EQUINOX GOLD CORP.

NI 43-101 Technical Report on the Santa Luz Project
Bahia State, Brazil

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Glossary

Abbreviations and Acronyms

a	annum
AA	Atomic Absorption
AISC	All-In Sustaining Cost
ALS Lima	ALS Chemex Laboratory in Lima, Perú
AMC	Continental Margin Arc
ANC	Acid Neutralizing Capacity
ANCcal	Calculated ANC
ANM	Agência Nacional de Mineração
ARD	Acid Rock Drainage
Ausenco	Ausenco Engineering Canada Inc.
BP	Piritiba Block
BRA	Back-Arc Basin
Brio	Brio Gold Inc.
BV	Bed Volume
C1/CS	Saúde and Itapicuru Complexes
CBPM	Companhia Baiana de Pesquisa Mineral
CC	Caraiba Complex
Ccarb	Carbonate C
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon-in-Pulp
CMS	Mairi-Seminha Craton
CN	Cyanide
CNWAD	weak-acid dissociable cyanide
COELBA	Companhia de Eletricidade do Estado da Bahia
COSIBRA	Companhia Sisal do Brasil
CRM	Certified Reference Material
CSIRO	Commonwealth Scientific and Industrial Research Organization
CVRD	Companhia Vale do Rio Doce
DDH	Diamond Drill Hole
dia	diameter
DNPM	Departamento Nacional de Produção Mineral
E&S	Environmental and Social
EIA-RIMA	Environmental Impact Assessment—Environmental Impact Report
EP	Exploration Permit
Equinox	Equinox Gold Corp.
Excel	Microsoft Excel

FCM	Mairi Craton fragment
FSR	Freight, Smelting, and Refining
G	giga (billion)
GJ	Jacobina Group
GRG	Gravity Recoverable Gold
GT	Grade Thickness
HDPE	High-Density Polyethylene
ICMS	Imposto Sobre Operações Relativas à Circulação de Mercadorias e Serviços de Transporte Interestadual de Intermunicipal e de Comunicações
ICP	Inductively Coupled Plasma
ID2	Inverse Distance Squared
ILR	Intensive Leach Reactor
INCRA	Instituto Nacional de Colonização e Reforma Agraria
INEMA	Instituto do Meio Ambiente e Recursos Hidricos
IPHAN-MinC	Instituto do Patrimônio Histórico e Artístico Nacional
IRR	Internal Rate of Return
Itasca	Itasca Consulting Group
LDPE	Low-Density Polyethylene
Leagold	Leagold Mining Corporation
LHD	Load-Haul-Dump
LIMS	Laboratory Information Management System
LOM	Life-of-Mine
Ma	million annum
MPA	Maximum Potential Acidity
MSE	Mineração Santa Elina
NAF	Non-Acid Forming
NAPP	Net Acid Production Potential
NN	Nearest Neighbour
NPV	Net Present Value
NSR	Net Smelter Return
OK	ordinary kriging
P_{80}	80% Passing
PAE	Plano de Aproveitamento Econômico
PEA or C1 Underground PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
ppb	part per billion
ppm	part per million
PRI	Pregnant Robbing Index
QA/QC	Quality Assurance/Quality Control
QP	Qualified Persons

R$	Brazilian Real
RAB	Rotary Air Blast Drill Holes
RC	Rotary Percussion Drill Holes
RIGB	Rio Itapicurú Greenstone Belt
RIL	Resin-in-Leach
RP	Reclamation Plan
RPA	Roscoe Postle Associates Inc.
SAG	Semi-Autogenous
SD	Standard Deviation
SGS	SGS Geosol Laboratório Ltda.
SI	International System of Units
SLR	SLR Consulting Ltd.
SMBS	Sodium Metabisulphite
SMU	Selective Mining Unit
SNUC	Sistema Nacional de Unidades de Conservação
SSJ	Sao Jose do Jacuipe Suite
SUDENE	Superintendência de Desenvolvimento do Nordeste
the Base Case	C1 Underground LOM base case
TOC	Total Organic Carbon
TSF	Tailings Storage Facility
US$	United States dollar
VLI	VLI Transportadora
VogBR	VogBR Recursos Hídricos e Geotecnia Ltda
WAD	Weak-Acid Dissociable
wk	week
WSF	Water Storage Facility
wt%	Weight-Percent
Yamana	Yamana Gold Inc.

Units of Measure

A	ampere
°C	degree Celsius
cm	centimetre
cm^2	square centimetre
d	day
dmt	dry metric tonne
g	gram
g/L	gram per litre
g/t	gram per tonne
L/s	litres per second
m	metre
m^2	square metre
m^3	cubic metre
µm	micron
mASL	metres above sea level
µg	microgram
m^3/h	cubic metres per hour
Mt	million tonnes

g/m^3	grain per cubic metre	Mt/a	million tonnes per annum
ha	hectare	min	minute
h	hour	mm	millimetre
Hz	hertz	MVA	megavolt-amperes
"	inch	MW	Megawatt
J	joule	MWh	megawatt-hour
k	kilo (thousand)	oz	Troy ounce (31.1035 g)
kg	kilogram	oz Au/a	ounces of gold per annum
km	kilometre	s	Second
km^2	square kilometre	t	metric tonne
km/h	kilometres per hour	t/a	metric tonne per year
kPa	kilopascal	t/d	metric tonne per day
kVA	kilovolt-amperes	V	volt
kW	kilowatt	W	watt
kWh	kilowatt-hour	wmt	wet metric tonne
L	litre	wt%	weight percent

1 SUMMARY

1.1 Executive Summary

Equinox Gold Corp. (Equinox) retained Roscoe Postle Associates Inc. (RPA), now part of SLR Consulting Ltd. (SLR), and Ausenco Engineering Canada Inc. (Ausenco) to jointly prepare with the Equinox Technical Services group a Technical Report on the Santa Luz Project (Santa Luz or the Project), located in Bahia State, Brazil. This Technical Report provides an update of a previously prepared Technical Report dated November 15, 2018 (RPA, 2018) which disclosed information regarding the Open Pit Feasibility Study and the underground mining scenario at the level of a Preliminary Economic Assessment (C1 Underground PEA). This Technical Report also provides an update on the ownership status of the Project, Mineral Resources (reported exclusive of Mineral Reserves), Mineral Reserves, and Project economics. The Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects.

On March 10, 2020, Equinox acquired Leagold Mining Corporation (Leagold), which owned Santa Luz as well as the Fazenda Brasileiro, Pilar, and Riacho dos Machados gold mines in Brazil, and the Los Filos gold mine in Mexico.

Yamana Gold Inc. (Yamana) operated the Santa Luz open pit mine from mid-2013 to mid-2014, when the mine was placed on care and maintenance following poor metallurgical recovery results from its carbon-in-leach (CIL) plant. Subsequent metallurgical testing programs, including the operation of a pilot-scale plant, has demonstrated that resin-in-leach (RIL) is a better method of gold recovery for the Project.

The Project is in the detailed engineering phase of process plant modifications to restart production, and will also require an increase in the storage capacities of the existing tailings and water storage facilities and a restart of open pit stripping and mining. The modifications and upgrades to the processing plant and tailings and water storage facilities are expected to be finished by the end of 2021. Open pit stripping is expected to recommence in February 2021.

The Project consists of six deposit areas: C1, Antas 2, Antas 3, Mansinha South, Mansinha North, and Mari, which are the only deposits used in the Mineral Resource and Mineral Reserve estimates. The Open Pit Feasibility Study is based on Proven and Probable Mineral Reserves of 24.9 million tonnes (Mt) grading 1.34 g/t gold (Au) (including stockpiles) as estimated at the effective date of June 30, 2020. The current Mineral Reserves are contained in the C1 and Antas 3 deposits, and existing stockpiles. Initial production is proposed to include ore mined from the C1 deposit; the Antas 3 deposit will be mined from 2023 to 2028 and the existing stockpiles will be mined from 2021 to 2030.

Santa Luz will be a conventional off-road truck and shovel open pit mining operation, utilizing a mining contractor for material movement. After the pre-production period, the nominal ore production rate over the following eight years is projected to be 2.77 million tonnes per annum (Mt/a), or 7,595 tonnes per day (t/d) excluding rehandling, plus 1.5 additional years at a lower rate from residual stockpile feed, over the total 9.5 years life-of-mine (LOM). The stripping ratio is 4.3:1 waste to ore including stockpiles (or 4.7:1 excluding stockpiles), and 6.9 Mt of pre-stripping is proposed (excluding the rehandling of old stockpiles), based on the mine schedule.

Processing will include crushing and grinding, gravity concentration, RIL, elution, and electrowinning. The Project has a targeted LOM processing rate of 2.7 Mt/a, or 7,400 t/d and 95,000 ounces of gold per annum (oz Au/a).

Table 1-1 summarizes the Santa Luz Mineral Resource estimate exclusive of Mineral Reserves, as of June 30, 2020. Table 1-2 presents a Mineral Reserve summary as of June 30, 2020. The Mineral Resource and Mineral Reserve estimates conform to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM Definition Standards, 2014).

No work has been completed at Santa Luz, nor has the topographic surface changed, subsequent to the effective date of the Mineral Resource and Mineral Reserve estimates.

Table 1-1: Mineral Resource Summary (Exclusive of Reserves)—June 30, 2020

Mineral Resource Category	Tonnes ('000s)	Gold Grade (g/t)	Contained Gold (oz)
Measured—Open Pit	9,986	1.22	390,306
Measured—Underground	121	1.94	7,561
Indicated—Open Pit	562	0.99	17,924
Indicated—Underground	5,913	2.55	484,066
Total Measured & Indicated	**16,582**	**1.69**	**899,857**
Inferred—Open Pit	694	1.29	28,748
Inferred—Underground	6,560	2.19	461,367
Total Inferred	**7,254**	**2.09**	**490,115**

Notes: **1.** CIM Definition Standards (2014) were followed for Mineral Resources.
2. Open Pit Mineral Resources are reported at a cut-off grade of 0.50 g/t Au.
3. Underground Mineral Resources are reported at a cut-off grade of 1.5 g/t Au.
4. Mineral Resources are exclusive of Mineral Reserves.
5. Mineral Resources are estimated using a gold price of US$1,500/oz and constrained by a pit shell.
6. Totals may not add due to rounding.

Table 1-2: Mineral Reserve Summary—June 30, 2020

Mineral Reserve Category	Tonnes ('000s)	Gold Grade (g/t)	Contained Gold (oz)
Proven—Open Pit	21,578	1.39	966,106
Probable—Open Pit	1,170	1.28	48,202
Probable—Stockpile	2,191	0.86	60,634
Total Proven & Probable	**24,939**	**1.34**	**1,074,941**

Notes: **1.** CIM Definition Standards (2014) were followed for Mineral Reserves.
2. Mineral Reserves were generated by Equinox based on the June 30, 2020 mining surface.
3. Mineral Reserves are quoted at cut-off grades of 0.52 g/t Au for dacite-leachable and carbonaceous ores for the C1 deposit; and cut-off grades of 0.54 g/t Au for dacite-leachable and carbonaceous ores and 0.45 g/t Au for dacite-high-sulphide ore for the Antas 3 deposit.
4. C1 and Antas 3 use a 10 m bench height (flitch height 5 m benches).
5. Process recovery of 84% for all types of ore.
6. Mineral Reserves were calculated using a long-term gold price of US$1,350/oz.
7. Totals may not add due to rounding.

Equinox and SLR are not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other relevant factors that could materially affect the Mineral Resource and Mineral Reserve estimates.

1.2 Technical Summary

1.2.1 Property Description and Location

The Santa Luz Project is located within the Maria Preta mining district, 35 kilometres (km) north of the town of Santa Luz, in Bahia State. The Project is approximately 240 km northwest of the state capital, Salvador, 115 km by road from the Fazenda Brasileiro gold mine, and 163 km from the Jacobina gold mine. The centre of the property has approximate latitude and longitude coordinates of 11°00'28" S and 39°18'28" W, respectively.

1.2.2 Land Tenure

The Santa Luz properties cover an area totalling 48,599.25 hectares (ha) including 36 exploration permits (42,666.41 ha), six mining concessions (2,611.69 ha), and four mining concessions in application (3,321.15 ha).

1.2.3 History

During the 1970s, Companhia Vale do Rio Doce (CVRD) invested in a regional prospecting program in Bahia State, while other private and state companies carried out intensive prospecting, geological mapping, and research programs. During this time, the Rio Itapicurú Greenstone Belt (RIGB) was identified.

Between 1979 and 1981, Companhia Baiana de Pesquisa Mineral (CBPM) conducted several geological and prospecting programs within the RIGB. These activities identified several gold-bearing trends and prospects including deposits within the Project area, which were mined between 1987 and 1995 by CBPM's subsidiary Rio Salitre Mineração Ltda.

In January 2005, Yamana completed an agreement with CBPM to acquire 7,000 ha of land over the C1 historical mine.

In May 2007, Yamana expanded its land ownership through the acquisition of mining concessions from Mineração Santa Elina (MSE), formerly owned by CVRD, which included the Antas 1 (now considered part of C1), Antas 2, and Antas 3 deposits and associated historical mine workings.

In December 2014, a new subsidiary, Brio Gold Inc. (Brio), was formed by Yamana to hold the Fazenda Brasileiro mine, the Pilar mine, and the Santa Luz properties, and later the acquired Riacho dos Machados Mine. Leagold acquired Brio on May 24, 2018 and became the owner of Santa Luz and Brio's other Brazilian mines. On March 10, 2020, Equinox acquired Leagold.

1.2.4 Geology and Mineralization

The Project area is hosted within the RIGB, which comprises the northeastern portion of the São Francisco Craton which was formed through the collision of several small Archean cratons during the Paleoproterozoic Trans-Amazon Orogeny (approximately 2 Ga).

The Paleoproterozoic aged RIGB is the largest greenstone belt in the São Francisco Craton. The north-south trending belt extends for approximately 100 km and ranges in width from 30 km to 50 km. The belt comprises three domains (mafic volcanic, felsic volcanic, and sedimentary), all intruded by later granitioid bodies.

Gold deposits and prospects in the Project area occur in shear and breccia zones at, or proximal to, the faulted contact of the volcanic and sedimentary domains in a continuous, north and locally north-easterly striking, mineralized zone. Mineralization is associated with quartz-carbonate-sulphide veining and breccia fillings. Significant gold targets and deposits at Santa Luz include C1 (historically called Maria Preta and included Antas 1), Antas 2, Antas 3, Mansinha South, Mansinha North, and Mari. The deposits are considered greenstone gold type deposits, a subgroup of the Orogenic Gold Deposit type.

Host rocks include a variety of epizonal dioritic and dacitic intrusive rocks, sedimentary rocks, and felsic to intermediate volcanic rocks. Volcanic and epizonal intrusive rocks are generally porphyritic with fine to medium grained quartz and felsdpar phenocrysts. Sedimentary rocks, including tuffaceous rocks contain variable quantities of organic carbon which appears to be a primary depositional component. More massive volcanic and epizonal intrusive rocks are relatively free of organic carbon. The organic carbon content is a major focus of geologic studies as the carbon interferes with cyanide leach gold recovery. Organic carbon-rich rocks require special treatment to facilitate gold recovery. All rocks of the RIGB have undergone greenschist to amphibolite grade metamorphism.

1.2.5 Exploration Status

From 1979 to 1995, CVRD and CBPM undertook several extensive stream sediment and soil geochemistry programs over the entire Maria Preta mining district. Encouraging results were followed up using geophysics and approximately 15,166 m of drilling.

From 2003 to 2013, Yamana explored the district with 201,379 m of drilling, including 126,658 m of diamond core drilling, spread across numerous deposit areas. Yamana also conducted soil and rock chip sampling and geologic mapping.

In 2015 and 2016, Brio conducted 20,590 m of exploration, geotechnical and metallurgical drilling, including 13,425 m of diamond core drilling for resource definition.

In late 2016 and early 2017, Brio conducted 4,036 m of exploration and geotechnical drilling.

Past owners have drilled a total of 3,884 drill holes collecting over 241,172 m of drill core and chip samples in the district.

1.2.6 Metallurgical Testing

The metallurgical testing programs for the Santa Luz processing facilities began in 2005 and supported the Yamana 2009 Feasibility Study. A pilot test program was performed in 2009, followed by further pilot plant testing in 2010. Production at Santa Luz commenced in 2013; however, it was discontinued in September 2014 and the facilities were put on care and maintenance, following a period of very low gold recoveries associated with the processing of carbonaceous ores. In late 2014, a metallurgical testing program was initiated by Brio to re-evaluate the existing process facilities, to determine the causes of the low gold recoveries and to develop a revised flowsheet to successfully process the carbonaceous material at Santa Luz.

The naturally occurring carbon was shown in the testwork to strongly adsorb gold from solution (preg-robbing). Kerosene was selected as a blinding agent to deactivate the natural carbon prior to RIL cyanide leaching. Gold recoveries were very low in leach tests performed without kerosene.

Gravity concentration testwork in 2015 and 2016 suggested potential gold recoveries around 20%.

More testwork was carried out in 2016 and 2017. This was designed to further develop the proposed whole ore leach flowsheet and formed the basis for preparing the design criteria, process flow diagrams, mass balance, and equipment sizing. The testwork was conducted by various laboratories including Commonwealth Scientific and Industrial Research Organization (CSIRO) in Perth, Australia, Hazen Research Inc. in the USA, RDI Minerals in the USA, SGS Geosol Laboratórios Ltda. in Brazil, as well as the Santa Luz on-site laboratory and pilot plant. The testwork program included:

- Bond Ball Mill Work index tests for bulk composites of dacite and carbonaceous ore
- Whole ore cyanidation using both CIL and RIL flowsheet variations
- Reagent optimization
- Variability testwork.

Further testwork was conducted in 2019 at Mintek in South Africa and at the Santa Luz on-site pilot plant to optimize the whole ore RIL processing circuit, to increase the gold grade (and reduce the copper grade) of the loaded resin and to optimize gold recovery from the resin.

The results of the programs show that the most favourable option is to process the dacitic and carbonaceous breccia ores together and to use RIL and a kerosene blanking circuit. Blending the dacitic breccia with the carbonaceous breccia results in slightly lower recoveries, due to preg-robbing by natural carbon in the carbonaceous ore. Gold recoveries based on combined feed and testwork is approximately 84%.

1.2.7 Mineral Resources

Mineral Resources for each of the deposits at Santa Luz were estimated by Santa Luz Project personnel in 2017 with the support of resource, geotechnical and metallurgical drilling and extensive metallurgical testwork conducted in 2015, 2016, and 2017. The Mineral Resources were reviewed by SLR.

Lithology, alteration, and mineralization domains were constructed over each deposit using gold grade thresholds specific to each area, in combination with lithology, alteration, and structural information. Variography and basic statistics were used to inform interpolation plans, which used Ordinary Kriging or Inverse Distance Squared methods to estimate gold values from capped gold composites within discrete block models in a series of interpolation passes. Density was averaged from on-site samples and applied to lithology and weathering domains in each deposit. Blocks were classified based on interpolation pass and Kriging variance. SLR conducted a series of block validation and data integrity tests on the block model. Mineral Resources were constrained using a Lerchs Grossmann pit.

The Mineral Resource is current and no additional work was undertaken after the estimate was completed. Mineral Resources exclusive of Mineral Reserves are summarized in Table 1-1 as reported by SLR and with an effective date of June 30, 2020.

1.2.8 Mineral Reserves

Equinox has generated Open Pit Mineral Reserve estimates for the C1 and Antas 3 deposits and reviewed the stockpile estimates prepared by Santa Luz Project personnel.

The Open Pit Mineral Reserves as estimated by Equinox as of June 30, 2020 are summarized in Table 1-2 using a gold price of US$1,350/oz and metal recoveries of 84% for dacite-leachable ore, 84% for dacite-high-sulphide ore, and 84% for carbonaceous ore. Mineral Reserves are quoted for dacite-leachable and carbonaceous ore at cut-off grades of 0.52 g/t Au for the C1 pit and 0.54 g/t Au for the Antas 3 pit, and 0.45 g/t Au for dacite-high-sulphide ore. Mineral Reserves are estimated only for C1, Antas 3 and stockpiles.

1.2.9 Mining Method

The Open Pit Feasibility Study is based on open pit mining with production from three pits: one pit at the C1 deposit and two small pits at the Antas 3 deposit. Pit bench heights will be 10 m and be mined in two 5 m flitches with a safety berm every 10 m. The ore and waste rock will be drilled and blasted, loaded with front end loaders, and hauled to either a crusher or waste rock dump. Haulage distances from the open pit to the crusher area will vary, with an average haul distance of approximately 3.9 km for C1 and 2.5 km for Antas 3. Mining will be carried out by contractors and mine technical services will be provided by Santa Luz personnel.

The mine will operate on a general production schedule of 24 h/d, 7 d/wk. The LOM is nine years for C1, and six years for Antas 3. The average mining rate will be approximately 15.3 Mt/a and the maximum mining rate will be approximately 22.0 Mt/a of ore and waste mined including some overlap of mining between deposits. The LOM is estimated to be 9.5 years, including 1.5 years of post-production processing of stockpiles.

Table 1-3 summarizes the open pit dimensions.

All material from the pit is excavated and is hauled by trucks to intermediate stockpiles. The material is then removed from these stockpiles with loaders and trucks and transported to the primary crusher. Considering that the ore blend is a critical factor to ensure the metallurgical recovery, the intermediate stock areas are designed to keep sufficient ore volumes and grades appropriate for blending the plant's feed. The mining plan and stock balance ensure the blend of 50% dacitic and 50% carbonaceous ores in the plant's feed, thus targeting the appropriate total organic carbon (TOC) level to achieve the desired metallurgical recovery throughout the LOM, as shown in Table 1-4.

Table 1-3: Santa Luz Pit Design Overview

			Years										
	Unit	**Total**	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Mined													
Dacite Leach	t '000s	8,803	215	728	815	1,051	1,318	1,648	1,258	1,487	282	0	0
	Au g/t	1.32	1.53	1.82	1.25	1.23	1.51	1.18	1.01	1.33	1.58	0	0
Dacite-High-Sulphide	t '000s	1,700	0	0	6	28	218	267	880	301	0	0	0
	Au g/t	0.96	0.00	0.00	0.92	0.95	0.75	0.76	1.01	1.15	0	0	0
Carbonaceous	t '000s	12,244	355	1,043	1,319	1,442	1,693	2,539	1,407	1,909	537	0	0
	Au g/t	1.50	1.40	1.67	2.06	1.54	1.33	1.40	1.13	1.48	1.71	0	0
Total Ore Mined	**t '000s**	**22,747**	**570**	**1,771**	**2,140**	**2,521**	**3,230**	**4,454**	**3,546**	**3,697**	**819**	**0**	**0**
	Au g/t	**1.39**	**1.45**	**1.73**	**1.75**	**1.40**	**1.36**	**1.28**	**1.06**	**1.39**	**1.66**	**0**	**0**
Stockpile Balance													
Initial Stockpile (DAC)	t '000s		1,481	1,697	1,188	659	387	574	1,139	1,927	2,365	1,297	0
	Au g/t		0.78	0.87	0.65	0.57	0.59	0.70	0.72	0.84	0.87	0.71	0.00
Ore Mined (DAC)	t '000s		215	264	202	204	661	875	1,112	722	113	0	0
	Au g/t		1.53	0.68	0.57	0.62	0.68	0.72	0.92	0.88	0.94	0	0
Milled Ore (DAC)	t '000s		0	773	730	475	475	309	323	284	1,181	1,297	0
	Au g/t		0.00	1.15	0.70	0.58	0.58	0.71	0.68	0.67	1.05	0.71	0.00
Final Stockpile (DAC)	t '000s	1,481	1,697	1,188	659	387	574	1,139	1,927	2,365	1,297	0	0
	Au g/t	0.78	0.87	0.65	0.57	0.59	0.70	0.72	0.84	0.87	0.71	0	0
Initial Stockpile (CARB)	t '000s		709	1,064	869	838	930	1,274	2,462	2,519	3,079	2,266	863
	Au g/t		1.03	1.14	1.69	1.79	1.79	1.17	1.18	1.18	1.16	1.35	0.69
Ore Mined (CARB)	t '000s		355	250	394	457	597	1,271	626	573	200	0	0
	Au g/t		1.36	3.82	1.21	1.21	0.00	1.20	1.15	1.05	1.16	0	0
Milled Ore (CARB)	t '000s		0	445	425	364	254	82	569	14	1,013	1,403	863
	Au g/t		0.00	1.58	1.05	1.05	0.69	1.23	1.15	1.05	0.73	0.69	0.69
Final Stockpile (CARB)	t '000s	709	1,064	869	838	930	1,274	2,462	2,519	3,079	2,266	863	0
	Au g/t	1.03	1.14	1.69	1.79	1.79	1.17	1.18	1.18	1.16	1.35	0.69	0
Processed													
Dacite Leach	t '000s	10,285	0	1,238	1,350	1,350	1,350	1,350	1,350	1,350	675	272	0
	Au g/t	1.24	0.00	1.64	1.05	1.09	1.46	1.28	1.00	1.39	1.04	0.64	0
Dacite-High-Sulphide	t '000s	1,700	0	0	0	0	0	0	0	0	675	1,025	0
	Au g/t	0.96	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.30	0.73	0
Carbonaceous	t '000s	12,953	0	1,238	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,403	863
	Au g/t	1.47	0.00	1.76	2.10	1.66	1.51	1.87	1.30	1.82	1.09	0.69	0.69
Total Ore Processed	**t '000s**	**24,938**	**0**	**2,475**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**2,700**	**863**
	Au g/t	**1.34**	**0.00**	**1.70**	**1.58**	**1.37**	**1.49**	**1.57**	**1.15**	**1.60**	**1.13**	**0.70**	**0.69**
	% TOC	**0.53**	**0.00**	**0.54**	**0.54**	**0.51**	**0.51**	**0.51**	**0.52**	**0.53**	**0.48**	**0.53**	**0.96**
Recovery	%	84	0	84	84	84	84	84	84	84	84	84	82
Recovered Gold	Au oz '000s	903	0	114	115	100	109	115	84	117	82	51	16

