is widely used and is considered best practice for assigning values to blocks containing polymetallic material. The cashflow model presented in Section 22 "Economic Analysis" uses the proposed mine plan tonnages and grades and the NSR approach to estimate annual revenues.

Table 49. Factors used to estimate Net Smelter Return

Item	Projected values	
Metal prices (3 year monthly trailing average prices to January 31, 2019)	Gold Price	$1,264 / troy oz Au
	Silver Price	$16.64 / troy oz Ag
	Copper Price	$2.68/ lb Cu ($5,909/ t Cu)
	Lead Price	$0.98/ lb Pb ($2,162/ t Pb)
	Zinc Price	$1.21/ lb Zn ($2,656/ t Zn)
Cu/Pb concentrate (con) mill recoveries (based on LCT1 test results)	Au % recovered	43.1%
	Ag % recovered	47.9%
	Cu % recovered	89.4%
	Pb % recovered	71.6%

Item	Projected values		
	Zn % recovered	20.0%	
Zn con mill recoveries (based on LCT1 test results)	Au % recovered	26.6%	
	Ag % recovered	26.1%	
	Cu % recovered	4.5%	
	Pb % recovered	13.4%	
	Zn % recovered	70.5%	
Estimated concentrate sales terms	Treatment	Cu/Pb con	Zn con
	Con grade	(20.9% Cu)	(51.2% Zn)
	Moisture content	8 wt%	8 wt%
	Treatment Charge	$132/ dt con	$188/ dt con
	Penalties	$13.10/ dt con	$9.45/ dt con
	Transportation	$95.45/wt con	$95.45/wt con
	Refining Charges (per unit of payable metal)	$0.135/ lb Cu $5/troy oz Au $2/troy oz/Ag	$5/troy oz Au $1/troy oz/Ag
	Payables	Cu/Pb con	Zn con
	Payment currency	USD	USD

Item	Projected values		
	Au payable	96%	65%
	Ag payable	90%	70%
	Cu payable	96.5%	0%
	Pb payable	0%	0%
	Zn payable	0%	85%
Net Smelter Return (LOM average head grades)	Metal	Net Smelter Return	
	Au payable	$23.75 / t /g Au.	
	Ag payable	$0.29 / t / g Ag.	
	Cu payable	$33.60 /t / % Cu	
	Pb payable	-$1.43 / t / % Pb[A]	
	Zn payable	$8.11 / t / % Zn	
	[A] Pb contained in the concentrates is not payable.		

Table 50. Estimated NSR's per tonne milled in Years 1 to 8

Year	Estimated NSR per tonne milled[A]
1	$200/t
2	$192/t

Year	Estimated NSR per tonne milled[A]
3	$187/t
4	$266/t
5	$250/t
6	$183/t
7	$135/t
8	$119/t
LOM average	$200/t

[A] Excludes taxes.

16.4.3 Stope tonnage and grade estimation

The process that was used to estimate the stope tonnages and grades is summarized in Figure 47. Readers are reminded that the PEA is preliminary in nature; scheduled tonnes and grades do not represent a Mineral Reserve estimate; and that the Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Figure 47. Stope tonnage and grade estimation process

The tonnage and grades of each potential stope were estimated based on preliminary stope outlines. The preliminary stope outlines were developed using Inferred Mineral Resource blocks for the South mineralized zone. In the Lower North zone and the Upper North zone a mine planning $244/t cut-off value (6 g/t Au Eq.) was employed to group Inferred Mineral Resource blocks into continuous potentially mineable areas. The grade of material contained within proposed stope limits was reduced by blocks having values below the mine planning cut-off. Estimated mining losses take anticipated mining conditions and other factors into consideration.

↓

The tonnages and grades of each proposed stope were tabulated. Waste blocks are included in the proposed mine plan in order to as examples: maintain a minimum 4 m mining width in drift and slash stopes; provide stope continuity along strike, and/or where lower grade material could be mined concurrent with better grade material.

Stope selection in the Upper North zone and Lower North zone:

During various trials to design conceptual drift and slash stopes in the Upper North zone and the Lower North zone, it was found that the average grade of these stopes was approaching the break even cut-off value taking dilution and mining losses into consideration. Subsequently, areas that offered better grade and that could allow contiguous mining blocks were identified. The MineSight software (version 15.4) used to examine the mineral resource block model data has the ability to limit displayed volumes based on a specified cut-off grade – this display is called a grade shell. Various cut-off value shells were visually examined and used to find grouping of resource model blocks that formed coherent potentially mineable areas albeit conceptual – see Figure 48. Limits were created from this display of the grade shell data, and the volumes of the mineralization solids were reduced to these limits. The widths of these resulting solids were adjusted to maintain a 4 m minimum mining width. Volumes of areas thought to have been previously mined in the Upper North zone were cut-out of these solids. The end result was a series of smaller solids with higher average grades.

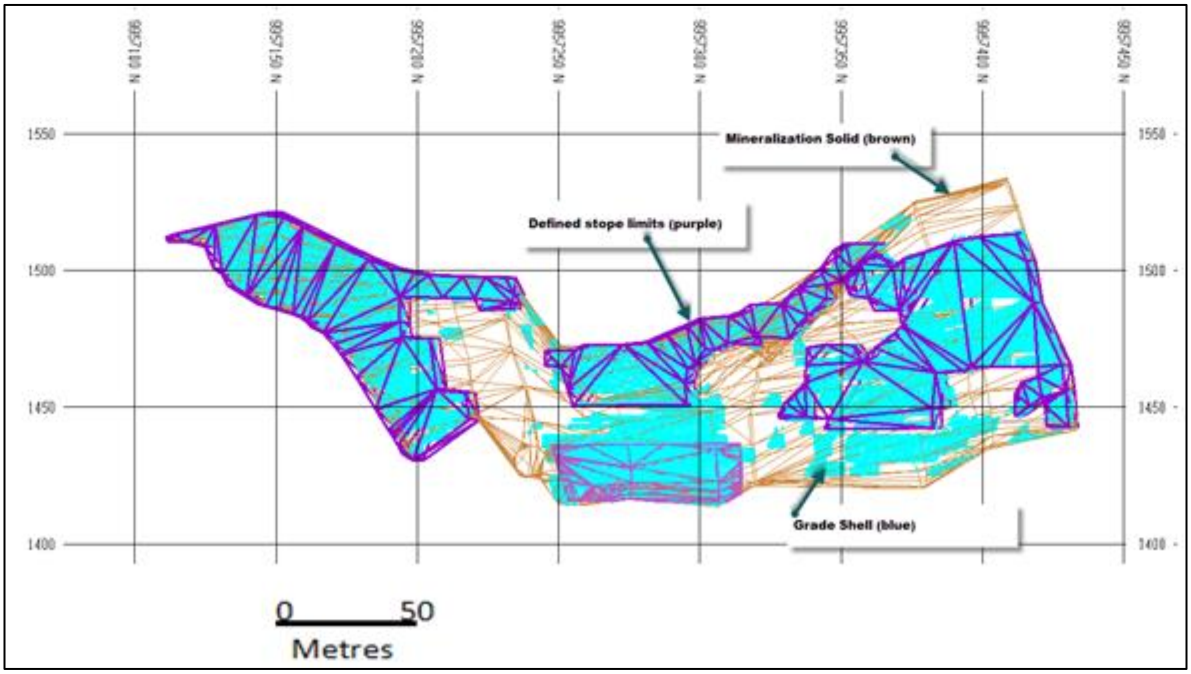


Figure 48. Conceptual drawing indicating how stope limits were defined

16.5 Geotechnical, Hydrogeological and Hydrologic Considerations

16.5.1 Geotechnical considerations

The geotechnical basis used to prepare the PEA was developed taking into account: the deposit geology including faulting; conceptual stope designs; the inspection of diamond drill core to understand possible rock conditions in the mineralized zones and the footwall and hangingwall; on-site discussions; experience at other projects and relevant guidelines. SGS also reviewed the horizontal widths of the mineralization and the ranges and averages of footwall and hangingwall dips in selected locations in the Upper North, Lower North and South zones. The results of that review are shown in Table 51.

Table 51. Projected apparent mineralization widths and footwall and hangingwall dips

	Upper North zone	Lower North zone	South zone
	Projected values		
Horizontal (apparent) width of mineralization in drift and slash stopes	8 m average (2 to 22 m range)	6 m average (2 to 10 m range)	8 m average (2 to 16 m range)
Footwall dip	46° average dip (7 to 62° range)	61° average dip (51 to 71° range)	65° average (47 to 88° range)
Hangingwall dip	49° average dip (9 to 68° range)	62° average dip (46 to 71° range)	64° average (38 to 80° range)

The conceptual stope ground control program encompasses:

- A stope development / ground support installation sequence;
- Pre-shear perimeter blastholes;
- Pattern bolting and screening;
- Backfilling using CRF;
- The use of multiple work places to allow for CRF curing; and
- The proposed use of cable bolts and shotcrete in the uppermost part of the South zone.

The conceptual ground support program is summarized in Table 52. In consideration of the preliminary nature of the PEA, recommendations aimed at adding to the geotechnical database for this Project are presented in Section 26.

Table 52. Conceptual ground support program

Conceptual Ground Support Program	
Development Headings	
Main ramp Stope access	Primary Ground Support Install split set bolts on a 1.2 m x 1.2 m pattern on the back and walls and screen to 1.5 m above the sill. Use 2.1 m long split set bolts and welded wire mesh screen. Use a bolter machine and ensure operators work from under secured ground. Secondary Ground Support The estimated mine development costs include a cost allowance for additional ground support at two locations where the ramps are projected to intercept the main fault.
Raises	Install 1.8 m resin rebar bolts on a 1.2 m x 1.2 m pattern with galvanized fencing fabric.
Drift and slash stopes	
Pilot drifts	Controlled blasting Pre-shear the perimeter (back and walls) blast holes. Primary Ground Support Install split set bolts on a 1.2 m x 1.2 m pattern on the back, and on a 1.5 m x 1.5 m pattern on the wall and screen to 1.5 m above the sill. Use 2.1 m long split set bolts and welded wire mesh screen. Install bolts and screen using stopers and jacklegs and work from under secured ground.
Footwall slash	Controlled blasting Pre-shear the perimeter (back and wall) blast holes. Primary Ground Support Install split set bolts on a 1.2 m x 1.2 m pattern on the back, and on a 1.5 m x 1.5 m pattern on the wall and screen to 1.5 m above the sill. Use 2.1 m long split set bolts

Conceptual Ground Support Program	
	and welded wire mesh screen. Install bolts and screen using stopers and jacklegs and work from under secured ground.
Hangingwall slash	Controlled blasting Pre-shear the perimeter (back and wall) blast holes. Primary Ground Support Install split set bolts on a 1.2 m x 1.2 m pattern on the back and the wall and screen to the sill. Use 2.1 m long split set bolts and welded wire mesh screen. Install bolts and screen using stopers and jacklegs and work from under secured ground.
CRF	CRF would be produced at an on-site plant and transported underground and pushed / jammed to the back. Lifts that would be later undermined by another lift would be backfilled using 8 wt% cement CRF. It is estimated that the overall average cement content of CRF used to backfill a stope would be 4 wt% cement.
Room and pillar stopes	
	Cable bolts and shotcrete the backs in the room and pillar stopes in the South zone. Also install split set bolts and screen on a 1.2 m x 1.2 m pattern on the back and walls and screen to 1.5 m above the sill. Use 2.1 m long split set bolts and welded wire mesh screen. Use a bolter machine and ensure operators work from under secured ground.

16.6 Hydrogeological considerations

The proposed mine dewatering system is based on preliminary mine dewatering pump power estimates that take the base mine water inflow rate (assumed to be 100 US gpm, 6.3 L/s) and assumed mine water recharge inflow due to wet season precipitation and faults and fractures into consideration. The projected mine pump power requirements over the life of the mine (LOM) are shown in Table 53. The Upper North zone stopes would not be developed through to surface, and a sill pillar would be left in place between the Upper North zone and the Lower North zone.

Table 53. Projected mine dewatering pump power over the LOM

Year	Projected installed pump power		
	Pump power to handle base mine water flow rates all year long.	Additional standby pump power available to deal with higher flow rates.	Estimated total installed pump power.
-1	100 kW (134 hp)	100 kW (134 hp)	200 kW (268 hp)
1	120 kW (160 hp)	120 kW (160 hp)	240 kW (320 hp)
2 to 8	200 kW (268 hp)	200 (268 hp)	400 (536 hp)

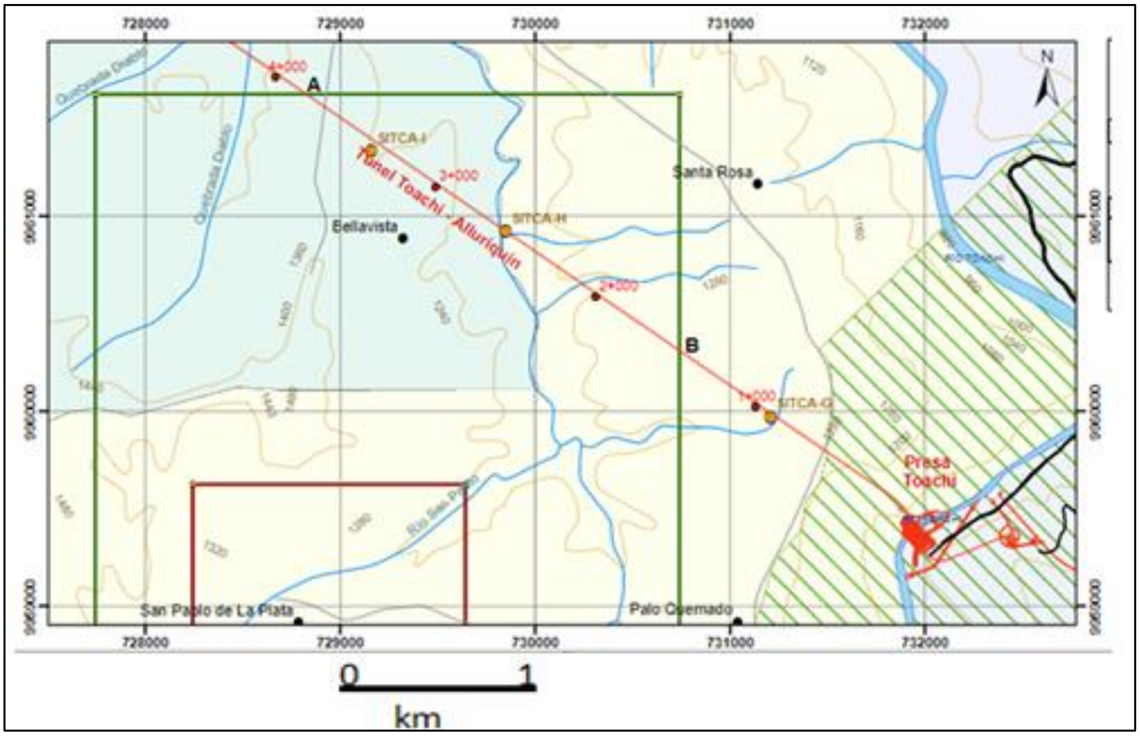
16.6.1 Hydrologic considerations

The potential impact of precipitation and surface water on the Project has been taken into consideration, where it is expected that precipitation that occurs during the summer wet season (October to May) would temporarily increase underground mine water inflow rates with a time lag. The mine ramp cost estimate includes a dewatering pipeline and a spare line.

It is assumed that water pumped from the operating mine would be piped to a water pond at the proposed engineered tailings storage facility for treatment prior to being discharged or recycled.

The Toachi-Alluriquin penstock tunnel is being developed across the northern part of the mining concession and distant from the Project as indicated in Figure 49. The development of this tunnel by others is not currently expected to impact the proposed Project.

Figure 49. Plan view showing the planned alignment of the Toachi-Alluriquin tunnel



16.7 Cemented Rock Fill

The proposed CRF plant would be constructed about 200 m from the Lower North zone ramp portal. It would include a bulk cement storage silo; a rock screening grizzly to remove +152 mm (+6 inch) rock particles, a cement slurry mixer with metered water supply; a CRF mixer and a truck load-out. It is envisaged that mine development waste rock would be trucked to surface and stockpiled adjacent to the CRF plant and reclaimed using a wheel loader. It is projected that scheduled mine waste rock production would at times need to be supplemented using imported non-acid generating rock.

The mixed CRF would be transported underground using the mine trucks and dumped near the stope lift being backfilled and then promptly reclaimed and placed in the lift using a load-haul-dump (LHD) unit. A LHD unit fitted with a boom would be used to push the CRF. A small dozer would also be available for use in the stopes.

16.8 Mine Ventilation System

The proposed mine ventilation system has been designed to a conceptual level with the objective of providing a safe atmosphere within the mine including development, stope and ancillary workplaces. The principle mine ventilation system components include:

- Main fans: Adjustable axial vane type fans would be installed on surface at the top of the ventilation / escapeway raises for the Upper North zone, and the Lower North zone and Upper North zone. The main fan installations would be equipped with carbon monoxide gas detection and stench gas worker alert systems.

- Fresh air distribution: Fresh air would be downcast the main ventilation / escapeway raises and distributed to the mine workplaces with the airflows controlled using ventilation bulkheads and auxiliary fans and ventilation ducting.

- Air quality: The levels of air-borne contaminants would be controlled to safe allowable levels by directing / exhausting workplace air to the open atmosphere via the main ramps. The level of airborne contaminants in the mine would also be controlled by: maintaining the diesel-powered equipment; using an appropriate grade of diesel fuel; maintaining the emission control devices; testing undiluted diesel engine exhaust; ambient dust level check measurements; providing fire suppression systems on equipment; and through the use of standard operating procedures.

- Blasting fumes removal: The mine would operate on a three eight hour shifts per day, seven days per week basis. Blasting would typically be conducted at the end of a shift.

- Monitoring: The mine supervisors and technical staff would inspect and assess the performance of the ventilation system and ensure that it is maintained. The skills training given to mine personnel would include training on ventilation related standard operating procedures.

The mine ventilation/escapeway raise system would be developed concurrent with ramp and stope access development. Mine ventilation fan power was estimated based on the numbers and projected utilization of the diesel-powered underground development and production equipment, engine brake horsepower, a 125 cfm (cubic feet per minute) per brake-horsepower [0.08 m3/sec per kW] factor, and anticipated booster and auxiliary fan requirements. The estimated fresh air requirement during operations is approximately 160,000 cfm (75 m3/sec). The fan power is summarized in Table 54.

Table 54. Estimated mine ventilation connected fan power

Year	Estimated mine ventilation connected fan power
-1	190 kW (250 hp)
1	225 kW (300 hp)
2 to 8	290 (390 hp)

16.9 Mine Development and Production

Readers are reminded that the PEA is preliminary in nature; scheduled tonnes and grades do not represent a Mineral Reserve estimate; and that the Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

16.9.1 Mine Development Sequence

The proposed mine development and production sequence is shown in Figure 50. Mine development is scheduled based on an average advance rate of 8 m/day. Each development drill jumbo is scheduled to drill-off up to approximately 125 m of advance / month. Mine production would commence in Year 1 (240 kt/yr) and increase to 288 kt/yr in Year 2. Readers are reminded that scheduled tonnes and grades do not represent an estimate of Mineral Reserves.