Table 1-4: Santa Luz LOM Production Schedule and Stockpile Balance

	Unit	Total	Years										
			0	1	2	3	4	5	6	7	8	9	10
Mined													
Dacite Leach	t '000s	8,803		304	667	1,051	1,101	1,258	1,374	1,201	1,630	217	0
	Au g/t	1.32		1.60	1.81	1.28	1.45	1.42	1.09	0.95	1.33	1.61	0.00
Dacite-High-Sulphide	t '000s	1,700		0	0	0	33	99	283	777	393	113	0
	Au g/t	0.96		0.00	0.00	0.00	0.96	0.76	0.72	0.96	1.10	1.22	0.00
Carbonaceous	t '000s	12,244		522	859	1,759	1,480	1,656	2,303	1,210	2,029	427	0
	Au g/t	1.50		1.44	1.69	1.92	1.62	1.30	1.37	1.06	1.49	1.76	0.00
Total Ore Mined	t '000s	22,747		825	1,526	2,809	2,614	3,014	3,961	3,188	4,052	757	0
	Au g/t	1.39		1.50	1.75	1.68	1.54	1.33	1.23	1.00	1.39	1.64	0.00
Stockpile Balance													
Initial Stockpile (DAC)	t '000s			1,481	1,448	765	466	250	258	565	1,193	1,867	847
	Au g/t			0.78	0.73	0.60	0.56	0.59	0.69	0.75	0.86	0.86	0.63
Ore Mined (DAC)	t '000s			182	220	144	108	315	669	825	754	108	0
	Au g/t			1.30	0.66	0.55	0.64	0.64	0.74	0.90	0.83	0.82	0.00
Milled Ore (DAC)	t '000s			216	902	444	324	307	361	196	81	1,128	847
	Au g/t			1.53	0.82	0.64	0.56	0.55	0.69	0.70	0.68	1.03	0.63
Final Stockpile (DAC)	t '000s		1,481	1,448	765	466	250	258	565	1,193	1,867	847	0
	Au g/t		0.78	0.73	0.60	0.56	0.59	0.69	0.75	0.86	0.86	0.63	0.00
Initial Stockpile (CARB)	t '000s			709	893	402	811	941	1,247	2,200	2,060	2,739	1,816
	Au g/t			1.03	0.88	0.48	0.69	0.45	0.52	0.60	0.49	0.54	0.46
Ore Mined (CARB)	t '000s			249	325	737	508	548	1,180	283	715	89	0
	Au g/t			0.75	0.78	0.82	0.74	0.71	0.81	0.65	0.73	0.75	0.00
Milled Ore (CARB)	t '000s			65	816	328	378	242	227	423	36	1,012	1,816
	Au g/t			2.04	1.04	0.71	1.36	0.69	1.29	1.17	1.10	0.71	0.46
Final Stockpile (CARB)	t '000s		709	893	402	811	941	1,247	2,200	2,060	2,739	1,816	0
	Au g/t		1.03	0.88	0.48	0.69	0.45	0.52	0.60	0.49	0.54	0.46	0.00
Processed													
Dacite Leach	t '000s	10,285		338	1,350	1,350	1,350	1,350	1,350	1,350	1,350	497	0
	Au g/t	1.24		2.00	1.32	1.09	1.31	1.36	1.07	0.92	1.47	1.08	0.00
Dacite-High-sulphide	t '000s	1,700		0	0	0	0	0	0	0	0	853	847
	Au g/t	0.96		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.20	0.72
Carbonaceous	t '000s	12,953		338	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,816
	Au g/t	1.47		1.80	1.73	1.95	1.83	1.46	1.76	1.11	1.87	1.04	0.68
Total Ore Processed	t '000s	24,938		675	2,700	2,700	2,700	2,700	2,700	2,700	2,700	2,700	2,663
	Au g/t	1.34		1.90	1.53	1.52	1.57	1.41	1.41	1.02	1.67	1.10	0.69
Recovery	%	84%		84%	84%	84%	84%	84%	84%	84%	84%	84%	84%
Recovered Gold	Au oz '000s	903		35	111	111	114	103	103	74	122	80	50

Notes: **1.** Numbers may not add due to rounding.
2. Dacite ore in stockpile balance includes high-sulphide ore.
3. Initial stockpile is shown in Table 15-3.

1.2.10 Mineral Processing

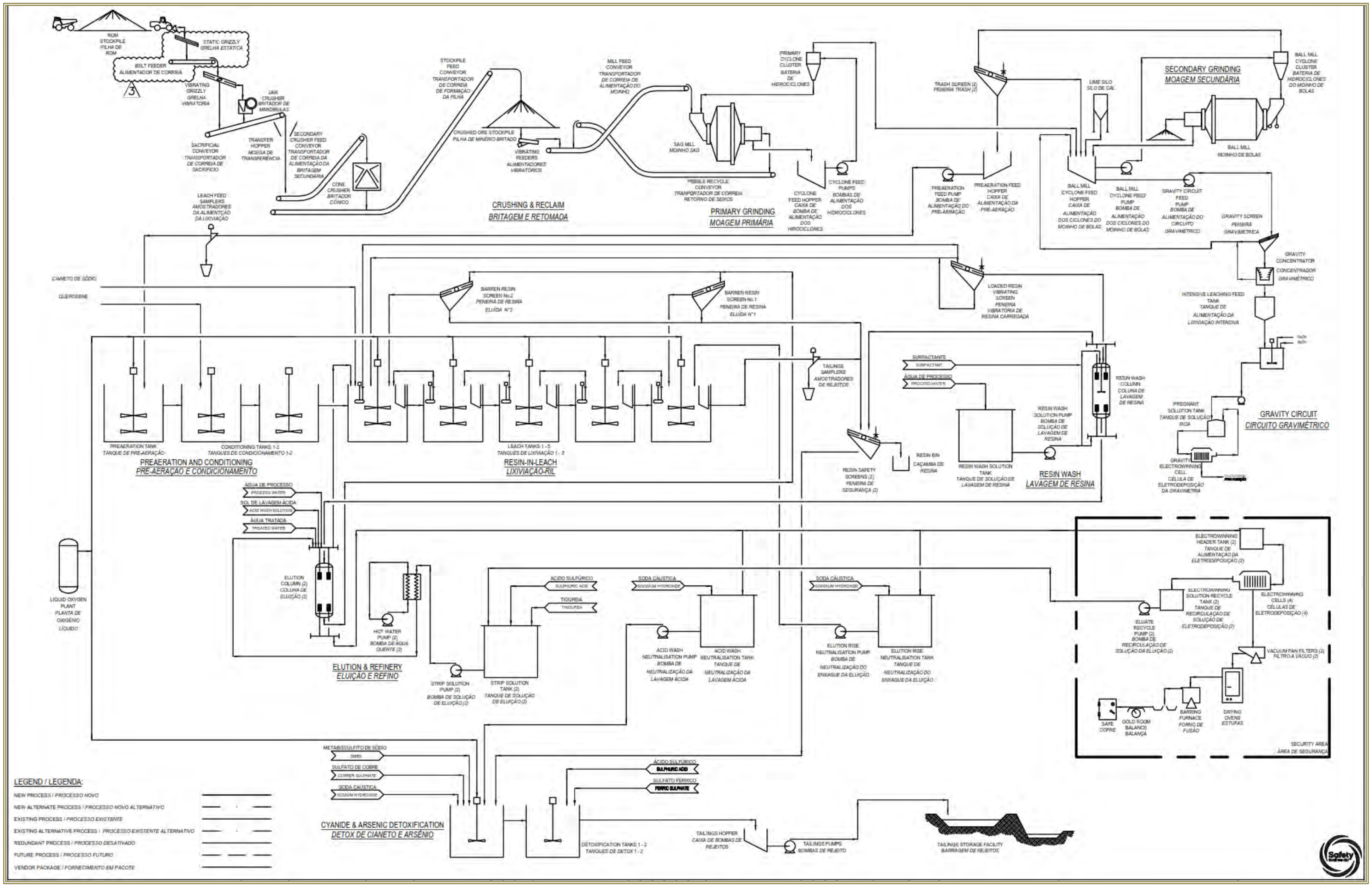
The Santa Luz processing facilities were commissioned in 2013, operated for approximately 14 months, and were put on care and maintenance in September 2014, due to a period of very low gold recoveries associated with the processing of carbonaceous ores. The existing plant is in reasonable physical condition, with some refurbishment and installation of additional equipment required to modify the plant for RIL processing. Additional grinding power will be installed to ensure design throughput and grind size are achieved.

From late 2014 to the present, a metallurgical testing program has been conducted to evaluate the existing process facilities, determine the causes of the low gold recoveries, and develop a new flowsheet and recommendations for plant modifications to successfully process the carbonaceous material at Santa Luz. The results of the testing program led to a decision to develop a preliminary design and economic assessment based on a whole ore CIL flowsheet rather than the original flotation and concentrate leaching flowsheet. In late 2015, a new testwork program was established to assist in flowsheet optimization, including the comparison of a RIL circuit versus a conventional CIL circuit. With the addition of variability testwork, it was decided to move forward with a RIL process.

A dedicated kerosene blinding circuit is included in the flowsheet to effectively use kerosene to deactivate the naturally occurring carbon that was the main cause for the gold recovery problems experienced during the 2013-2014 operations. The design will utilize as much of existing equipment as possible and either add or modify equipment as required. The process circuit now includes:

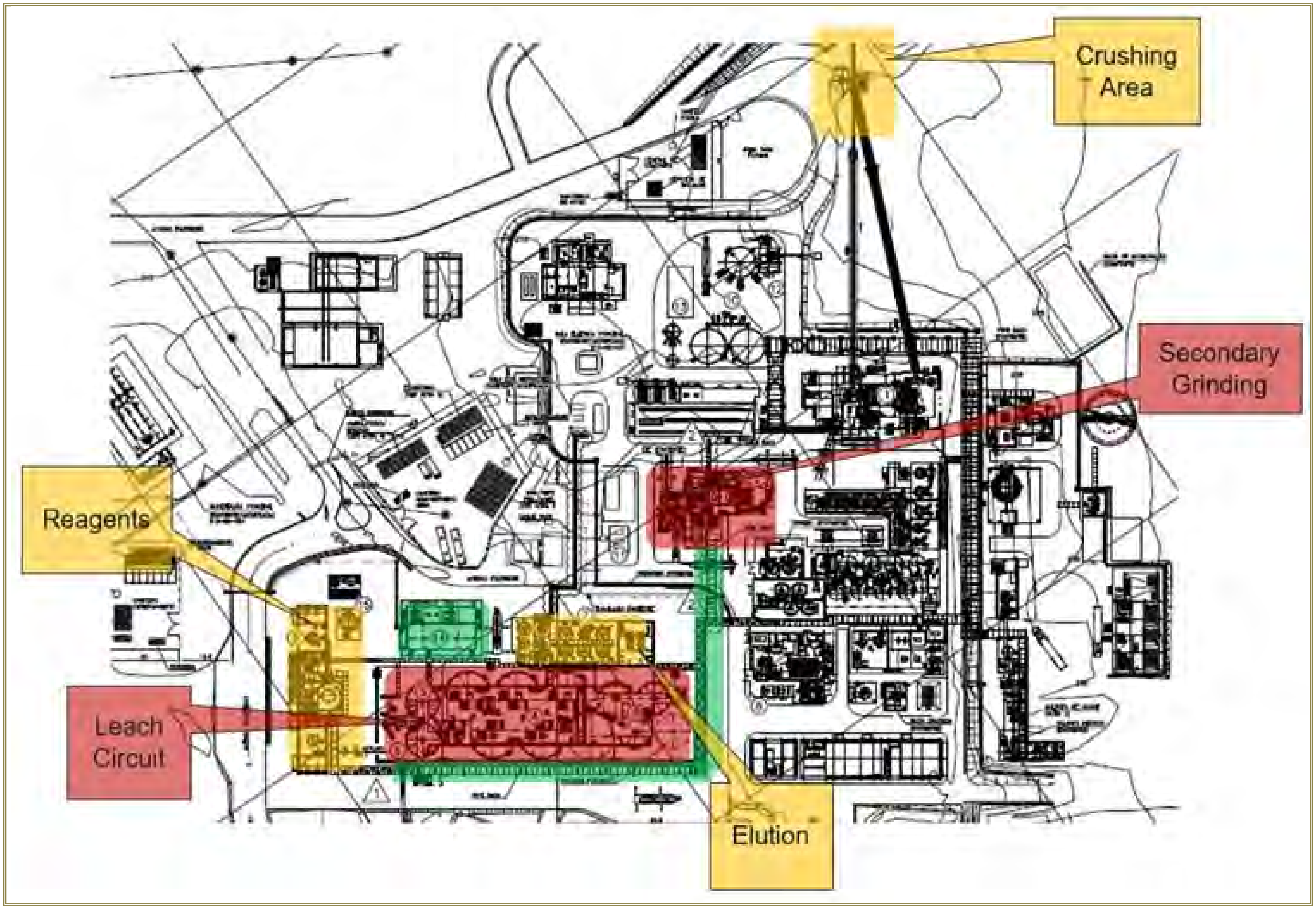
- Primary and secondary crushing
- Primary semi-autogenous grinding (SAG) mill grinding
- Secondary grinding using a conventional ball mill
- Gravity concentration
- Cyclone classification
- Kerosene pre-treatment in a dedicated circuit prior to RIL leaching
- Whole ore RIL leaching
- Cyanide destruction
- Resin acid washing, elution, and resin regeneration
- Electrowinning of the gold
- Doré casting
- Tailings storage facility (TSF), which has been geosynthetically lined, will be used for storage of whole ore leach tailings
- Water storage facility (WSF) will be used for storage of raw water.

A simplified flow diagram of the processing plant is shown in Figure 1-1, and a general arrangement of the plant is shown in Figure 1-2 and Figure 1-3.



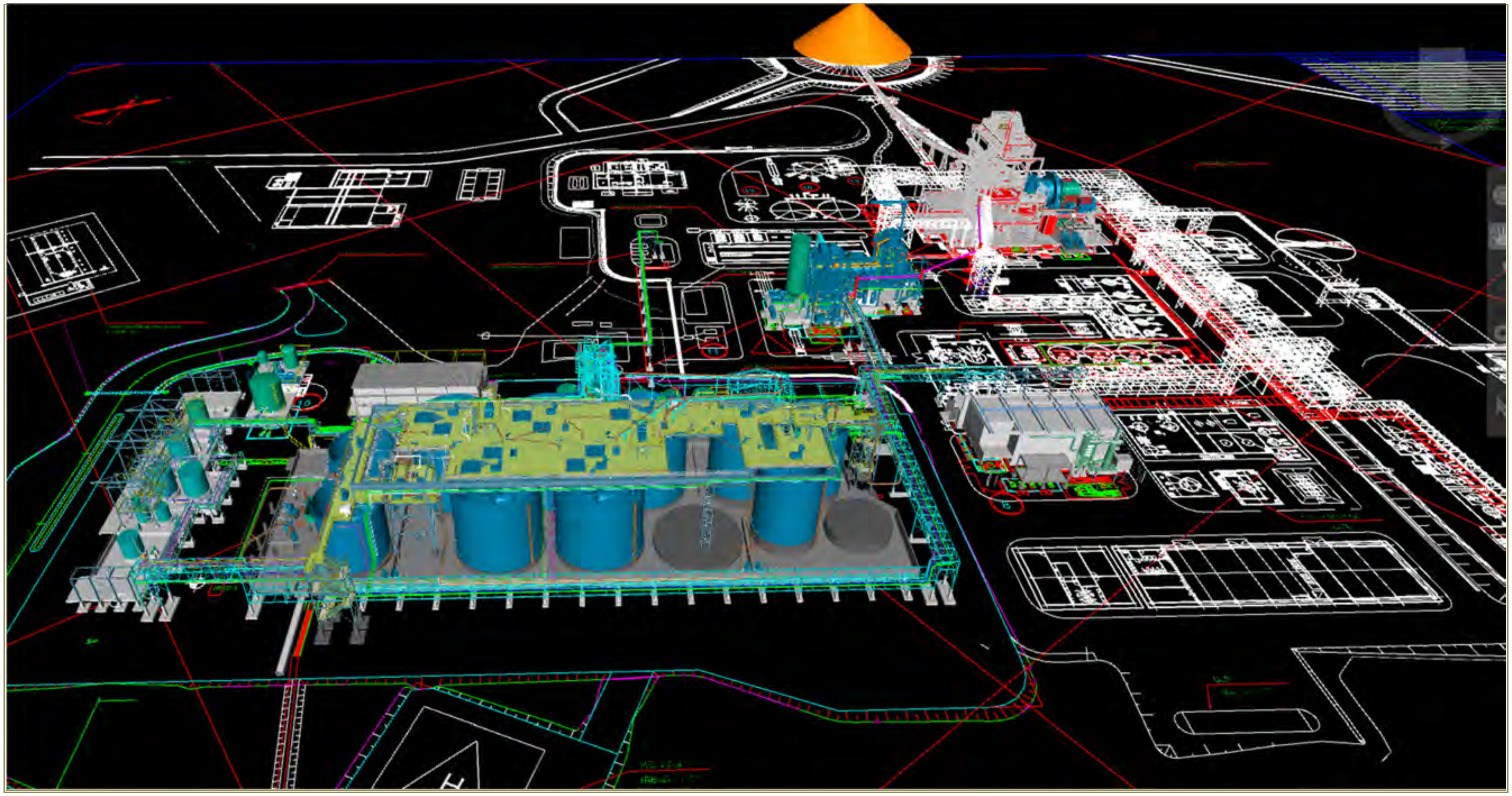
Source: Ausenco

Figure 1-1: Simplified Process Flowsheet



Source: Ausenco

Figure 1-2: Process Plant General Arrangement



Source: Ausenco

Figure 1-3: Process Plant 3D Isometric View

1.2.11 RIL Plant Operating Parameters

The process operating parameters for the Santa Luz plant, modified for whole ore leaching, are presented in Table 1-5 and are the basis for this RIL process flowsheet and project feasibility. Average processing operating costs are estimated at US$13.74/t for the combined dacite-carbonaceous blended ore and US$9.17/t for dacite with high sulphides, for an average LOM processing cost of $13.43/t.

Table 1-5: Santa Luz RIL Process Operating Parameters

Parameter	Unit	Value
Throughput Rate		
Annual	t/a	2,700,000
Daily	t/a	7,400
Operating Period	years	9.5
Ore Grade (average LOM)		
Gold (including stockpiles)	g/t	1.34
Total Organic Carbon (TOC)	%	0.6
Arsenic	g/t	500
Gold Recovery	%	84
Gold Production	oz/a	95,000
Ore Physical Characteristics		
Work Index	kWh/t	19
Abrasion Index		0.5
Primary Crush Size	80% passing, mm	150
Secondary Crush Size	80% passing, mm	50
Primary Mill Grind Size	80% passing, µm	860
Secondary Mill Grind Size	80% passing, µm	75
Gravity		
Recovery	%	20%
Retention Times		
Conditioning	hours	6
Leaching	hours	20
Detoxification	hours	3
Employees		
Management	number	12
Operation	number	71
Maintenance	number	74
Utilities Consumption		
Power	kWh/t	42
Fresh Water (make-up)	m^3/t	0.40
Consumables		
Resin	m^3/t	0.00003
Grinding Balls	kg/t	1.80
Quick Lime	kg/t	1.00
Kerosene	kg/t	2.00
Sodium Cyanide	kg/t	0.75
Sodium Metabisulphite (SMBS)	kg/t	0.75
Thiourea	kg/t	0.25
Operating Cost (LOM, all ores)	US$/t	13.43

1.2.12 Project Infrastructure

The operation includes open pit workings and gold ore processing facilities, as well as other necessary buildings and infrastructure. This infrastructure includes:

- Mine workings and equipment.
- A 7,400 t/d processing plant.
- Power supplied from a 138 kV power line extending from the Coelba power station to the main substation at the Santa Luz plant site.
- Water for use on the Project site is sourced from the Itapicurú River, the main drainage system in the area, and will be stored in the WSF, which was previously used to store leached flotation concentrate tailings, in the Antas 3 pit, which will be drained prior to mining and the water transferred to the WSF, and the TSF, which was previously used to store flotation tailings.
- The WSF has a current storage capacity of 0.3 Mm^3 and a planned total storage capacity of 3.0 Mm^3.
- The TSF has a current storage capacity of 3.0 Mm^3 (or 4.2 Mt), and a planned total storage capacity of up to 25.2 Mm^3 (or 35.3 Mt), for tailings from the new RIL plant.

Santa Luz site infrastructure is shown on the site map provided in Figure 1-4.

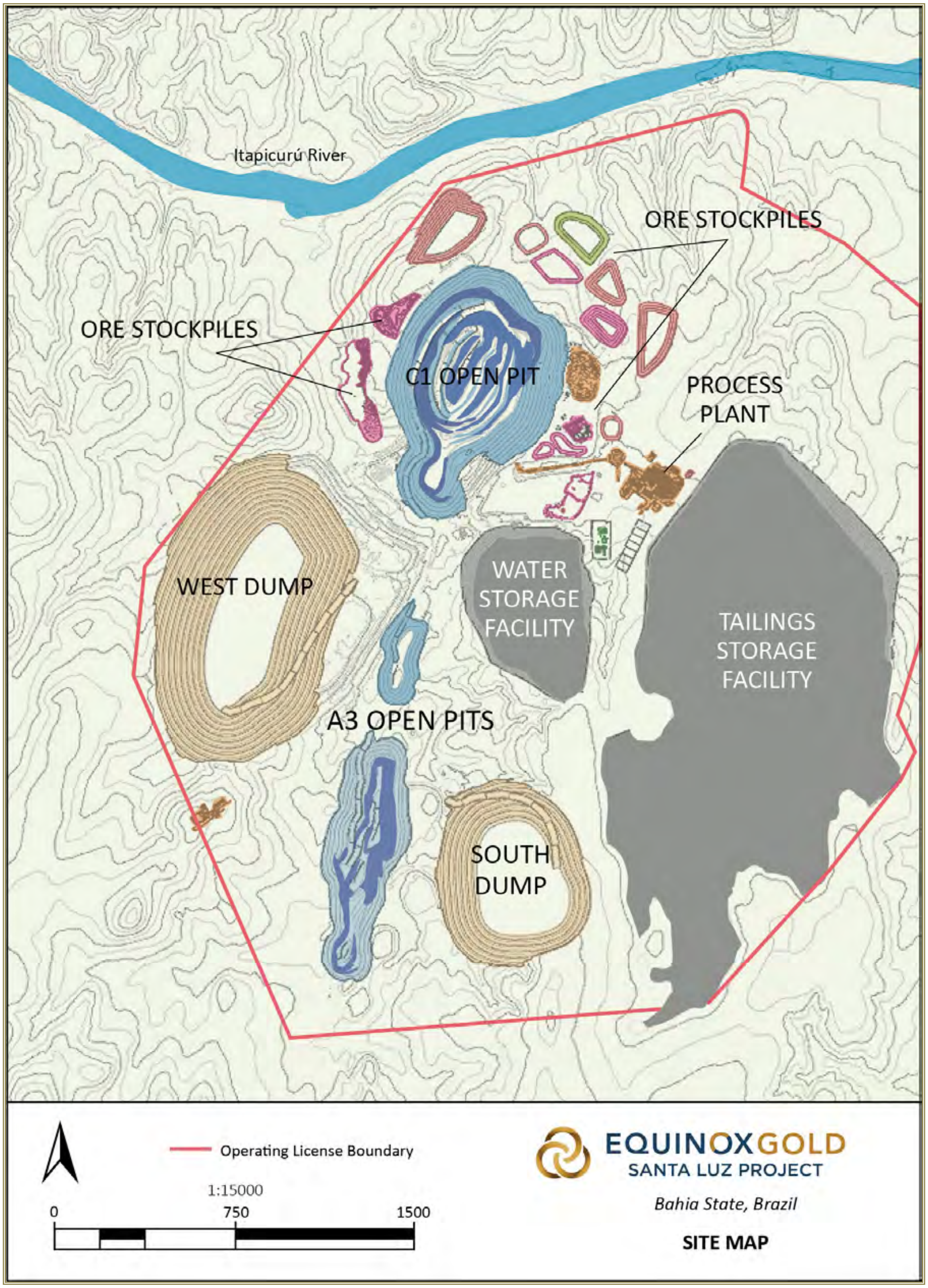


Figure 1-4: Site Infrastructure

1.2.13 Market Studies

Gold is the principal commodity at Santa Luz and is freely traded, at prices that are widely known, so that prospects for sale of any production are virtually assured. Prices are often quoted in United States dollars (US$) per troy ounce.

Santa Luz will use contractors for mining, drilling and road maintenance, similar to contracts that were in place during the previous operations. The explosive supply will include blasting services at the mine. There will be supply contracts for electricity, fuel, spare parts, steel balls, and processing reagents following a similar supplier structure as in the previous operation.

Santa Luz will prioritize sourcing goods and services through local suppliers, to contribute to the sustainable economic development of local communities.

1.2.14 Environmental, Permitting, and Social Considerations

The company complies with established environmental technical requirements. The environmental and social (E&S) impacts of the Project, such as noise level, alteration of the morphology, increase in dust levels, surface and groundwater quality, and deforestation, among others, have been assessed and appropriate mitigation measures have been put in place.

The Project has all relevant permits in place and no environmental liabilities associated with the property were identified. Equinox is currently renewing its operational licences and has requested an adjustment in general terms to reflect amendments to the operational process.

Yamana had previously committed to a number of community concessions for the nearby village of Nova Esperança, including village relocation, community compensation, and other environmental considerations. The village construction and relocation was completed in 2018.

The current estimate for the reclamation and closure of the mine is US$8.8 million based on current land disturbances.

1.2.15 Capital and Operating Cost Estimates

Capital costs for the Project are summarized in Table 1-6.

A summary of the Project's operating costs is shown in Table 1-7.

Table 1-6: Santa Luz Summary of Project Capital Costs

Capital Category	Pre-Production (US$ '000s)	Year 1 (US$ '000s)	Year 2 (US$ '000s)	Years 3 to 10 (US$ '000s)	Total (US$ '000s)
Plant Alterations	37.5	-		-	37.5
TSF and WSF Raises	7.5	-		-	7.5
EPCM and others	4.8	-		-	4.8
Owner's Costs	10.2	-		-	10.2
Pre-Stripping	20.5	-		-	20.5
Contingency	9.5	-		-	9.5
Working Capital	13.0	-	-	(5.6)	7.4
Salvage	-	-	-	(15.0)	(15.0)
Deferred-Stripping Capital Cost	-	22.8	17.6	20.2	60.6
Sustaining Capital Cost	-	6.9	1.7	12.5	21.0
Reclamation Cost	0.1	0.1	0.0	8.6	8.8
Total Capital Cost	**103.1**	**29.8**	**19.3**	**20.7**	**172.9**

Note: **1.** LOM exchange rate R$5.00:US$1.00.

Table 1-7: Santa Luz Summary of Project LOM Operating Costs

Total Operating Costs	LOM Total (US$ '000s)	Unit Costs (US$/t Processed)
Mining Cost	262,724	10.54
Grade Control	4,357	0.17
Ore Re-handle (ROM Pad to Crusher)	11,222	0.45
Ore Re-handle (Stockpiles)	16,921	0.68
Processing	334,875	13.43
Fixed G&A	68,579	2.75
Total Operating Costs	**698,678**	**28.02**

1.3 Economic Analysis

The economic analysis contained in this Technical Report is based on Proven and Probable Mineral Reserves only.

The after-tax cash flow projection is summarized in Table 1-8 and is based on the Open Pit LOM production schedule and capital and operating costs. A more detailed cash flow summary is presented in Table 1-9.

Table 1-8: Santa Luz Cash Flow Summary (US$1,500/oz Au)

Description	Unit	Value
After-tax IRR	%	57.6
After-tax NPV at 0.0% discount rate	US$ M	436.0
After-tax NPV at 5.0% discount rate	US$ M	305.1
After-tax NPV at 8.0% discount rate	US$ M	248.1

A summary of the key criteria is provided below.

1.3.1 Economic Criteria

Revenue

- Approximately 7,400 t/d of ore processed (approximately 2.7 Mt/a).
- Processing gold recoveries of 84% were used in the cash flow for a blended feed of high carbonaceous material, low carbonaceous material, and dacitic ore. Gold recovery for dacites with high-sulphides is also projected to be 84%.
- Metal prices for cash flow: US$1,500/oz Au.
- Salvage value of US$15 million was applied to equipment or infrastructure at the end of the LOM.
- 9.5-year project life during production.
- Yearly revenues were calculated by subtracting the applicable refining charges and transportation costs (US$10/oz) from the payable metal value generated by carbonaceous and dacitic ore and US$177/oz from dacites with high-sulphide ore.
- Revenue is recognized at the time of production.
- Production schedule includes only Proven and Probable Mineral Reserves costs.
- There are 6.9 Mt mined excluding stockpile rehandle as pre-stripping prior to the start of commercial production.
- Unit operating costs for mining, processing, rehandle, grade control, and G&A were applied to determine the overall yearly operating cost.
- Closure costs for the Project have been estimated at US$8.8 million and these costs are included in the cash flow.
- Initial capital cost totals US$103.1 million.
- Local currency denominated capital and operating costs are based on a nominal exchange rate of R$5.00:US$1.00.
- Project LOM all-in-sustaining cost (AISC) is US$877/oz.

Royalties

An existing royalty agreement with the Federal Government for 1.5% gross revenue, and another agreement for 1% gross revenue with Companhia Sisal do Brasil (COSIBRA), was included in the cash flow and pit optimization analysis. An additional 2% royalty was included for the CBPM area of the C1 deposit, which represents a royalty on 397,810 oz in the production schedule.

Taxation

For the calculation of income taxes, it has been assumed that a government economic stimulus program (the SUDENE) mining tax incentive would be approved for the duration of the LOM, which results in an income tax rate of 15.25%. An average rate of 9.25% was assumed for operating and capital costs subject to Brazilian federal value-added-taxes (PIS and Cofins) and 12% was assumed for items subject to state value-added taxes (ICMS).

1.3.2 Cash Flow Analysis

Table 1-9 shows cash flow summary results for Santa Luz. The financial model was established on a 100% equity basis, which does not include debt financing and loan interest charges.

Considering the Project on a stand-alone basis, the undiscounted after-tax cash flow totals US$436.0 million over the LOM.

The after-tax NPV at a 5% discount rate is US$305.1 million, with an IRR of 57.6%.

Table 1-9: Santa Luz Cash Flow Summary Results

	Unit	LOM Total
Total Ore Mined	kt	22,747
Total Waste Mined	kt	106,519
Total Material Moved	kt	129,266
Strip Ratio	w:o	4.7
Au Grade	g/t	1.39
Contained Gold	oz	1,014,263
Stockpiled Ore Processed	kt	2,191
Au Grade	g/t	0.86
Contained Gold	oz	60,654
Total Ore Processed	kt	24,938
Processed Au Grade	g/t	1.34
Contained Gold	oz	1,074,917
Recovery	%	84
Recovered Gold	oz	902,549
Mine Life	year	9.5
Initial Capital	US$M	103.1
Sustaining Capital (excluding capitalized stripping)	US$M	21.0
Average Annual Production (LOM)	oz	95,000
Average Annual Production (2022–2026)	oz	110,500
Average Annual Production (2022–2029)	oz	104,500
Average Annual EBITDA (LOM)	US$M	68.7
Average Annual EBITDA (2022–2024)	US$M	84.6

	Unit	LOM Total
Average Annual Net Cash Flow (LOM, after tax)	US$M	56.9
Net Cumulative Cash Flow (LOM, after tax)	US$M	436.0
NPV 5% (after tax)	US$M	305.1
IRR (after tax)	%	57.6
Payback Period	year	1.6
Cash Costs (LOM, including royalties)	US$/oz	776
AISC[1]	US$/oz	877

Note: [1] AISC includes mine cash costs per oz sold, royalties, sustaining capital costs, and operational waste stripping costs.

1.3.3 All-In Sustaining Cost

The LOM plan shows production averaging 110,500 oz of gold per year for the first five years (2022-2026) with an average AISC of $922/oz. Production for the first eight years (2022-2029), when the mine is processing predominantly fresh ore, averages 104,500 oz of gold per year with an average AISC of $858/oz, followed by 1.5 years processing residual stockpile feed for a LOM production average of 95,000 oz of gold per year and a LOM average AISC of $877/oz. The Project's AISC includes capitalized stripping and reclamation costs. Annual gold production and AISC are shown on Figure 1-5.

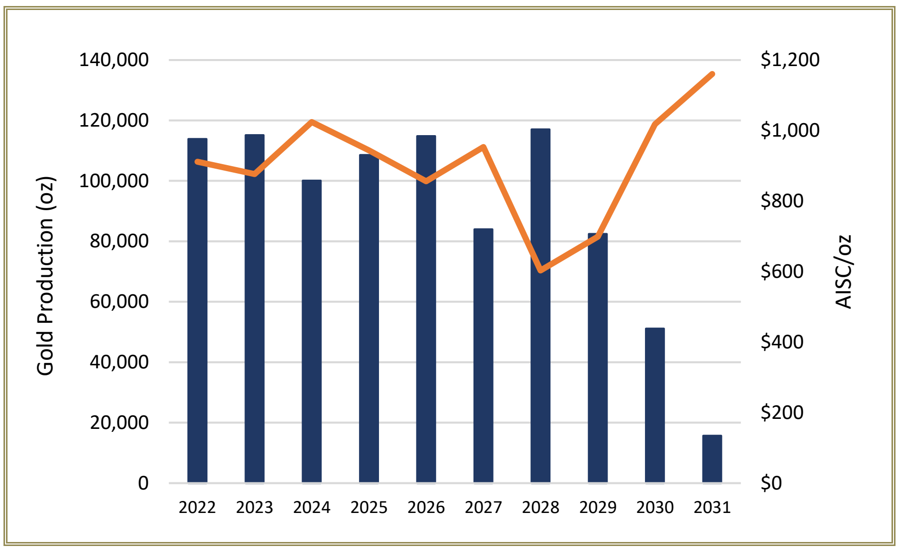


Figure 1-5: Annual Gold Production and All-In Sustaining Cost

1.3.4 Sensitivity Analysis

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities for the following:

- Gold price
- Operating costs
- Foreign exchange
- Capital costs.

The NPV at 5% discount rate sensitivities are shown in Figure 1-6, Figure 1-7, and Table 1-10. The Project NPV is most sensitive to changes in the gold price, followed by changes in the operating costs, foreign exchange, and capital costs.

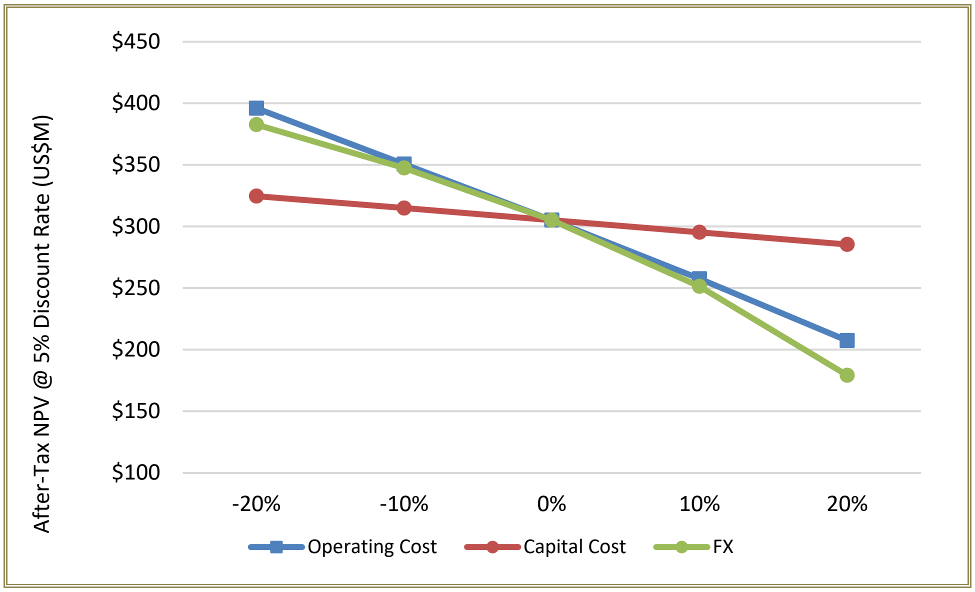


Figure 1-6: Operating Cost, Capital Cost, and Foreign Exchange Sensitivity Analysis on After-Tax NPV

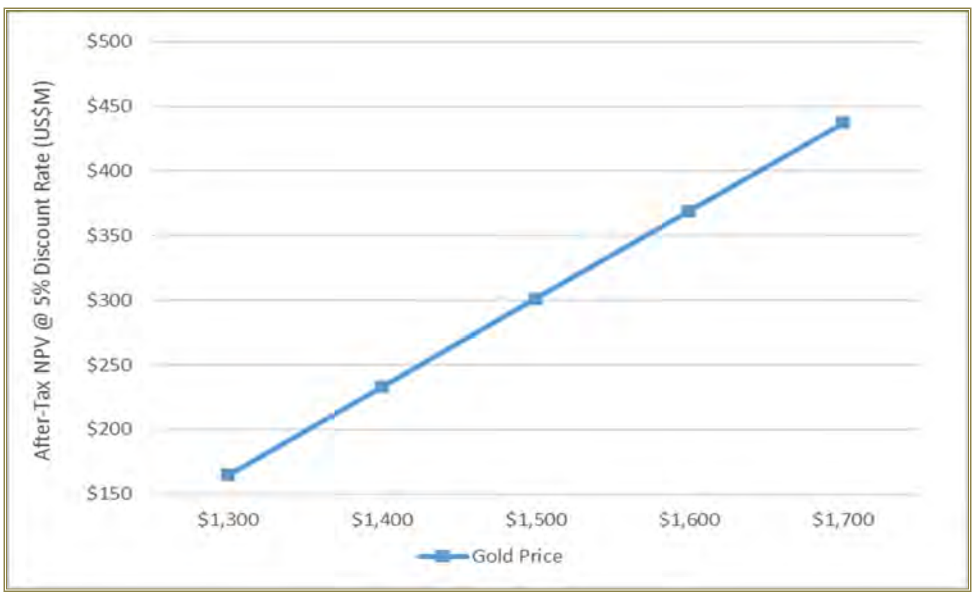


Figure 1-7: Gold Price Sensitivity Analysis on After-Tax NPV

Table 1-10: Santa Luz After Tax NPV and IRR Sensitivity Analyses

Factor	After-Tax NPV at 5% (US$M)	IRR (%)
Gold Price		
US$1,300	186	38
US$1,400	247	48
US$1,500	305	58
US$1,600	362	67
US$1,700	419	76
US$1,800	475	85
Operating Cost		
-20%	396	73
-10%	351	65
0%	305	58
10%	258	49
20%	207	41
Capital Cost		
-20%	325	73
-10%	315	65
0%	305	58
10%	295	52
20%	285	47
Foreign Exchange		
-20%	383	79
-10%	347	69
0%	305	58
10%	251	46
20%	179	32

1.4 C1 Underground Preliminary Economic Assessment

SLR updated a PEA-level study of the potential to exploit the Mineral Resources below the C1 open pit using underground mining methods. The C1 Underground resources are a proximal down-dip extension of the Mineral Resource exploited by the C1 open pit.

The C1 Underground Mineral Resource, as of June 30, 2020, in the PEA are summarized in Table 1-11.

Table 1-11: Santa Luz C1 Underground Mineral Resource—June 30, 2020

Category	Tonnes ('000s)	Grade (g/t Au)	Contained Gold (oz)
Measured	121	1.94	7,561
Indicated	5,913	2.55	484,066
Measured & Indicated	**6,034**	**2.53**	**491,627**
Inferred	6,560	2.19	461,367

Notes: **1.** CIM Definition Standards (2014) were followed for Mineral Resources.
2. Underground Mineral Resources are reported at a cut-off grade of 1.5 g/t Au.
3. Bulk density of 2.70 t/m^3 used.
4. No minimum thickness was used in the resource estimation.
5. Mineral Resources are estimated using a gold price of US$1,500/oz.
6. Totals may not add due to rounding.

Host rocks to the underground resource include carbonaceous metasedimentary rocks, dioritic and dacitic intrusive rocks, and metavolcanic rocks. Most of the underground resource is classified as carbonaceous breccia. The mineralization style is quartz-carbonate-sulphide veins and breccia fillings hosted in a major, district-scale shear zone, typical of orogenic gold deposits.

The shear zone is north to northeast trending and dips at 30° to 40° to the west. The shear zone and mineralization range in thickness from several metres to over twenty metres.

The C1 Underground Mineral Resources considered in this study exist in four separate mining zones (A, B, C, and F). The largest is the B-Zone.

Primary and secondary long hole stoping using paste backfill is considered the most practical and economic method for extracting the C1 Underground Mineral Resources.

The design anticipates a nominal 2,500 t/d underground long hole mining operation using cemented paste backfill to allow for maximum extraction of the deposit. Over the potential 9.5-year LOM, a total of 7.1 Mt of mill feed would be extracted at a grade of 2.65 g/t Au.

The preliminary development access and mining method design for the C1 Underground is based on current practices at Equinox's Fazenda Brasileiro mining operation located 115 km by road southeast of Santa Luz. SLR has utilized the same development heading profiles, stope drilling, blasting patterns and mobile equipment fleet for the C1 Underground as are currently in use at the Fazenda Brasileiro mine. Unit productivities (except for development) and unit costs for all component development and stoping activities (except for backfilling) proposed for the C1 Underground are based on the actual Fazenda Brasileiro mine 2016 and 2017 results.

If Equinox elects to develop the C1 Underground, development of the main decline will take two years. Production would begin to ramp-up in Year 3 and mining would be completed by Year 10. A summary project schedule is shown in Figure 1-8.

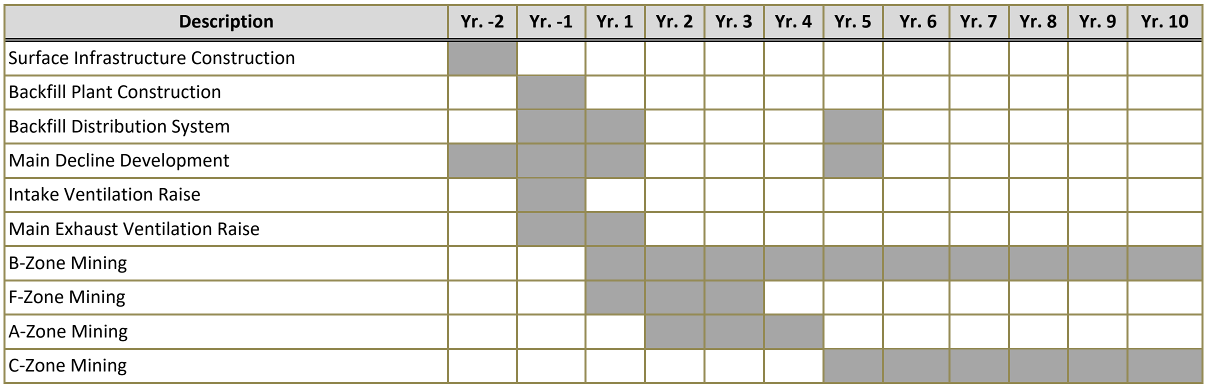


Figure 1-8: C1 Underground Summary LOM Schedule

The mill feed from the C1 Underground would be blended with open pit ore in the proposed 7,400 t/d process plant and no modifications to the process plant are included in this analysis. Over the expected 9.5-year LOM, the C1 Underground is forecast to contribute a total production of 511,000 oz Au.

A large proportion of the tailings generated from the processing of C1 Underground mill feed will be returned underground as paste backfill for the mined-out stopes. Paste fill production is estimated at 5.1 Mt. The remaining tailings (2.0 Mt) will be placed in the existing TSF.

The estimated pre-production capital cost for the C1 Underground is US$74.1 million and the total project capital is US$98.3 million, including sustaining and closure capital. The estimated operating cost is US$50.28/t. The key project parameters, based on a foreign exchange rate of R$5.00:US$1.00, are shown in Table 1-12.

Table 1-12: C1 Underground PEA—Key Project Metrics

Description	Unit	Value
Tonnes Mined and Processed	Mt	7.132
Mine Life (including production ramp-up)	years	9.5
Mill Throughput (full production)	t/d	2,500
Mill Throughput (annual)	Mt/a	0.75
Average Grade Gold	g/t	2.65
Gold Price	US$/oz	1,500
Average Operating Cost	US$/t	50.28
Pre-production Capital Cost	US$ M	74.1
Sustaining Capital Cost	US$ M	23.2
Closure Allowance	US$ M	1.0
Undiscounted Pre-Tax Cash Flow	US$ M	278
Pre-tax NPV@5%	US$ M	189
After-Tax NPV@5%	US$ M	178
After-Tax IRR	%	39

Mineral Reserves have not yet been estimated for the C1 Underground Project; however, the PEA results indicate that it has the potential to improve the overall cash flow profile of the Santa Luz Project. The economic analysis of the C1 Underground is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. Additional drilling and technical studies will be required to convert the C1 Underground Mineral Resources to Mineral Reserves. There is no certainty that the results contemplated in the PEA will be realized.

1.5 Conclusions

Equinox, Ausenco and SLR offer the following conclusions based on the open pit mining feasibility study and the C1 Underground PEA.

1.5.1 Geology and Mineral Resources

- The regional and property geology is well understood by Equinox and the style of mineralization is consistent with an orogenic gold type deposit.
- The majority of the concessions at Santa Luz are at an early exploration stage with limited exploration activity other than regional mapping, regional geochemistry surveys, and airborne surveys which were completed by the previous owners. Many of these concessions remain prospective for gold.
- Drilling and logging methods and the sample preparation, analysis, and security procedures at Santa Luz are adequate for use in the estimation of Mineral Resources.
- Based on the data validation and the results of the standard, blank and duplicate analyses, SLR is of the opinion that the assay and bulk density databases are of sufficient quality for Mineral Resource estimation for the Santa Luz deposits.
- The resource database is sufficiently reliable for grade modelling and Mineral Resource estimation.
- The variograms and correlograms are appropriate for the style and nature of the gold mineralization at Santa Luz and are suitable for grade interpolation in the block models.
- SLR reviewed the Mineral Resources dated June 30, 2020 and is of the opinion that the parameters, assumptions, and methodology used for Mineral Resource estimation are appropriate for the style of mineralization. The classification of Measured, Indicated, and Inferred Resources conform to CIM (2014) definitions. Validation work indicates that the block models are reasonable and acceptable; however, they should be reviewed on an ongoing basis.

1.5.2 Mining and Mineral Reserves

- Conventional open pit mining methods (drilling, blasting, loading, and hauling) will be employed to extract the ore and waste. Due the higher open pit production, off- road trucks with approximately 100-t capacity and excavators of 9 m^3 and 12 m^3 bucket capacity should be employed.

- C1 and Antas 3 pits were previously mined by Yamana in 2013-2014. Both pits have stable highwalls, and there is currently water in the bottom of both pits. The water will be pumped to a WSF prior to the resumption of mining of the pits.
- Mining will start at C1, with production from a single pit, followed by the mining of two open pits at Antas 3. Pit benches will be mined by 10 m benches in two 5 m high flitches for C1 and for Antas 3, with a safety berm every 10 m.
- Mining will be conducted by contractors with oversight by Equinox personnel.
- Equinox prepared the Mineral Reserves dated June 30, 2020.
- The Open Pit Feasibility Study is based on Mineral Reserves with reasonable prospects for economic extraction by open pit mining and whole ore leaching using RIL.
    - Approximately 24.9 Mt at an average grade of 1.34 g/t Au, factored for dilution and extraction, are potentially mineable by open pit methods. This includes approximately 21.6 Mt of Proven Mineral Reserves grading 1.39 g/t Au and 1.17 Mt of Probable Mineral Reserves grading 1.28 g/t Au from the C1 and Antas 3 pits and a low-grade stockpile of Probable Mineral Reserves of approximately 2.19 Mt grading 0.86 g/t Au.
    - The LOM stripping ratio is 4.3:1 waste to ore, including stockpiles (4.7:1 excluding stockpiles), with pit designs including benches and roads.
    - At an average processing production rate of approximately 7,400 t/d, or 2.7 Mt/a, the Project life is approximately 9.5 years (a partial year for ramp-up of production, eight years of mining plus an additional 1.5 years of processing stockpiles).
- Topographical relief, climate, haul distances, and political jurisdiction present no issues to the Project.

1.5.3 Metallurgical Testwork and Mineral Processing

- Production at the Santa Luz mine and mill was discontinued with the facilities put on care and maintenance in September 2014, following a period of low gold recoveries associated with the processing of carbonaceous ores.
- Gold mineralization in the Santa Luz deposits occurs in two main rock types, the carbonaceous schist and dacite, both of which have been variably brecciated along a moderately dipping structural zone. The occurrence of gold and their amenability to processing vary significantly between the two lithologies.
- Since late 2014 there have been a series of metallurgical testing programs to evaluate the existing process facilities, to determine the causes of the low gold recoveries and develop a revised flowsheet to successfully process the carbonaceous ore.
- The 2014 test program included both bench and pilot-scale testing in the following areas:
    - Comminution, including Bond and JK SimMet drop weight testing
    - Gravity separation, both gravity recoverable gold (GRG) and heavy liquid testing
    - Flash flotation

- Conventional flotation
- Solid liquid separation
- Pregnant gold solution robbing factor tests and kerosene addition to blind naturally occurring carbon
- Concentrate CIL leaching
- Whole ore CIL leaching
- Pilot plant testing of flotation and concentrate leaching, and whole ore leaching.

- In late 2015, a new drilling program commenced, enabling new samples to be collected for further metallurgical investigation and flowsheet development. The drilling program also enabled a variability testwork program to be undertaken (at a bench scale) that enhanced the knowledge of the metallurgical parameters within each ore type. The testwork program included:
  - Whole ore CIL leaching
  - Whole ore RIL leaching
  - Resin metals stripping and electrowinning
  - Variability testing of both carbonaceous and dacitic ore types.
- In 2016 the Santa Luz pilot plant was restarted. Results from the pilot plant showed that even with a mop up circuit to capture excess kerosene, whole ore CIL processing was unable to achieve the target gold recovery at realistic gold loadings on the carbon, leaving RIL as the favoured option.
- The results of the 2017 pilot plant test program showed that with adequate kerosene addition, whole ore RIL processing of blended carbonaceous and dacite ores was a viable option that could achieve the target 84% Au recovery.
- Further testwork was conducted in 2019 at Mintek in South Africa and at the Santa Luz on-site pilot plant to optimize the whole ore RIL processing circuit, to increase the gold grade (and reduce the copper grade) of the loaded resin and to optimize gold recovery from the resin.

1.5.4 Process Plant

- The new ore-processing facility will incorporate the crushing, crushed-ore storage, and SAG mill of the original plant. The rest of the plant, except for the refinery, will be new.
- The new plant will consist of the following sequence of processes:
  - Two-stage crushing
  - Two-stage grinding
  - Gravity concentration
  - Pre-aeration and kerosene conditioning
  - RIL leaching
  - Detoxification of leached tailings
  - Containment of tailings in a TSF

- Detergent and acid washing of the loaded resin
- Resin elution with simultaneous electrowinning of gold from the eluate
- Refining to generate copper-gold bars.

- Plant throughput will be 2.7 Mt/a ore (nominal 7,400 t/d ore) grading 1.39 g/t Au (or 1.34 g/t with stockpiles). Projected overall gold recovery is 84%.
- In the final year of operation, high-arsenopyrite, high-sulphide ore from a portion of the Antas 3 deposit will be processed. This ore will be processed by flotation in the existing flotation circuit to recover a gold-arsenic flotation concentrate that will be shipped off-site for custom processing.