Figure 50. Proposed mine development and production sequence

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	LOM Total(1)
Upper North zone										
Development										
Production		240 kt	168 kt	91 kt						**499 kt**
Lower North zone										
Development										
Production			120 kt	177 kt	15 kt	22 kt		60 kt	11 kt	**405 kt**
South zone										
Development										
Production				20 kt	273 kt	266 kt	288 kt	228 kt	28 kt	**1,103 kt**
LOM Total (1)		**240 kt**	**288 kt**	**288 kt**	**288 kt**	**288 kt**	**288 kt**	**288 kt**	**39kt**	**2,007 kt**

(1) Potentially mineable mineralization

16.9.2 Mine Mobile Equipment

The mine development equipment fleet over the LOM is shown in Table 55.

Table 55. Mine mobile equipment fleet

Item	Projected number of pieces of equipment in mine fleet								
	Year -1	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**
Two boom jumbo	2	2	2	2	2	2	2	2	2
Single boom jumbo		1	1	2[A]	2[A]	1	1	1	1
LHD unit (10 t capacity)	2	2	2	2	2	2	2	2	2
LHD unit (6.8 t capacity)		2	2	2	2	2	2	2	2

Item	Projected number of pieces of equipment in mine fleet								
	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
LHD unit (3.5 t capacity)			1	1	1	1	1	1	1
Truck (30 t capacity)	2	3	3	3	3	3	3	3	3
Blast truck	1	1	1	1	1	1	1	1	1
Mechanical scaler	1	1	1	1	1	1	1	1	1
Rock bolter	1	1	1	2	2	2	2	2	2
Cable bolter				1	1	1	1	1	1
Shotcrete unit				1	1	1	1	1	1
Shotcrete transmixer				1	1	1	1	1	1
Scissorlift	1	2	2	2	2	2	2	2	2
Personnel carrier	1	2	2	2	2	2	2	2	2
Supervisor vehicle	1	1	1	1	1	1	1	1	1
Tech Services vehicle	1	1	1	1	1	1	1	1	1
Maintenance vehicle	1	2	2	2	2	2	2	2	2
Mine grader	1	1	1	1	1	1	1	1	1

Item	Projected number of pieces of equipment in mine fleet								
	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
CRF rammer		1	1	1	1	1	1	1	1
CRF dozer		1	1	1	1	1	1	1	1
CRF surface loader	1	1	1	1	1	1	1	1	1
Truck loading chutes (South zone stopes)				2	3	3	3	3	3
Surface rock truck		1	1	1	1	1	1	1	1

[A] The second single boom jumbo shown in Years 3 and 4 would be rented.

16.9.3 Mine Production Schedule

The projected tonnes and grades of the potentially mineable mineralization included in the PEA mine production schedule are shown in Table 56. Multiple production workplaces would be mined in each year as indicated in Tables 57, 58 and 59. The mine plan allows for multiple production faces.

This PEA was prepared in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"). The PEA is preliminary in nature and the PEA mine production schedule 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 PEA will be realized.

Table 56. Proposed LOM Mine Plan

Production Year	kt[A]	Estimated Run-Of-Mine Grades				
		Au g/t	Ag g/t	Cu%	Pb%	Zn%
1	240	2.85	31	2.83	0.47	3.55
2	288	2.70	26	2.94	0.31	2.76
3	288	2.44	26	3.01	0.29	2.62
4	288	4.82	59	3.05	0.78	4.09
5	288	4.30	57	2.96	0.68	4.09
6	288	3.31	51	1.79	0.54	3.70
7	288	2.19	34	1.46	0.40	3.07
8	39	2.13	39	0.93	0.49	3.27
TOTAL	2,007	3.22	41	2.54	0.50	3.41

[A] Potentially mineable mineralization. Totals may not sum exactly due to rounding.

Table 57. Upper North zone plan

Production Year	kt[A]	Estimated Run-Of-Mine Grades				
		Au g/t	Ag g/t	Cu%	Pb%	Zn%
1	240	2.85	31	2.83	0.47	3.55
2	168	2.89	29	3.38	0.31	2.46

Production Year	kt[A]	Estimated Run-Of-Mine Grades				
3	91	2.61	28	4.12	0.21	1.80
4	-	-	-	-	-	-
5	-	-	-	-	-	-
6	-	-	-	-	-	-
7	-	-	-	-	-	-
8						
TOTAL	499	2.82	30	3.25	0.37	2.86

[A] Potentially mineable mineralization. Totals may not sum exactly due to rounding.

Table 58. Lower North zone plan

Production Year	kt[A]	Estimated Run-Of-Mine Grades				
		Au g/t	**Ag g/t**	**Cu%**	**Pb%**	**Zn%**
1	-	-	-	-	-	-
2	120	2.42	22	2.32	0.32	3.18
3	177	2.15	22	2.57	0.28	2.84
4	15	1.44	22	3.78	0.14	1.24
5	22	1.44	22	3.78	0.14	1.24
6	-	-	-	-	-	-

Production Year	kt[A]	Estimated Run-Of-Mine Grades				
		Au g/t	Ag g/t	Cu%	Pb%	Zn%
7	60	1.82	17	2.26	0.15	1.84
8	12	2.22	47	0.59	0.41	2.85
TOTAL	405	2.12	22	2.50	0.26	2.65

[A] Potentially mineable mineralization. Totals may not sum exactly due to rounding.

Table 59. South zone plan

Production Year	kt[A]	Estimated Run-Of-Mine Grades				
		Au g/t	Ag g/t	Cu%	Pb%	Zn%
1	-	-	-	-	-	-
2	-	-	-	-	-	-
3	20	4.20	53	1.83	0.72	4.34
4	273	5.01	61	3.01	0.81	4.25
5	266	4.53	60	2.89	0.72	4.33
6	288	3.31	51	1.79	0.54	3.70
7	228	2.29	39	1.25	0.46	3.39
8	27	2.09	36	1.08	0.53	3.45
TOTAL	1,103	3.80	53	2.23	0.64	3.93

[A] Potentially mineable mineralization. Totals may not sum exactly due to rounding.

16.10 Mine Personnel

16.10.1 G&A Staffing

The General and Administration labour in Year 2 as an example is shown in Table 60.

Table 60. G&A labour in Year 2

Area	G&A staff position	Number on payroll
Administration	Toachi General Manager	1
	Executive Assistant	1
	Accountant	1
	Payroll	1
	Purchasing	1
	Concentrate sales	1
	Human Resources	1
	Clerk	1
	IT Technician (part time)	1
	HSES Manager	1

Area	G&A staff position	Number on payroll
Health-Safety-Environmental-Social (HSES) Security Kitchen	Environmental technician	1
	Nurse	1
	Social Director	1
	Community Social Worker	1
	Community Relations Assistant	1
	Security	4
	Kitchen staff	3
Total		22

16.10.2 Mine direct labour

A breakdown of mine direct labour in Year 2 as an example is shown in Table 61.

Table 61. Mine direct labour in Year 2

Area	Mine direct labour	Number on payroll[A]
Mine development	Jumbo operators	6
	Mechanical scaler operator	6

Area	Mine direct labour	Number on payroll[A]
	Rockbolter operator	6
Stoping	Jumbo operators	8
	Manual bolters	16
Total		42

A. Mine crews would work on rotating 8 hour shifts.

16.10.2.1 Mine indirect labour

A breakdown of mine indirect labour in example Year 2 is shown in Table 62.

Table 62. Mine indirect labour in Year 2

Area	Indirect labour	Total number on payroll[A]
Mine Supervision and training	Mine supervisor	6
	Trainer	1
Technical Services	Mining engineer	1
	Mine technician	1
	Mine surveyor	1

Area	Indirect labour	Total number on payroll[A]
	Mine geologist	1
	Grade control geologist	1
Blasting	Blasting leader	6
	Blasting helper	6
Excavation and haulage	LHD operators	6
	Mine rock truck drivers	12
	Surface truck driver	4
Cemented Rock Fill	CRF plant operator	4
	CRF wheel loader operator	4
	CRF rammer operator	4
Mine maintenance	Maintenance supervisor	2
	Maintenance planner	1
	Lead mechanic	2
	Mechanic	8
	Welder	2
	Lead electrician	2

Area	**Indirect labour**	**Total number on payroll**[A]
	Electrician	3
	Instrumentation technician	1
	Warehouse attendant	1
Total number on payroll		**80**

[A] Crews would work on rotating shifts.

17.0 Recovery Methods

17.1 Introduction

This section describes the process design criteria applied to the crushing, grinding, flotation, and dewatering facilities for the proposed 800 tonnes per day copper, lead, zinc, gold and silver mineral processing plant.

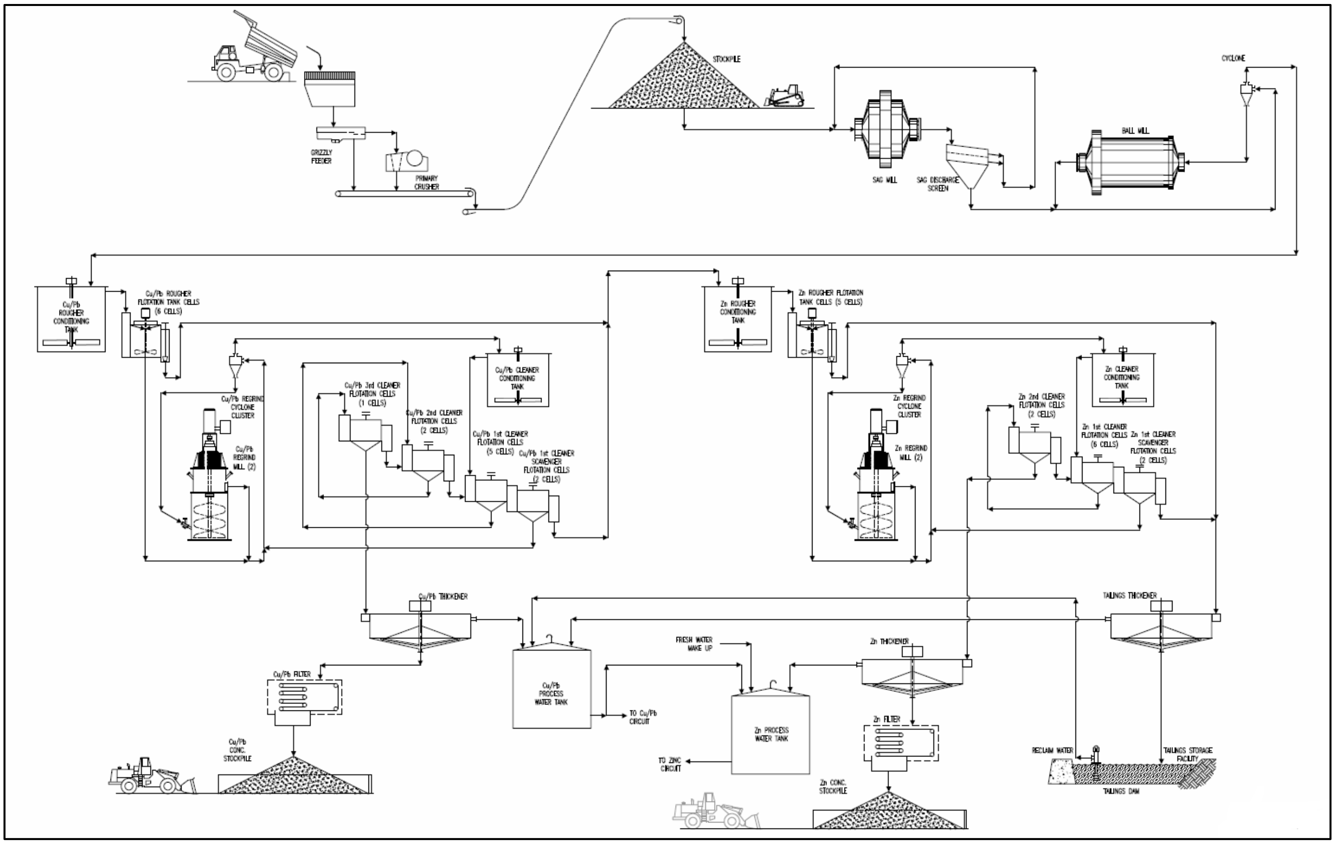
In plant conceptual design:

6. The crushing plant would process the run-of-mine (ROM) material by using a primary jaw crusher to reduce the material from a nominal 30-45 cm (12-18 inches) to approximately 100% passing (P_{100}) of 249 mm (P_{80} of 114 mm).
7. The grinding circuit would be a semi-autogenous (SAG) mill - ball mill grinding circuit with subsequent processing in a flotation circuit. The SAG mill would operate in closed circuit with a vibrating screen. The ball mill would operate in closed circuit with hydrocyclones.
8. Cyclone overflow, the grinding circuit product, is fed to the flotation plant. The flotation plant would consist of copper/lead and zinc flotation circuits. The copper/lead flotation circuit would consist of rougher flotation and three-stage cleaner flotation. The zinc flotation circuit would consist of rougher flotation and two-stage cleaner flotation.
9. Both copper/lead and zinc concentrates are thickened, filtered, and stored in concentrate storage facilities prior to being bagged and loaded into containers for shipment to port.
10. Zinc rougher flotation tailing and zinc first cleaner scavenger tailing would be the final tailing. Tailing thickener underflow would be pumped to a tailings storage facility (TSF).

Plant water stream types include: copper/lead process water, zinc process water, fresh water, and potable water.

The overall flowsheet is shown in Figure 51.

Figure 51. Process plant overall flowsheet



17.2 Primary crushing

ROM material is transported to the crushing plant area by rear-dump trucks. The primary crushing line consists of a dump hopper, grizzly screen, rock breaker, crusher and associated dust collection and transfer equipment. ROM material is dumped into the dump hopper by rear-dump trucks. The grizzly screen oversize feeds the jaw crusher. One mobile rock breaker is available to service the crusher or screen. The crusher reduces the ROM size from a maximum of 450 mm to approximately P_{100} of 249 mm. Crushed material drops onto a belt conveyor that transports it to a stockpile.

The crushing production rate is monitored by a belt scale mounted on the conveyor. Tramp iron is removed using a magnet located at the discharge of the primary crusher discharge conveyor. A metal detector would be installed over the conveyor. Dust is controlled in the dump pocket with water sprays and dust collector vents positioned at the conveyor transfer points. An air compressor and instrument air dryer are installed for operation and maintenance. A mobile crane is installed for maintenance of the primary crusher.

17.3 Crushed material conveying, transport and storage

Primary crushed material is stockpiled on the ground. A reclaim tunnel is installed beneath the stockpile. The stockpile would contain approximately 800 tonnes of “live” storage. When required, the material is moved from the “dead” storage area to the “live” storage area by a front-end loader.

Material is withdrawn from the coarse reclaim stockpile by variable speed belt feeders. The feeders discharge to the transfer conveyor belt. The transfer conveyor discharges to the SAG mill in the grinding circuit. The reclaim rate is monitored by a belt scale mounted on the conveyor.

Dust control in the stockpile area is achieved using a wet type dust collector system. One of the two dust collector systems is installed to control dust at the discharge of the stockpile feed conveyor and another one is installed to control dust in the reclaim tunnel.

17.4 Grinding

The mineralized material would be ground in a SAG mill primary grinding circuit and a ball mill secondary grinding circuit.

The SAG mill would operate in closed circuit with a vibrating screen. Water is added to the SAG mill to produce a slurry and the material feed size is reduced as it traverses the SAG mill. The SAG mill discharges onto a double deck screen with 8.0 mm sized bottom openings. Screen oversize is recirculated to the SAG mill feed chute by a series of conveyors. Screen undersize flows by gravity to the cyclone feed pump box. A belt scale mounted on the recycle conveyor is used to monitor the SAG mill recycle rate. The target SAG grind would be P_{80} of 1,354 microns.

Secondary grinding is performed in a ball mill. The ball mill operates in closed circuit with hydrocyclones. Ball mill discharge is combined with vibrating screen undersize in the cyclone feed pump box and pumped to hydrocyclone clusters. Combined slurry is pumped using variable speed horizontal centrifugal slurry pumps (one operating and one standby) to the cyclone clusters.

Hydrocyclone overflow (final grinding circuit product at 80% minus 70 microns) flows by gravity to the tramp oversize screen positioned prior to the flotation circuit.

Cyclone overflow would be sampled by primary samplers and analysed by the copper/lead and zinc on-stream analyser for metallurgical control prior to flotation. Cyclone overflow from the cyclone cluster is also be monitored for particle size distribution by a particle size monitor.

Zinc sulfate ($ZnSO_4$), sodium sulfide (Na_2S) and sodium metabisulfite, $Na_2S_2O_5$ (SMBS) would be added into the ball mill.

Grinding balls would be added to the SAG mill and ball mill by ball loading systems. Air compressors and an instrument air dryer would provide service and instrument air for operations and maintenance. An overhead crane is installed for maintenance of the grinding mills.

17.5 Copper/Lead flotation and regrind

Hydrocyclone overflow would flow by gravity to the copper/lead flotation circuit. The copper/lead flotation circuit would consist of one row of rougher cells and one row of cleaner cells. The rougher row consists of six (6) 10 m^3 tank type rougher flotation cells with a drop between each cell. The copper/lead rougher concentrate are sampled by a rougher concentrate primary sampler and pumped (one operating pump and one spare) to the copper/lead regrind mill circuit. Reground copper/lead rougher concentrate flow by gravity from the lead cleaner conditioning tank to the copper/lead first cleaner flotation cells. The copper/lead cleaner row consists of ten (10) flotation cells; five (5) 1.4 m^3 forced air first cleaner cell, two (2) 1.4 m^3 forced air first cleaner scavenger cells, two (2) 0.71 m^3 forced air second cleaner cells, and one (1) 0.71 m^3 forced air third cleaner cells. The copper/lead first cleaner concentrate is pumped (one operating pump and one spare) into the second cleaner flotation cells. Copper/lead rougher tailing and copper/lead first cleaner scavenger tailing flow by gravity into the zinc rougher conditioning tank. The copper/lead second cleaner concentrate would be pumped to the copper/lead third cleaner flotation cell. The copper/lead third cleaner concentrate flow by gravity to the copper/lead concentrate thickener.

The concentrate samples cut by the samplers are analysed for process control by the copper/lead and zinc on-stream analyser. Tailing from rougher flotation cells and first cleaner scavenger cells are combined together and sampled with primary samplers and analysed by the copper/lead and zinc on-stream analyser.

Copper/lead rougher concentrate would be pumped to the lead regrind cyclone feed pump box and combined with the regrind mill discharge. The combined slurry is pumped using horizontal centrifugal slurry pumps (one operating and one spare) to a hydrocyclone cluster. Overflow from the regrind cyclone cluster (final regrind circuit product) is sampled for particle size distribution analysis by the copper/lead regrind cyclone particle size monitor. It is then analysed by the copper/lead and zinc on-stream analyser and flows by gravity to the copper/lead cleaner conditioning tank. The cyclone underflow flows by gravity to the copper/lead regrind mill. Product from the regrind mill reports to the copper/lead regrind cyclone feed pump box.

Air compressors, air receivers, and instrument air dryer are installed for general plant operation and maintenance.

A bridge crane is installed for maintenance of the flotation and regrind equipment.