1.5.5 Infrastructure

- Existing site services and infrastructure will be reused as part of the new processing facility. This includes water storage and distribution, maintenance workshop, laboratory, site administration buildings, warehousing, messing facilities, etc.
- Practically all the infrastructure requirements for the Project are already in place as a result of the prior operations. The mine is only 35 km from the established town of Santa Luz, and is easily accessible from the port of Salvador, minimizing infrastructure requirements; consequently, the Project is not remote or difficult to access.
- The Project is in a semi-arid area and water must be carefully managed. Raw water is obtained from an adjacent river and will be stored in the leach TSF, which will be expanded and converted into a WSF, an existing small open pit, and in the existing flotation TSF. Process water will be recycled from the TSF to the processing plant during operations.
- Power is provided by an existing 138 kV connection to the national power grid.
- All the service systems and buildings required for the operation are already in place.
- The operation will employ approximately 749 personnel, approximately 60% of whom will be contractors. Most of the workers live near the mine site (within 75 km).

1.5.6 Environmental, Permitting, and Social Considerations

- The environmental impacts of the Project have been assessed and appropriate mitigation measures have been put in place.
- The Project has all relevant permits in place. There are no identified environmental liabilities associated with the property.

1.5.7 Capital and Operating Costs

- The initial mine stripping, plant, and infrastructure capital cost estimate is US$103.1 million including contingencies. Total LOM sustaining capital is estimated at US$21.0 million, and the capitalized stripping is US$60.6 million.
- Reclamation and closure costs have been estimated to be US$8.8 million at an exchange rate of R$5.0:US$1.00.

- Average processing operating costs are estimated at US$13.74/t for the combined dacite-carbonaceous blended ore and US$9.17/t for dacite with high sulphides, for an average LOM processing cost of US$13.43/t.
- The estimated LOM costs for the Project are as follows:
  - Total Capital Cost.................................. US$166.1 million
  - Total Operating Cost .... US$28.02/t of ore processed
  - Off-site Cost (RIL).................................. US$9.0 million
  - Off-site Cost (High-Sulphide Dacite)....... US$7.8 million
- The capital costs are adequate for the size of the Project, considering that much of the plant and infrastructure is already in place from the prior operation, and Equinox plans to use a mining contractor.
- The operating costs are comparable with the costs of similar operations of similar size.
- The off-site costs comprise the freight, smelting, and refining (FSR) costs for dacitic-sulphide ore concentrates that will be processed in the final year of operation.

1.5.8 Economics

Project

- 24.9 Mt at 1.34 g/t Au head grade producing a yearly average of 95,000 oz Au over 9.5-year LOM for a total of 903,000 oz with an AISC of US$877/oz, including capitalized stripping and reclamation cost.
- Positive after-tax net present value (NPV) at a 5% discount rate of US$305.1 million with an internal rate of return (IRR) of 57.6% using a US$1,500/oz Au price. Payback period is 1.6 years from the re-start of the Project.
- Both capital and operating costs are based on a LOM exchange rate of R$5.00:US$1.00. Both capital and operating costs are sensitive to changes in the exchange rate.
- The Project return is most sensitive to changes in the gold price, followed by changes in operating costs, exchange rate, and capital costs.

Risks/Opportunities

- There are no significant risks and uncertainties that could reasonably be expected to affect the reliability or confidence in the Mineral Resource or Mineral Reserve estimates on the projected economic outcomes.

1.5.9 C1 Underground PEA

- PEA-level designs and analyses indicate that a significant portion of the C1 Underground Mineral Resource at Santa Luz has the potential for economic extraction and warrants more detailed and specific analysis.
- PEA results are summarized in Section 24 of this Technical Report.

1.6 Recommendations

Equinox, Ausenco and SLR offer the following recommendations.

1.6.1 Geology and Mineral Resources

- Limit the number of Certified Reference Material (CRM) in any drill program to four: one approximating the cut-off grade, two proximal to the average grade, and one representative of high-grade material at the Project site.
- Field duplicate analyses are consistent with the natural variability often seen in orogenic gold deposits; however, Equinox should continue to monitor results of high-grade duplicate samples and consider submitting larger half core samples in place of quarter core samples.
- Use a minimum thickness of five metres for future resource estimation to remove narrow intersections that may not be economic when "diluted out." The high grades from narrow intersections will influence block grades in the interpolation process.
- Complete an in-fill drilling program to acquire information on TOC, sulphur, and pregnant robbing index (PRI) solution, critical for the understanding of the mineralization and mineral processing of the C1 and Antas 3 deposits. Further gold analyses from this program will provide higher confidence in gold continuity.

1.6.2 Mining

- Continue to examine the optimal waste dump locations.
- Initiate evaluation of the outlying Mineral Resource areas of Mansinha and Mari.
- Complete a hydrogeology study to determine the pit dewatering parameters.
- Continue to study the Project limits based on the proposed waste dump designs and optimizing truck cycle times for the LOM plan.

1.6.3 Metallurgical Testwork and Mineral Processing

- Confirm that carbonaceous/dacite ore blend(s) processed in the Santa Luz pilot plant over test Stages 11-17 can be achieved over the LOM plan (apart from final Antas 3 high sulphide phase).
- Add a gravity treatment stage to the Santa Luz pilot plant to clarify the effect of gravity pre-treatment on whole ore RIL processing and confirm that the 20-hour, five tank RIL circuit offers the most cost-effective option.
- Conduct further pilot plant testwork to firm up the gold recovery mass balance to confirm that mass recovered from resin corresponds with extraction determined from head/tailings assays on the ore.

- Conduct further testwork on gold recovery from resin to:
  - Confirm materials of construction for the elution heat exchanger
  - Clarify the effect of multiple cycle eluate reuse on gold electrowinning performance and the most appropriate treatment of bleed solutions from elution to maximize gold recovery.

1.6.4 Infrastructure

- Establish the overburden dewatering systems needed for the design of surface diversions and drainage systems.

1.6.5 Environmental, Permitting, and Social Considerations

- Maintain renewal of the Operating Licences. As required by Brazilian law, the renewal application must be submitted at least 120 days before the expiration date. The Santa Luz Project has two Operating Licences that will expire on August 22 and 28, 2020 and their renewal applications have been submitted on February, 28 and March 17, 2020, respectively.

1.6.6 C1 Underground PEA

- Complete further drilling along strike and at depth to expand and better define the current Underground Mineral Resources and upgrade the Inferred Mineral Resources to Indicated Mineral Resources.
- Review the current block model to determine and implement necessary modifications which would better define the deposit for underground mining method design purposes.
- Carry out drilling of specifically oriented geotechnical holes to identify the joint sets in the hanging wall rocks. These holes should be inclined at approximately 30° to 40° from vertical and drilled with triple tube core barrels using oriented coring techniques.
- Review the stope designs based upon the updated geotechnical analysis including:
  - An assessment of the potential ore losses (mining extraction) in the shallow dipping zones
  - A detailed evaluation of the dilution from the wall rock and backfill
  - A review of the overall stability of the hanging wall as the extent of stope extraction increases to determine whether pillars are required.
- Consider alternative stoping methods.
- Undertake a follow-up hydrogeological evaluation of the planned mining area.
- Undertake laboratory testing of the milled tailings for use as underground paste fill. If amenable, advance the design of a backfill plant and distribution system.
- Determine which existing surface infrastructure installations (including electrical power supply and distribution) can be utilized or expanded to meet the needs of the underground operation. Advance the design of the surface infrastructure.
- Advance the mine design and scheduling including a review of the development rates.
- Advance the ventilation system design including consideration of the heat loads within the mine.

- Review the capital and operating costs and increase the level of detail in these estimates.
- Undertake further metallurgical testing using drill core from the underground resource to compare with previous work completed for the open pit to confirm that metallurgical performance will be the same.
- Complete the environmental permitting process with the appropriate involvement of stakeholders.

2 INTRODUCTION

Equinox Gold Corp. (Equinox) retained Roscoe Postle Associates Inc. (RPA), now part of SLR Consulting Ltd (SLR), and Ausenco Engineering Canada Inc. (Ausenco) to jointly prepare with the Equinox Technical Services group a Technical Report on the Santa Luz Project (Santa Luz or the Project), in Bahia State, Brazil. This Technical Report provides an update of a previously prepared Technical Report dated November 15, 2018 (RPA, 2018) that disclosed information regarding the Open Pit Feasibility Study and the underground mining scenario at the level of a Preliminary Economic Assessment (PEA) (C1 Underground PEA). This Technical Report also provides an update on the ownership status of the Project, Mineral Resources (reported exclusive of Mineral Reserves), Mineral Reserves and Project economics. The Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects. RPA personnel visited the Project in June 2015 and May 2016.

On March 10, 2020, Equinox acquired Leagold Mining Corporation (Leagold), which owned Santa Luz as well as the Fazenda Brasileiro, Pilar, and Riacho dos Machados gold mines in Brazil, and the Los Filos gold mine in Mexico.

Yamana Gold Inc. (Yamana) operated the Santa Luz open pit mine from mid-2013 to mid-2014, when the mine was placed on care and maintenance following poor metallurgical recovery results from its carbon-in-leach (CIL) plant. Subsequent metallurgical testing programs, including the operation of a pilot-scale plant, has demonstrated that resin-in-leach (RIL) is a better method of gold recovery for the Project.

The Project is in the detailed engineering phase of process plant modifications to restart production, and will also require an increase in the storage capacities of the existing tailings and water storage facilities and a restart of open pit stripping and mining. The modifications and upgrades to the processing plant and tailings and water storage facilities are expected to be finished by the end of 2021. Open pit stripping is expected to recommence in February 2021.

The Project consists of six deposit areas: C1, Antas 2, Antas 3, Mansinha South, Mansinha North, and Mari, which are the only deposits used in the Mineral Resource and Mineral Reserve estimates. The Open Pit Feasibility Study is based on Proven and Probable Mineral Reserves of 24.9 million tonnes (Mt) grading 1.34 g/t gold (Au) (including stockpiles) as estimated at the effective date of June 30, 2020. The current Mineral Reserves are contained in the C1 and Antas 3 deposits, and existing stockpiles. Initial production is proposed to include ore mined from the C1 deposit; the Antas 3 deposit will be mined from 2023 to 2028 and the existing stockpiles will be mined from 2021 to 2030.

Santa Luz will be a conventional off-road truck and shovel open pit mining operation, utilizing a mining contractor for material movement. After the pre-production period, the nominal ore production rate over the following eight years is projected to be 2.77 million tonnes per annum (Mt/a), or 7,595 tonnes per day (t/d) excluding rehandling, plus 1.5 additional years at a lower rate from residual stockpile feed, over the total 9.5 years life-of-mine (LOM). The stripping ratio is 4.3:1 waste to ore including stockpiles (or 4.7:1 excluding stockpiles), and 6.9 Mt of pre-stripping is proposed (excluding the rehandling of old stockpiles), based on the mine schedule.

Processing will include crushing and grinding, gravity concentration, RIL, elution, and electrowinning. The Project has a targeted LOM processing rate of 2.7 Mt/a, or 7,400 t/d and 95,000 ounces of gold per annum (oz Au/a).

Mineral Reserves have not yet been estimated for the potential C1 Underground Project; however, the PEA results indicate that it has the potential to improve the overall cash flow profile of the Project.

2.1 Sources of Information

Table 2-1 provides a list of Qualified Persons (QP), their respective sections of responsibility, and the name of the contributors for this Technical Report. The QPs' certificates are included in Section 29.

Table 2-1: Qualified Persons and their Respective Sections of Responsibility

Section	Title of Section	Qualified Persons
Section 1	Summary	Hugo Ribeiro de Andrade Filho, Equinox Tommaso R. Raponi, Ausenco Stephen La Brooy, Ausenco Mark Mathisen, SLR Robert Michaud, SLR
Section 2	Introduction	Hugo Ribeiro de Andrade Filho, Equinox
Section 3	Reliance on Other Experts	Hugo Ribeiro de Andrade Filho, Equinox
Section 4	Property Description and Location	Mark Mathisen, SLR
Section 5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Mark Mathisen, SLR
Section 6	History	Mark Mathisen, SLR
Section 7	Geological Setting and Mineralization	Mark Mathisen, SLR
Section 8	Deposit Types	Mark Mathisen, SLR
Section 9	Exploration	Mark Mathisen, SLR
Section 10	Drilling	Mark Mathisen, SLR
Section 11	Sample Preparation, Analysis, and Security	Mark Mathisen, SLR
Section 12	Data Verification	Mark Mathisen, SLR
Section 13	Mineral Processing and Metallurgical Testwork	Stephen La Brooy, Ausenco
Section 14	Mineral Resource Estimates	Mark Mathisen, SLR
Section 15	Mineral Reserve Estimates	Hugo Ribeiro de Andrade Filho, Equinox
Section 16	Mining Methods	Hugo Ribeiro de Andrade Filho, Equinox
Section 17	Recovery Methods	Tommaso R. Raponi, Ausenco
Section 18	Project Infrastructure	Tommaso R. Raponi, Ausenco
Section 19	Market Studies and Contracts	Hugo Ribeiro de Andrade Filho, Equinox
Section 20	Environmental Studies, Permitting, and Social or Community Impact	Hugo Ribeiro de Andrade Filho, Equinox
Section 21	Capital and Operating Costs	Tommaso R. Raponi, Ausenco Hugo Ribeiro de Andrade Filho, Equinox
Section 22	Economic Analysis	Hugo Ribeiro de Andrade Filho, Equinox
Section 23	Adjacent Properties	Hugo Ribeiro de Andrade Filho, Equinox
Section 24	Other Relevant Data and Information	Robert Michaud, SLR
Section 25	Interpretation and Conclusions	Hugo Ribeiro de Andrade Filho, Equinox Tommaso R. Raponi, Ausenco Stephen La Brooy, Ausenco

Section	Title of Section	Qualified Persons
		Mark Mathisen, SLR Robert Michaud, SLR
Section 26	Recommendations	Hugo Ribeiro de Andrade Filho, Equinox Tommaso R. Raponi, Ausenco Stephen La Brooy, Ausenco Mark Mathisen, SLR Robert Michaud, SLR
Section 27	References	Hugo Ribeiro de Andrade Filho, Equinox Tommaso R. Raponi, Ausenco Stephen La Brooy, Ausenco Mark Mathisen, SLR Robert Michaud, SLR

2.2 Qualified Persons Site Visits

The following QPs visited the site in relation with this work:

- Hugo Ribeiro de Andrade Filho, MAusIMM(CP), Equinox Mine Engineer, visited the site from June 22 to 26, 2020
- Mark Mathisen, C.P.G., as RPA's Principal Geologist, visited the site from May 2 to 7, 2016
- Stephen La Brooy, FAusIMM, as Ausenco's Metallurgist, visited the site from February 19 to 20, 2016.

2.3 Units of Measure and Currency

Units of measure used in this report conform to the International System of Units (SI) (metric system). All currency in this report is expressed as United States dollars (US$) unless otherwise noted.

a	annum	L/s	litres per second
A	ampere	m	metre
°C	degree Celsius	Ma	million annum
		m^2	square metre
cm	centimetre	m^3	cubic metre
cm^2	square centimetre	µm	micron
d	day	mASL	metres above sea level
dia	diameter	µg	microgram
dmt	dry metric tonne	m^3/h	cubic metres per hour
g	gram	Mt/a	million tonnes per annum
G	giga (billion)	min	minute
g/L	gram per litre	mm	millimetre
g/t	gram per tonne	MVA	megavolt-amperes
g/m^3	grain per cubic metre	MW	Megawatt

ha	hectare	MWh	megawatt-hour
h	hour	oz	Troy ounce (31.1035 g)
Hz	hertz	ppb	part per billion
J	joule	ppm	part per million
k	kilo (thousand)	R$	Brazilian Real
kg	kilogram	s	Second
km	kilometre	t	metric tonne
km^2	square kilometre	t/a	metric tonne per year
km/h	kilometres per hour	t/d	metric tonne per day
kPa	kilopascal	US$	United States dollar
kVA	kilovolt-amperes	V	volt
kW	kilowatt	W	watt
kWh	kilowatt-hour	wmt	wet metric tonne
L	litre	wt%	weight percent

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by Equinox's qualified persons and is supported by SLR and Ausenco. The information, conclusions, opinions, and estimates that SLR and Ausenco have provided and are contained herein are based on:

- Information available to SLR and Ausenco at the time of report preparation
- Assumptions, conditions, and qualifications as set forth in this report.

For the information contained in this report, SLR and Ausenco have relied on ownership information provided by Equinox. SLR and Ausenco have not researched property title or mineral rights for Santa Luz, and express no opinion as to the ownership status of the property.

SLR and Ausenco have relied on Equinox for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Project.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Santa Luz Project is located within the Maria Preta mining district, 35 km north of the town of Santa Luz, in Bahia State (Figure 4-1), Brazil. It is approximately 240 km northwest of the state capital, Salvador, 115 km by road from the Fazenda Brasileiro mine, and 163 km from the Jacobina Mine. The centre of the property has latitude and longitude coordinates of approximately 11°00'28" S and 39°18'28" W.

4.2 Mineral and Surface Rights in Brazil

The exploration and exploitation of mineral deposits in Brazil are defined and regulated by the 1967 Mining Code and overseen by the National Mining Agency (Agência Nacional de Mineração or ANM), which is the former National Department of Mineral Production (Departamento Nacional de Produção Mineral or DNPM). There are two main legal regimes under the Mining Code regulating exploration and mining in Brazil: the Exploration Permit (Autorização de Pesquisa), and the Mining Concession (Concessão de Lavra).

Applications for an Exploration Permit (EP) are made to ANM and are available to any company incorporated under Brazilian law and maintaining a main office and administration in Brazil. EPs are granted following submission of required documentation by a legally qualified geologist or mining engineer and include an exploration plan and evidence of funds or financing for the investment forecast in the exploration plan. An annual fee per hectare ranging from US$0.35 to US$0.70 is paid by the holder of the EP to the ANM, and reports of exploration work performed must be submitted. During the period when a formal EP application has been submitted by a company, but not yet granted, exploration works are permitted except for drilling. In this document, areas covered by the pending EP applications are referred to as Exploration Claims.

EPs are valid for a maximum of three years, with a maximum extension equal to the initial period, issued at the discretion of the ANM. The annual fee per hectare increases by 50% during the extension period. After submitting a positive Final Exploration Report, the EP holder can request a mining concession. Mining concessions are granted by the Brazilian Ministry of Mines and Energy, are renewable annually, and have no set expiry date. The concessions remain in good standing subject to submission of annual production reports and payments of royalties to the federal government.

When the maximum extension of an EP for an area has been reached, if a positive Final Exploration Report and mining concession request have not been submitted, then the Exploration Claim expires and the area is once again considered to be *Available*. Following expiry, the ANM will accept new EP applications from the public, including from the first owner, for a period of 60 days. If any valid EP applications are submitted during this period in addition to the first owner's application, ANM will review, with consideration of the work completed, and decide to whom they will issue the permit. Before a decision is reached for a competitive area, the claim status is considered *In Dispute*.

Surface rights can be applied-for if the land is not owned by a third party. The owner of an EP is guaranteed, by law, access to perform exploration fieldwork, provided adequate compensation is paid to third-party landowners and the explorer accepts all environmental liabilities resulting from the exploration work.

Figure 4-1: Santa Luz Project Location

4.3 Land Tenure

The Santa Luz properties (Figure 4-2) cover an area totalling 48,599.25 ha, including 36 EPs (42,666.41 ha), six mining concessions (2,611.69 ha), and four mining concessions in application (3,321.15 ha).

The current status of the EPs is indicated below:

- Eight are at exploration stage, with Partial Exploration Report submitted to ANM requesting the deadline extension of its permission (9,849.47 ha).
- Two are at final exploration stage, with the Positive Final Exploration Report already submitted to ANM (1,885.88 ha), indicating reasonable prospects to continue with economical analyses and subsequent mining concession application after ANM's approval of reports.
- Five are at final exploration stage, with the Negative Final Exploration Report already submitted to ANM (6,711.28 ha), which means that these areas should be considered Available after ANM's approval of reports.
- Twenty-one are at an exploration stage (24,219.78 ha).

The EPs are listed in Table 4-1, and the mining concessions and mining concessions in application are listed in Table 4-2.

One of the exploration permits expired during 2020, and is either in the process of submission of reports or will lapse. This exploration permit does not impact the Mineral Resources or Mineral Reserves, or future operations.

The Santa Luz claims cover several farms. Agreements were signed between Yamana and the landowners to allow mining and exploration activities, and these agreements have been transferred to Equinox.

Equinox has verified that there are no environmental liabilities on the property. Equinox has all required permits to conduct work on the properties. These permits and their status are listed and described in Section 20. Equinox has verified that there are no other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the property.

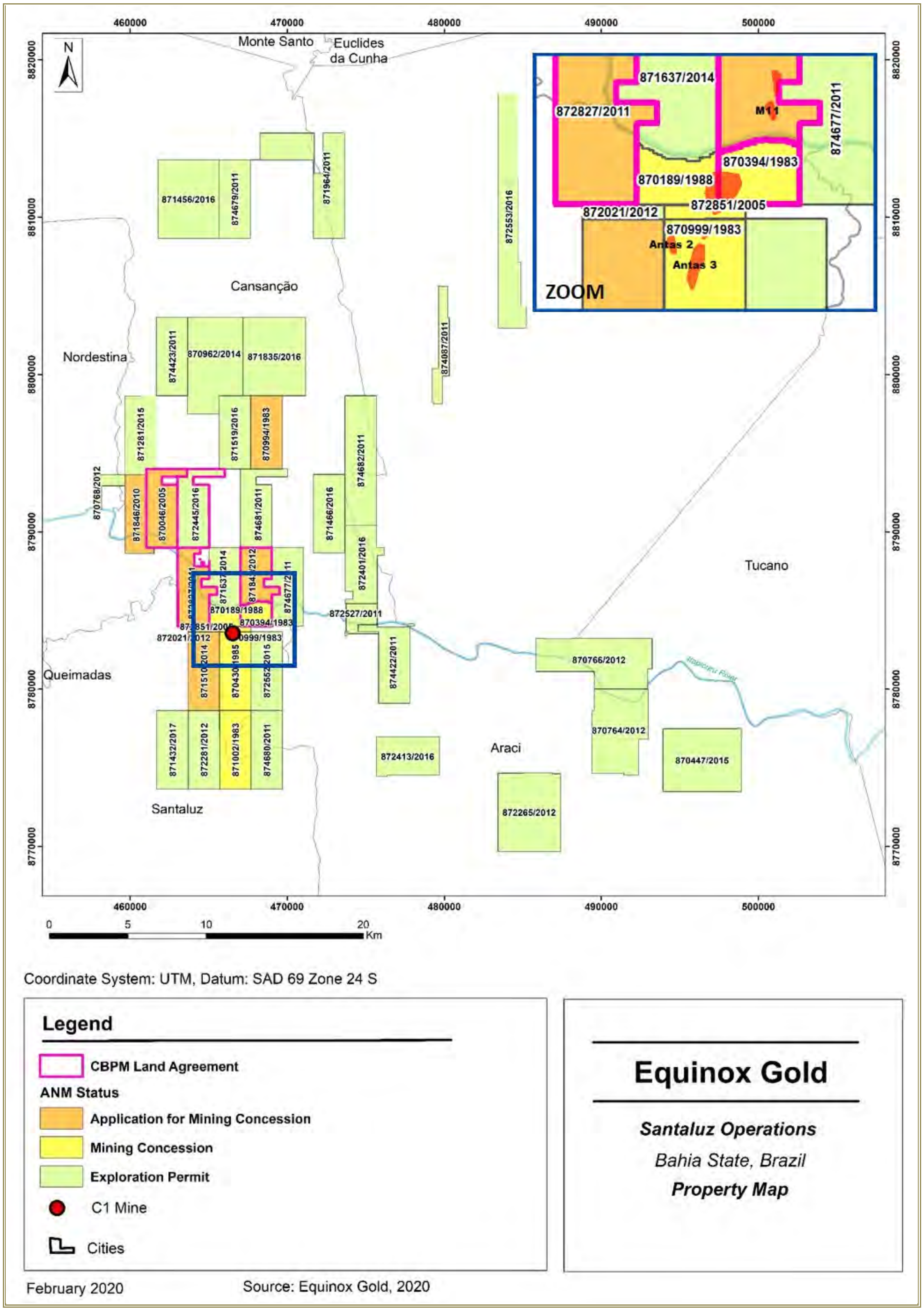


Figure 4-2: Current Mining and Exploration Areas

Table 4-1: Exploration Permit and Claim List

ANM Number	Owner	Expiry Date	Area (ha)
870.764/12	SLDM	03-Oct-21	1,790.75
870.766/12	SLDM	03-Oct-21	1,929.53
870.768/12	SLDM	03-Oct-21	218.00
872.265/12	SLDM	03-Oct-21	1,992.79
872.281/12	SLDM	22-Aug-21	998.52
872.527/11	SLDM	22-Aug-21	323.00
874.087/11	SLDM	22-Aug-21	502.32
874.422/11	SLDM	22-Aug-21	1,042.20
872.021/12	SLDM	22-Aug-21	68.15
874.423/11	SLDM	03-Oct-21	998.88
874.677/11	SLDM	22-Aug-21	999.92
874.679/11	SLDM	22-Aug-21	991.71
874.680/11	SLDM	22-Aug-21	999.03
874.681/11	SLDM	22-Aug-21	1,000.06
874.682/11	SLDM	22-Aug-21	1,657.15
871.637/14	SLDM	22-Aug-21	758.70
870.447/15	SLDM	14-nov-21	1,999.87
870.962/14	SLDM	7-jun-22	1,988.36
871.835/16	SLDM	15-May-23	1,992.86
872.445/16	SLDM	05-May-23	970.66
871.432/17	SLDM	21-Dec-20	997.32
Total			**24,219.78**
Partial Report Submission			
871.281/15	SLDM	No expiration date	977.50
872.552/15	SLDM	No expiration date	999.79
871.456/16	SLDM	No expiration date	1,949.64
871.466/16	SLDM	No expiration date	999.88
871.519/16	SLDM	No expiration date	931.82
872.401/16	SLDM	No expiration date	1,014.32
872.413/16	SLDM	No expiration date	977.24
872.553/16	SLDM	No expiration date	1,999.28
Total			**9,849.47**
Negative Final Report Submission			
871.964/11	SLDM	13-jul-20	1,779.48
870.767/12	SLDM	14-Sep-18	1,000.00
872.148/13	SLDM	08-Sep-18	1,749.80
872.557/15	SLDM	14-Apr-19	182.00
871.489/16	SLDM	17-Oct-19	2,000.00
Total			**6,711.28**
Positive Final Report Submission			
870.046/05	CBPM	No expiration date	979.42
872.827/11	CBPM	No expiration date	906.46
Total			**1,885.88**

Notes: **1.** CBPM = Companhia Baiana de Pesquisa Mineral (mineral rights held under agreement with SLDM).
2. SLDM = Santa Luz Desenvolvimento Mineral Ltda. Is a 100% Equinox Gold owned Brazilian company.
3. Table 4-1 is current as of the effective date of the report. As part of the ANM process, claims are reapplied on expiration.

Table 4-2: Mining Concession List

ANM Number	Owner	Final Report Submission Date	Date of Award	Status	Area (ha)
870.189/88	SLDM	12-May-08	27-Dec-13	Mining Concession	241.15
870.394/83	CBPM	07-Jul-88	06-Jun-14	Mining Concession	298.06
870.430/85	SLDM	12-Aug-88	13-Feb-92	Mining Concession	1,000.00
871.002/83	SLDM	22-Nov-88	29-Aug-95	Mining Concession	1,000.00
872.851/05	SLDM	12-May-08	03-Jul-14	Mining Concession	4.28
870.999/83	SLDM	12-Aug-88	9-Apr-18	Mining Concession	68.20
870.994/83	SLDM	09-Aug-88	-	Application Submitted	931.80
871.510/14	SLDM	-	-	Application Submitted	999.99
871.842/12	UML	-	-	Application Submitted	701.69
871.846/10	SLDM	-	-	Application Submitted	687.67
Total					**5,932.84**

Notes: **1.** CBPM = Companhia Baiana de Pesquisa Mineral (mineral rights held under agreement with SLDM).
2. SLDM = Santa Luz Desenvolvimento Mineral Ltda. is a 100% Equinox Gold owned Brazilian company.
3. UML = Utinga Mineração Ltda is a subsidiary company of CBPM (mineral rights held under agreement with SLDM).

4.4 Royalties

The Brazilian government collects a 1% gross revenue royalty on all gold operations in Brazil. In addition, a 1% gross revenue royalty is payable to Companhia Sisal do Brasil (COSIBRA), a large surface-rights owner, over the C1, Antas 2, and Antas 3 areas.

A 2% royalty is payable to the previous owner, Companhia Baiana de Pesquisa Mineral (CBPM), on any gold extracted within Mining Permit 870.394/1983, covering the east portion of the C1 deposit. The CBPM mineral rights are held under agreement.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located within the Maria Preta mining district, 35 km north of the town of Santa Luz (population 36,000), and approximately 240 km northwest of the state capital, Salvador (population 2.675 million). Access from Salvador is by way of highway BR-324 to Feira de Santana, BR-116 to Serrinha, BA-409 to Conceição do Coité, and finally BA-120 to Santa Luz. From Santa Luz, the property is accessed by way of a municipal dirt road.

A railway (former Viação Férrea Federal Leste Brasileiro) operated by VLI Transportadora (VLI), links Salvador and the sister cities Juazeiro and Petrolina, and has a station in Santa Luz.

A few gravel runways in the region can handle small aircraft, the closest being at the cities of Valente and Serrinha, approximately 20 km and 90 km from Santa Luz, respectively. Since early 2015, the Feira de Santana airport, which is 153 km away from Santa Luz, started operation of daily flights from Campinas City, São Paulo state.

5.2 Climate and Physiography

The climate in the area is semi-arid and hot. There are two rainy seasons, from September to December and from April to June. Outside the rainy seasons, the weather is dry and hot. Temperatures range from 16°C in July and August to 40°C in January and February, and averages 24°C. The predominant wind direction in the region is from the southeast.

The average annual air humidity is 71.6 g/m^3. The potential evaporation of the region is approximately 2,000 mm/a, representing a rain/evaporation deficit for the Project area. The Project would be able to operate year-round.

5.3 Topography and Vegetation

The elevation of the Project area ranges from 250 mASL to 300 mASL. The regional drainage system is characterized by small ephemeral streams within the Itapicurú River hydrographic basin.

Ground cover at the Project area consists of small- to medium-sized desert vegetation. It is mostly composed of bromeliads, cacti, and legumes, which are highly resistant to long dry periods. Sisal, a fibrous plant used in making rope and twine, is farmed in the area.

5.4 Existing Infrastructure

5.4.1 Infrastructure

The current mine includes open pit workings and gold ore-processing facilities, as well as other necessary buildings and infrastructure. This infrastructure, which is on a care and maintenance basis, includes, but is not limited to:

- Mine workings and equipment
- A 7,400 t/d processing plant
- Power supplied from a 138 kV power line extending from the Coelba power station to the main substation at the Santa Luz plant site
- A WSF, formerly the leach flotation tailings facility, with a remaining storage capacity of 0.3 Mm^3 and a planned total storage capacity of 3.0 Mm^3
- A TSF, formerly the flotation tailings facility, with a remaining storage capacity of 3.0 Mm^3 (4.2 Mt) and a planned total storage capacity of up to 25.2 Mm^3 (or 35.3 Mt), for tailings from the new RIL plant
- Other buildings and supporting facilities including workshops, a storeroom, a fuelling station, offices, dry facilities, a cafeteria, a medical clinic, and a laboratory.

5.4.2 Water, Power, and Local Resources

Water for use on the Project site is currently sourced from the Itapicurú River, the main drainage system in the area, at a rate of 1.17 Mm^3/a. The water supply infrastructure includes an intake structure and pumps at the river. Currently there are 3.9 Mm^3 of water stored in the C1 pit that will be transferred to the WSF (formerly the leach TSF), the Antas 3 pit, and the former flotation TSF. Most of the current water storage came from water abstraction in the Itapicurú River during 2018 and 2019.

Santa Luz facilities power consumption is approximately 100500 MWh/a, from a total installed load of approximately 12 MW. Power is supplied from a 138 kV power line that extends from the Coelba power station to the main substation at the plant site.

The mine area has had a short history of mining activity. Mining suppliers and contractors are available within the regional area, and support other active mines within Bahia State, such as the Fazenda Brasileiro and Jacobina mines. Both experienced and general labour are readily available from the nearby town of Santa Luz, with an estimated population of approximately 21,300 inhabitants, and Valente, with 15,800 inhabitants, according to the July 1, 2020 Bahia State census.

6 HISTORY

During the 1970s, Companhia Vale do Rio Doce (CVRD), at the time a state-owned mining company (now known as Vale), invested in a regional prospecting program targeting base metals in Bahia State. Other private and state companies, including Ferbasa, Caraíba Metais, and Anglo American, carried out intensive prospecting, geological mapping, and research programs. During this time, the Rio Itapicurú Greenstone Belt (RIGB) was identified through field mapping and geochemical surveys conducted by CVRD.

Between 1979 and 1981, CBPM, a state-owned mineral exploration company mandated with identifying mineral investment opportunities in Bahia, conducted several geological and prospecting programs within the RIGB. These activities identified several gold-bearing trends and prospects, including the C1 and Mansinha North deposits within the Project area. These deposits were subsequently mined between 1987 and 1995 by CBPM's subsidiary Rio Salitre Mineração Ltda.

In the 1980s, CVRD focused exploration in gold-bearing mineralized prospects of the RIGB, resulting in several gold discoveries, three of which were developed to the mining stage. The Fazenda Maria Preta mine at Santa Luz (including the Antas 1, Antas 2, and Antas 3 ore bodies) and the nearby Fazenda Brasileiro mine were developed by CVRD. Mining was conducted simultaneously for some time at the C1 deposit, by CBPM, and at the adjacent A1 deposit, by CVRD. All mining was open pit. The CVRD Santa Luz mines were closed in 1994 due to depleting reserves, low gold prices, and reduced gold recoveries. Equinox now owns and operates the third CVRD operation, Fazenda Brasileiro mine.

6.1 Ownership History

In January 2005, Yamana completed an agreement with CBPM to acquire 7,000 ha of land over the C1 historical mine (formally called Maria Preta). Under this agreement, CBPM retains a 2% royalty interest in these concessions.

In May 2007, Yamana expanded its landownership through the acquisition of mining concessions from Mineração Santa Elina (MSE), formerly owned by CVRD, which included the Antas 1 (now considered part of C1), Antas 2, and Antas 3 deposits, and associated historical mine workings. The 2007 agreement also retained a royalty interest that was transferred from MSE to Callix Finance Inc. in April 2014, and was finally extinguished through an agreement between Yamana and Callix Finance Inc. in March 2015.

In December 2014, it was announced that Yamana had formed a new subsidiary, Brio Gold Inc. (Brio), to hold the Fazenda Brasileiro, Pilar, and Santa Luz properties, as well as some related exploration concessions, all of which were held as non-core assets within Yamana. In December 2016, Brio became an independent, publicly traded company. Leagold acquired Brio on May 24, 2018, and became the owner of the Santa Luz Project. On March 10, 2020, Equinox acquired Leagold, and assumed ownership of the Santa Luz Project.

6.2 Historical Production

Estimates of historical production from the Project area include:

- CBPM reported historical production of 1,772 kg, or 56,971 oz, from the C1 orebody and nearby deposits.
- CVRD mined Antas 1 as a 450 m long open pit, exploiting a mineralized zone reported to have an 8 m average width. Mine production was 400,000 tonnes with an average grade of 4.5 g/t. Antas 3 was mined as an open pit to 35 m over 400 m of strike. A carbon-in-pulp (CIP) plant treated 221,149 tonnes of oxidized ore at an average grade of 2.88 g/t Au, and 86,366 tonnes were heap leached at an average grade of 1.64 g/t. Antas 2 was mined over 250 m of length, at an average width of 10 m, and to a depth of 60 m. A total of 270,000 tonnes of ore were produced with an average grade of 2.5 g/t Au.

Since 1995, artisanal miners have extracted gold from the area on a relatively small scale, using a variety of methods, including panning, small underground workings, and some small-scale processing plants. These operations are peripheral to the main deposit areas. While gold production data from these artisanal mines are not available, this mining is believed to be significant for the local economy.

Yamana operated the Santa Luz open pit mine from mid-2013 to mid-2014, when it was placed on care and maintenance following poor metallurgical recovery results. Gold production from Santa Luz in 2013 and 2014 is listed in Table 6-1.

Table 6-1: Santa Luz Historical Production (2013–2014)

Year	Tonnes Processed	Gold Grade (g/t Au)	Gold Produced (oz)	Recovery (%)
2013 (June to December)	844,000	1.55	13,000	30.5
2014 (January to August)	1,084,000	1.69	20,400	33.6
Total	**1,928,000**	**1.63**	**33,400**	**32.3**

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The RIGB comprises the northeastern portion of the São Francisco Craton, which underlies most of Bahia State in addition to the surrounding states of Minas Gerais, Sergipe, and Goias, Brazil. The São Francisco Craton formed from the collision of several smaller Archean cratons during the Paleoproterozoic Trans-Amazon Orogeny (approximately 2 Ga).

The RIGB is the largest greenstone belt in the São Francisco Craton. It extends for approximately 100 km in a north–south trend, and ranges in width from 30 km to 50 km. The RIGB's dimensions were dilated by large granitoid batholitic intrusions and granite-gneiss domes. During the Trans-Amazon Orogeny, the belt was folded, and experienced upper greenschist facies metamorphism, and, locally, amphibolite facies metamorphism.

The RIGB (Figure 7-1) has been subdivided by Davison et al. (1988) into three different lithological domains:

- A mafic volcanic domain composed mainly of pillowed, massive and/or sheared tholeiitic basalts and andesitic basalts
- A felsic volcanic domain composed of calc-alkaline andesites, rhyodacites, and pyroclastic sequences (including cinder cone tuffs, andesitic tuffs, lapilli tuffs, crystal tuffs, agglomerates, and pyroclastic breccias), diorite intrusions, subordinate dacitic volcanic units, and epizonal intrusions
- A sedimentary domain represented mostly by chemical sediments (cherts and limestones) and turbiditic sequences (metapelites).

The rocks of the RIGB are thought to have formed in a back-arc tectonic setting, as shown in Figure 7-2 (Melo, 1995; Padilha & Melo, 1991; Silva et al., 2001). Silva et al. (2001) are somewhat uncertain of the arc's location; however, it may be to the west of the belt. Several gold-mineralized areas occur in the RIGB, most notably the Santa Luz group of deposits and Fazenda Brasileiro mine (Figure 7-1).

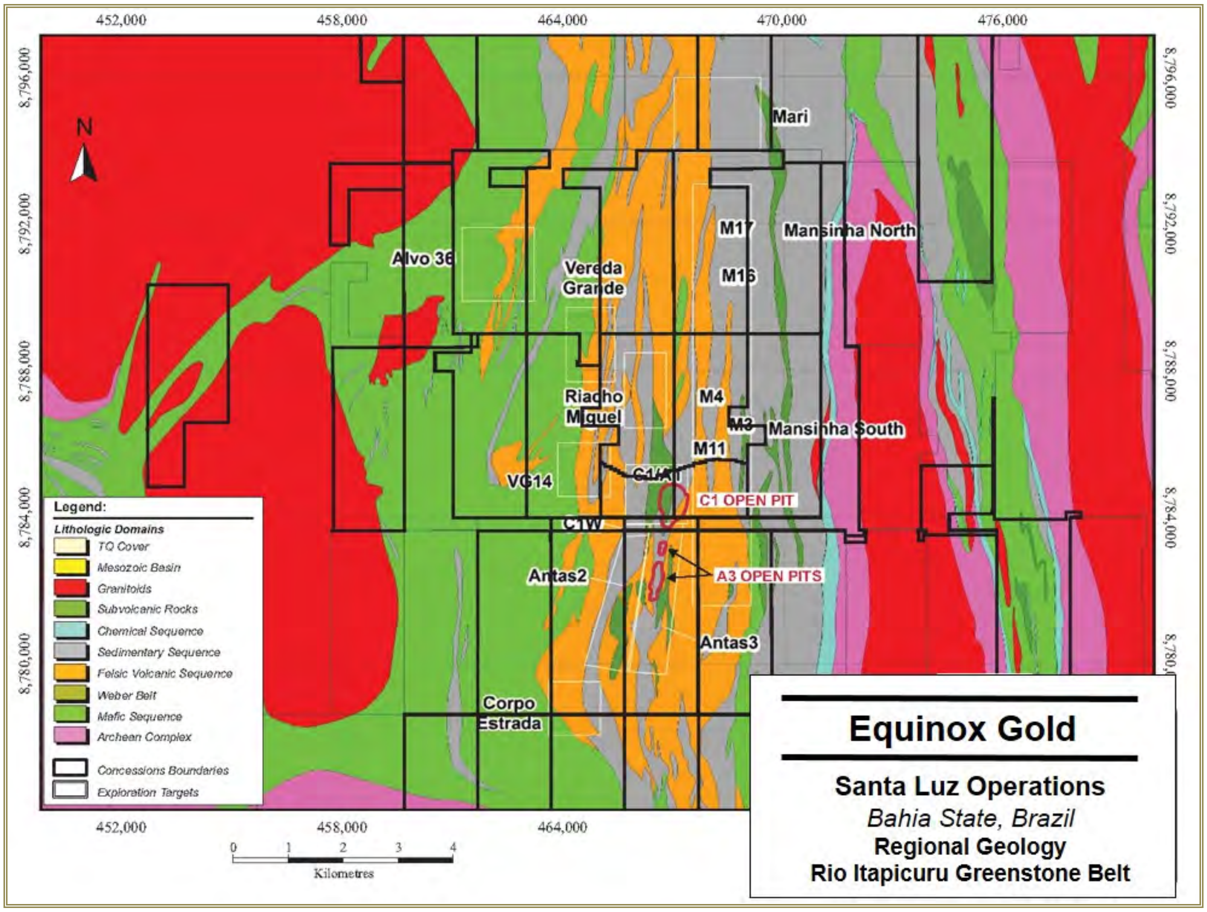
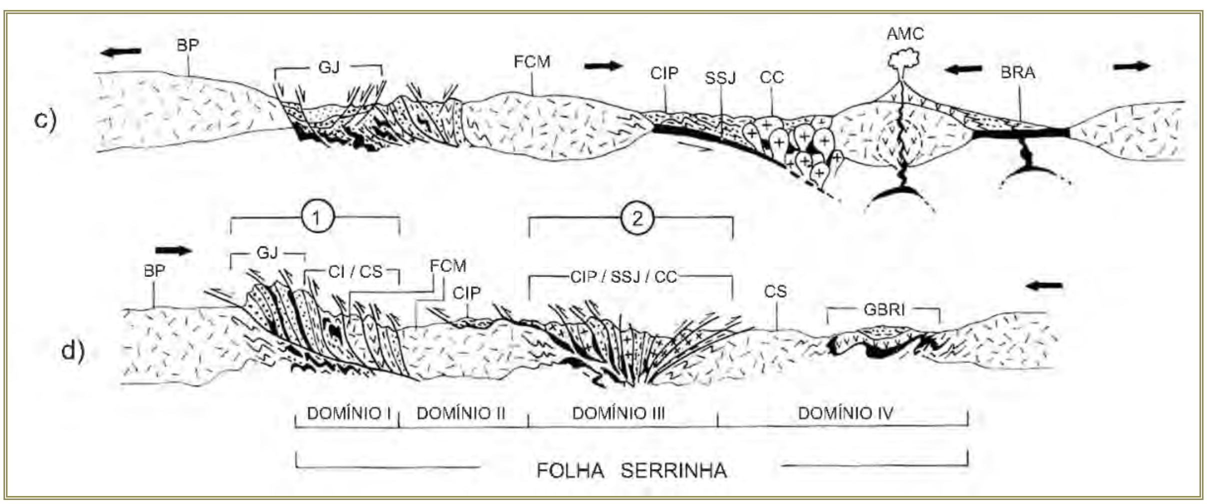


Figure 7-1: Regional Geology—Rio Itapicurú Greenstone Belt



Notes: BP = Piritiba Block; C1/CS = Saúde and Itapicurú Complexes; CMS = Mairi-Seminha Craton; FCM = Cratonic Fragments of Mairi Craton; CIP = Ipira Complez; SSJ = Sao Jose do Jacuipe Suite; CC = Caraiba Complex; ACM = Continental Margin Arc; BRA = Back-Arc Basin; GJ = Jacobina Group; BGRI = Rio Itapicurú "Greenstone Belt"

Figure 7-2: Geologic Evolution Model Proposed for the Rio Itapicurú Greenstone Belt

Isotopic dating for the RIGB defines the following geologic history:

- Mafic volcanism at 2,209 Ma (Silva et al., 2001)
- Felsic volcanism at 2,170 Ma (Silva et al., 2001)
- Emplacement of arc-related granodioritic intrusions—the Treado intrusion—at 2,155 Ma (Rios et al., 2003), and of tonalitic to granodioritic intrusions—the Barrocas and Teofilandia plutons—at 2,130 Ma (Chauvet et al., 1997; Melo et al., 2000)
- Emplacement of syntectonic intrusions—the Itareru tonalitic body—accompanying the main metamorphic event (Carvalho and Oliveira, 2002), and the formation of synchronous domes—the Ambrosio Dome—at 2,080 Ma (Melo et al., 1999)
- Emplacement of post-tectonic granitic intrusions at 2,072 Ma—the Morro do Lopes intrusion (Rios et al., 2000).

7.1.1 Structure

The structural evolution of the RIGB includes three main deformational phases, all of which are associated with transpressive tectonics caused by an oblique continental collision. The first deformational phase is characterized by east-northeast to east vergent compressional structures represented by intense folding and thrusting, with reverse faults on the axial planes of the folds. The folds are gentle to tight asymmetric folds with sub-horizontal, north–south striking axes. Fold axes of this phase are refolded and transposed by later deformational phases. The second deformational phase is dominated by strike–slip tectonics along north to north-northeast striking regional shear zones, which developed as a continuation of the former deformational phase. The shear zones are well developed, reaching several metres thick and several kilometres long, and the sense of movement is dominantly sinistral. Associated with this phase are gentle asymmetric folds with steeply dipping axes. Kink folds are also present and are probably related to late stages of brittle deformation. The third and latest deformational phase consists of northeast-striking, dominantly sinistral faults that crosscut all other structural features in the belt. These are associated with great volumes of quartz veining and brecciation. The structural evolution of these northeast-striking faults is not certain; however, it is possible they are closely related to the later stages of strike–slip shearing.

7.2 Local Geology

Gold deposits and prospects in the Project area occur in silicified breccia zones at or proximal to the faulted contact of the volcanic and sedimentary domains of the RIGB. Significant gold targets and deposits in the Santa Luz trend include the C1 (formally called Maria Preta and including Antas 1), Antas 2, Antas 3, the Mansinha Trend (South, including M11 and M3-M4, and North, including M16 and M17), and the Mari Deposit. Maps of the local geology and mineralized areas for C1 and Antas 3 are shown in Figure 7-3 and Figure 7-4, respectively. Representative cross sections of the C1 and Antas 3 resource areas are shown in Figure 7-5 and Figure 7-6, respectively.