17.6 Zinc flotation and regrind

Copper/lead rougher tailing and copper/lead first cleaner scavenger tailing would flow by gravity to a zinc rougher conditioning tank. The zinc flotation circuit consists of one row of rougher cells and one row of cleaner cells. The rougher row consists of five (5) 10 m^3 tank type rougher flotation cells. The zinc rougher concentrate would be sampled by the zinc rougher concentrate primary sampler and pumped (one operating pump and one spare) to the zinc regrind mill circuit. The zinc cleaner row consists of ten (10) flotation cells; one bank of six (6) 1.4 m^3 forced air first cleaner flotation cells, two (2) 1.4 m^3 forced air first cleaner scavenger flotation cells, and two (2) 0.71 m^3 forced air second cleaner flotation cells. Tailings from zinc rougher cells and zinc first cleaner scavenger are pumped to the tailing sample box, then to the tailing thickener.

Reground zinc rougher concentrate flow by gravity from the zinc cleaner conditioning tank to the zinc first cleaner flotation cells. The zinc first cleaner concentrate is pumped (one operating pump and one spare) into the zinc second cleaner flotation cell. The zinc secondary cleaner flotation concentrate is pumped to the zinc concentrate thickener.

The concentrate samples cut by the samplers would be analysed for process control by the copper/lead and zinc on-stream analyser. Tailing from rougher flotation cells and first cleaner scavenger cells are sampled with primary samplers and analysed by the copper/lead and zinc on-stream analyser.

Zinc rougher concentrate would be pumped to a zinc regrind hydrocyclone feed pump box and combined with the zinc regrind mill discharge. The combined slurry is pumped using horizontal centrifugal slurry pumps (one operating and one spare) to the zinc regrind hydrocyclone cluster. Overflow from the zinc regrind cyclone cluster is sampled by sampler for particle size distribution analysis by the zinc regrind cyclone particle size monitor. It is then analysed by the copper/lead and zinc on-stream analyser and flows by gravity to the zinc cleaner conditioning tank. The underflow flows by gravity to the zinc regrind mill. Product from the regrind mill reports to the zinc regrind cyclone feed pump box.

17.7 Copper/Lead concentrate dewatering

Concentrate from the copper/lead third cleaner flotation cell would be pumped to a copper/lead concentrate thickener. The concentrate thickener overflow is pumped back to the thickener feed for dilution and the thickener spray bar; to control froth, or to the copper/lead process water tank. The concentrate thickener underflow is pumped (one operating pump and one spare) to an agitated storage tank and then to a pressure filter. Filter cake is discharged to a covered stockpile.

Concentrates, both copper/lead and zinc, would be reclaimed and bagged.

17.8 Zinc concentrate dewatering

Concentrate from the zinc secondary cleaner flotation cell would be pumped to a zinc concentrate thickener. The concentrate thickener overflow is pumped back to the thickener feed for dilution and the thickener spray bar; to control froth, or to the zinc process water tank. The concentrate thickener underflow is pumped to an agitated storage tank and then to a pressure filter. Filter cake is discharged to a covered stockpile.

17.9 Tailing dewatering

Tailings from the zinc rougher flotation and zinc first cleaner scavenger would be pumped to a high rate tailings thickener. Thickener overflow flows by gravity from the tailings thickener overflow tank to the copper/lead process water tank. Thickener underflow is pumped by variable speed horizontal centrifugal slurry pumps (one operating and one stand-by) to the TSF.

17.10 Reagents

Reagents requiring receiving, handling, mixing, and distribution systems include:

- Sodium Sulfide (Na_2S)
- Zinc Sulfate ($ZnSO_4 \cdot 7H_2O$)
- Aerofloat 242 (Promoter)
- Aerophine 3418A (Promoter)
- Sodium Metabisulfite, $Na_2S_2O_5$ (SMBS)
- Copper Sulfate ($CuSO_4 \cdot 5H_2O$)
- Sodium Isopropyl Xanthate (SIPX)
- Methyl Isobutyl Carbinol (MIBC, frother)
- Flocculant
- Lime

17.11 Water system

17.11.1 Fresh water

The Plant's fresh water requirement would primarily be met using wells and by recycling water from the effluent treatment plant water pond.

17.11.2 Process water

17.11.2.1 Process water – copper/lead circuit

The copper/lead process water tank would receive overflow from the copper/lead concentrate thickener, tailing thickener and water reclaimed from the TSF. The copper/lead process water is used as makeup water in the primary cyclone feed sump. Fresh water can be added to the copper/lead process water tank if necessary. This copper/lead process water is not suitable for general distribution throughout the process plant. Water would be reclaimed from the effluent treatment plant pond using barge-mounted reclaim water pumps.

17.11.2.2 Process water – zinc circuit

Overflow from the zinc concentrate thickener and copper/lead process water excess overflow would be recycled to the zinc process water tank, and used as makeup water in the zinc flotation circuit. Fresh water can be added to the zinc process water tank.

18.0 Project Infrastructure

This section describes the Project infrastructure which has been designed to a conceptual level.

18.1 Project logistics

The Project is situated approximately 13 km by road from the village of La Unión del Toachi which is located on a paved Road E20. The road from the village to the Project site is unpaved and at times narrows to a single lane. The PEA includes road improvements include such as adding culverts and road surfacing material to selected sections of road, truck pull-off areas and signage. The access road would be used to transport heavy equipment to the site in Year -1, and transport metal concentrate containers to Road E20 and hence to Guayaquil Port commencing in Year 1.

18.2 Infrastructure

The following infrastructure would be developed as part of the Project.

18.2.1 Main power line and substation

An approximately 7 km long power line would be constructed from an existing substation on the National hydropower grid to a location adjacent to the proposed mill site.

In 2018, Toachi representatives met with La Empresa Eléctrica Pública Estratégica Corporación Nacional de Electricidad (CNEL EP) Unidad de Negocio Santo Domingo and obtained preliminary cost information from CNEL EP for the construction of the power line and new substation near at the mill based on a then-contemplated 12-15 MWA range power system.

CNEL EP estimated that it would take two years for it to complete an environmental assessment for the power line and obtain approvals and permits for construction. This PEA assumes that this preparatory work would be conducted by CNEL prior to power line construction in Year -1.

18.2.2 On-site power distribution

Pole lines would be used to distribute power to the mine, mill and infrastructure transformers. The costs for on-site power distribution are included in the mining, processing and infrastructure capital expenditure estimates.

18.2.3 Back-up diesel generator

A diesel generator would provide electrical power for mine development in Year -1 when the mine load is projected to be 600 kW. The generator would become a standby power unit in Year 1.

18.2.4 Mill infrastructure

The mill and mine would share the main shop, training and change house facilities. The mill would have a laydown yard and small parts and consumables storage containers. The mill substation would be installed adjacent to the mill.

Process water sourced from wells and the effluent treatment plant pond would be pumped to the mill. Tailings would be pumped to the TSF using a suitable pipeline equipped with pipe pressure monitoring system.

The Cu/Pb and Zn concentrates would be loaded into bulk bags and then placed into containers within a roof covered area.

18.2.5 Fresh water source

Mill fresh water would primarily be obtained from wells and the effluent treatment plant water pond.

18.2.6 Tailings storage facility

The PEA assumes that the mill tailings would be pumped approximately 1 km to a 1.7 Mt capacity tailings storage facility (TSF) developed in a north-east trending valley. The tailings would be constrained by the natural topography and engineered containment structures at the north-eastern end of the TSF. The TSF would be engineered to withstand design precipitation and seismic events.

Ditches would be used to divert clean run-off around the TSF. Tailings drainage would be collected in an engineered pond and pumped to a conventional lime-based technology effluent treatment plant for metal precipitation, effluent pH adjustment and suspended solids settling removal.

The TSF would be developed in phases over the project life. The tailings which are expected to be net acid generating would be kept sufficiently moisture saturated over the operating life of the mine to inhibit sulphide oxidation and acid generation. The tailings would be capped at closure.

18.2.7 Mine water pipeline

The mine development costs include the excavation of mine sumps and pump installations. The mine dewatering pump system would include a back-up set of pumps and a spare dewatering pipeline. One set of pumps would be used to handle normal mine water inflows. The back-up set of pumps would allow the mine to double its dewatering capacity in the event of additional inflow. The water would be pumped to a lined pond and reclaimed for treatment.

18.2.8 Final effluent treatment

The PEA assumes a proven, conventional lime-based final effluent treatment process including suspended particle settling and treated water pH adjustment would be used to produce a final treated effluent that is acceptable for discharge. The PEA includes a treated water reclaim pump and pipeline to allow for treated water re-use.

18.2.9 Mine shop

The proposed mine shop would be constructed using shipping containers and a roof structure. It would be equipped with a reinforced concrete floor, lighting, work benches, and an oil storage container, hydraulic work clean room container, warehousing areas, and a separate equipment wash bay. The maintenance office, lunch room and washrooms would also be containerized. The estimated shop costs also include the mine air compressor, shop tools and spill prevention / spill kit costs.

18.2.10 Fuel storage and dispensing

Fuel would be stored in commercially available double walled steel tanks equipped with dispensing meters and hoses. The fuel station which would be equipped with fire extinguishers and a spill kit would be routinely inspected.

18.2.11 CRF plant

The proposed CRF plant would be constructed about 200 m from the Lower North zone portal and include: a bulk cement storage silo; a rock feed screening grizzly designed to remove +152 mm (+6 inch) rock particles, a cement slurry mixer with metered water supply; and a CRF mix /truck loading unit. The rock screening, mixing and CRF discharge areas of the facility would be roof covered. Mine development waste rock would be trucked to surface and stockpiled adjacent to the CRF plant and reclaimed using a wheel loader. It is projected that scheduled mine waste rock production would need to be supplemented using imported non-acid generating rock.

18.2.12 Mine office complex

The mine supervisors, trainers, nurse and technical personnel would work from the mine office. This complex would also include a training classroom, mine rescue room, and separate male / female change rooms, and a miner cap lamp charging station. This complex also includes the kitchen and dining room.

18.2.13 Social office

Toachi’s social consultation group would work from its current office situated in the Palo Quemado village.

18.2.14 Gate house

The security gatehouse would be located at the main gate.

18.2.15 Construction office

Construction management and technical staff would work from office trailers in Year -1.

18.2.16 Construction camp

A 25 person capacity camp would be constructed in Year -1.

18.2.17 Communications

The project would have phone, mobile, internet and radio services. The mine would use a radio system.

18.2.18 Potable water

Potable water for the project would be obtained from a well and treated using a commercially available potable water treatment system.

18.2.19 Grey water and sewage treatment

The proposed Project would use a commercially available grey water / sewage treatment system.

18.2.20 Port

It is assumed that Cu/Pb and Zn concentrate super sacks would be loaded into containers, road transported to Guayaquil Port, and ocean transported to smelter ports in Asia.

19.0 Market Studies and Contracts

19.1 Products

The PEA assumes that the Project would generate income from the sale of a copper/lead concentrate and a zinc concentrate to smelters in Asia. The projected metal concentrations in the Cu/Pb and Zn concentrates are shown in Tables 63 and 64 respectively.

Table 63. Cu/Pb concentrate - estimated metal concentrations

Metal	Estimated total amount of metal reporting to the Cu/Pb concentrate over the life of mine	Estimated average metal concentrations in the Cu/Pb concentrate
Gold (Au)	90,000 troy oz Au	0.4 troy oz Au/t concentrate
Silver (Ag)	1.26 Million troy oz Ag	5.8 troy oz Ag / t concentrate
Copper (Cu)	100.4 million lbs Cu	461 lbs Cu / t concentrate
Zinc (Zn)	30.1 million lbs Zn	138 lbs Zn / t concentrate
Lead (Pb)	15.7 million lbs Pb	72 lbs Pb / t concentrate

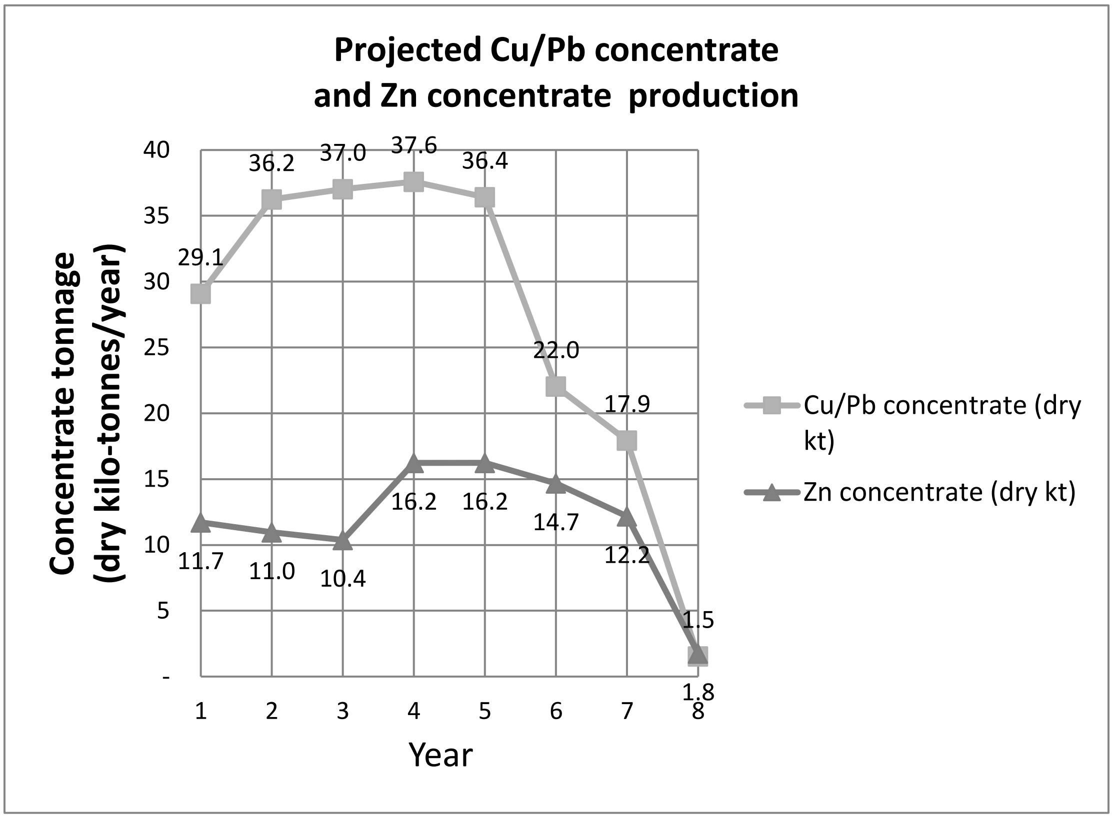
Table 64. Zn concentrate - estimated metal concentrations

Metal	Estimated total amount of metal reporting to the Zn concentrate over the life of mine	Estimated average metal concentrations in the Zn concentrate
Gold	55,000 troy oz Au	0.6 troy oz Au/t concentrate
Silver	687,000 troy oz Ag	7.3 troy oz Ag / t concentrate
Zinc	106.3 million lbs Zn	1,129 lbs Zn / t concentrate

Metal	Estimated total amount of metal reporting to the Zn concentrate over the life of mine	Estimated average metal concentrations in the Zn concentrate
Copper	5.1 million lbs Cu	53 lbs Cu / t concentrate
Lead	2.9 million lbs Pb	31 lbs Pb / t concentrate

The estimated concentrate annual tonnages shown in Figure 52 are based on the mine schedule.

Figure 52. Projected concentrates production



19.2 Product marketing

Metallurgical testing has produced saleable Cu/Pb and Zn concentrates.

<u>Copper/lead concentrate</u>

The copper/lead concentrate would be transported to a copper smelter for treatment. For the purposes of this PEA, it is assumed that the copper concentrate would be bagged and loaded into containers and trucked to the Guayaquil port and ocean-transported to a copper smelter port in Asia. It is assumed that Toachi Mining would enter into an agreement with the smelter and receive payment in US dollars subject to treatment charges (TC), minimum deduction, penalties, smelter schedule payables, refining charges (RC), concentrate transportation and handling costs, and other commercial terms.

<u>Zinc concentrate</u>

The zinc concentrate would be transported to a zinc smelter facility for pyro-metallurgical or hydro-metallurgical treatment. This PEA assumes that the zinc concentrate would be bagged and loaded into containers and trucked to the Guayaquil port and then ocean-transported to a zinc smelter port in Asia. It is assumed that Toachi Mining would enter into an agreement with the smelter and receive payment in US dollars subject to smelter schedule treatment charges, minimum deductions, penalties, payables, concentrate transportation and handling costs, and other commercial terms.

19.3 Net Smelter Return

The net smelter returns for the concentrates was estimated taking publically disclosed information about smelter terms, smelter market conditions, and input from concentrate / metal traders into consideration. The PEA assumes that the concentrates would be sold under CIF (e.g. Cost, Insurance and Freight) conditions where Toachi would pay for the insured transport of the concentrates to the destination port(s).

Preliminary budget quotes were obtained from selected logistics companies in Ecuador for the transportation of containerized 25 t loads of bagged concentrate from the Project to Guayaquil Port, Ecuador to a proxy port (Quigdao Port, China).

19.4 Metal price forecasts

The three year trailing average metal prices used in this PEA are shown in Table 65.

Table 65. Projected metal prices used in the PEA

Metal	Projected Metal Prices used in the PEA[A]
Gold	$1,264/troy oz Au
Silver	$16.64/troy oz Ag
Copper	$5,909/t Cu, $2.68/lb Cu
Lead	$2,163/t Pb, $0.98/lb Pb
Zinc	$2,657/t Zn, $1.21/lb Zn

[A] Three year trailing monthly average prices to end of January 2019.

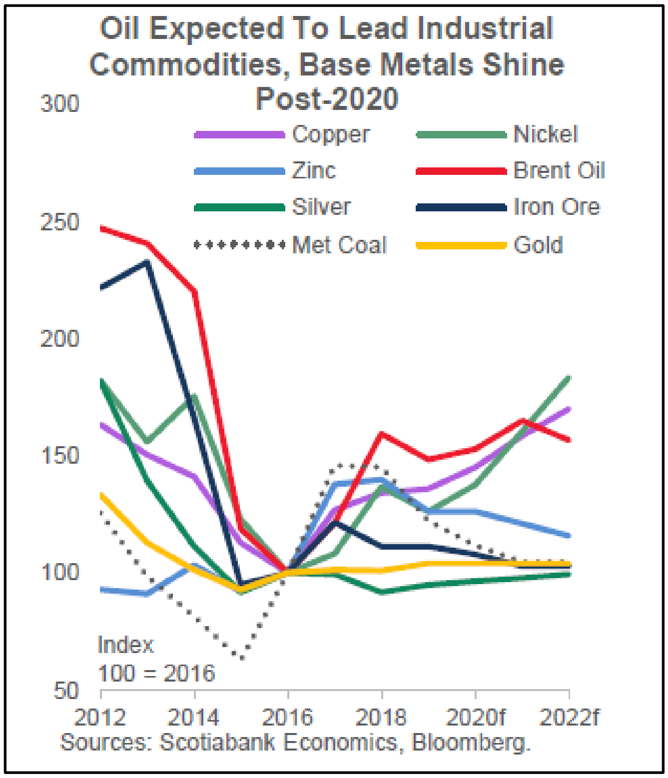
Metals price projections are the results of marketing studies that take into account metal supply and producers, anticipated consumption, global demand and supply, and specific aspects not limited to developments at producing operations, new metal uses, existing inventories, precious metal investments, recycled metal, political uncertainties, foreign exchange rates, and other factors affecting metal trading prices, macroeconomics, etc.