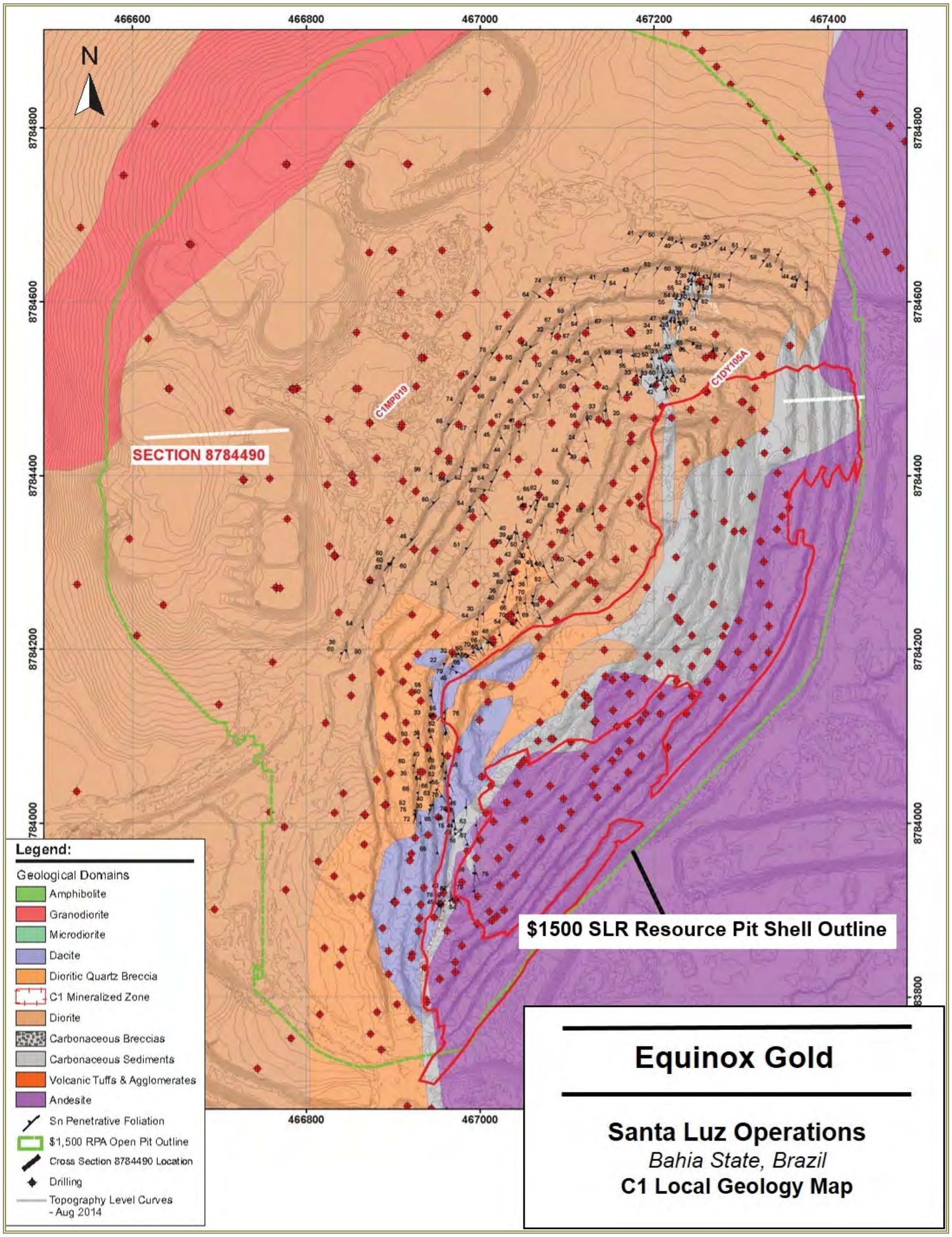


Figure 7-3: Santa Luz C1 Local Geology

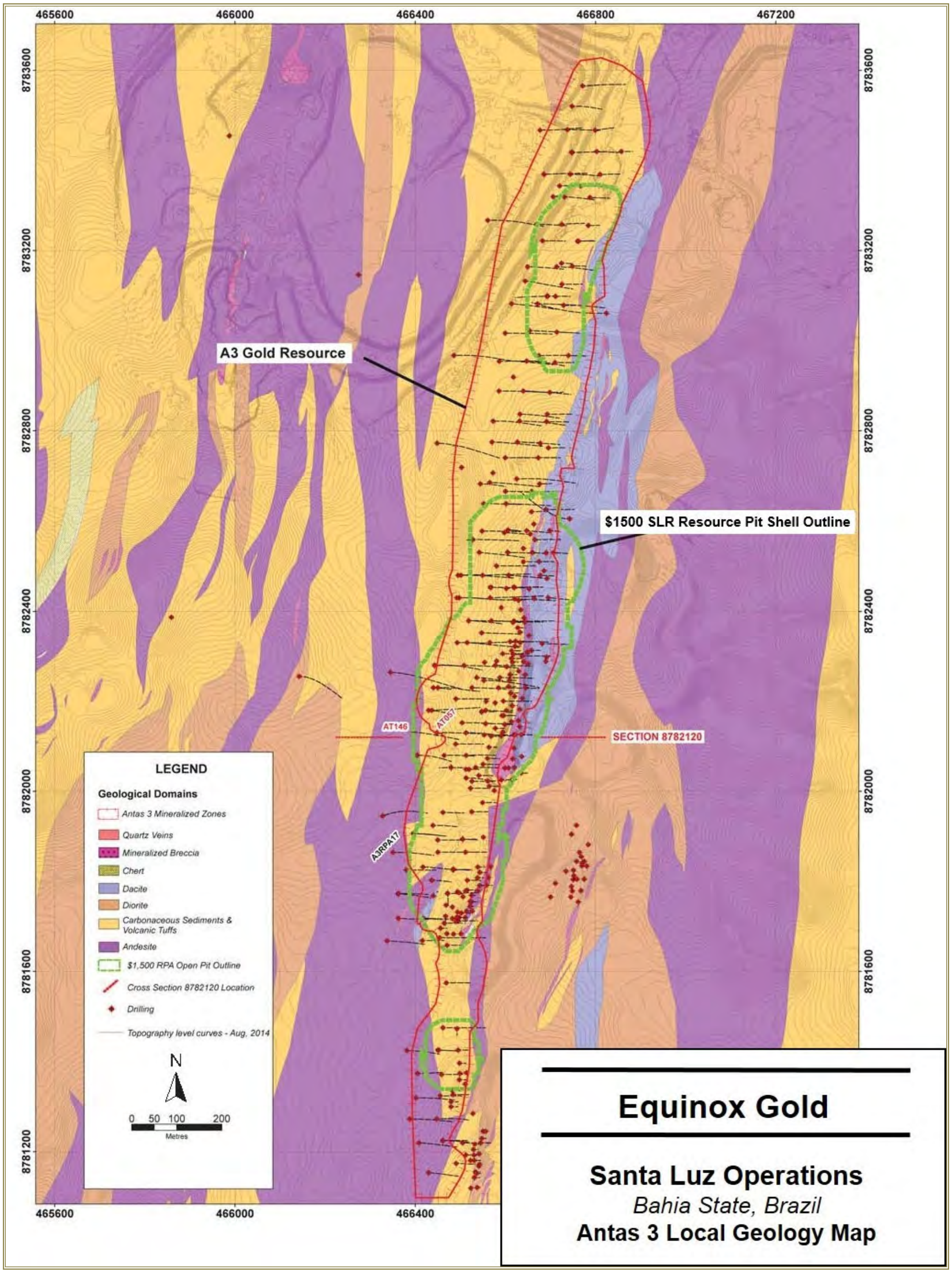


Figure 7-4: Santa Luz Antas 3 Local Geology

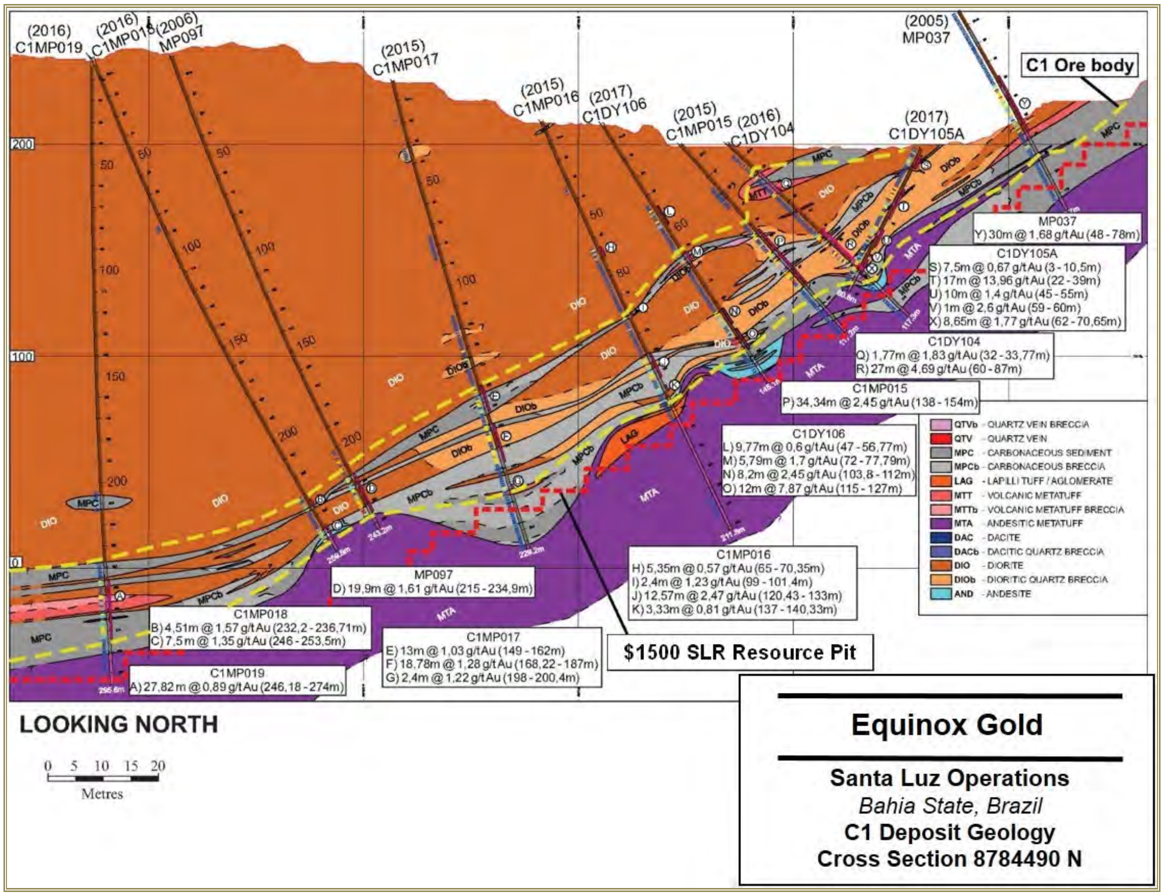


Figure 7-5: C1 Deposit Geologic Cross Section 878 4490

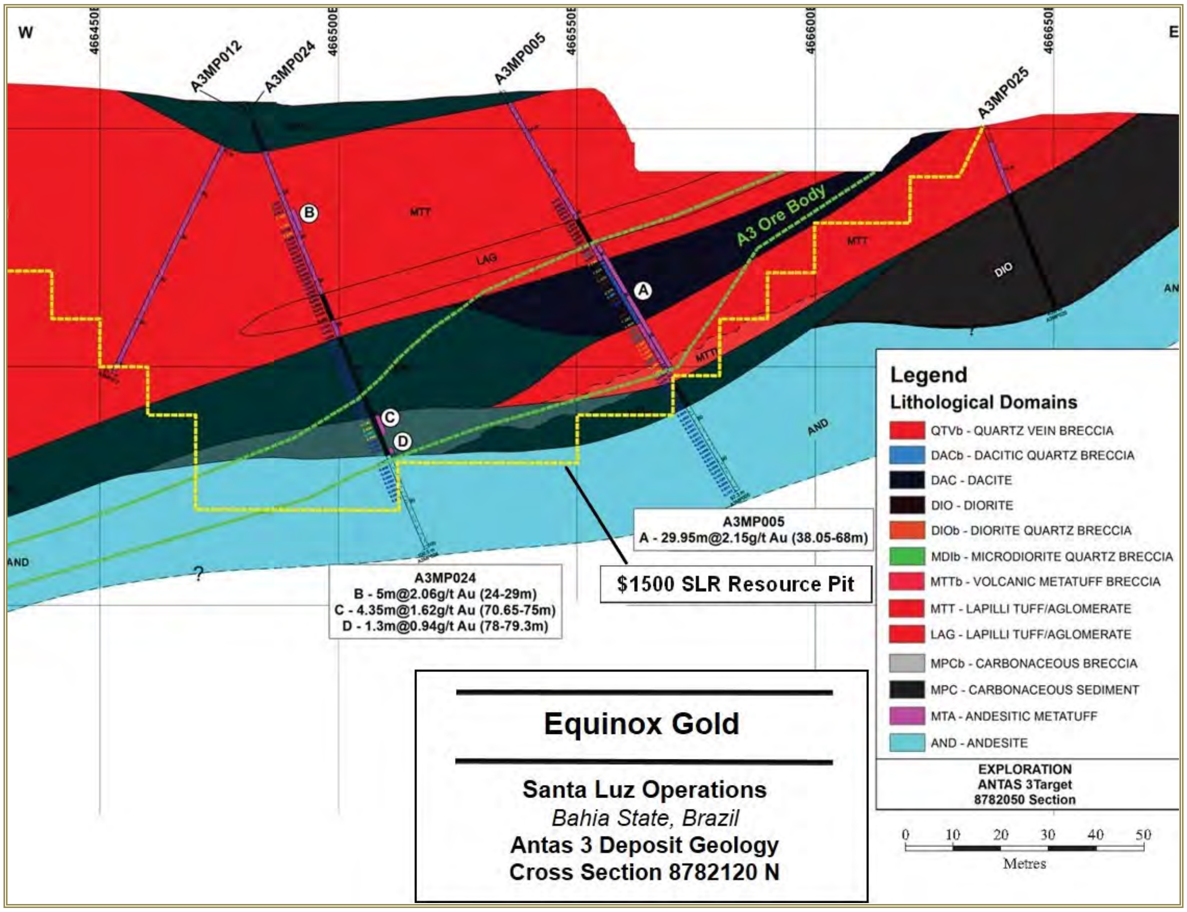


Figure 7-6: Antas 3 Deposit Geologic Cross Section 878 2120

Important host rocks include dioritic and dacitic intrusive rocks, sedimentary rocks, and a lesser amount of other volcanic and tuffaceous rocks. The dioritic and dacitic intrusions are epizonal, and are typically porphyritic, with fine- to medium-grained quartz and feldspar phenocrysts. The contacts with sedimentary rocks are commonly sharp, concordant with the main regional foliation, and exhibit finer-grained marginal phases (chill margins). The sedimentary and tuffaceous rocks are derived from, and are closely related to, the dacitic volcanic rocks. Sedimentary rocks and tuffaceous rocks commonly contain organic carbon, which appears to be a primary sedimentary component. More massive volcanic rocks and intrusive rocks are relatively free of organic carbon. The main gold-hosting lithologies from the C1 and Antas 3 resource areas and some averaged geochemical values from multi-element analyses conducted on 2015–2017 mineralized zone drill core are provided in Table 7-1. Referring to Table 7-1, carbonaceous rocks at C1 are sub-equal in abundance with dacitic rocks and other non-carbonaceous rocks, while at Antas 3 the non-carbonaceous rocks are more abundant. Both the overall gold grade and total organic carbon (TOC) content are higher at C1 than at Antas 3. Gold is notably higher in brecciated rocks.

Table 7-1: Geochemistry of Host Rocks—C1 and Antas 3 Resource Areas

Area	Lithology	Au (g/t)	Au Re-Adsorbed (g/t)	TOC (%)	S (%)	Ag (g/t)	Cu (ppm)	As (ppm)	# Samples	% of Pit
C1	Dacite Group	1.11	0.26	0.07	0.60	0.38	73	900	347	27
	Dacite Quartz Breccia	1.93	0.32	0.16	0.80	0.40	57	2256	253	20
	Carbonaceous Sediment	1.38	3.24	1.63	1.24	0.69	135	598	224	18
	Carbonaceous Breccia	2.24	3.22	1.27	1.47	0.64	114	709	375	30
	Volcanic Tuff Group	1.26	2.15	0.70	0.95	0.54	94	938	67	5
Antas 3	Dacite Group	1.45	0.13	0.04	0.52	0.44	19	1932	304	54
	Dacite Quartz Breccia	1.40	0.31	0.07	0.64	0.27	24	1094	70	12
	Carbonaceous Sediment	0.92	2.97	1.18	0.95	0.47	158	464	34	6
	Carbonaceous Breccia	3.02	3.07	0.89	1.05	0.68	95	463	66	12
	Volcanic Tuff	1.24	2.24	0.50	0.66	0.33	83	603	88	16

Host rocks are grouped for metallurgical purposes into two main groups, carbonaceous rocks and non-carbonaceous dacitic rocks. The carbonaceous rocks contain more TOC, which can interfere with cyanide leach gold recovery if measures are not implemented during processing that mitigate its effect. Carbonaceous rocks also contain higher contents of copper and sulphur related to higher modal abundances of sulphide minerals, possibly in part diagenetic in origin. Histograms showing the ranges of TOC content in dacitic and carbonaceous rock types are provided in Figure 7-7.

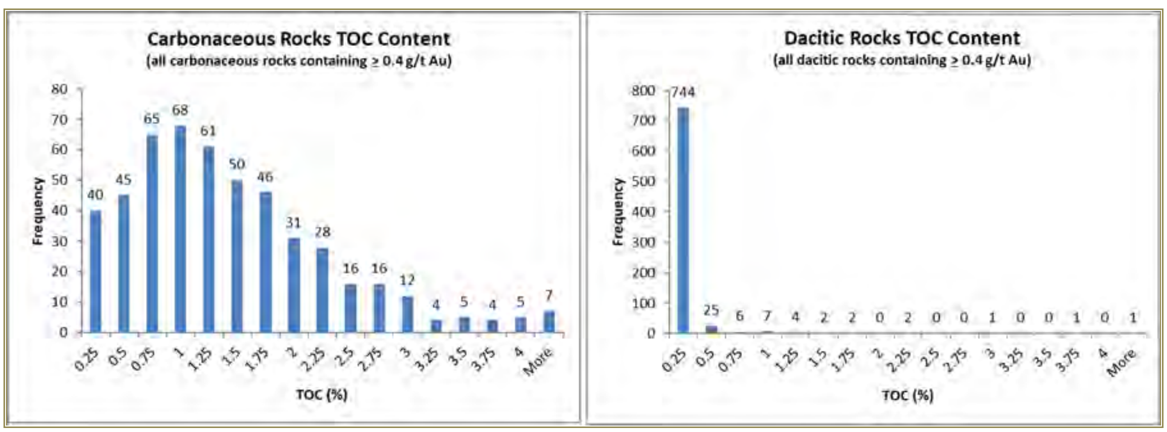
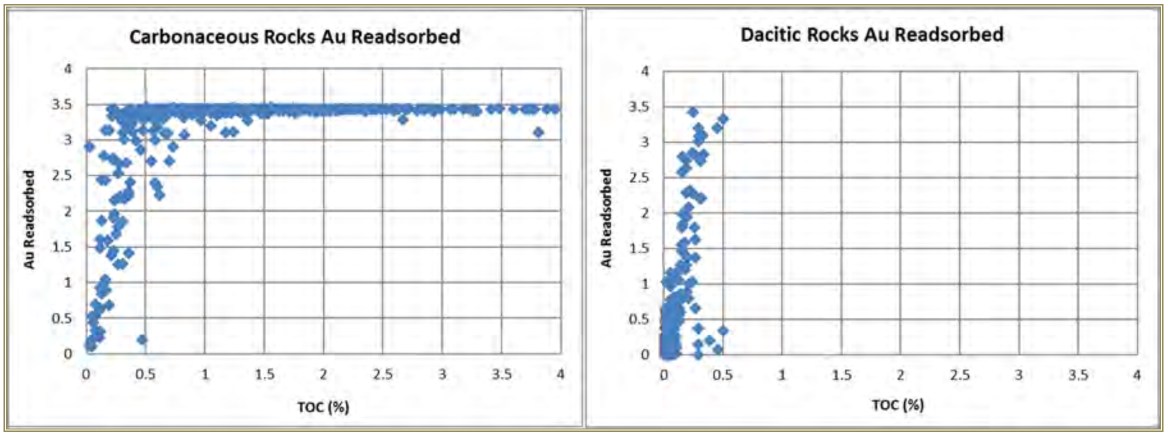


Figure 7-7: Carbonaceous and Dacitic Rocks TOC Contents

One analytic measure of the effect of the organic carbon content on cyanide gold leach recovery is the capacity for a sample to re-adsorb gold from a cyanide solution spiked with 3.4 ppm Au. High numbers returned from the gold re-adsorbed test indicate that a sample will have gold leach recovery problems (preg-robbing) if no mitigating factor is introduced. This test is routinely conducted on all drill core samples from the Project to identify which rocks require TOC mitigation. The results of the gold re-adsorbed testing for carbonaceous and dacitic rocks are provided in Table 7-1 and Figure 7-8. The mitigation of the impact of TOC is described in Section 13.



Note: Due to the re-adsorption test protocol, 3.5 g/t is the effective maximum that can be determined from the test.

Figure 7-8: Carbonaceous and Dacitic Rocks TOC vs. Au Re-Adsorbed

Dacitic rocks overall contain higher arsenic contents than do the carbonaceous rocks, indicative of a higher modal abundance of arsenopyrite. A particular subtype of dacite has been identified at the Antas 3 deposit, which contains an unusually high arsenic content. In this dacite, the gold is especially fine-grained and tends to be encapsulated in sulphide minerals, interfering with the gold cyanide leach recovery. This dacite subtype, referred to as the *high-sulphide dacite,* has been modelled as discrete lenses, separate from other Antas 3 dacite, as it will require different processing to recover the contained gold mineralization.

Silver contents are low (less abundant than gold) in all mineralized lithologies. While silver values occasionally reach several tens of grams per tonne in highly mineralized samples, the silver values are more commonly <1 g/t in the mineralized zone.

In general, gold mineralization at the Mansinha Trend is characterized by sulphide-rich quartz veining at the sheared contacts between lenses of microdiorite and carbonaceous and tuffaceous metasedimentary rocks, including crystal tuffs and agglomerates. At the Mari deposit, gold mineralization is associated with an extensive ductile shear zone (dominantly strike-slip), which is the geological contact between diorite and metasedimentary rocks. A schematic cross section of mineralization at the Mari deposit is shown in Figure 7-9. Oxidation at the C1 deposit can reach depths below surface of up to 60 m.

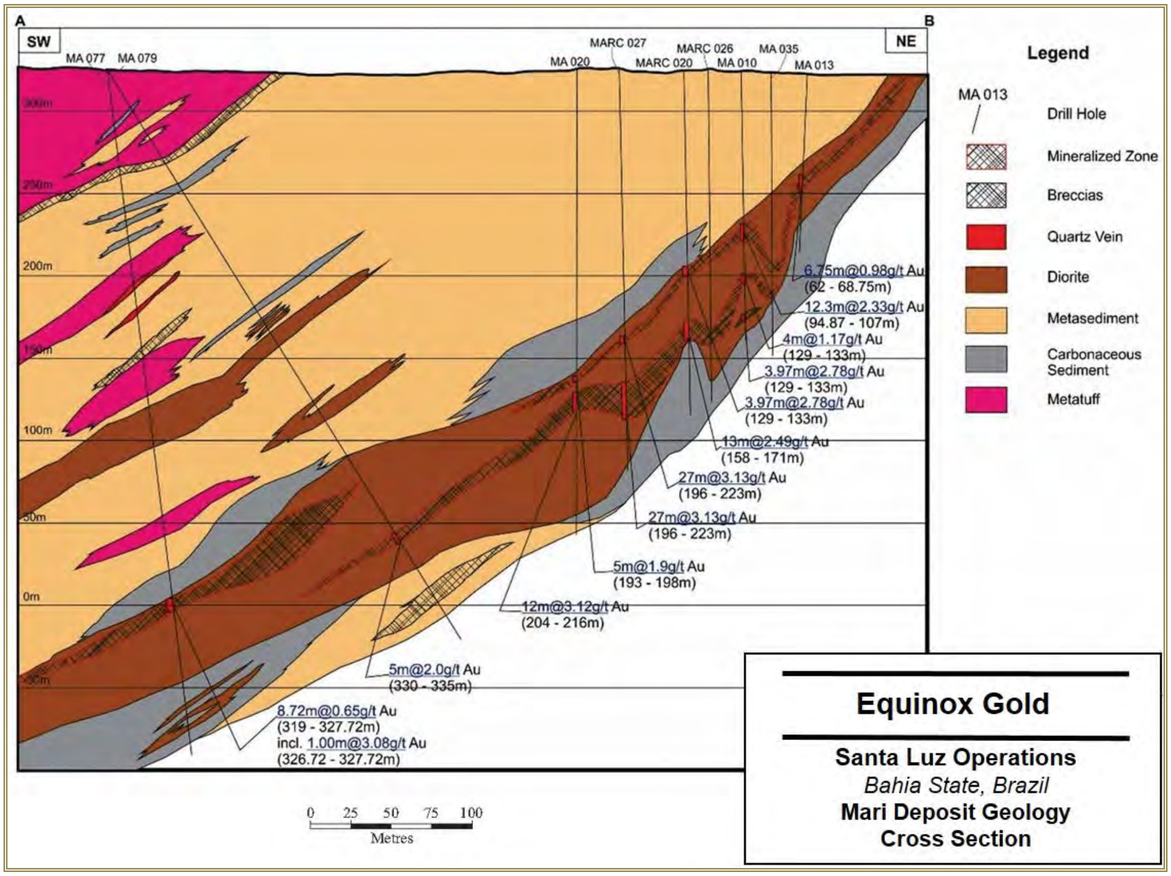


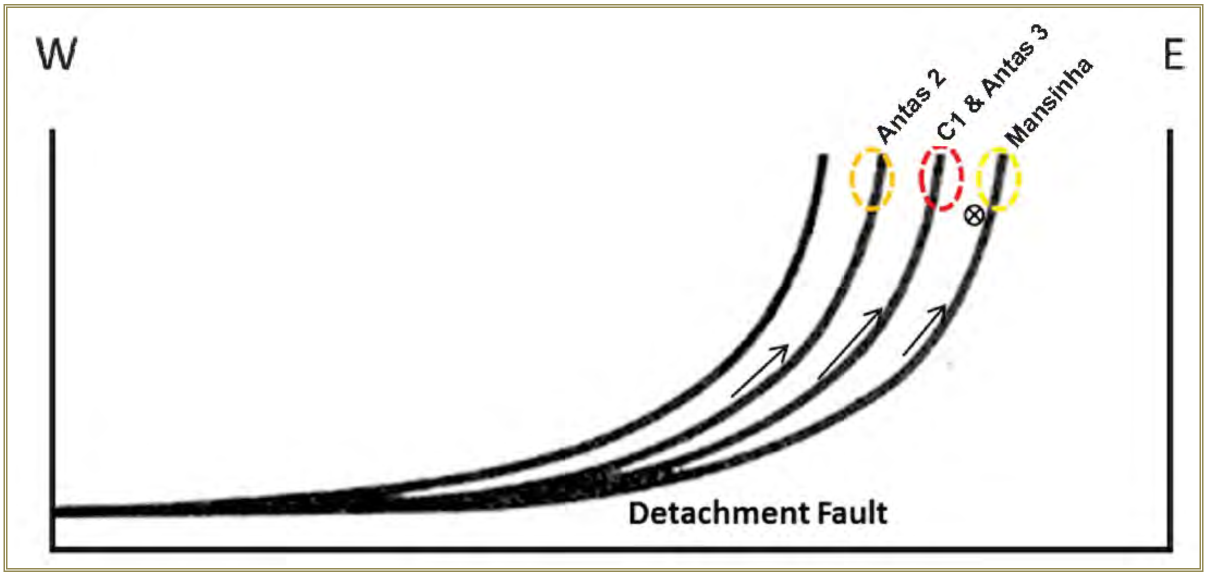
Figure 7-9: Mari Deposit Geologic Cross Section

7.2.1 Structure

The C1, Antas 2, Antas 3, Mansinha South (including the M11 and M3-M4 zones), Mansinha North (including the M16 and M17 zones), and Mari deposits are hosted in a set of three parallel, gently to moderately dipping, reverse and/or transcurrent fault zones, which extend for over 20 km along a north-south trend, and are thought to lie within an extensive reverse fault system. A schematic model of the fault system is shown in Figure 7-10.

Mineralization is best developed in areas where the faults deviate in orientation, and where the faults comprise the contact between host rocks having contrasting geochemical and/or rheological properties.

The dominant fault system at C1 exhibits a steep dip at surface, which flattens down dip. Down, dip stretch lineations and slickensides suggest that fault movement was dominantly vertical. Asymmetric folds and folded quartz veins demonstrate reverse movement. An increase in breccia thickness and gold grade is observed with the flattening of the fault. Mineralized zones range in thickness from a few metres to tens of metres and have relatively long strike and dip lengths.



Source: SLR, 2018

Note: E-W schematic cross section on Maria Preta District showing hypothetical model of reverse listric fault system. Strike-slip movements also occur along faults ⊗

Figure 7-10: Schematic Cross Section of the Reverse Listric Fault System

7.3 Gold Mineralization

At the Project, gold mineralization is closely related to quartz-carbonate-sulphide veining and breccia filling hosted in sheared and hydrothermally altered rocks. There are two main types of quartz veins. The most common type is foliation-parallel, white to grey coloured, north–south striking, and generally <1 m wide. These may be deformed and boudinaged, or may appear undeformed. Where deformed, they are associated with stretch lineations related to transcurrent or thrust movements. The second type of commonly observed quartz vein is most likely related to late northeast-striking structures. These structures are in brittle or ductile-brittle faults that crosscut all other structures, including foliation, and display a general dextral sense of movement. The veins associated with these structures are normally black or dark grey, and are thicker than the north-south striking veins. Fluid assisted hydraulic breccia is commonly associated with these veins.