Toachi Mining has not conducted a marketing study or analysis for the Project. SGS has reviewed publicly available information regarding forecast consumption and prices for gold, silver, copper, zinc and lead. SGS has determined that: publicly available information including current metal prices and price forecasts prepared by reputable organizations; the three year trailing average prices for gold, silver, copper, zinc and lead; and the results of metallurgical testing completed to date provide a sufficient basis for forecasting metal prices and assessing the Project's product market access also given that:

- The markets for the gold, silver, copper, zinc and lead are well established.
- The Project's copper and zinc concentrates are expected to be marketable to smelters internationally on a contract or spot market basis. The net smelter return estimates presented in Section 22 "Economic Analysis" include cost penalty allowances for minor elements in the copper and zinc concentrates.
- The proposed Project is small on a global scale and its output would not affect metal prices.
- The Project would receive payment for product sales in US dollars - Ecuador's adopted currency.

In January 2019, Scotiabank (2019) indicated that the prices of industrial commodities are expected to broadly rise over the coming years - see Figure 53.

Figure 53. Price trends (after Scotiabank (2019), Chart 1)



19.4 Contracts

Toachi has not entered into any contracts for mine development, mining, concentrating, transportation, handling, sales and hedging, or forward sales contracts or arrangements.

20.0 Environmental Studies, Permitting and Social or Community Impact

20.1 Summary

SGS has reviewed the proposed La Mina VMS Project (Project); the regulatory regime for the environmental assessment and permitting of mine projects; permits and authorizations; citizen consultation and social license aspects; environmental reports; and potential impacts. In summary:

- The regulatory framework for environmental assessment and mine permitting is established.

- Toachi has the environmental license and authorizations needed to conduct advance exploration activities on the La Plata mineral concession which is a registered small scale (up to 300 tpd throughout) mining property. The PEA assumes that that the Project would produce 800 tpd for processing, and that Toachi would obtain a concession registration for the property as a medium scale (301-1,000 tpd throughput) mining property. There is a possibility that after clarifying medium scale mining registration requirements Toachi may find it preferable to apply for large scale (e.g. >1,000 tpd mined material) registration.

- Environmental studies conducted on the mineral concession indicate that the native fauna and flora and wildlife in the project area have mostly been exterminated by human activity. No species at risk were identified.

- The Project includes engineered controls designed to prevent or mitigate impacts to the environment and protect human health and safety.

- The net impact of the proposed Project is seen as positive.

It is internationally accepted that any new mining project needs to earn its "social acceptance to operate". This is often achieved through a proponents' community social engagement efforts, public participation in environmental assessment and approvals / permitting processes, the involvement of local people in social and environmental monitoring programs during operations, and the demonstrated good performance of the company. In Ecuador, one also needs to take additional aspects into consideration as examples:

- The 2008 Constitution assigned rights to people and Nature. Nature's rights can be defended by a person, entity or organization acting on behalf of Nature.

- The requirement to inform and then consult (e.g. prior consultation) with people about the environmental and social aspects of a proposed project is assigned to government officials by legislation. This allows a proponent such as Toachi from informing people about their potential project and soliciting their input.

- The legislation affecting the environmental assessment, permitting and regulatory oversight of mining projects has been reformed to include the involvement of local committees, the re-organization of some government departments, and the increased inclusion of citizens in decision making for mine projects.

- Ecuador is strongly encouraging and open to mining sector investments as indicated by the recent removal of the Windfall tax and the levels of interest and investment in its mining sector. The courts have been relied upon to make decisions on project-by-project basis - examples are a decision to suspend a project and a decision to cancel the referendum on another project.

Toachi has a social consultation Director and has proactively informed local people about its programs and is gaining support for its plans to develop a mine. Toachi reports that it has local public support for a project with few dissenters. Toachi plans to continue to consult with local people and others who could be potentially impacted by the Project. Toachi has begun to assess local social-economic aspects and identified that Indigenous persons account for 1% of the Palo Quemado township population.

The author is of the option that the legal framework needed to develop and operate the Project is in place, and the permitting and development of the Project will largely depend on the success of the company's community social consultation efforts. Delays could occur.

The proposed Project has been designed to a conceptual level and may be revised and improved when it is further evaluated in the next technical study stage. The PEA makes use of Inferred Mineral Resources which are not mineral reserves and there is no guarantee that the Project described in this PEA can be implemented.

20.2 The Project

The Project includes an underground mine and an on-site mill that would produce a zinc concentrate and a copper/lead concentrate that would be shipped out-of-country for treatment. As described in Table 66 the Project would use conventional technologies and equipment.

Table 66. Project concept

Item	Concept
Underground mine	The mine would use cut and fill mining methods. It is assumed that the stopes would be backfilled using cemented rock fill. The mine is scheduled to produce 240 kt in Production Year 1 and 288 kt/yr (800 tpd) starting in Year 2.
Mine life	• The Project would be developed in Year -1 and operated for 7.1 years.
Mill	• The proposed 800 tpd mill would produce two concentrates – a zinc concentrate and a copper/lead concentrate.
Concentrate processing	• The concentrates would bagged and placed in shipping containers and truck-transported to Guayaquil port and then ocean- transported to smelters in Asia.
Electrical power	• A power line would be constructed between an existing substation and a new substation at the mine property. There may be an opportunity to also make use of wind and solar powered energy at this site and the nearby hydropower plant under construction – see recommendations.
Mill tailings	• A new engineered tailings storage facility (TSF) would be constructed. The TSF would be designed taking design severe seismic and precipitation events into consideration. This PEA includes a recommendation to examine at other tailings storage options not limited to the use of dry stacked tailings or using paste tailings for mine backfill.

Item	Concept
Water management	• Non-contact water: Clean run-off will be diverted using ditches to natural drainage areas. • Contact water: Water collected from the mine workings, mill site and the tailings storage area would be collected and treated prior to being released or recycled. Excess water would be treated and tested before it is released to the environment. • Potable water: Drinking water would be obtained from a well on the property and treated.
Training, Health and safety	• Most workers would reside in communities within the region. Toachi would provide health and safety training to its employees. Toachi would also organize transportation for workers living in surrounding communities.
Consultation	• Toachi would continue to consult with people including Indigenous people who may potentially be affected by the Project.
Hazardous waste	• Toachi would have procedures designed to help it avoid producing hazardous wastes where possible. Used batteries or waste chemicals will be properly disposed of using licensed specialist contractors.
Mine closure	• At the completion of mining, the facilities would be sold or demolished and the site would be cleaned-up before it is allowed to return to a natural state. The site would also be progressively rehabilitated during operations.

20.3 Permitting

20.3.1 Licences and authorizations already in place

The licenses and authorizations that are already in place include:

- Environmental license 1477 for the advanced exploration phase of the La Plata metallic minerals concession issued by the Ministry of the Environment in November 2011. The company also holds water use and land use permits for its camp / shop area.

- The concession is a registered small-scale mining property.

- The La Plata concession was inspected by regulatory authorities in April 2018;

- The Ministry of the Environment (Ministerio del Ambiente) reported on conditions and requested an Environmental Remediation Plan for historic mine and mill wastes.

- The Water Secretariat SENAQUA (Subsecretaría de la Demarcación Hidrográfica del Guayas) concluded that water resources were not being impacted.

- The Mining and Regulation and Control Agency ARCOM (Agencia de Reglación y Control Minero) assessed health and safety and general site conditions and identified minor items that the company is addressing.

- The Ministry of Mining (Ministerio de Minería) also assessed site conditions and reported that the concession is not located in a protected area, the registered small scale mining status of this concession and that the concession owner plans to develop a remediation plan for current environmental liabilities after the completion of a feasibility study.

20.3.2 Projected permitting requirements

The currently projected permitting requirements for the Project are listed in Table 67. These requirements may change as the new Environmental Code is implemented and by other requirements that may be introduced by Autonomous Decentralized Governments.

Table 67. Projected permitting / Authorization requirements

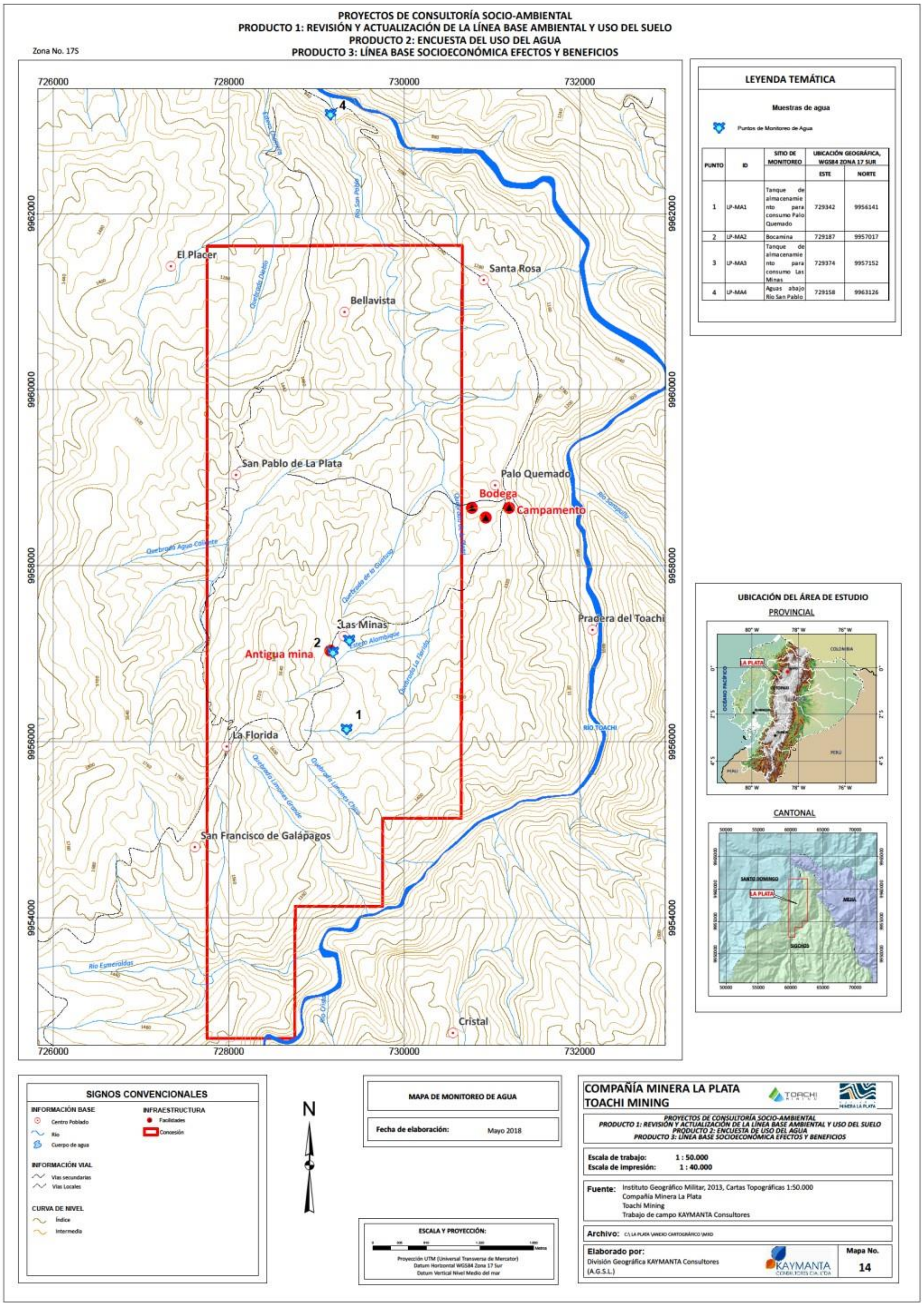
Projected Permitting / Authorization Requirements
Local Approvals:
Surface access agreements with land owners in the concession.
Regulatory permits / authorizations:
Certificate of validity of mining rights.
Mining easement.
Environmental license for mining and metallic mineral processing. Plus a supporting environment impact study and environmental management plan.
Authorization to develop and operate a process plant.
Authorization to develop and operate a tailings management area.
Water use permit- process water.
Water use permit- human consumption.
Authorization to extract borrow materials.
Authorization from a local municipality to deposit non-hazardous waste in its landfill.
Project and infrastructure construction permits.
Fire department approval for project facilities.
Hazardous waste generator registration.
Explosive storage / transportation/ blasting permits.
Fuel purchase permit.
National grid powerline:
Environmental permit for power line and substation
Agreement - Power line and substations.
Land access rights.
Construction permit.

20.4 Recent environmental and social studies

The environmental and social aspects of the concession have been studied as part of permitting processes. A general plan of the concession area showing local waterways and the Toachi Pilatón Protected Forest is shown in Figure 54. The Los Ilinizas ecological reserve (Reserva Ecológica Los Ilinizas) is located immediately south of the concession.

The results of environmental field studies to date indicate that the native flora and fauna in the concession area were largely exterminated by people over the years with relicts of secondary vegetation only found along watercourses. The abundance of terrestrial and aquatic wildlife fauna was found to be low with no species at risk.

Figure 54. Plan of the concession



20.4.1 Environmental and soil use base line

Toachi completed a qualitative update to the environmental and soil use base line for the concession in July 2018.

- As part of this work: the environmental aspects of the property were evaluated; areas of influence were updated; potential impacts were evaluated; current and applicable environmental regulations and information and knowledge gaps were assessed in comparison to requirements under the IFC (International Financial Corporation) Performance Standards on Environmental and Social Sustainability.

- Satellite images were used to compare land use and vegetative cover in 2012 and 2017. Information was also obtained from a 2016 Environmental Impact Study and a 2014-2016 environmental compliance audit. At the time of Toachi's study, its activities were mostly being carried out at its camp, shop and fuel storage area in the Palo Quemado village, situated in cantón Sigchos, Cotopaxi Province.

Toachi reports that the Sultana del Ecuador company's exploration activities were suspended in 2008 by the national Mining Mandate in effect at that time. Advanced exploration activities resumed in 2016 with additional drilling and an Environmental Audit of the concession was conducted in 2017. The report also indicates that:

- Significant changes had occurred in the biological and social baseline since the Environmental Impact Study in 2006. About 30 ha of forest had been converted to grassland / pasture or for growing crops such as sugarcane. Surface rights and surface land uses are separate from mineral patents.

- The social perception regarding mining activities was found to include differing points of view and opinions. Additional measures to maintain positive links with local people involved in these agricultural activities were recommended.

- The ecology in the concession area has been severely affected by economical activities other than mining. Patches of natural and secondary vegetation were found in the main rivers.

20.4.2 Hydrological study

Toachi recently completed a hydrological study that characterized:

- Local agricultural water use as well as rainfall and evapotranspiration levels.
- Projected project water use and possible water sources.

- The water requirements of the Toachi Pilatón hydroelectric project which would not be affected by the Project.
- Water quality at four sampling stations. Additional testing is required to assess the sulphide oxidation potential of historic mine rock that had been disposed on surface, and to predict mine water quality.

20.4.3 Socio-Economic base line

Toachi's socio-economic effect and benefit base line (línea base socioeconómica effecto y beneficios). report (Toachi, 2008b) provides useful information for the development of a comprehensive community consultation program. Eight inhabitants who were identified as an Indigenous person in the last demographic census represent 1% of the Palo Quemado township's population.

20.4.4 Community Consultation

In Ecuador, a person's right to be informed in advance (e.g. Prior Consultation) on environmental matters that can affect them / their community is enshrined in the 2008 Constitution. The consultation process must ensure that relevant information is circulated in a timely matter to people / communities so that they can express informed opinions. If the majority of the community opposes a project, a decision to undertake it must be taken by the maximum authority of the competent public institution through a motivated decision. Public consultation requirements are defined in legislation including the 2009 Organic Citizen Participation Law (Ley Orgánica de Participación Ciudadana).

Under Ecuadorian legislation, the need to conduct community prior consultation is exclusively assigned to the Ecuadorian State. Toachi has proactively engaged with local communities in order to inform the communities about its project and to gain / keep public support.

Consultative activities undertaken by government officials and Toachi have included as examples:

- In 2017-2018, the Ministry of Mining held two sessions. The first one was held in July 2017 in Palo Quemado – where the legal frame for mining in Ecuador, the importance of an envisaged mining project and the benefit this industry could bring to the community were reviewed. A similar meeting took place at the office of the Governor of Cotopaxi, at Latacunga in October 2017. There were peaceful demonstrations on both occasions.

- In 2017, CMLP's environmental director made two presentations on its Environmental Management Plan (Plan de Manejo Ambiental or PMA) for the then contemplated Project, and explained the plan components including civil control of the project, grievances policy, fulfillment of the PMA commitments and answered questions raised. The first meeting was intended for members of the Parish Council of Palo Quemado, while the second one was an open call for people to attend. The presentation also reviewed the local environment and good mining practices.

- Toachi has consulted with people in each of the neighborhoods located in a possible direct influence zone.

Toachi has made pubic commitments including:

- Maintaining a section of road linking La Unión del Toachi with Las Pampas;
- Providing furniture for use at a future nursing home;
- Supporting the Ecuadorian Electric Corporation's (CELEC EP) plans to provide potable water to a local community. CELEC EP is developing the Toachi-Pilatón hydro-power project.

Toachi has entered into numerous agreements with local landowners. During its 2016-2018 exploration campaign, the company signed compensation agreements with nine landowners in regard to establishing diamond drilling platforms on their properties. Another 50+ land owner agreements were obtained to enable mineral exploration traverses along creeks located on private land within the mineral concession.

20.5 Mine Closure (remediation and reclamation)

A conceptual mine closure strategy has been developed for the purpose of this PEA based on mine closure requirements defined in 2016 environmental regulations for mining activities (reglamento ambiental de actividades mineras). Mine operators are required to fund the closure of their projects. The estimated cost to close the Project at the completion of mining and mineral processing is shown in Table 68.

The overall objective of the closure works is to leave the closed site in safe and physically and chemically stable condition and allow it to return to a natural state for possible use by local people and wildlife. The closure works would be based on a closure plan that would be progressively refined over the life of the project. Toachi has initiated a laboratory testing program to assess the metal leaching and acid rock drainage potential of the historic mine wastes on the property and plans to characterize the waste rock and mineralized rock zones that would be exposed by the Project.

Table 68. Estimated closure cost

Item	Closure Activity	Estimated Cost (US$)
Mine	• Remove equipment, and unused fuel, oil, explosives, detonators, parts, tires and waste materials such as waste oil from the underground mine. • Salvage mine pipe and cables. • Block the mine portal entrance. • Construct engineered caps over raise collars at surface.	$500k
Mill	• Remove wastes and unused reagents. • Demolish mill. • Clean-up the mill site and cover with clean soil and re-vegetate.	$1,500k
Tailings	• Cover the tailings. The tailings cover would be designed to prevent or mitigate metal leaching and acid generation.	$1,000k
Infrastructure	• Remove the infrastructure including the power line and substations, offices, shops, fuel and chemical storage facilities, etc.	$1,000k
Other	• Water management and treatment.	$500k

Item	Closure Activity	Estimated Cost (US$)
	• Environmental monitoring during closure works. • Post-closure environmental monitoring.	
	Total estimated closure cost	$4.5 million

20.6 Regulatory Regime

20.6.1 Current status

The country has a regulatory regime for the environmental assessment and permitting of mine projects which is continuing to be refined. While the government is strongly promoting Ecuador as a mining investment destination, the sector has experienced challenges. Valencia (2019) reports that the implementation of two mining projects have stalled due to the constitutional requirement to get approval from local communities where:

- Operations at the Rio Blanco gold-silver mine project were suspended in 2018. It had been argued that the mine had failed to have a prior consultation with Indigenous communities as required by the Constitution and international law.