Mixed veins, characterized by black and milky-coloured quartz, are found in strongly broken zones, mainly at the intersections of the north-south structures with the northeast-southwest structures. These intersections host voluminous and high-grade gold quartz veins and silicified breccia.

The host rocks of the gold mineralization exhibit weakly to moderately strong hydrothermal alteration. The most common type is quartz-sericite alteration. Also common are locally-pervasive albitization, carbonatization, and sulphidation.

Mineralized zones range in thickness from a few metres to tens of metres and have relatively long strike and dip lengths. Mineralization at C1 has a north-south strike length of approximately 1,550 m, width of approximately 1,400 m orthogonal to strike, and a depth extension of at least 840 m. Mineralization ranges in thickness from 15 m to 55 m with good continuity along strike and down dip.

Dacitic mineralization accounts for approximately 45% of the deposit, with carbonaceous mineralization accounting for 55% of the deposit.

Mineralization at Antas 3 has a north-south strike length of approximately 1,575 m, width of approximately 200 m to 400 m orthogonal to strike, and depth extension of 150 m. Mineralization ranges in thickness from <5 m to 15 m with good continuity along strike and down dip. Dacitic mineralization accounts for approximately 67% of the deposit, with carbonaceous mineralization accounting for 33% of the deposit.

Mineralization for C1 and Antas 3 is open at depth beyond the current limits of drilling and along strike, although it becomes relatively weaker.

QEMSCAN analysis found that gold is mainly found in association with pyrite and arsenopyrite as fine-grained inclusions, on grain boundaries and on fractures of sulphide grains. Gold grains may also be found in and on the surfaces of gangue minerals, especially hydrothermal quartz or carbonate. Gold grains commonly range from <5 µm to over 30 µm in size, with increasingly rare occurrences at progressively larger sizes. QEMSCAN work performed in 2017 found that the high arsenic, high-sulphide dacite found in part of the Antas 3 deposit has relatively finer-grained gold particles than the other dacitic rocks. Figure 7-11 displays QEMSCAN images illustrating typical modes of occurrence of gold grains. Where mineralization has been subject to near surface weathering and oxidation, gold commonly occurs in secondary iron oxides formed from the original sulphide grains.

Regional, property geology, and the style of the mineralization that is consistent with an orogenic gold deposit, are well understood by Equinox.

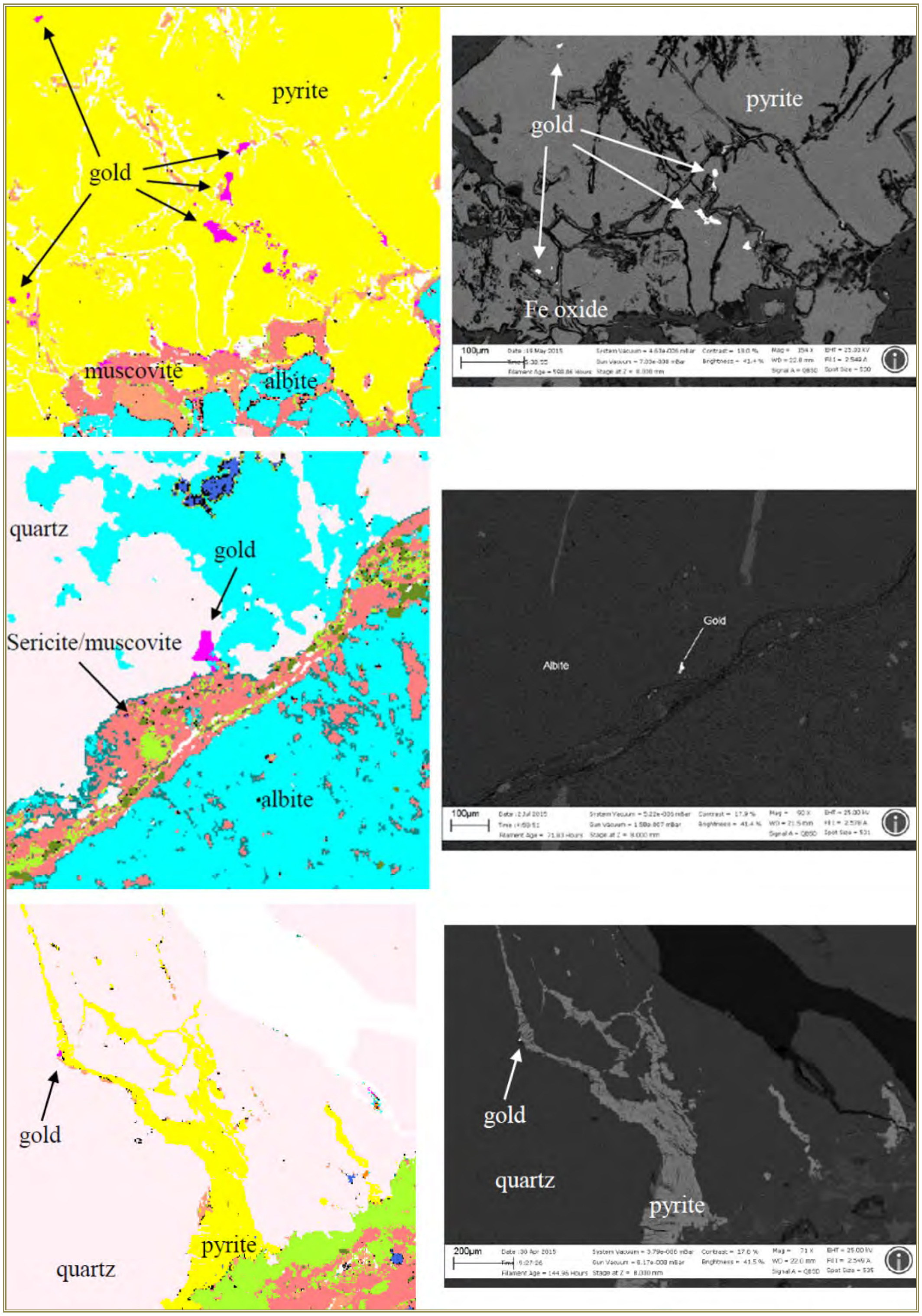


Figure 7-11: QEMSCAN Images

7.4 Exploration Potential

There are several satellite targets in the Project area, including the Mansinha North, Mansinha South, Mari, and Alvo 36 areas, where gold resources have been historically defined and are believed to show potential for the development of additional resources for Santa Luz. These areas are proximal to the C1 processing facilities, have returned positive drill results, and, in some cases, have been historically mined as open pits with heap leach gold recovery. The locations of these areas are shown in Figure 7-12.

Additional drilling would be required to potentially confirm the resources at these target areas. In addition, there are numerous significant and active artisanal mining areas in the Project area, such as the Treado Pit mined by Vale or CBPM, and areas such as Serra Branca North.

Significant gold mineralization is known to exist in the C1 resource in the down-dip extension of the deposit outside of the modelled C1 open pit. This resource is referred to as the C1 Underground on maps and cross sections and in the current Mineral Resource tabulation (Figure 7-6, Table 14-1, and Table 14-16), and is the subject of a PEA-level study summarized in Section 24 of this Technical Report. The C1 Underground has been explored with drill holes at 100 m to 200 m spacing, generally becoming less densely spaced with depth. Additional drilling could upgrade the classification of this resource and could potentially expand the resource along strike and further down dip. Equinox Gold plans to conduct in-fill and step-out drilling for the C1 Underground resource, with the goal of further developing underground resources.

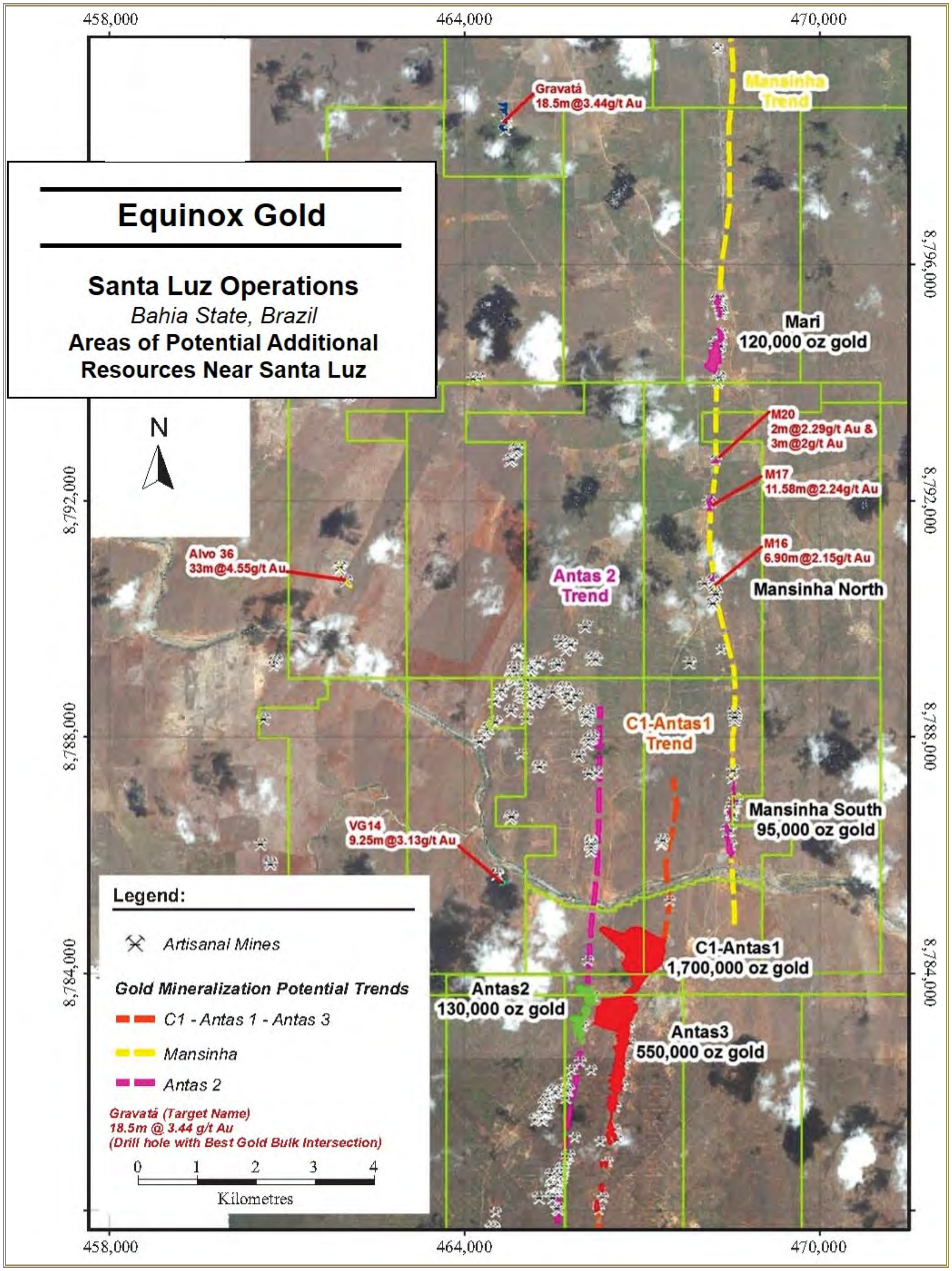


Figure 7-12: Areas of Potential Additional Resources Near Santa Luz

8 DEPOSIT TYPES

Mineralization at the Santa Luz Project is consistent with an orogenic gold deposit type. Orogenic gold deposits are distributed along major compressional to transpressional crustal-scale fault zones, in deformed greenstone terranes commonly marking the convergent margins between major lithological domains, such as volcano-plutonic and sedimentary domains (Dubé and Gosselin, 2006). These types of deposits are most abundant and significant, in terms of total gold content, in Archean terranes; however, a significant number of world-class deposits are also found in Proterozoic and Paleozoic terranes. Where found in Precambrian greenstone belts, these deposits are commonly classified as greenstone gold deposits, although the characteristics are fundamentally the same as those of later orogenic gold deposits.

Greenstone gold deposits are structurally controlled, epigenetic deposits characterized by simple to complex networks of gold-bearing, laminated quartz-carbonate fault-fill veins. These veins are hosted by moderately to steeply dipping, compressional, brittle-ductile shear zones and faults with associated extensional veins and hydrothermal breccias. These deposits are hosted by greenschist to locally amphibolite-facies metamorphic rocks of dominantly mafic composition and formed at intermediate crustal depth (5 km to 10 km). The mineralization is syn- to late-deformation and typically post-peak greenschist-facies or syn-peak amphibolite-facies metamorphism. It is typically associated with iron carbonate alteration. Gold is largely confined to the quartz-carbonate vein network; however, it may also be present in significant amounts within iron-rich sulphidized wall rock selvages or within silicified and arsenopyrite-rich replacement zones.

In the Project area, gold deposits are hosted within the Paleoproterozoic aged RIGB, which was deformed and metamorphosed during the Trans-Amazon Orogeny (approximately 2 Ga). Gold mineralization mainly occurs with fault-related quartz-sulphide and quartz-carbonate-sulphide veining and quartz-sulphide breccia. Alteration includes sericitization, carbonate alteration, albitization, sulphidation, and silicification.

Regional, property geology and the style of the mineralization that is consistent with an orogenic gold deposit, is well understood by Equinox.

9 EXPLORATION

From 1979 to 1995, CVRD and CBPM undertook several extensive stream sediment and soil geochemistry programs over the entire Maria Preta Gold District. Encouraging results were followed up using geophysics and drilling. Numerous deposits were discovered and mined, commonly focusing on the shallow, oxidized portions of these deposits. Possessing a wealth of historical exploration data, Yamana conducted extensive drilling to develop the C1 and Antas 3 deposits as well as several other prospects in the district.

From September 2015 through April 2017, Brio's work at Santa Luz was conducted in two phases of resource, metallurgical, and geotechnical drilling in support of this Open Pit Feasibility Study. This drilling is described in Section 10.

Most of the concessions at Santa Luz are at an early exploration stage, with limited exploration activity other than regional mapping, regional geochemistry surveys, and airborne surveys completed by the previous owners.

9.1 Additional Resource Potential

Additional resources could potentially be developed in the area of the C1 Underground Resource, down-dip from the C1 open Pit Resource. Future drilling could potentially upgrade and increase the known resource, which is poorly constrained by the existing, widely spaced exploration drill holes. Other gold prospects are also known in the Project area, some of which have had historical gold production and have modelled resources as described in SLR (2015). Future drilling on these prospects could potentially develop additional resources as well.

10 Drilling

Drilling on the Santa Luz Project area has been conducted in phases by several companies since 1975. Very limited information on the historical drilling details is available. From 2003 to 2017, Yamana, and subsequently Brio, carried out diamond drilling for resource definition. In addition, metallurgical and geotechnical drilling were conducted in support of the Open Pit Feasibility Study in 2015 and 2017. A drilling summary, including historical drill logs collected by Yamana, is included in Table 10-1. Maps of drill hole collars are shown in Figure 10-1 to Figure 10-3.

Leagold and Equinox have not carried out any drilling at the Project.

Table 10-1: Santa Luz Drilling

	DDH		RAB		RC		Metallurgical		Geotechnical		Wells		Total Drilling	
Deposit	**No. Holes**	**Metres**	**No. Holes**	**Metres**	**No. Holes**	**Metres**	**No. Holes**	**Metres**	**No. Holes**	**Metres**	**No. Holes**	**Metres**	**No. Holes**	**Metres**
Historical (1975—1995)[1,2]														
C1	67	3,184	-	-	-	-	-	-	-	-	-	-	67	3,184
Antas 2	70	5,588	-	-	-	-	-	-	-	-	-	-	70	5,588
Antas 3	109	6,394	-	-	-	-	-	-	-	-	-	-	109	6,394
Subtotal	**246**	**15,166**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**246**	**15,166**
Yamana (2003—2013)														
C1	203	57,855	301	7,871	6	983	-	-	-	-	-	-	510	66,709
Antas 2	29	5,272	131	3,275	-	-	-	-	-	-	-	-	160	8,547
Antas 3	150	21,389	374	8,939	-	-	-	-	-	-	-	-	524	30,328
Mansinha	86	10,558	761	21,709	8	1,197	-	-	-	-	-	-	855	33,464
Mari	59	6,795	234	6,476	22	3,039	-	-	-	-	-	-	315	16,310
Other Targets[4,5]	159	24,789	418	11,696	57	7,050	-	-	-	-	-	-	634	43,535
Wells	-	-	-	-	-	-	-	-	-	-	43	2,486	43	2,486
Subtotal	**686**	**126,658**	**2,219**	**59,966**	**93**	**12,269**	**-**	**-**	**-**	**-**	43	**2,486**	**3,041**	**201,379**
Brio (2015—2016)														
C1	50	9,348	-	-	-	-	2	287	8	1,368	-	-	60	11,003
Antas 2	8	1,225	-	-	-	-	-	-	4	439	-	-	12	1,664
Antas 3	24	2,852	-	-	-	-	3	289	5	440	-	-	32	3,580
Plant Site	-	-	-	-	-	-	-	-	12	176	-	-	12	176
Stockpiles	-	-	-	-	446	4,166	-	-	-	-	-	-	446	4,166
Subtotal	**82**	**13,425**	**-**	**-**	**446**	**4,166**	**5**	**576**	**29**	**2,423**	-	-	**562**	**20,590**
Brio (2016—2017)														
C1	10	1,595	-	-	-	-	-	-	-	-	-	-	-	-
Antas 3	20	2,358	-	-	-	-	-	-	-	-	-	-	-	-
Plant Site	5	83	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal	**35**	**4,037**	-	-	-	-	-	-	-	-	-	-	-	-
Grand Total	**1,049**	**159,285**	**2,219**	**59,966**	**539**	**16,435**	**5**	**576**	**29**	**2,424**	43	**2,486**	**3,849**	**237,135**
C1	330	71,982	301	7,871	6	983	2	287	8	1,368	0	0	637	80,896
Antas 2	107	12,085	131	3,275	0	0	0	0	4	439	0	0	242	15,799
Antas 3	303	32,993	374	8,939	0	0	3	289	5	440	0	0	665	40,302
Mansinha[3]	86	10,558	761	21,709	8	1,197	0	0	0	0	0	0	855	33,464
Mari	59	6,795	234	6,476	22	3,039	0	0	0	0	0	0	315	16,310

Deposit	DDH		RAB		RC		Metallurgical		Geotechnical		Wells		Total Drilling	
	No. Holes	Metres	No. Holes	Metres	No. Holes	Metres	No. Holes	Metres	No. Holes	Metres	No. Holes	Metres	No. Holes	Metres
Other Targets[4]	159	24,789	418	11,696	57	7,050	0	0	0	0	0	0	634	43,535
Wells	0	0	0	0	0	0	0	0	0	0	43	2,486	43	2,486
Plant Site	5	83	0	0	0	0	0	0	12	176	0	0	12	176
Stockpiles	0	0	0	0	446	4,166	0	0	0	0	0	0	446	4,166
Grand Total	**1,049**	**159,285**	**2,219**	**59,966**	**539**	**16,435**	**5**	**576**	**29**	**2,423**	43	**2,486**	**3,849**	**237,135**

Notes: **1.** Type of historical drilling is unknown.
2. DDH indicates diamond drill holes; RAB indicates rotary air blast drill holes; RC indicates rotary percussion drill holes.
3. Summary includes digitally captured historical drilling only and is not thought to be comprehensive and not used in resource estimate.
4. Collectively referred to as Mansinha South (M3M4, M11) and Mansinha North (M17).
5. Includes 26 gold targets on the property.
6. Metres are rounded.