- The Loma Larga gold project. The development of the mine was temporarily stalled by a requirement to hold a referendum on the project which was followed by a legal challenge. On March 13, 2019 INV Metals Inc. reported the immediate suspension of a referendum on the Loma Larga project.

Two mines projects that being developed towards production are:

- The Fruta del Norte gold mine. First gold production expected in late 2019.

- The Mirador copper project. Wood Mackenzie (2019) reports that the construction of this 60,000 tpd project commenced in late 2015 with commissioning projected to start in late 2019 with a latent risk of delays.

20.6.2 Regulatory regime developments

The country's regulatory regime for mining projects is continuing to be refined reflecting political, social, economic and cultural changes. The key drivers for recent changes have included but are not limited to: increasing interest in the country's rich natural resource base; a need to increase government revenues; government's encouragement of large scale mining project investments; rights assigned in the 2008 Constitution including the assignment of rights to nature; public memories of earlier times when Indigenous people and others were not consulted; and developments in the country's oil and mineral sectors.

Events and developments that shaped the regulatory regime are described below.

The 1960's

The discovery of oil in Ecuador's Amazon region in 1967 transformed the economy which was previously dependent on banana, cacao, coffee and sugar exports.

The 1970's

A military junta overthrew the national government in 1972. The leader was replaced by another military government in 1976. Ecuador returned to a constitutional based system of government and elected a new president in 1979.

The 1990's

In 1991, the reformed Mining Law established a mechanism to access government mining concessions, and delineated the environmental obligations of government and concessionaires (World Bank, 2007).

The 2000's

Ecuador adopted the U.S. dollar as its national currency in 2000. In 2007, Rafael Correa was elected President following a decade of political and economic instability. Examples of his successes are: i) He declared the country's national debt illegitimate arguing that it had been contracted for by corrupt and despotic prior regimes and he eventually had the debt significantly reduced; ii) He obtained public approval, via a referendum, for the new Constitution introduced in 2008; and iii) His government heavily invested in the country's infrastructure.

- The 2008 national Constitution recognizes a citizen's right to a healthy environment, and provides separate rights for nature. As a result, a citizen can protect their right to a healthy environment, and a person, community or nationality can assert constitutional rights pertaining to nature by applying the principles enshrined in the Constitution. Ecuador was the first country to enshrine the rights of nature into its constitution. Other environmental rights and obligations established by the Constitution as identified by Beveridge and Diamond (2009) include but are not limited to the right to water; prohibitions on the introduction of persistent organic pollutants and genetically modified organisms; and Indigenous rights related to the use of natural resources and biodiversity.

- In 2008, a Mining Mandate suspended exploration activities for 6 months or until a new Mining Act could be approved (USGS, 2010). This resulted in the revocation of 80% of the nation's mineral concessions with the balance suspended, and the cancellation of mineral concession applications in process at that time. In late 2008, the President announced that a mining bill would be sent to the legislative commission of Ecuador to change the 3% to 8% mineral sales royalty to at least 5% without specifying a maximum amount, and require exploitation and services contracts between mining companies and the government. A separate Windfall tax would be levied on revenue earned from metals sales subject to a fixed price(s) to be established in a contract with the government.

- In early 2009 a new Mines Law (2009 Law) classified mineral exploration as initial or advanced. An initial stage would last for four years with an opportunity to apply for two 2-year extensions in the advanced stage. No single concession could hold more than 5,000 hectares during the exploration and exploitation stages. After initial exploration a concessionaire could be required to reduce their concession size, with a further reduction following advanced exploration, to meet guidelines that had not yet been defined. Concessionaires needed to negotiate exploitation contracts with government on a case-by-case basis and submit environmental impact studies to obtain permits.

- The Mining Law established in 2009 provided a new administrative and regulatory framework for government oversight and authority over the mining sector. Oversight and enforcement responsibilities were given to a Mining Regulation and Control Agency, and responsibilities for mining information and technologies were assigned to a National Institute for Geological Investigation. The law also created a National Mining Company.

- Beveridge & Diamond (2009) reports that under the 2008 Constitution the State has patrimony over non-renewable resources, and mining rights may be transferred through procedures including public tendering and auctions for concessions, contracts for mining extraction operations, and licences and permits for exploration and extraction operations. Concessionaires' obligations included the need to: conduct environmental impact assessments; comply with water quality and waste management requirements; community consultation requirements; obtain agreements with surface rights holders; and make payments to the government based on a percentage of the sale of materials extracted from the concession. The government may declare the expiration of a concession for reasons listed in the Law, and has the right to nullify titles to mining rights for violation of the Law.

<u>2010 - 2017</u>

- Kinross Gold Corp. (Kinross) acquired the Fruta del Norte gold project in Ecuador for US$1.2B in 2008. The government's insistence on a 70% windfall profits tax led Kinross to decide to abandon its project in 2013. Kinross subsequently sold its Fruta del Norte project to Fortress Minerals Corp. a member of the Lundin Group of Companies (Lundin) for US$0.24B in 2014. LePin (2017) reports that Lundin obtained approval for the Fruta del Norte environmental impact study in October 2016, and signed an exploitation agreement and an investment protection contract with the Ecuadorian government in December 2016 allowing construction to commence. Ecuador's economy contracted 1.6% in 2016 largely due to the combined effects of reduced depressed oil revenue, and earthquake reconstruction and humanitarian assistance costs.

Kinross' exit from its Fruta del Norte project in particular is considered to have caused the government to overhaul its mineral sector approach with input from consultants Wood Mackenzie and create a mining ministry in 2015.

The key aspects of Lundin's Exploitation Agreement for the Fruta de Norte project are (Smith, 2017):

- The right to develop and produce gold at Fruta del Norte for 25 years with renewal possible.
- An advance royalty payment of US$65M to Ecuador, plus two US$20M royalty payments due on first and second year signing anniversaries.
- A royalty equal to 5% of the net smelter revenue from production with advance royalty payments being deductible against future royalties.

- The 2009 Mining law was amended in July 2013. The General Regulations to the Mining Law, the Environmental Mining Regulations and the Regulations for Small-Scale Mining were issued in November 2009 and amended in 2014 and 2015.

- In 2018

  - A national referendum led to the abolishment of indefinite re-election; limited presidential tenure to two terms; prohibited mining in protected areas and urban zones; and expanded protection against oil development in a national park.

  - The government encouraged international investment in its mining sector as did the previous government. Winning social approval for mining projects and conflict with some local communities are acknowledged challenges.

  - The Moreno government had to address social challenges at the Rio Blanco mine, canton Cuenca, province of Azuay. The mining license for this medium scale project was revoked in June 2018 reportedly due to a possible need for more public consultation, concerns about its location in relation to a protected forest and National Park, and potential impacts to surface water quality. This project has since been suspended.

  - Ecuador's new Environmental Code came in effect in April 2018 and essentially replaced the country's previous environmental legislation. A number of the key aspects of the new law as identified by PHP (2017) are summarized in Table 69.

Table 69. Selected aspects of the Environmental Code

Area	Aspect
Permits, environmental studies and authorizations	The types of permits, procedures, environmental studies and administrative authorizations are to be determined via secondary law. The Code makes references to: administrative authorizations; good environmental practices guidelines; environmental impact studies; environmental management plans; insurance policies or security to cover environmental liability; the shutdown of operations; activity suspensions; permit or environmental authorization revocation; and environmental audits.
Type of environmental permit	A project's potential environmental impact will be classified as significant, low, moderate, or high and used to determine the type of environmental permit that should be granted.
Project Operator obligations	The operator of a project is obligated to prevent, avoid, reduce, and, whenever possible, eliminate the environmental impact and risk that this activity could cause. When an impact is inflicted on the environment, the operator will establish mechanisms to restore it.
Guiding principles	• *In Dubio Pro Natura* principle - When in doubt regarding the scope of legal provisions, chose the one that most favors the environment and nature. • *Pollutor Pays* principle - The Code establishes the obligation to provide comprehensive redress and to indemnify injured parties, and pay the respective sanctions.

Environmental incentives	Ecuador is refining its legislation for the electrical energy sector and is strongly supportive of protecting air quality and fighting climate change. As indicated in the new Environmental Code, Ecuador's new national environmental authority will develop guidelines for evaluating and granting possible environmental incentives.
Autonomous Decentralized Governments	Autonomous Decentralized Governments (GADs, for their initials in Spanish) with jurisdiction will monitor and regulate environmental management, primarily by focusing on: • Civil works—minimize visual impact or impact to the landscape. • Activities that affect ground quality or stability. • Hazardous chemical substances. • Comprehensive waste and residue management. Environmental administrative violations of the Code's environmental rules will be classified as minor, serious, or very serious. Administrative violations will be sanctioned in accordance with their seriousness, the offender's recidivism, the impact or magnitude of the violation, the offender's financial capacity, and mitigating or aggravating factors (PBP, 2017). The Code also provides for liability for environmental damages (LSE, 2017).

Sources: PBP(2017) and LSE(2017)

21.0 Capital and Operating Costs

This section provides the estimated capital and operating costs used in the PEA cashflow.

The PEA is preliminary in nature. The projected capital expenditures comprise upfront capital costs (incurred in Year -1 and Year 1, Q1) and sustaining capital costs. The projected operating costs include direct and indirect operating costs including but not limited to direct labour and supervision costs, maintenance and other indirect labour costs, consumable costs, equipment operating and maintenance costs, and electrical power costs. The General & Administration (G&A) costs include project management, engineering and geology, site security, and site kitchen personnel costs and related project-wide operating costs. Costs have been estimated to ±35% accuracy.

21.1 Capital Expenditures

The capital expenditure estimates were developed using: from first principles estimates; preliminary budget quotes for key equipment and consumables; factored estimates; and cost allowances based on experience on other projects and project requirements. The capital cost estimates include, as relevant, direct labour costs; direct materials costs; equipment purchase and/or lease costs; equipment operating costs; maintenance labour and consumable costs; and indirect personnel and indirect project operating costs. The PEA does not include salvage values.

The estimated capital expenditures exclude the following:

- Sunk costs not limited to costs that would be incurred to obtain required surface rights and/or access.
- Pre-feasibility / feasibility study costs including associated diamond drilling and assay costs; geotechnical and environmental investigations and evaluation costs; additional metallurgical sampling and testing costs; concentrate marketing and agreement costs; the costs of other engineering, surveys and technical evaluations; and technical studies costs incurred prior to a potential production decision.
- Surface land access and property acquisition costs.
- Community consultation, social and environmental assessment, and project licensing/permitting costs.
- Value added taxes and import duties.

21.1.1 Upfront CAPEX

The upfront capital costs are summarized in Table 70.

Table 70. Upfront CAPEX summary

Item	Upfront CAPEX 15 month preproduction time line		
	Year -1	Year 1 Q1	Total
Mine capex			
• Mine equipment lease costs	$4.25M		$4.25M
• Mine pre-production development costs	$5.32M		$5.32M
Mill capex			
• Mill capex	$43.03M	$7.17M	$50.2M
Tailings storage facility (TSF)			
• TSF capex	$2.00M		$2.00M
• Effluent treatment capex	$0.95M		$0.95M
Infrastructure			
• Mine infrastructure	$3.52M		$3.52M
Indirect costs	$3.95M		$3.95M
Closure bond (Payment in Year -1)	$0.10M		$0.10M
Total pre-production CAPEX	**$63.1M**	**$7.17M**	**$70.27M**

Totals may not sum correctly due to rounding.

21.1.2 Mine equipment lease and purchase costs

The major pieces of mine equipment would be leased under assumed 20% down payment, 8%/yr interest, 72 month, and lease-to-own terms. In the year that a piece of equipment is acquired, the lease costs would encompass a 20% down payment, the equipment transportation and assembly costs, and the first 12 months of lease payments. Additional monthly lease payments would be made in the subsequent 60 months. The estimated equipment costs are based on preliminary budget quotes and other reference cost data. Other equipment such as jackleg and stoper drills, mine rescue equipment and mine ventilation fans would be purchased.

21.1.3 Mine development costs

The mine development requirements (i.e. metres of ramp development, metres of stope access drifts, metres of raise, etc.) are based on the preliminary mine layout and were scheduled along with proposed stoping. The mine development quantities include an allowance for the development of muck bays, sumps, and other cut-outs. The estimated mine development costs in a year are based on the scheduled metres of development and relevant mine development unit costs.

Mine development unit costs were estimated from first principles and include the following direct cost components: direct labour; drilling consumables; explosives, blasting agent and detonators; ground control materials; mine services (i.e. process water, dewatering and compressed air lines; power cable; isolated blasting cable, communications cable, ventilation ducting as applicable); equipment lubricant, parts, tires, and wear parts; and other items (i.e. signage, road bed dressing material, and grab samples of development rock for acid rock drainage / metal leach characterization testing). The unit costs for explosives, blasting agent and detonators and ground support materials were based on preliminary budget quotes. Key unit development costs are shown in Table 71.

Table 71. Key mine development unit costs

Item	Description	Estimated unit cost	
Mine ramp	5 m high x 4.5 m wide. Gradients from -15% to +15%.		**Direct costs $1,890/m**
		Mine indirects (hourly personnel)[A]	$275/m
		Mine indirects (staff & indirect OPEX)[B]	$415/m
		Mine power (genset power)[C]	$265/m
		Mine equipment lease costs[D]	$1,455/m
			Total $4,300/m[E]
Stope access heading	5 m high x 4.5 m wide. Gradients from -15% to +15%. (Excludes back slash cost)		**Direct costs $1,795/m**
		Mine indirects (hourly personnel)[A]	$275/m
		Mine indirects (staff & indirect OPEX)[B]	$415/m
		Mine power (genset power)[C]	$265/m
		Mine equipment lease costs[D]	$1,455/m
			Total $4,205/m[E]
Raise	Ventilation/escapeway raise. 50° to 72° inclination.		**Direct costs $3,900/m[E]**

[A] Mine indirects (hour personnel) costs are $803k in Year -1. At 8m/day, the estimated cost per metre advance in Year -1 is approximately $275/m.

[B] Mine indirect (staff & indirect OPEX) costs are $1,210k in Year -1. At 8m/day, the estimated cost per metre advance in Year -1 is approximately $415/m.

[C] A diesel-powered generator would be used to generate electrical power for mine development in Year -1 at an estimated cost of $773k. At 8m/day, the estimated cost per metre advance in Year -1 is approximately $265/m.

[D] The estimated mine equipment lease costs are $4,248k in Year -1. At 8m/day, the estimated cost per metre advance in Year -1 is approximately $1,455/m.

[E] The PEA assumes the mine development work would be done by Toachi using its own labour and equipment.

21.1.4 Mill capital costs

A summary of the upfront capital costs for the process plant and associated infrastructure is shown in Table 72. This cost estimate was based on equipment costs and multiplied by factors for installation.

Table 72. Processing Plant CAPEX

Item	Plant CAPEX estimate
Processing plant:	
Area 10: Crushing, conveying, stockpile	$1,768k[A]
Area 15: Grinding	$6,890k[A]
Area 25: Copper/lead flotation	$3,330k[A]
Area 26: Zinc flotation	$3,169k[A]
Area 27: Multiplexor	$1,211k[A]
Area 30: Concentrate thickening & filtration	$3,043k[A]
Area 60: Tailings thickening	$551k[A]
Area 65: Reagents	$720k[A]
Area 66: Water distribution on-site	$655k[A]
Area 67: Plant air	$136k[A]
Subtotal	$21,473k
Plant direct costs:	
Site development	$1,534
Process & overland piping on-site	$1,534
Buildings (process & non-process)	$2,300
Electrical power distribution at Process Plant	$1,534
Water source and transmission to site	$650k
Total direct costs	$29,025k
Plant indirect costs:	
EPCM (12% of direct cost)	$3,483k
Construction indirect cost (10% of direct cost)	$2,903k
Owner's costs	$2,322k
Spare parts	$767k
Initial fill and reagents	$435k
Equipment insurance & freight cost	$1,227k
Total indirect costs	$11,137k
Total direct and indirect costs	$40,162k
Contingency at 25%	$10,040k
Total	$50,202k $50.2M
Plant CAPEX would be incurred over 15 months.	
Plant CAPEX distribution used in PEA cashflow:	$43.0M in Year-1
	$7.2M in Year 1, Q1

[A] Installed equipment cost shown in Table = estimated purchased equipment cost x 1.40 factor

The Plant direct capital costs were based on the following SGS documents:

- Preliminary design criteria
- Equipment list
- Process flowsheet
- Mining cost service source quote data for minor equipment
- SGS engineering equipment database for recent similar projects
- Preliminary budget quotations from vendors for major equipment
- Conceptual drawings performed by SGS

The direct costs exhibited in this estimate include, but are not limited to, labour, equipment and materials for the construction activities set forth below:

Equipment costs

An equipment list was developed and incorporated into the cost estimate. The estimate for equipment was developed from the following sources:

- Written or e-mailed budgetary estimates from vendors for major equipment.
- Historical data and budget costs from recent similar projects for miscellaneous equipment.

The cost for "installed equipment" was estimated using a factor of forty percent (40%) of purchased equipment costs. This factor reflects typical costs to install equipment and covers labour, concrete foundations, steel, and other services and construction materials associated with equipment foundations, erection, and placement.

Site Development

General site development costs include excavations, backfills, grading, roads, and fencing. The initial construction site development cost was estimated using a ten percent (10%) factor of the purchased plant equipment cost. The factor was selected based on the mountainous nature of the proposed project site, and the type of native soils in the area. The project would require development at the following major locations:

- On site access roads
- Primary crusher area
- Stockpile area
- Process plant areas

Process and Overland Piping On-Site

Process piping costs include materials and installation of all piping within the process plant. The initial process piping cost was estimated using a ten percent (10%) factor of purchased equipment costs. The factored costs were based on the size of the plant, as well as the distribution of piping within the plant area. The water system included pumping and piping to distribute approximately 7.5 liters per second on a continuous basis. Fresh, potable and process water pumps, storage tanks and distribution pipelines are included in the capital cost estimate.

Buildings (Process and Non-Process)

Building costs include materials, labour, and other miscellaneous costs associated with erecting covered structures within the project site. The initial construction building cost was estimated using a 15% factor of the purchased plant equipment cost. The factor was selected to reflect the projected costs of the buildings based on building type and square footage. The project would require the following buildings:

- Grinding and flotation area
- Control rooms and offices
- Tailings filter area
- Reagent storage area
- Laboratory

The main office complex including change rooms and the main shop, which are not included in this estimate, would be shared by the mine and plant.

Electrical Power Distribution at Process Plant

Electrical distribution costs include transformation, wiring, cable tray, instrumentation, lighting and grounding within the process plant. The initial electrical cost was estimated using ten percent (10%) of purchased equipment costs. The factors were selected based on preliminary equipment power requirements. The project would require the following electrical power distribution items:

- 4.16 kV distribution on site (switchgear, distribution lines)
- Pad-mounted distribution transformers
- Medium voltage (4160 V) switchgear
- Low voltage (480 V) motor control center
- Back-up diesel generators for essential systems

The Main substation transformers and circuit protection would be provided by the utility.

Plant construction indirect costs

Certain indirect costs exhibited in this estimate include, but are not limited to, labour, equipment and materials for the detailed activities set forth below:

- EPCM for the process facilities and associated infrastructure was estimated using 12% of the direct costs and includes detailed engineering, procurement and construction management costs.

- Construction indirect costs for the initial construction and mill expansion were estimated using a ten percent (10%) factor of the total direct costs and includes: construction supervision, equipment rental, field office expenses, mobilization / demobilization, and consumables.

- Owners cost for the plant are estimated using an eight percent (8%) factor of the total direct costs.

- Spare parts costs were estimated using a five percent (5%) factor of the purchased plant equipment cost.

- Initial fill and reagents costs were estimated using a one and one-half percent (1.5%) factor of the total direct costs.

- Equipment insurance and freight costs were estimated using an eight percent (8%) factor of the purchased plant equipment costs.

Process plant contingency and accuracy

The SGS crushing and process plant portion of the cost estimate includes a 25% contingency for project unknowns and identified risks. Contingency is a necessary part of the cost estimate and is based on the fact that less than three percent (< 3%) of the engineering is completed to date. SGS believes the estimated contingency amount will be spent during the construction period of the plant site and associated infrastructure for identified risks and unknown items.

While SGS has not performed a statistical analysis of the crushing plant and process plant accuracy of the capital cost estimate, SGS has a high confidence, based on previous experience with similar projects, that the accuracy of the process portion of the PEA capital cost estimate will end up between minus thirty-five percent and plus thirty-five percent (-35% / +35%) of the SGS capital cost estimate.

21.1.5 TSF and effluent treatment plant capital expenditures

The mill tailings would be pumped approximately 1 km to a 1.7 Mt capacity tailings storage facility (TSF) constructed in a north-east trending valley. The tailings would be constrained by the natural topography and engineered containment structures at the north-eastern end of the TSF. Clean runoff from adjacent areas would be diverted around the TSF using diversion ditches. Tailings drainage and TSF seepage would be collected and pumped to a conventional lime-based technology effluent treatment plant for treatment prior to recycle or release. The TSF would be designed taking severe precipitation and seismic design events into consideration.

The TSF would be constructed in phases over the life of the Project. The tailings which are expected to have a net acid generating potential would be kept sufficiently saturated over the operating life of the mine to inhibit sulphide oxidation and acid generation. The tailings would be capped using an engineered cover at closure. The estimated TSF capital expenditures are summarized in Table 73.

Table 73. Estimated TSF and effluent treatment plant CAPEX

Year	Projected capital expenditures	
	TSF	Effluent treatment plant
-1	$2.00M	$0.95M
1	0.39M	
2	0.39M	
3	0.39M	
4	0.39M	
5	0.39M	
6	0.39M	
7	0.39M	
Total	$4.73M	$0.95M

21.1.6 Infrastructure capital expenditures

The estimated infrastructure capital expenditures are summarized in Table 74.

Table 74. Projected infrastructure CAPEX

Item	Estimated infrastructure CAPEX	
	Year -1	Year 1
Access road improvements	$196k	
Power line from the national hydropower grid to new substation that would be constructed near the mill, a pole line to the mine substation, and the mine substation	$1,697k	
Construction office and furnishings	$126k	
Fuel storage	$40k	$43k
Mine complex - office, training, mine rescue room, change house and site kitchen	$502k	$105k
Mine shop including work benches, tools, a separate wash bay, and storage containers	$503k	$70k
Mine camp (25 rooms)	$250k	
Potable water supply/septic system	$140k	
Communications system	$71k	$234k
Total	$3,525k $3.52M	$452k $0.45M

21.1.7 Project indirect costs in Year -1

The indirect costs in Year -1 are shown in Table 75. Additional information is provided in Tables 76 to 78.

Table 75. Estimated indirect costs in Year -1

Item	Estimated indirect costs in Year -1
Indirect labour (hourly) cost	$803k
Diesel generator cost[A]	$773k
Mine supervision and support personnel cost	$1,210k
G&A cost	$1,165k
Total	$3,951k $3.95M

[A] A diesel generator would provide electrical power for mine development work in Year -1. The generator lease cost is included in the mine equipment cost. The generator operating cost in Year -1 is $773k or $0.16/kWh.

Table 76. Indirect labour (hourly) cost

Indirect labour (hourly) personnel	Number on payroll	Estimated Cost in Year -1
Lead blaster	6	
Blaster helper	6	
LHD operator	6	
Truck driver	12	
Total	30	$803k

Table 77. Mine supervision and support personnel cost

Item	Indirect labour	Total number on payroll[A]	Estimated Cost in Year-1
Mine Supervision and training	Mine supervisor	6	
	Trainer	2	$348k/yr
Technical Services	Mining engineer	1	
	Mine surveyor	1	
	Mine geologist	1	$166k/yr
Mine maintenance	Maintenance supervisor	2	
	Maintenance planner	1	
	Lead mechanic	2	
	Mechanic	8	
	Welder	2	
	Lead electrician	2	
	Electrician	3	
	Instrumentation technician	1	$425k/yr
Hand tools, safety, daily travel and meal costs			$204k/yr
Indirect vehicle operating costs			$30k/yr
Mine fan, pump and compressor maintenance costs			$37k/yr
		Total	$1,210k/yr

Table 78. G&A cost in Year -1

Area	G&A staff position	Number on payroll	Estimated G&A cost in Year-1
Administration	Toachi General Manager	1	$388k
	Accountant	1	
	Payroll	1	
	Purchasing	1	
	Human Resources	1	
	Clerk	1	
	IT Technician (part time)	1	
Health-Safety-Environmental-Social (HSES) Security Kitchen	HSES Manager	1	$459k
	Environmental technician	1	
	Nurse	1	
	Social Director	1	
	Community Social Worker	1	
	Security	4	
	Kitchen staff	3	
Safety, daily travel and meal costs			$67k
G&A vehicle operating costs			$11k
Project-wide indirect costs			$240k
		Total	$1,165k

21.2 Operating Costs

21.2.1 Operating cost summary

The operating costs are summarized in Table 79.

Table 79. Operating cost summary

Item	Estimated LOM operating cost ($/t milled)
Mining cost	$37.94/t
Mine equipment lease cost	$8.21/t[A]
Processing costs	$26.62/t
Tailings disposal cost	$0.53/t[B]
Effluent treatment cost	$0.70/t[C]
Subtotal	$74.00/t
G&A	$5.15/t
Total	$79.15/t

[A] Mine equipment lease costs are included as a sustaining capex item in the PEA cashflow.
[B] Tailings are placed in an engineered TSF.
[C] Effluent would be treated using a conventional lime-based process.

21.2.2 Mine Operating Costs

The mining costs were developed from first principles assuming that the mine would operate on a three 8 hour shift per day, seven days per week basis with crew rotation. The LOM average mining cost is summarized in Table 80.

Table 80. Mining cost summary

Mining direct cost component	Estimated LOM average cost $/t milled
Direct stoping cost	$18.39/t
Direct cemented rock fill (CRF) cost	$7.75/t
Stope indirect labour	$3.16/t
CRF backfill labour and operating cost	$1.07/t
Mine staff, support personnel & indirect costs	$5.68/t
Electrical power cost	$1.89/t
Total	$37.94/t

Direct stoping costs

The $18.39/t milled direct stoping cost shown in Table 81 is the weighted average of the mining costs shown in Table 82.

Table 81. Direct stoping costs by zone

Zone	Direct stoping unit cost $/t milled
Upper North zone - drift and slash stoping	$16.58/t milled
Lower North zone - drift and slash stoping	$18.93/t milled
South zone - room and pillar stoping	$33.67/ t milled
South zone - drift and slash stoping	$16.31/t milled

Mining direct cost components

The mining direct costs include direct labour, direct consumables, and equipment operating costs. The cost breakdown for the Upper North zone mining cost is provided as an example in Table 21.13.

Table 82. Direct stoping cost – Upper North zone stopes

Item	Direct Cost ($/t)
Direct labour cost	$2.19[A]
Vacation / absentees cost allowance	$0.04
Direct supplies -pilot drift	$8.73 [B]
Direct supplies - footwall slash	$3.67 [B]
Direct supplies - hangingwall slash	$1.95 [B]
Total	$16.58[C]

[A] Based on 6 jumbo drill operators and 16 manual bolters on the mine payroll.
[B] Includes the cost of drilling consumables, explosives and detonators, ground support materials, equipment operating costs and installed services.
[C] The cost of blasters, LHD operators, truck drivers, supervisors are included in the mine indirect operating costs.

Mine indirect operating costs

The annual costs for mine indirect hourly personnel and mine staff and support personnel in Year 2 as an example are shown in Tables 83 and 84 respectively.

Table 83. Mine indirect (hourly personnel) cost in Year 2

Item	Number on payroll	Estimated Cost in Year 2
Indirect labour (hourly)		
Lead blaster	6	
Blaster helper	6	
LHD operator	6	
Mine truck driver	12	
Surface truck driver (haul to mill)	4	
Total	34	$894k

Table 84. Mine staff, support personnel and indirect costs in Year 2

Item	Indirect mine staff and support personnel	Total number on payroll	Estimated Cost in Year 2
Mine Supervision and training	Mine supervisor	6	
	Trainer	1	$208k
Technical Services	Mining engineer	1	
	Mine technician	1	
	Mine surveyor	1	
	Mine geologist	1	
	Grade control geologist	1	$271k
Mine maintenance	Maintenance supervisor	2	
	Maintenance planner	1	
	Lead mechanic	2	
	Mechanic	8	
	Welder	2	
	Lead electrician	2	
	Electrician	3	
	Instrumentation technician	1	
	Warehouse	1	$579k
Hand tools, safety, daily travel and meal costs			$286k
Indirect vehicle operating costs			$50k
Mine fans, pumps and compressor maintenance costs			$236k
		Total	$1,630k

CRF backfilling

The basis of the CRF backfilling cost is summarized in Table 85.

Table 85. Estimated CRF costs

Item	Estimated cost
CRF direct costs • The PEA assumes the CRF would be produced using mine waste rock and imported non-acid generating rock sourced from within 4 km of the on-site CRF plant. Imported fill rock is projected to account for two-thirds of the rock used. The CRF cement content averages 4 wt%. • The CRF would be back-hauled underground by production truck drivers. The estimated cost includes CRF plant consumable costs and CRF placement equipment operating costs.	$7.75/t milled
CRF indirect costs • CRF production and placement labour costs. • There are 12 CRF production and placement personnel on the Payroll in Year 2.	$1.07/t milled
Total	$8.82/t milled

<u>Mine electrical power</u>

The power cost in Year 2 is estimated based on a 1.5 MW connected load, projected 52% utilization and a $0.084/kWh power cost.

21.2.3 Processing operating cost estimate

Annual and unit process operating cost estimates for a 800 tonnes per day milling operation are summarized in the following Table 86. Support tables for the cost estimates are shown in Tables 86 through 93.

Table 86. Summary of plant operating cost by cost item

Item	Estimated Annual Cost ($M)	Cost ($/tonne)
Power	$1,215k	$4.22
Labour	$1,353k	$4.70
Reagents	$3,336k	$11.58
Grinding media	$1,022k	$3.55
Repair materials and operating supplies	$460k	$1.60
Liners and wear materials	$280k	$0.97
Total	$7,666k $7.67M	$26.62

The preliminary power consumption estimate is based on the installed power with estimates of the operating power draft and operating time, and power unit cost of US$ 0.084/kWh. The process power consumption and power cost estimates are summarized in Table 87 and Table 88. respectively.

Table 87. Power consumption summary

Area	kWh/tonne
Area 10 - Primary Crushing	2.57
Area 15 - Grinding	20.57
Area 25 - Lead Flotation	6.58
Area 26 - Zinc Flotation	5.77
Area 27 - Multiplexer	0.30
Area 30 - Thickening and Filtration	4.90
Area 60 - Tailings Thickening	0.43
Area 65 - Reagents	0.93
Area 66 - Water	3.56
Area 67 - Plant Air	4.63
Total	50.24

Table 88. Estimated power cost

Usage	
kWh per tonne	50.24
Power Cost, US$ per kWh	$0.084
Power Cost, US$ per tonne	$4.22
Estimated Annual Power Cost	$1.215M/yr

The labour cost estimate for mill operations is shown in Table 89. The labour rates and burden are based on the rates for similar mill operation.

Table 89. Labor cost

Function	Per Crew	Total	Total hrs/Yr	Total
Operations Shift Crews (4 crews req'd)				
Crusher Operator	1	4	8,760	
Grinding Operator	1	4	8,760	
Flotation Operator	1	4	8,760	
Filter Operator (Concentrate)	1	4	8,760	
Labour	2	8	8,760	
Conc Handling	1	4	8,760	
Sub total		**28**		**$431k**
Operations Day Crew				
Reagent Mixing (10 and 4)	2	2	4,160	
General Cleanup; Ball Charging	2	2	4,160	
Sub total		**4**		**$68k**
Maintenance				
Mechanics	4	4	8,320	
Electrician	4	4	8,320	
Sub total		**8**		**$197k**
Technical				
Shift Sample Prep / Sampler	1	4	8,760	
Day sample prep	2	2	4,160	
Assayers (day only)	2	2	4,160	
Sub total		**8**		**$145k**
Salaried Personnel				
Mill Superintendent	1			
Maintenance Supervisor	1			
Plant Supervisor	3			
Metallurgist	1			
Process Technician	1			
Instrument Technician	1			
Process Supervisor	1			
Sub total	**9**			**$512k**
Grand Total		**57**		**$1,353k**

Reagent cost estimates are shown in Table 90. The reagent consumption rates are based on SGS Lakefield metallurgical test work data in 2019.

Table 90. Reagent cost

Reagents	Estimated usage kg/tonne milled	Cost $/year	Cost $/tonne milled
Sodium Sulfide (Na_2S)	1.475	$375k	$1.30
Zinc Sulfate ($ZnSO_4 \cdot 7H_2O$)	3.700	$1,321k	$4.59
Aerofloat 242	0.023	$44k	$0.15
Aerophine 3418A	0.080	$270k	$0.94
Sodium Metabisulfite,$Na_2S_2O_5$ (SMBS)	11.800	$884k	$3.07
Copper Sulfate ($CuSO_4 \cdot 5H_2O$)	0.350	$274k	$0.95
Sodium Isopropyl Xanthate (SIPX)	0.020	$25k	$0.09
Methyl Isobutyl Carbinol (MIBC)	0.030	$32k	$0.11
Lime	1.200	$66k	$0.23
Flocculant	0.040	$45k	$0.16
Total		**$3,336k** **$3.34M**	**$11.58**

The wear material and grinding media and liner cost estimates are provided in Table 91 and Table 92. The consumption estimates are based on assumed abrasion index.

Table 91. Wear material operating cost estimates

	Cost, $/tonne Milled	Cost, $/year
SAG Mill liners	0.229	$66k
Ball Mill liners	0.568	$164k
Regrind Mill liners	0.174	$50k
Total Wear Material	0.970	$280k

Table 92. Grinding media operating cost estimates

	Cost, $/tonne Milled	Cost, $/year
SAG Mill Balls	0.761	$219k
Ball Mill Balls	2.135	$615k
Regrind Mill Balls	0.654	$188k
Total Grinding Media	3.549	$1,022k

The repair materials and operating supplies is estimated using empirical factor based on total equipment cost, SGS has assumed 3.0 percent for this 800 tonnes per day plant (Table 93).

Table 93. Repair materials and operating supplies

New Equipment Capital Estimate	$15,337k
Repair Materials and Supplies (Percentage of Equip)	3.00%
Annual Maintenance Cost	$460k
Cost per tonne	$1.60

21.2.4 General and Administration Cost

The estimated General and Administration (G&A) cost in Year 2 as an example is shown in Table 94.

Table 94. G&A Cost in Year 2

Area	G&A staff position	Number on payroll	Estimated G&A Cost in Year 2
Administration	Toachi General Manager	1	$479k
	Executive assistant	1	
	Accountant	1	
	Payroll	1	
	Purchasing	1	
	Concentrate marketing	1	
	Human Resources	1	
	Clerk	1	
	IT Technician (part time)	1	
Health-Safety-Environmental-Social (HSES) Security Kitchen	HSES Manager	1	$259k
	Environmental technician	1	
	Nurse	1	
	Social Director	1	
	Community Social Worker	1	
	Community Social Assistant	1	
	Security	4	$178k
	Kitchen staff	3	$59k
Vacation/absentee replacements			$10k
Safety, daily travel and meal costs			$85k
G&A vehicle operating costs			$11k
Project-wide indirect operating costs including insurance			$385k
		Total	$1,466k $1.47M

21.3 Mine closure cost

The estimated mine closure cost is $4.5M as described in Section 20.5.

22.0 Economic Analysis

The potential economic viability of the mineral resource was assessed using a discounted cashflow model based on assumed 100% equity financing. The Project's internal rate of return (IRR), net present value (NPV) and payback were estimated on a pre-tax and after-tax basis. The sensitivity of the after-tax IRR to incremental changes in income, capital costs and operating costs was assessed.

The metal prices used in the cashflow model are based on three year monthly trailing average prices as of January 31, 2019 and remain constant over the project life and are not escalated. Preproduction capital, sustaining capital, and operating costs are in reported in 2019 Q1 US dollars and are not escalated in the cashflow.

The PEA cashflow makes 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 PEA will be realized.

22.1 Cashflow basis and assumptions

It is assumed the Project would be constructed and commissioned over a 15 month time line comprised of 12 months in Year -1 and the first 3 months in Year 1. Costs incurred prior to Year -1 such as but not limited to technical study costs, land use agreement costs, environmental assessment costs and Project permitting costs are assumed to be sunk costs and are not included in the cashflow model. It is assumed that Toachi would develop and operate the Project using its own labour and equipment.

The capital and operating costs used in the cashflow are estimated to ±35% accuracy.

22.1.1 Income

It is assumed the Project would produce and sell Cu/Pb and Zn concentrates to smelters in Asia and receive payment in US dollar currency. The Republic of Ecuador's currency is the US dollar currency.

Annual revenues are based on projected recovered tonnages, diluted head grades, mill recoveries, NSR terms and Au, Ag, Cu and Zn prices. Readers are reminded that the PEA is preliminary in nature; the scheduled tonnes and grades do not represent a Mineral Reserve estimate; and that the Mineral Resources used in the PEA cashflow are not Mineral Reserves and do not have demonstrated economic viability.

Based on the results of metallurgical test LCT1 the mill would recover approximately 85% of the lead and that the lead would report in different portions to the Cu/Pb and Zn concentrates. The PEA is based on the LCT1 metallurgical testing results and includes the cost to bulk bag and transport lead containing Cu/Pb and Zn concentrates from the mill to the smelters. The cashflow includes a projected smelter \$2 per 1% content penalty when the combined Pb & Zn content of the Cu/Pb concentrate exceeds 3% (e.g. \$2 per 1% when above 3%). As indicated in Section 26 "Recommendations" additional metallurgical testing is recommended to optimize metal recoveries, the number of concentrates, concentrate grades and the mill process.

The PEA cashflow assumes that lead is not a payable metal based on SGS's review of possible NSR terms. The projected annual tonnages of concentrates are minor in comparison to global concentrate production levels, and developments affecting the Cu and Zn concentrate smelting marketplaces.

The NSR factors used in the cashflow model are summarized in Table 95.

Table 95. NSR factors used in the cashflow model

Item	Projected values		
Metal prices (3 year monthly trailing average prices to January 31, 2019)	Gold Price	$1,264 / troy oz Au	
	Silver Price	$16.64 / troy oz Ag	
	Copper Price	$2.68/ lb Cu ($5,909/ t Cu)	
	Lead Price	$0.98/ lb Pb ($2,162/ t Pb)	
	Zinc Price	$1.21/ lb Zn ($2,656/ t Zn)	
Cu/Pb concentrate mill recoveries (based on LCT1 test results)	Au % recovered	43.1%	
	Ag % recovered	47.9%	
	Cu % recovered	89.4%	
	Pb % recovered	71.6%	
	Zn % recovered	20.0%	
Zn con mill recoveries (based on LCT1 test results)	Au % recovered	26.6%	
	Ag % recovered	26.1%	
	Cu % recovered	4.5%	
	Pb % recovered	13.4%	
	Zn % recovered	70.5%	
Estimated concentrate (con) sales terms	Treatment	Cu/Pb con	Zn con
	Con grade	(20.9% Cu)	(51.2% Zn)
	Moisture content	8 wt%	8 wt%
	Treatment Charge	$132/ dt con	$188/ dt con
	Penalties	$13.10/ dt con	$9.45/ dt con
	Transportation	$95.45/wt con	$95.45/wt con
	Refining Charges (per unit of payable metal)	$0.135/ lb Cu $5/troy oz Au $2/troy oz/Ag	$5/troy oz Au $1/troy oz/Ag
	Payables	Cu/Pb con	Zn con
	Payment currency	USD	USD
	Au payable	96%	65%
	Ag payable	90%	70%
	Cu payable	96.5%	0%
	Pb payable	0%	0%
	Zn payable	0%	85%

22.1.2 Preproduction CAPEX

The cashflow includes estimated Project pre-production costs: mine development costs; initial mine equipment lease costs; CRF backfill plant construction costs; mill construction costs; tailings storage facility (TSF) construction costs, effluent treatment plant construction costs; infrastructure costs including the power line and main substation costs; pre-development engineering, indirect operating and G&A costs; mill commissioning cost; and a mill cost contingency. The cashflow does not include working capital.

22.1.3 OPEX

The cashflow includes the estimated underground mining cost; surface haul cost; milling cost; TSF operating cost; effluent treatment cost; concentrate transportation and smelting and refining costs; indirect cost and G&A cost.

22.1.4 Sustaining CAPEX

The cashflow includes the following estimated sustaining capital expenditures for: sustaining mine development costs; mine equipment lease costs; mill sustaining capex; TSF expansion costs; closure surety bond costs; and closure work costs. It is assumed that $108k in closure surety bonding would be required in Year -1 and that the surety bonding coverage would be progressively increased over the operating life of the Project.

22.1.5 Salvage value

The estimate closure cost assumes there is no salvage value as there is no certainty that the Project materials and equipment can be salvaged and sold after production ceases.

22.1.6 Taxation

Corporate taxes were estimated after a general review of Ecuador's corporate tax requirements and with assistance from Toachi. It is projected that after taking initial CAPEX amortization into account, the Republic of Ecuador would receive at least 50% of the project's profit through receipt of 22% corporate taxes, Republic of Ecuador royalty, and a mandated 15% share of profit including employee share of profit. The costs used in the cashflow do not include recoverable Ecuadorian VAT (Value Added Tax).

22.2 Project cashflow

The results of the economic analysis (i.e. the Internal Rate of Return "IRR, Net Present Value "NPV", and Payback) are shown in Table 96. The cashflow for the Project is summarized in Table 97.

Table 96. PEA cashflow results

	Before Tax	After Tax
IRR	38%	24%
NPV(5%)	$100M	$52M
NPV (8%)	$81M	$39M
NPV (10%)	$70M	$32M
Payback	2.8 years	3.5 years

Table 97. PEA cashflow

Item	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Mined material processed (kt)		240	288	288	288	288	288	288	39		2007
Au grade (g/t Au)		2.85	2.70	2.44	4.82	4.30	3.31	2.19	2.13		3.22
Ag grade (g/t Ag)		31	26	26	59	57	51	34	39		41
Cu grade (%/tonne)		2.83	2.94	3.01	3.05	2.96	1.79	1.46	0.93		2.54
Pb grade (%/tonne)		0.47	0.31	0.29	0.78	0.68	0.54	0.40	0.49		0.50
Zn grade (%/tonne)		3.55	2.76	2.62	4.09	4.09	3.70	3.07	3.27		3.41
Cu/Pb Concentrate Production											
Cu/Pb concentrate tonnage (kdt)		29	36	37	38	36	22	18	2		218
Cu/Pb concentrate tonnage (kwt)		32	39	40	41	40	24	19	2		237
Cu/Pb Concentrate Gross Revenue											
From Au ($k)		$10,385	$11,645	$10,367	$21,899	$19,403	$15,164	$9,899	$1,335		$100,098
From Ag ($k)		$1,319	$1,187	$1,169	$3,365	$3,266	$3,071	$2,010	$333		$15,720
From Cu ($k)		$32,999	$41,126	$42,019	$42,663	$41,338	$25,010	$20,352	$1,752		$247,259
From Pb ($k)		$0	$0	$0	$0	$0	$0	$0	$0		$0
From Zn ($k)		$0	$0	$0	$0	$0	$0	$0	$0		$0
Cu/Pb concentrate tranportation costs, TCs, RCs, deductions and penalties ($k)		-$9,208	-$11,236	-$11,434	-$12,280	-$11,876	-$7,523	-$6,046	-$575		-$70,178
Cu/Pb concentrate NSR ($k)		**$35,495**	**$42,721**	**$42,121**	**$55,647**	**$52,131**	**$35,723**	**$26,216**	**$2,845**		**$292,898**
Zn Concentrate Production											
Zn con tonnage (kdt)		12	11	10	16	16	15	12	2		94
Zn con tonnage (kwt)		13	12	11	18	18	16	13	2		102
Zn Concentrate Gross Revenue											
From Au ($k)		$4,349	$5,023	$4,524	$9,120	$8,059	$6,115	$3,946	$514		$41,650
From Ag ($k)		$327	$342	$359	$1,089	$1,040	$924	$536	$89		$4,707
From Cu ($k)		$0	$0	$0	$0	$0	$0	$0	$0		$0
From Pb ($k)		$0	$0	$0	$0	$0	$0	$0	$0		$0
From Zn ($k)		$11,434	$10,691	$10,136	$15,839	$15,840	$14,317	$11,883	$1,712		$91,852
Zn concentrate transportation costs, TCs/RCs,deductions & penalties ($k)		-$3,567	-$3,341	-$3,169	-$4,992	-$4,985	-$4,500	-$3,717	-$536		-$28,805

Zn concentrate NSR ($k)		$12,544	$12,714	$11,850	$21,057	$19,954	$16,856	$12,650	$1,779		$109,403
Total gross income ($k)		**$48,039**	**$55,435**	**$53,971**	**$76,704**	**$72,085**	**$52,578**	**$38,866**	**$4,624**	**$0**	**$402,302**
Operating costs:											
Stoping directs opex ($k)		$3,979	$5,057	$5,539	$7,369	$4,754	$4,697	$4,854	$668	$0	$36,916
Stoping indirects opex ($k)		$894	$894	$894	$894	$894	$894	$894	$89	$0	$6,347
CRF backfilling direct opex ($k)		$1,862	$2,235	$2,235	$2,235	$2,235	$2,235	$2,235	$273	$0	$15,545
CRF backfilling indirect opex ($k)		$137	$329	$329	$329	$329	$329	$329	$33	$0	$2,145
Mine staff & indirect OPEX ($k)		$1,457	$1,630	$1,630	$1,630	$1,630	$1,630	$1,630	$163	$0	$11,400
Mine and G&A power cost ($k)		$428	$552	$552	$552	$552	$552	$552	$55	$0	$3,797
G&A ($k)		$1,398	$1,466	$1,466	$1,466	$1,466	$1,466	$1,466	$147	$0	$10,339
Processing cost ($k)		$6,389	$7,667	$7,667	$7,667	$7,667	$7,667	$7,667	$1,039	$0	$53,427
Tailings storage facility OPEX ($k)		$127	$153	$153	$153	$153	$153	$153	$21	$0	$1,064
Effluent treatment cost ($k)		$168	$202	$202	$202	$202	$202	$202	$101	$0	$1,478
Total opex ($k)		**$16,840**	**$20,183**	**$20,665**	**$22,495**	**$19,881**	**$19,824**	**$19,980**	**$2,588**	**$0**	**$142,457**
EBITDA ($k)		**$31,198**	**$35,252**	**$33,305**	**$54,209**	**$52,204**	**$32,754**	**$18,886**	**$2,035**	**$0**	**$259,844**
Preproduction CAPEX											
Mine indirects hourly ($k)	-$803										-$803
Mine direct indirect staff & indirect operating costs ($k)	-$1,210										-$1,210
Genset power costs ($k)	-$773										-$773
G&A during procurement & construction ($k)	-$1,165										-$1,165
Infrastructure capex ($k)	-$3,525										-$3,525
Mine equipment lease costs ($k)	-$4,248										-$4,248
Mine development costs ($k)	-$5,318										-$5,318
Mill capex ($k)	-$43,031	-$7,172									-$50,202
Tailings storage facility capex ($k)	-$2,000										-$2,000
Effluent treatment capex ($k)	-$946										-$946
Closure surety bond cost ($k)	-$108										-$108

Pre-production CAPEX ($k)	**-$63,128**	**-$7,172**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**-$70,300**
Sustaining CAPEX											
Infrastructure costs ($k)		-$452									-$452
Mine equipment lease costs ($k)		-$4,379	-$2,318	-$3,124	-$2,520	-$2,327	-$1,037	-$415	-$363	$0	-$16,483
Mine sustaining development ($k)		-$5,871	-$1,914	-$6,065	-$5,192	-$1,358	-$904	-$847	-$109	$0	-$22,261
Tailings sustaining capex ($k)		-$390	-$390	-$390	-$390	-$390	-$390	-$390	$0	$0	-$2,730
Closure surety bond costs ($k)		-$121	-$134	-$148	-$161	-$174	-$187	-$200	$0	$0	-$1,126
Closure work expenditures ($k)		$0	$0	$0	$0	$0	$0	$0	-$4,400	-$100	-$4,500
Sustaining CAPEX ($k)	**$0**	**-$11,213**	**-$4,757**	**-$9,726**	**-$8,264**	**-$4,249**	**-$2,518**	**-$1,852**	**-$4,873**	**-$100**	**-$47,551**
Before-Tax Cashflow ($M)	-$63.1	$12.8	$30.5	$23.6	$45.9	$48.0	$30.2	$17.0	-$2.8	-$0.1	$142.0
Estimated taxes and Ecuadorian royalty ($M)	$0.0	-$3.3	-$9.5	-$7.1	-$15.4	-$15.9	-$9.3	-$4.4	$3.5	$0.0	-$61.2
After-Tax Cashflow ($M)	**-$63.1**	**$9.6**	**$21.0**	**$16.5**	**$30.6**	**$32.1**	**$21.0**	**$12.7**	**$0.6**	**-$0.1**	**$80.8**

22.3 Sensitivity Analysis

The sensitivity of the after tax Internal Rate of Return (IRR) to incremental changes in revenue, operating cost and upfront capital cost was assessed. The results of this assessment are summarized in Table 98 and graphically depicted in Figure 55. The Project IRR is most sensitive to changes in income and less sensitive to changes in opex and capex.

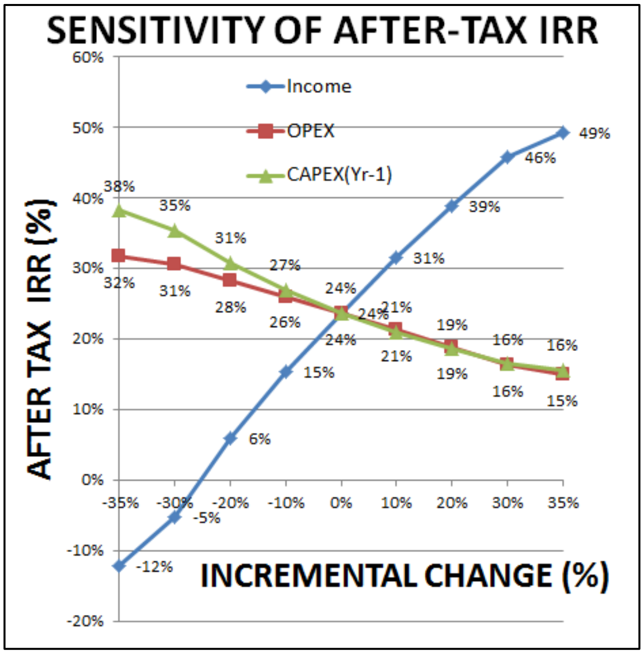
The IRR is the interest rate at which the Net Present Value (NPV) of all the positive and negative cashflows equals zero.

Table 98. After Tax IRR Sensitivity Analysis Results

Parameter	**Incremental change in parameter**								
	-35%	-30%	-20%	-10%	0%	10%	20%	30%	35%
	After tax IRR								
Income	-12%	-5%	6%	15%	24%	31%	39%	46%	49%

OPEX	32%	31%	28%	26%	24%	21%	19%	16%	15%
CAPEX(Yr-1)	38%	35%	31%	27%	24%	21%	19%	16%	16%

Figure 55: After-Tax IRR sensitivity to changes in income, OPEX and CAPEX.



23.0 Adjacent Properties

There are no operating mines or active exploration areas adjacent to the La Mina project.

24.0 Other Relevant Data and Information

There is no other relevant information known to SGS that would make this report more understandable or if undisclosed would make this report misleading.

25.0 Interpretation and Conclusions

The Inferred Mineral Resources used in this PEA 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 PEA will be realized. The words “production”, “mineable” and mine plan” are used in the PEA in an operational sense and not in an accounting sense.

The Project warrants being carried forward to the Pre-Feasibility Study (PFS) stage (based on the proposed approach, the use of proven conventional technologies and equipment, and the positive results of the PEA) with a view to advancing the Project towards an eventual production decision.

The independent Qualified Persons responsible for assigned portions of this PEA are not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Updated Mineral Resources Estimate presented in this PEA.

25.1 Conceptual Project

The Project has been developed to a conceptual level and additional technical studies are required to provide information for the PFS study.

25.2 Risks and Uncertainties

Potential significant risks and uncertainties that could affect the updated mineral resources estimate and/or the projected economic viability of the Project are summarized in Table 99.

Table 99. Risks and uncertainties

Area	Potential Risks / Uncertainties
Mineral Resources	• The updated mineral resources estimate presented in the PEA does not include the 173kt historic mined tonnage which is based on available historic information. While the exact historic mine tonnage is uncertain, the potential for a significant impact to the updated mineral resources estimate and proposed mine plan is considered low.
Exploration	• Surface exploration diamond drilling aimed at intercepting potential addition mineralization below the La Mina VMS deposit South Block is warranted. The PEA does not include any additional mineral resources that may be identified as a result of recommended exploration drilling described in Section 26.
Mining	• The mine development and production plan is based on the use of conventional mining equipment and technologies. The underground workings would be accessed using ramps and would have required mine services. The proposed ground support requirements in the development headings and stopes are projections that are to be reviewed as part of the mine geotechnical study described in Section 26. There is a possibility that additional ground

Area	Potential Risks / Uncertainties
	support requirements that may be introduced by the study and significantly increase mine operating costs. • The PEA assumes that the mine would hire experience lead miners and maintenance personnel and provide training to others. It is expected that most mine personnel would be hired from the Project region. There is a possibility that the Project could find it difficult to attract and retain people from the region. • The PEA assumes that the mine would be dewatered using conventional mine sumps and pumps with standby pumps and a spare dewatering pipe line to handle higher flows. There is a possibility that the results of the recommended hydrological/hydrogeological study described in Section 26 may identify a need for additional mine pumping capacity and add to the mine capital and operating costs. • There is a possibility that actual wages and salaries could be higher than estimated in the PEA. Possible reasons include as examples: higher wage levels for experienced lead hands; and additional social costs due to governmental factors.
Metallurgy	• Metallurgical testing on the 4 substantial sample composites provided (Composites 100, 200,300 and 400) has demonstrated that saleable Cu/Pb and Zn concentrates can be produced using the proposed process. • Gravity separation: A significant upgrading of the gold distribution present into a low weight (<0.2% of the feed) has been demonstrated – however further work will be required to quantify the benefits to the currently proposed circuit. For the current phase of project development, the flowsheet has not included a specific gravity circuit. However, as this has been important in the various previous processing activities on site, the final decision will be considered as part of the next phase of the project development. • Flotation: Significant development work was undertaken through the program to minimise the misreporting of Sphalerite into the Copper concentrate. The combination of high dosages of sphalerite/pyrite depressants (Na_2S, SMBS, and $ZnSO_4$) was found to produce optimal results. This has been confirmed by Locked Cycle Testing. • There is the likelihood that the additional metallurgical testing work described in Section 26 will provide a basis for metal recoveries and concentrate quality improvements. Changes if any to the plant process may increase plant capital and operating costs.
Processing	• The proposed Plant would process material using conventional flotation recovery methods. The material would be crushed and conveyed to the processing plant. The material would be ground to 80% passing 70 microns in a SAG/Ball mill grinding circuit. The ground material would then be floated with the rougher concentrates being reground to 80% passing 20 microns prior to cleaning to produce high-value separate copper/lead (with gold and silver) and zinc (with gold and silver) concentrates. Concentrates would be bagged, containerized and trucked to port for ocean shipment to smelters. ▪ The following future evaluations may improve the project economics: perform a variability study to improve confidence in the metallurgical response of the deposit; and optimize flotation reagent addition rates to lower the process plant operating cost.
Infrastructure	• The powerline and a main substation near the plant would be permitted and constructed by CNEL EP. There is a risk that a significant delay in connecting the Project to the National hydropower grid could increase the estimated upfront capital cost.

Area	Potential Risks / Uncertainties
NSR	• The PEA assumes that Toachi would produce and sell Cu/Pb and Zn concentrates to at least two different smelters in Asia at projected preliminary smelting, refining and payable / foreign exchange terms. The estimated net smelter return (NSR) terms used in the PEA are expected to change with time. There is a possibility that the estimated concentrate transportation costs, treatment costs and refining costs may increase. • The preliminary NSR terms used in the cashflow are based on projected smelter market conditions / NSR terms and it assumed that the Cu/Pb and Zn concentrates would be sold to smelters in Asia. There is a possibility that future market conditions / NSR terms may be significantly harsher or more favourable than projected, and/or that the Project would sell its concentrates to smelters in other parts of the world.
Tailings	• Exploration diamond drilling results could affect the waste rock/tailings storage program. If exploration drilling leads to an increase in the potentially mineable resources tonnage and tailings quantities, significant additional costs could be incurred to provide additional tailings storage capacity if available within a reasonable distance from the plant and /or include the use of tailings in the mine backfill program. The waste rock/tailings options study described in Section 26 would be expected to address such a scenario.
Environmental	• It is anticipated that the mine waste rock and tailings would be net acid generating and measures to inhibit acid drainage generation and conventionally lime-treat mine water and TSF drainage and seepage are included in the PEA. The mine geotechnical and mine waste rock/tailings disposal options studies described in Section 26 and the social and environmental assessment and permitting process may identify the need for additional engineered controls and add to the estimated capital and operating costs.
Permitting	• While the legal framework needed to develop and operate the Project is in place, it is expected that the permitting and development of the Project will largely depend on the success of the company’s community social consultation efforts. There is a latent risk of project permitting and development delays.
Projected potential economic viability	• The cashflow is based on projected revenues and costs. There is a risk that income from concentrate sales may be lower than projected due to higher concentrate transportation costs and harsher NSR terms.
Exclusion	• Land use agreement costs cannot be estimated with confidence at this time. These costs have been treated as sunk costs in the PEA.

26.0 Recommendations

Based on the positive results of the PEA, and in the absence of any currently identified material obstacle to advancing the Project towards an eventual production decision, it is recommended that the Project be advanced to the Pre-Feasibility Study (PFS) stage.

The estimated cost of the PFS and supporting work / studies is shown in Table 100.

Table 100. Recommended budget

Area	Recommendation	Recommended work budget
PFS supporting work / studies:		
Geology	• Additional diamond drilling to convert Inferred Mineral Resources	$4.0M
Mine planning	• Mine geotechnical study • Waste rock/ tailings storage options study	$0.24M $0.4M
Mineral processing	• Metallurgical testwork to optimize metal recoveries/concentrate grades/mill process	$0.25M
Environmental	• Hydrological / hydrogeological water management study • Energy conservation	$0.25M
Social	• Expand socio-economic baseline	$0.25M
	Subtotal PFS supporting studies	**$5.39M**
Pre-Feasibility Study prepared in accordance with NI 43-101 requirements:		**$2.0M**
	Work budget	**$7.39M**

26.1 Geology

Conduct additional diamond drilling to convert Inferred Mineral Resources

It is recommended that Toachi conduct a surface diamond drilling program to convert where possible Inferred Mineral Resources to the Indicated Mineral Resources and/or Measured Mineral Resources categories and to explore for a possible deeper extension of the La Mina VMS deposit South block. The proposed budget ($4M) is based on 15,900 m of drilling at $250/m including QA/QC and assaying costs.

26.2 Mine planning

Mine geotechnical study

It is recommended that Toachi engage a qualified firm to conduct a mine geotechnical characterization and design study to provide geotechnical data for use in the PFS. The key components include:

- A geotechnical review of existing drill core to gain an improved understanding of rock mass characteristics not limited to degree and extent of fracturing in the hangingwall and footwall zones and immediately adjacent to the mineralization; rock characteristics in the mineralized zones; and the characteristics of other fracture zones and intercepted faults.

- Geotechnical drilling and sampling including groundwater measurements and core logging.
- Geotechnical testing and assessment: rock strength testing; rock quality designation; orientation, spacing and nature of discontinuities; groundwater conditions; rock mass rating.
- A technical re-assessment of the proposed mine development and operating plan, and ground support requirements for development headings, drift and slash with CRF stopes, and room and pillar with CRF stopes taking backfilling options into consideration. It is expected that the mine geotechnical study would be done in parallel with the separately recommended geotechnical tailings disposal options study.
- The development of geotechnical criterion for mine planning.

The estimated budget is shown in Table 101.

Table 101. Projected budget – mine geotechnical study

Mine geotechnical study	**Estimated cost**
Geotechnical labour:	
Senior geotechnical engineer	$30k
Intermediate geotechnical engineer	$80k
Technician / clerical support /lab testing labour	$30k
Expenses:	
Geotechnical drilling	$60k
Lab testing	$15k
Other disbursements	$25k
Total	$240k

Waste rock/ tailings storage options study

It is recommended that Toachi engage a qualified firm to conduct a waste rock / tailings storage options study. Possible options are to: use cemented and uncemented rock fill in stopes; use cemented paste to backfill stopes; co-disposal of waste rock and hydraulic/filtered/paste tailings in a TSF; and combinations of the above. Paste tailings are a de-watered and non-segregating material that can be stacked.

The above options would be compared on a technical, comparable cost and potential risk basis. A hydrological side study would be used to size the surface waste storage area. The metal leaching and acid rock drainage potential of the tailings and waste rock would also be assessed along with a possible change to the conceptual closure plan. The design of the selected option would be advanced to the level needed to support the cost estimating accuracy used in the PFS. The estimated budget is shown in Table 102.

Table 102. Projected budget – tailings / waste rock storage options study

Tailings / waste rock storage options study	Estimated Cost
Specialist labour:	
Senior geotechnical engineer	$30k
Intermediate geotechnical engineer	$100k
Hydrologist/hydrogeologist	$80K
Design engineer / cost estimator / clerical support	$70k

Expenses: Field investigations Lab testing Software	 $70k $40k $10k
Total	$400k

26.3 Mineral processing

Metallurgical testwork to optimize metal recoveries / concentrate grades / mill process

It is recommended that Toachi conduct additional metallurgical testwork to optimize metal recoveries, concentrate grades and mill process. This work should be undertaken on new diamond drill core, on indivual drill core samples/composites selected to represent near, medium and long term aspects of the mine plan.

Further studies aimed to help improve the Project economics would include:

- Evaluate the possibility of producing a separate lead concentrate for sale.
- Additional locked cycle testing for the North Block and South Block mineralization. These additional studies will allow for validation of the final estimated recoveries and the selected concentrate grades.
- Verify that potential smelters have the capacity and ability to accept the proposed quantity and quality of produced copper/lead and zinc concentrates. As part of this program, additional concentrate analysis should be completed to further define the concentrate qualities. Transportation, treatment charges, and refinery charges should be confirmed.

The proposed budget for this work is $250k.

26.4 Environmental

Water management

It is recommended that Toachi engage a qualified firm to conduct a hydrological / hydrogeological study to evaluate the mill water balance and process water supply options, and TSF water pond storage and final effluent treatment plant capacity requirements. The proposed budget for this work is $250k.

Energy conservation

It is recommended that the PFS assess: the possibility of using on-site generated wind and solar power to supply energy to isolated and low power electrical loads; and possibility of using battery-powered mine equipment available from reputable suppliers.

It is recommended Toachi proactively as part of the PFS revisit the possibility of obtaining electrical power from the Toachi-Alluriquin hydropower project under construction.

26.5 Social

Expand socio-economic baseline

It is recommended that Toachi carry out a study to expand the Company's socio-economic baseline. The proposed budget for this work is $250k.

26.6 Closure planning

Sustainability

It is recommended that Toachi assess if there could be an opportunity for a sustainable activity that could be done by the local community by making use of the project's infrastructure after the Project has ceased production.

26.7 Potential significant opportunities for improvement

Several of the recommendations represent potential significant opportunities for improvement to the Project as described in Table 103.

Table 103. Potential significant opportunities for improvement

Area	Potential significant opportunities for improvement
Metallurgical testing	The recommended metallurgical testing program is needed to optimize the mill process. Copper and gold metal sales represent the major sources of income in the cashflow. The recommended metallurgical testing offers significant opportunities for improvements to the Project income. Separate lead concentrate There is a possibility that a separate lead concentrate can be produced. • The PEA assumes that some 15.7 million pounds of lead contained in Cu/Pb and Zn concentrates would be transported to smelters in Asia. The cashflow includes the costs of transporting the weight of the lead to the smelters, an excessive lead concentration penalty cost, but excludes revenue from lead.

<table>
<tr><td></td><td>The production of a separate lead concentrate would be expected to benefit the Project in a number of ways:
A marked reduction in the Pb content of the Cu and Zn concentrates would be expected to make the concentrates marketable to additional smelters internationally, and could make more favourable Net Smelter Return terms available to the Project.
Reducing lead concentrations in the Cu and Zn concentrates would reduce their transportation costs.
<u>Separate gravity circuit</u>

There is a possibility that a gold-silver product can be produced for sale by adding a gravity circuit.
The PEA assumes that all of the recovered gold and silver report to the Cu/Pb and Zn concentrates. As such, the gold and silver values in the PEA are subject to minimum smelter deductions and smelter payables for gold and silver.
The production of a separate gold-silver gravity circuit product would reduce the amount of gold and silver reporting to the Cu and Zn concentrates. It is expected the Project income would be increased by separately processing a gold and silver gravity circuit product in-country or elsewhere.</td></tr>
<tr><td>Integrated geotechnical assessments</td><td>Integrated geotechnical assessments are required to provide technical data for use in the recommended Pre-Feasibility Study. The results of these studies can in concept lead to net savings in tailings storage and mine backfilling costs, and may reduce potential risks.</td></tr>
<tr><td>Social baseline</td><td>Obtaining social acceptance has been a challenge for some new projects in Ecuador. The recommended social baseline work represents a valuable opportunity to add to the existing social baseline and help avoid some of the pitfalls encountered at other projects.</td></tr>
<tr><td>Diamond drilling</td><td>The recommended surface diamond drilling program includes $1M for exploration drilling aimed at if possible extending the downward extension of the South zone mineralization.</td></tr>
</table>

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CERTIFICATE OF DANIEL LEROUX, M.SC., P.GEO., GÉO

I, Daniel Leroux, M.Sc., P. Geo., géo (ON, QC, SASK), of Dalhousie Québec, do hereby certify that:

1) I am the Global Business Manager – Geological and Mining Consulting Services for SGS Canada Inc. located at 10 boul. de la Seigneurie Est, Suite 203, Blainville, Québec, Canada, J7C 3V5 since February 2017. This certificate applies to the technical report titled “Amended ands Restated NI 43-101 Preliminary Economic Assessment of the La Mina VMS Project, Cotapaxi, Ecuador (the “Technical Report”) dated July 17, 2019.” with an effective date of March 30, 2019.
2) I graduated with a Bachelor of Science, Geology degree from Laurentian University in 1993 and a Master of Science degree in Mineral Exploration in 2013 from Laurentian University and have practiced the profession of geoscience since my Bachelor of Science graduation. Prior to joining SGS Canada Inc, I was previously the Vice President with the firm of A.C.A. Howe International Limited, Mining and Geological Consultants (“Howe”) located at 365 Bay St., Suite 501, Toronto, Ontario, Canada, M5H 2V1. I was employed with Howe since 1993; since 2007 as Vice President, from 2005 to 2007 as a Senior Consulting Geologist, from 1999 to 2004 as an associate consulting geologist and from 1993 to 1999 as Project Geologist. I have a total of 26 years' experience in the mining industry including a background in international mineral exploration, evaluation and valuation studies for precious metals, base metals, diamonds and industrial minerals projects. Additional experience includes the completion of various National Policy 2A and NI 43-101 technical reports for base metal and gold projects located worldwide.
3) I am a Professional Geoscientist (P. Geo.) registered with the Association of Professional Geoscientists of Ontario (APGO, No. 742), the Ordre des géologues du Québec (OGQ, No. 1648), and the Association of Professional Geoscientists of Saskatchewan (APEGS, No. 10475).
4) I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5) I have not visited the La Mina VMS Project.
6) I am responsible for co-authoring Sections 2 to 12 and Sections 25 and 26 of the Technical Report along with those sections of the Executive Summary pertaining thereto.
7) I am independent of issuer applying all of the tests in section 1.5 of National Instrument 43-101.
8) I have had no prior involvement with Toachi Mining Inc. nor with the properties that is the subject of this report. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.
9) As of the effective date of this report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: March 30, 2019

Signed Date: July 17, 2019

[SIGNED AND SEALED]

{Daniel Leroux}

Daniel Leroux, M.Sc., P. Geo.

CERTIFICATE OF QUALIFIED PERSON – BRIAN R. WOLFE

I am an author of the report entitled " Amended and Restated NI 43-101 Resource Report: La Plata VMS Project Cotopaxi Province Ecuador, South America" (the "Report"), effective date March 30, 2019, and a signature date of July 17, 2019 and I hereby state:-

1. My name is Brian Richard Wolfe and I am the Principal Resource Geologist of the firm of International Resource Solutions Pty Ltd of 71 Watkins St, White Gum Valley, WA 6162, Australia.
2. I am a practicing geologist registered with the Australian Institute of Geoscientists. My membership number is 4629.
3. I graduated with a BSc Degree (Hons) in Geology in 1992 from the National University of Ireland, Dublin, and hold a Postgraduate Certificate in Geostatistics from Edith Cowan University (2007).
4. I have practiced my profession for a total of 23 years since 1993. I have experience in exploration geology, mining geology and geostatistical modelling and estimation of mineral resources.
5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
6. I have personally visited the La Plata VMS Project between 20th May 2017 and 23th May 2017.
7. I have not previously been engaged with respect to the La Plata VMS Project by Toachi Mining Inc or any other company in any capacity.
8. I am responsible or jointly responsible for preparing or supervising the preparation of sections 1 to 12, 14 and 23 to 27 of the Report.
9. I am not aware of any material fact or material change with respect to the subject matter of the Study which is not reflected in the Study, the omission of which would make the Study misleading.
10. I am independent of the La Plata VMS Project and the Issuer (Toachi Mining Inc) pursuant to section 1.5 of the Instrument.
11. I have read the National Instrument and Form 43-101F1 (the "Form") and the parts of the Study for which I am responsible have been prepared in compliance with the Instrument and the Form.
12. I do not have nor do I expect to receive a direct or indirect interest in the La Plata VMS Project of Toachi Mining Inc, and I do not beneficially own, directly or indirectly, any securities of Toachi Mining Inc or any associate or affiliate of such company.
13. At the effective date of the Study, to the best of my knowledge, information and belief, the Study contains all scientific and technical information that is required to be disclosed to make the Study not misleading.

Dated at Perth, Western Australia, on July 17, 2019.

[SIGNED AND SEALED]

{Brian R. Wolfe}

Brian R. Wolfe
Principal Resource Geologist
International Resources Solutions Pty Ltd.

CERTIFICATE OF QUALIFIED PERSON - DAVID A. ORAVA, P.ENG.

I, David A. Orava M.Eng., P.Eng., residing at 19 Boulding Drive, Aurora, Ontario L4G 2V9 do hereby certify that:

1. I am an Associate Mining Engineer at SGS Geological Services and Senior Mining Engineer at Orava Mine Projects Ltd.

2. This Certificate applies to the Technical Report titled "Amended and Restated NI 43-101 Preliminary Economic Assessment of the La Mina VMS Project, Cotopaxi Province, Ecuador" (the "Technical Report)" having an effective date of March 30th, 2019.

3. I am a graduate of McGill University located in Montreal, Québec, Canada at which I earned my Bachelor Degree in Mining Engineering (B.Eng. 1979) and Masters in Engineering (Mining – Mineral Economics Option B) in 1981. I have practiced my profession continuously since graduation. I am licensed by Professional Engineers Ontario (License No. 34834119).

I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

My summarized career experience is as follows:

- Mining Engineer – Iron Ore Company of Canada 1979-1980
- Mining Engineer –J.S. Redpath Limited /J.S. Redpath Engineering 1981-1986
- Mining Engineer & Manager Contract Development – Dynatec Mining Limited 1986-1990
- Vice President – Eagle Mine Contractors 1990
- Senior Mining Engineer – UMA Engineering Ltd. 1991
- General Manager – Dennis Netherton Engineering 1992-1993
- Senior Mining Engineer – SENES Consultants Inc. 1993-2003
- Senior Mining Engineer & President – Orava Mine Projects Ltd. 2003 to present
- Associate Mining Engineer – SGS Geological Services 2018 to present

4. I visited the Property that is the subject of this Technical Report on September 4 and 5, 2018.

5. I am responsible for authoring Sections 15, 16, 18, 19, 20 and 22 and co-authoring Sections 2, 3, 21, 25, 26 and 27 of the Technical Report along with those sections of the Summary pertaining thereto.

6. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

7. I have had no prior involvement with the project that is the subject of this Technical Report.

8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.

9. As of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: March 30, 2019
Signed Date: July 17, 2019

[SIGNED AND SEALED]

{David Orava}

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David Orava, M.Eng., P.Eng.

CERTIFICATE OF QUALIFIED PERSON – SIMON MEIK, FAUSIMM

I am an author of the report entitled " Amended and Restaed NI 43-101 Preliminary Economic Assessment of the La Mina VMS Project, Cotopaxi Province, Ecuador, South America" (the “Report”), effective date March 30, 2019, and I hereby state:-

14. My name is Simon Strickland Meik and I am an independent Mineral Processing Engineer, resident at 76 Salvington Hill, Worthing, West Sussex, BN133BB, United Kingdom.
15. I am a practicing metallurgist, registered with the Australian Institute of Mining and Metallurgy. My membership number is 106145.
16. I graduated with a BSc Degree (Hons) in 1974, and a PhD Degree in 1978 in Minerals Engineering from the University of Birmingham.
17. I have practiced my profession for a total of 40 years since 1978. I have experience in laboratory and pilot plant testing; feasibility studies including plant design and commissioning; project and plant management; the study and engineering of unit processes and equipment selection in many aspects of small, medium and large mineral processing operations.
18. I am a “qualified person” as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the “Instrument”).
19. I have not personally visited the La Mina VMS Project.
20. I have not previously been engaged with respect to the La Mina VMS Project by Toachi Mining Inc or any other company in any capacity.
21. I am jointly responsible for preparing section 13 of the Report.
22. I am not aware of any material fact or material change with respect to the subject matter of the Study which is not reflected in the Study, the omission of which would make the Study misleading.
23. I am independent of the La Mina VMS Project and the Issuer (Toachi Mining Inc) pursuant to section 1.5 of the Instrument.
24. I have read the National Instrument and Form 43-101F1 (the “Form”) and the parts of the Study for which I am responsible have been prepared in compliance with the Instrument and the Form.
25. I do not have, nor do I expect to receive, a direct or indirect interest in the La Mina VMS Project of Toachi Mining Inc, and I do not beneficially own, directly or indirectly, any securities of Toachi Mining Inc or any associate or affiliate of such company.
26. At the effective date of the Study, to the best of my knowledge, information and belief, the Study contains all scientific and technical information that is required to be disclosed to make the Study not misleading.

Dated at Worthing, West Sussex, United Kingdom

Effective Date: March 30, 2019

Signed Date: July 17, 2019

“signed and sealed”

Simon S. Meik, BSc(Hons), PhD, FAusIMM (CP)
Process Consultant to Toachi Mining, Inc.

CERTIFICATE OF AUTHOR - QINGHUA JIN, P.E.

I, Qinghua Jin, P.E., of Tucson, Arizona, do hereby certify that:

1. I am currently employed as a Principal Process Engineer with SGS North America Inc. with an office at 3845 N. Business Center Drive, Suite 115, Tucson, AZ. 85705, USA;
2. This certificate applies to the technical report titled “Amended and Restated NI 43-101 Preliminary Economic Assessment of the La Mina VMS Project, Cotopaxi, Ecuador” for Toachi Mining Inc., with an effective date of March 30, 2019, (the “Technical Report”) prepared for Toachi Mining Inc. (“the Issuer”);
3. I am a graduate of Northeastern University in Shenyang, China with a Bachelor of Engineering degree in Mineral Processing Engineering in 1990. I obtained two Master of Science degrees in Mining Engineering and Statistics both from West Virginia University, USA, in 2002 and 2006, respectively. I am a member in good standing of the Association of Arizona State Board of Technical Registration (License #53463), and a registered member of the Society for Mining, Metallurgy & Exploration (04138753RM). I have practiced mineral processing for 29 years. I have worked on scoping, prefeasibility and feasibility studies for mining projects in the North America, South America, Europe, Africa, and Asia, as well as worked on the design phases of some of these projects.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
5. I have not visited the La Mina VMS Project;
6. I am responsible for 1 (part), 17, 21 (part), 25 (part), 26 (part) of the Technical Report;
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
8. I have not had prior involvement with the property that is the subject of the Technical Report;
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
10. As of the effective date of the Technical Report and the date of this certificate, 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: March 30, 2019

Signing Date: July 17, 2019

[SIGNED AND SEALED]

{Qinghua Jin}

Qinghua Jin, P.E.,

Principal Process Engineer

SGS North America Inc.