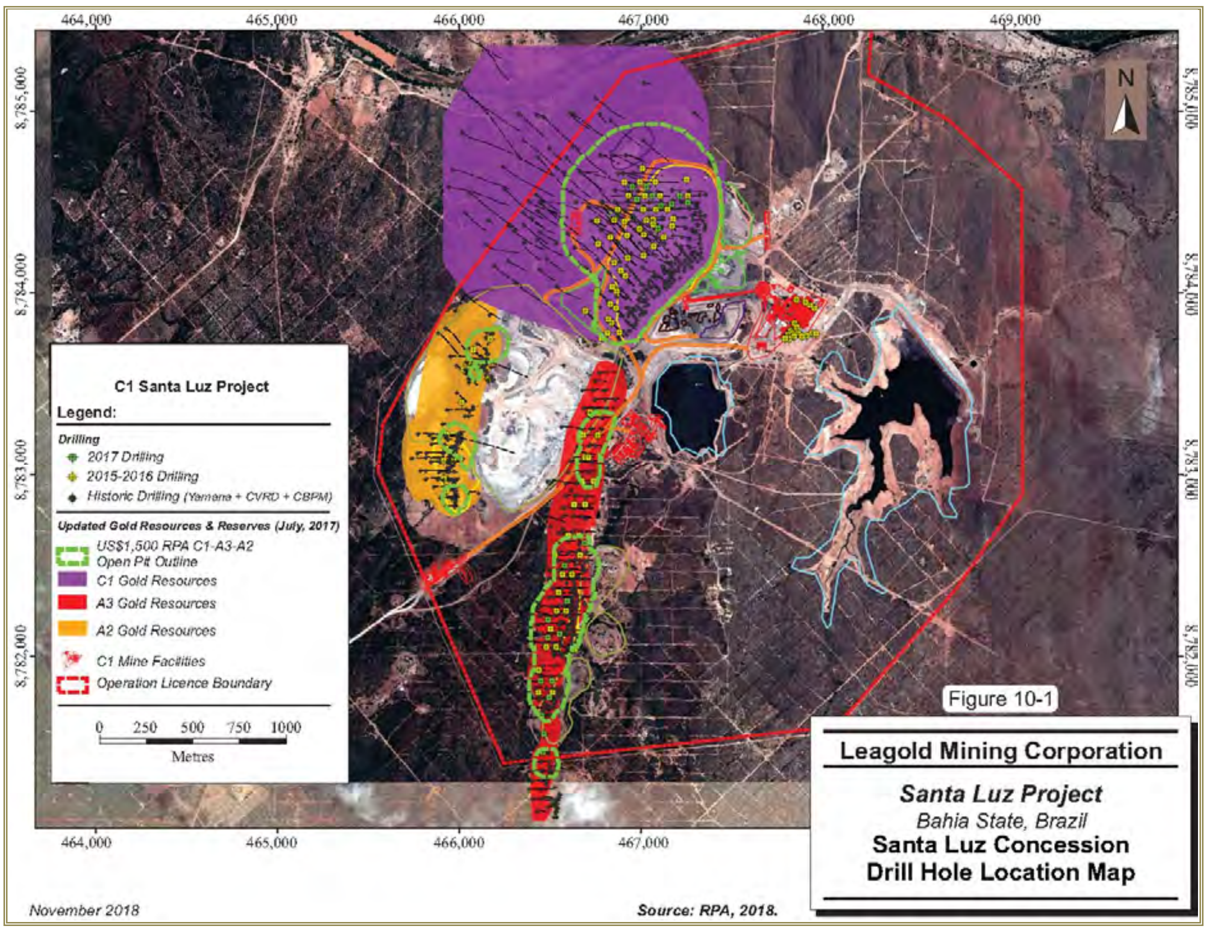


Figure 10-1: Santa Luz Concession Drill Hole Location

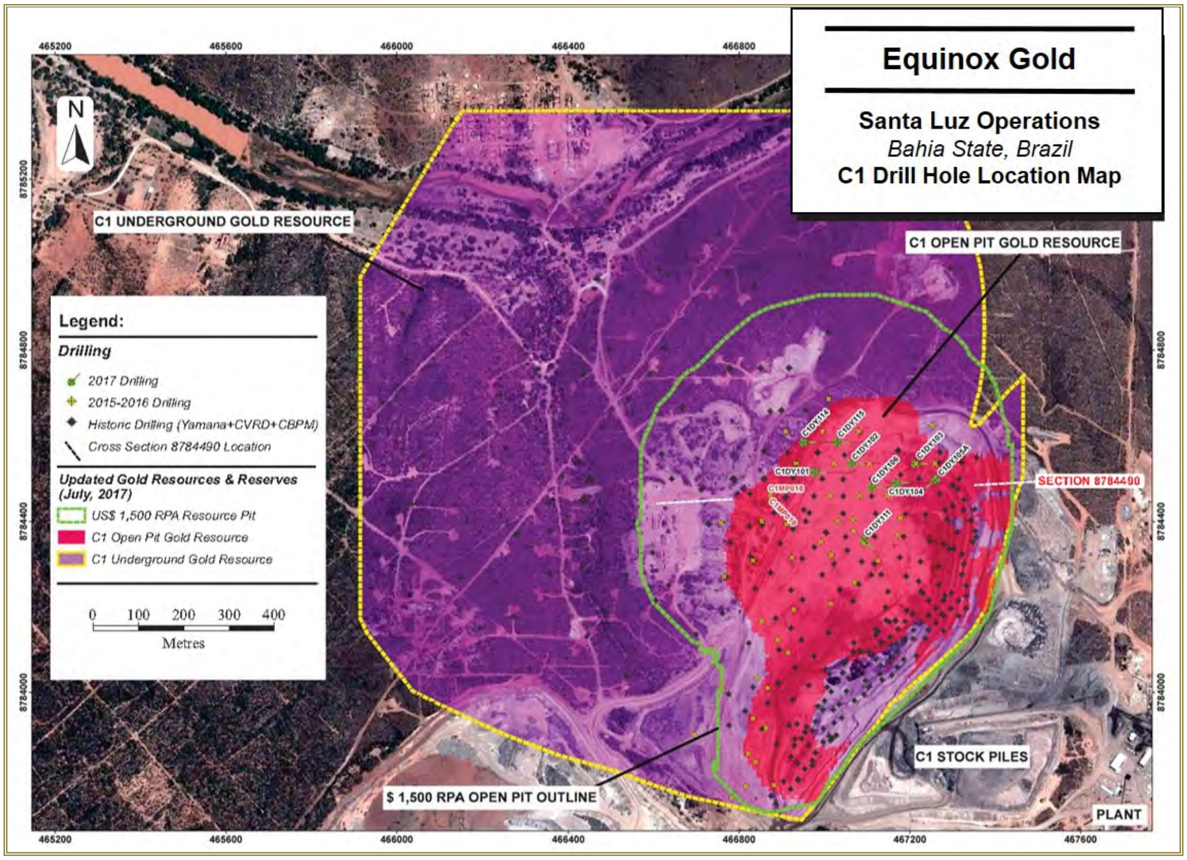


Figure 10-2: C1 Drill Hole Location

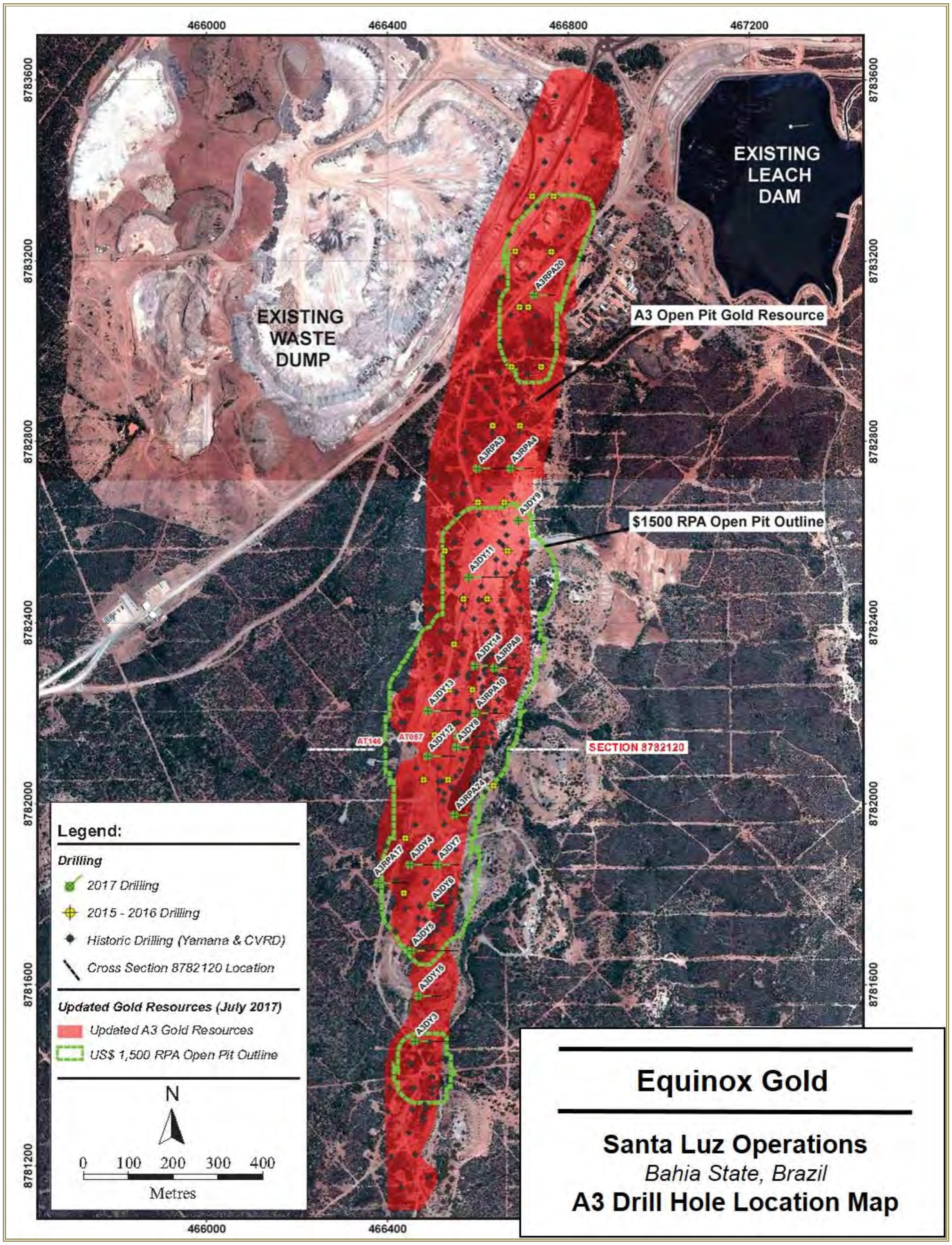


Figure 10-3: Antas 3 Drill Hole Location

10.1 Previous Drilling (1975—1995)

Limited detail is available on the CVRD and CBPM drilling programs. Approximately 34,500 m of surface diamond drilling and reverse circulation (RC) (air-percussion) drilling was completed from 1975 to 1995, the majority of which was over now-depleted open pit mine areas, including C1, Antas 2, Antas 3, C1W, the Mansinha Trend, and Mari. Yamana digitally captured a portion of the information for this drilling, reflected in Table 10-1. Due to data-quality concerns, data from these drilling campaigns have not been included in the current Mineral Resource database.

10.2 Yamana (2003—2013)

Yamana conducted drilling on the property from 2003 to 2013, principally using diamond drilling to define gold resources. Rotary air blast (RAB) drilling was used to generate shallow mineralization targets between known mineralization zones, and RC drilling was used to pre-collar deep targets at Santa Luz and to test some shallow targets.

10.3 Brio (2015—2016)

In support of the 2016 Pre-feasibility Study (PFS), Brio conducted core and RC drilling from September 2015 through July 2016. Core drilling included the resource definition, metallurgical and geotechnical core drilling tabulated in Table 10-1. Drillgeo Geologia e Sondagem Ltda. drilled all core using HQ diameter (63.5 mm). Resource definition holes were logged, split, and analyzed for geochemistry and specific gravity and results are used for resource modelling. Geotechnical holes are oriented core holes that provided structural data used in geomechanical modelling to derive pit slope parameters. Metallurgical holes provided core samples for ore processing tests. Brio also completed 4,166 m of RC drilling to better define stockpiled material from the period that Yamana was mining the deposit.

10.4 Brio (2016—2017)

In support of the Open Pit Feasibility Study, from December 2016 through April 2017 Brio conducted core drilling that included the resource definition and plant site geotechnical core drilling tabulated in Table 10-1. The drilling company, Servitec, produced HQ core. Resource definition holes were logged, split, and analyzed for geochemistry and specific gravity, and results were used for resource modelling. Significant intercepts from the resource definition drilling are provided in Table 10-2.

Table 10-2: Significant Intercepts from 2016–2017 Resource Definition Drilling

Drill Hole	From	To	Length (m)	Au (g/t)
C1 Significant Intercepts				
C1DY101	212.00	222.00	10.00	1.03
C1DY101	224.92	233.00	8.08	1.53
C1DY102	167.00	178.52	11.52	4.83
C1DY103	50.22	55.00	4.78	1.15
C1DY103	113.68	118.56	4.88	1.51
C1DY104	60.00	87.00	27.00	4.69
C1DY105A	22.00	39.00	17.00	13.96
C1DY105A	45.00	55.00	10.00	1.40
C1DY105A	62.00	70.65	8.65	1.77
C1DY106	72.00	77.79	5.79	1.70
C1DY106	103.80	112.00	8.20	2.45
C1DY106	115.00	127.00	12.00	7.87
C1DY111	85.00	104.80	19.80	1.09
C1DY114	226.00	232.59	6.59	1.22
C1DY114	238.00	253.00	15.00	1.73
C1DY115	174.00	196.00	22.00	1.22
C1DY115	207.00	217.12	10.12	6.02
Antas 3 Significant Intercepts				
A3DY3	32.00	38.00	6.00	0.90
A3DY4	64.00	72.00	8.00	1.19
A3DY4	84.20	87.38	3.18	3.20
A3DY5	19	26.04	7.04	1.17
A3DY5	53.00	54.67	1.67	1.49
A3DY6	40.89	45.55	4.66	2.51
A3DY7	41.00	45.48	4.48	1.26
A3DY8	55.37	65.00	9.63	2.19
A3DY8	77.00	81.45	4.45	1.16
A3DY9	10.00	14.00	4.00	2.22
A3RPA3	95.00	101.00	6.00	1.24
A3RPA8	43.00	52.00	9.00	1.40
A3RPA8	56.00	62.00	6.00	1.16
A3RPA10	42.50	46.00	3.50	2.25
A3RPA10	70.85	73.46	2.61	1.72
A3RPA13	58.80	71.00	12.20	1.95
A3RPA17	95.00	103.79	8.79	2.26
A3DY11	43.00	51.16	8.16	1.26
A3DY11	85.00	91.07	6.07	1.39
A3DY11	94.00	105.00	11.00	1.36
A3DY12	75.00	87.00	12.00	1.53
A3DY13	106.67	116.00	9.33	2.28
A3DY14	63.88	67.52	3.64	1.41

10.5 Drilling and Logging Procedures

At Santa Luz, diamond drill programs have been completed by Geosol, Rede, and Servitec drilling companies, in addition to Yamana-owned drill rigs. Various sized drill hole diameters have been employed, including AQ (27 mm), BQ (36.5 mm), HQ, NQ (47.6 mm), NQ2 (50.6 mm), LTK48 (35.3 mm), and HWL (63.5 mm); however, the majority of drilling was completed using HQ diameter core. RC drilling on site was contracted to Geosedna. No information was available on the RAB drilling campaigns, which were used in preliminary exploration to generate targets for diamond drill holes.

At Santa Luz, drill hole collars have been surveyed using the UTM South American Datum 1969. All RC and infill diamond drill holes as well as holes located on significant exploration targets were sighted using a total station theodolite, which was also used to set azimuth orientation. A check of the azimuth and dip angles was performed prior to the completion of the hole. Initial collar locations of exploratory RAB, RC, and diamond drill holes were sighted using a hand-held GPS, with drill hole dip and azimuth of each hole set using a hand-held compass.

Downhole surveys of RC and diamond drill holes were completed at 3 m intervals using magnetic and non-magnetic survey equipment, including DeviFlex, REFLEX-MAXIBOR, Pee Wee, and REFLEX-GYRO. Results were duplicated and compared. The survey was performed a third time if a discrepancy above a 2% tolerance limit was found. Downhole surveys of RAB holes were not performed.

Drill core was placed in wooden and/or plastic core boxes with a nominal capacity of 4 m for BQ, NQ, or NQ2 diameter drill core and 3 m for HQ- or HWL-diameter drill core. The drill hole name, target name, box number identification, and downhole depths were stamped onto an aluminum tag and affixed to the edge of the box. The driller placed wooden downhole core depth markers in the core box, affixed with an aluminum tag stamped with the depth, the length of the interval, and the length of the recovered sample.

Upon receipt of the drill core at the logging shed, sample recovery was measured and recorded. Where total drill hole recovery was <80%, or recovery of mineralized zones was poor and not representative, the drill hole was discarded and duplicated at the drill company's expense. The recovery in the mineralized zones was generally better than 90%. Hole depth, in metres, was recorded in black permanent ink on the core box, with consideration to core loss and over-drill in weathered zones. Intervals for use in bulk density determinations were selected and marked.

Geologists then marked lithological boundaries and contacts on the core, and samples were marked down the entire length of the drill core, both directly on the core and on the core boxes. Following drill core mark up, core was photographed, sawn in half with an electric diamond-blade rock saw, and sampled by trained samplers.

After core sampling, site geologists logged lithology and recorded lithological contacts, structure, mineralization, and hydrothermal alteration features in a graphic log. Codes assigned for oxidation state, alteration, sulphide presence and content, and presence of visible gold were recorded on paper core logs. The rock codes used are listed in Table 10-3. Core angles related to structures such as foliation, faults, or quartz veins were also recorded on the logs, as were sample intervals and numbers.

Once logging and measurements for a drill hole were complete, geology technicians typed results into a Microsoft Excel table, validated, and imported using scripts into the primary Microsoft Access database.

Table 10-3: Santa Luz Rock Codes

Group Code	Rock Code	Lithology	Grouped Lithology
1	0	Without Recovery	INDEFINITE
	3	Waste	
	10	Quartz Vein	
	75	Basaltic Breccia	
	77	Sediment Breccia	
	88	Rhyolite	
	84	Pyroclastic Sequence	
	97	Chert	
	100	Basalt	
	107	Sericite Schist	
	108	Diorite Quartz Porphyry	
	117	Volcanic Cristal Tuff	
2	1	Soil/Laterite/Cover	SOIL
	2	Saprolite/Weathered Rock	
3	71	Carbonaceous Breccia	CARBONACEIOUS BRECCIA
	94	Lapilli-Tuff	
4	80	Andesite	ANDESITE
	74	Andesite Breccia	
5	81	Dacite	DACITE
	70	Dacitic Quartz Breccia	
	72	Quartz Breccia	
	73	Dioritic Quartz Breccia	
6	83	Volcanic Tuff	VOLCANIC TUFF
	85	Volcanic Agglomerate	
	86	Volcanic Tuff Breccia	
	90	Sediment	
7	105	Diorite	DIORITE
	106	Granodiorite	
	87	Metapelite Volcanic	
8	91	Carbonaceous Sediment	CARBONACEOUS SEDIMENT

Logged and sampled core boxes were placed on pallets and moved to a permanent core storage area (core shed).

Drilling recovery is contractually stipulated to be above 95% in all mineralized zones. No drilling, recovery, or sampling factors were encountered that negatively impacted the accuracy and reliability of the resource estimate. In Equinox's opinion, the drilling and logging methods are acceptable for the purposes of a Mineral Resource estimate.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Method and Approach

Sampling of the 2016 and 2017 drill holes focused on the mineralized zones and a significant length of core above and below the targeted mineralization was sampled to ensure that the mineralized zone was captured. Samples have a nominal length of 1 m; however, the length was adjusted so that sample endpoints respected geological contacts. Samples were tagged with a plasticized paper tag indicating sample number, a duplicate of which was stapled inside the core box. Quality assurance/quality control (QA/QC) samples, including duplicates, blanks, and standards, were incorporated into the sample stream.

Diamond drill core was sawn in half lengthwise with an electric diamond-blade core saw and sampled by a trained sampler, returning half of split core to the core box, and submitting the other half for sample preparation and analysis. Half core samples were placed in a marked plastic bag with their paper sample tag. Bags were securely closed and sealed with a tie to avoid leakage of the sample during delivery to the laboratory. Sample weight was approximately two kilograms.

Core and chips are stored within two purpose-built core sheds on site, both of which are locked at night.

11.2 Density Analysis

For density determination, core samples were collected from each drill hole in the various lithologies to represent most rock types at the C1, Antas 3, and Antas 2 areas. One representative sample of mineralized and non-mineralized lithologies was taken from the remaining half HQ drill core of each hole. Usually, the sample was 10 cm to 15 cm long, preferably collected at the beginning of a geochemical sample interval. In total, 300 core samples were analyzed for bulk density: 150 samples from the C1 open pit resource, 126 samples from Antas 3, and 24 samples from Antas 2.

Santa Luz personnel used independent and internationally recognized laboratories for sample preparation and analysis. The density test samples were sent to the independent ALS Chemex Laboratory in Lima, Perú (ALS Lima). Density data obtained were reported by ALS Lima in digital reports accompanied by their respective certificates. This laboratory is ISO 9001:2000 and ISO 17025:2005 accredited. The analytical procedure used was the ALS Chemex OA-GRA09as, in which the core samples are coated in paffin wax, weighed in air, and then weighed while submerged in water. The density is calculated from the following equation:

After density testwork was completed, all core samples were returned and put back in their respective core boxes.

OA-GRA09as:

$$\boxed{\text{Density} = \frac{A}{B - C - [(B - A) / \text{Dwax}]} \text{ x Density of water [at temperature (°C)]}}$$

where: A = weight of sample in air
B = weight of waxed sample in air
C = weight of waxed sample suspended in water
D = density of wax

11.3 Sample Preparation

Sample preparation was completed at ALS Chemex in Vespasiano, Minas Gerais. This laboratory is ISO 9001:2000 and ISO 17025:2005 accredited, and independent of Equinox Gold. For shipping from the mine site to the laboratory, the samples were placed in large plastic shipping sacks, accompanied by documentation of the batch and samples, loaded onto a truck owned and driven by a locally based transport company, and driven to the laboratory. Each sample was logged into a laboratory information management system (LIMS), weighed, dried, and then crushed to better than 90% passing a 2 mm screen (10 mesh) using a jaw crusher. A 300 g split was taken using a Jones riffle or rotary splitter, and subsequently pulverized using a steel pulverizer to better than 95% passing a 150 mesh. Following pulverization, a 100 g split was taken using a rotary splitter or spatula. Between samples, crushers and pulverizers were cleaned using compressed air, and between batches, with certified blank material. After the samples were crushed and pulverized, pulp splits were sent for geochemical analysis at ALS Lima. Remaining sample material was returned to the Santa Luz Project for storage.

In SLR's opinion, the sample preparation methods are acceptable for the purposes of a Mineral Resource estimate.

11.4 Sample Analysis

Assays were processed at ALS Lima. Gold content was determined by 50 g fire assay with an atomic absorption (AA) finish. Silver and other elements were analyzed using a four-acid digestion, 48 elements inductively coupled plasma (ICP) method. TOC and sulphur contents were determined using a LECO furnace.

Also, at ALS Lima, the capacity of samples to re-absorb gold from a cyanide leach solution was analyzed using special wet chemical cyanide leach tests that measure the concentration of gold in leach solutions with and without an added gold spike, determining the amount of gold that is reabsorbed from the leach solution by the rock sample.

In addition to the analyses performed at ALS Chemex, 5% of analyzed samples were also sent to SGS Geosol in Vespasiano and Minas Gerais for an inter-laboratory check of the fire assay.

Following a 45-day period, pulp and coarse rejects were returned to the Santa Luz Project site for storage in a core storage facility.

11.5 Database Management

Geology technicians type logging results into a Microsoft Excel (Excel) table, which is then validated and imported using scripts into the primary Microsoft Access database. The laboratories provided assay data in Excel format, then imported using-purpose built scripts with integrated validation tools. The final database for use in Mineral Resource estimation is imported using *.csv tables into Maptek's Vulcan software.

In SLR's opinion, the sample preparation, analysis, and security procedures at Santa Luz meet industry standards and are adequate for use in estimating Mineral Resources.

11.6 Quality Assurance/Quality Control

Quality assurance (QA) consists of evidence to demonstrate that the assay data have precision and accuracy within generally accepted limits for the sampling and analytical method(s) used to have confidence in a resource estimate. Quality control (QC) consists of procedures used to ensure that an adequate level of quality is maintained in the process of collecting, preparing, and assaying the exploration drilling samples. In general, QA/QC programs are designed to prevent or detect contamination and allow assaying (analytical), precision (repeatability), and accuracy to be quantified. In addition, a QA/QC program can disclose the overall sampling-assaying variability of the sampling method itself.

11.6.1 Quality Assurance/Quality Control Protocols

Historical (1975–1995)

No information on QA/QC protocols is available for historical data. The historical data were not included in the resource database.

Yamana (2003–2013)

Yamana initiated a QA/QC protocol at the Santa Luz site in 2005. Assays collected during the 2004 program were not accompanied by QA/QC samples.

RPA reviewed internal reports prepared for the Project area on QA/QC results Yamana collected and analyzed from 2005 to 2008 (Yamana, 2008), and from 2008 to 2012 (Yamana, 2012). RPA described these QA/QC programs and results in its PEA report dated September 15, 2015, made recommendations for future QA/QC programs, and concluded that the associated drill results are acceptable for use in Mineral Resource and Mineral Reserve estimates.

Brio (2015–2107)

Brio drilling programs at the property from October 2015 through April 2017 implemented the QA/QC protocol for drill hole sampling described below. Using standard geologic practices in accordance with industry guidelines, Santa Luz personnel verified the accuracy and precision of its geochemical analyses for the drilling program by inserting standards of known metal content in the sample stream, by preparing and analyzing duplicate samples, and by reanalyzing approximately 5% of all sample pulps at a second laboratory. Details of the Santa Luz QA/QC procedure are as follows:

- One gold standard was submitted per 20 drill core samples.
- One blank was submitted per 20 drill core samples. The blanks are samples of coarsely-ground quartz submitted in sufficient mass to be subjected to the entire sample preparation process as would a drill core sample.
- One field duplicate (two quarter-split core samples are prepared from the sample interval) was submitted per 20 core samples.
- One pulp duplicate (two pulps prepared from one core sample) was prepared for every 20 core samples.
- 5% of all drill core sample pulps were submitted to a secondary laboratory for reanalysis (check assay).

The results of the standards, blanks, and duplicates are summarized in the following sections.

11.6.2 Certified Reference Material

Brio (2015–2017)

Nine certified reference materials (CRM) were used, including eight gold standards and one blank. The certified values, acceptable ranges of values (±2 Standard deviations [SD]), and other statistics for the CRMs are presented in Table 11-1.

Table 11-1: CRM Certified Values (2015–2017)

Gold Standard	Certified Grade (ppm)	Standard Deviation	Acceptable Range (ppm) ±2 SD		Source
G912-5	0.38	0.02	0.34	0.42	1[a]
G311-5	1.32	0.06	1.20	1.44	1[a]
G997-3	1.41	0.08	1.25	1.57	1[a]
G911-4	2.43	0.09	2.25	2.61	1[a]
G900-5	3.21	0.13	2.95	3.47	1[a]
G398-10	4.07	0.19	3.69	4.45	1[a]
G312-9	5.84	0.25	5.34	6.34	1[a]
G307-7	7.87	0.28	7.31	8.43	1[a]
Quartz 403/002P				<0.025[b]	2[c]

Notes: [a] GEOSTATS Pty Ltd, O'Connor, WA, Australia.
[b] 5x lower detection limit used as pass/fail.
[c] Química Brasileira Ltda, Belo Horizonte, Brazil.

There is a good correlation between the CRMs used and the average economic metal concentration in the drill samples. Only 0.3% of the gold concentrations in the drill samples lie above the highest gold concentration in the CRM suite (7.87 ppm Au), and the failure rate is low, at 0.39%).

The performance of the laboratory in analyzing the mineralized CRMs inserted in the analytical stream is excellent for gold. The overall bias of assays for all the CRMs is <1%, and the highest indicated bias is <-2.6%. Notably, that bias is negative, suggesting a slight underestimation of actual gold values; however controlled in -1SD//-2SD. Details of the laboratory's performance analyzing each CRM are provided in Table 11-2 and Figure 11-1.

Table 11-2: CRM QA/QC (2015–2017)

Standard ID	Gold (ppm)		N	Bias (%)	> ± 2SD	
	Expected	Found			No.	%
G912-5	0.38	0.37	90	-2.55	0	0
G311-5	1.32	1.33	19	0.62	0	0
G997-3	1.41	1.41	78	0.37	0	0
G911-4	2.43	2.47	29	1.53	0	0
G900-5	3.21	3.17	75	1.23	1	1.33
G398-10	4.07	4.06	66	-0.24	0	0
G312-9	5.84	5.74	74	-1.72	0	0
G307-7	7.87	7.89	81	0.29	1	1.23
Total			**512**		**2**	

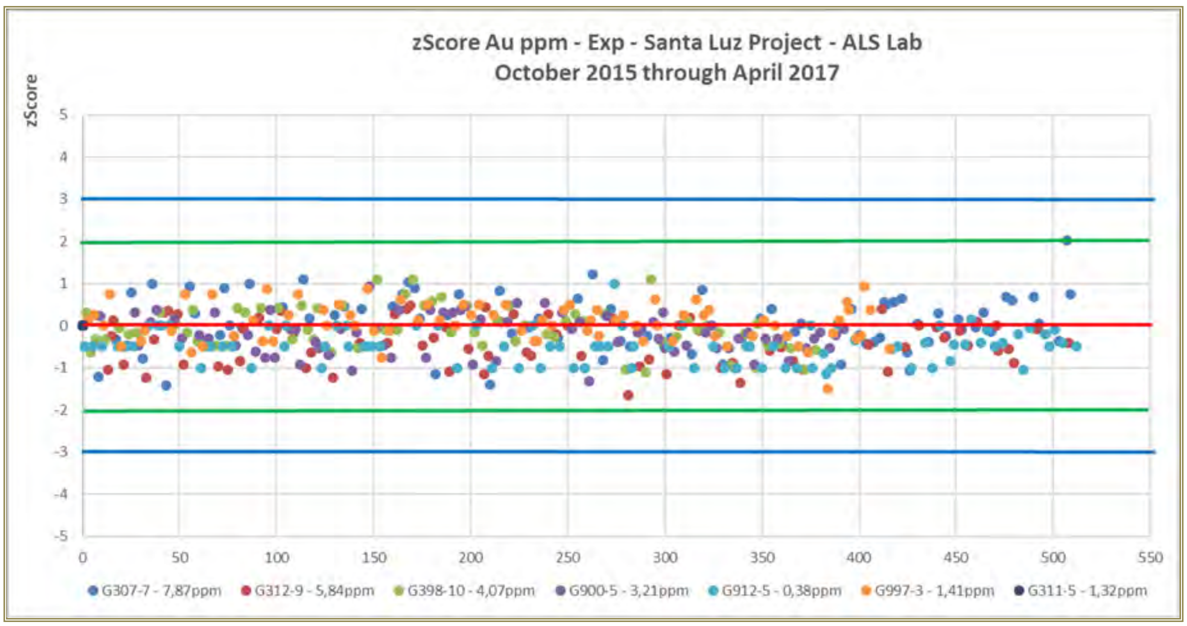


Figure 11-1: Score Chart of Au CRM Results (2015–2017)

11.7 Blanks

In processing and analyzing the blank standard for gold, the laboratory performed very well. Of the 533 blanks analyzed, no samples reported results above the lower detection limit for gold (100% approved), and the highest gold value reported was 0.014 g/t Au. No blanks returned results above the acceptable limits for gold, which is set at five times higher than the lower detection limit (LD = 0.025 g/t Au). Details of the performance of blanks are provided in Figure 11-2.

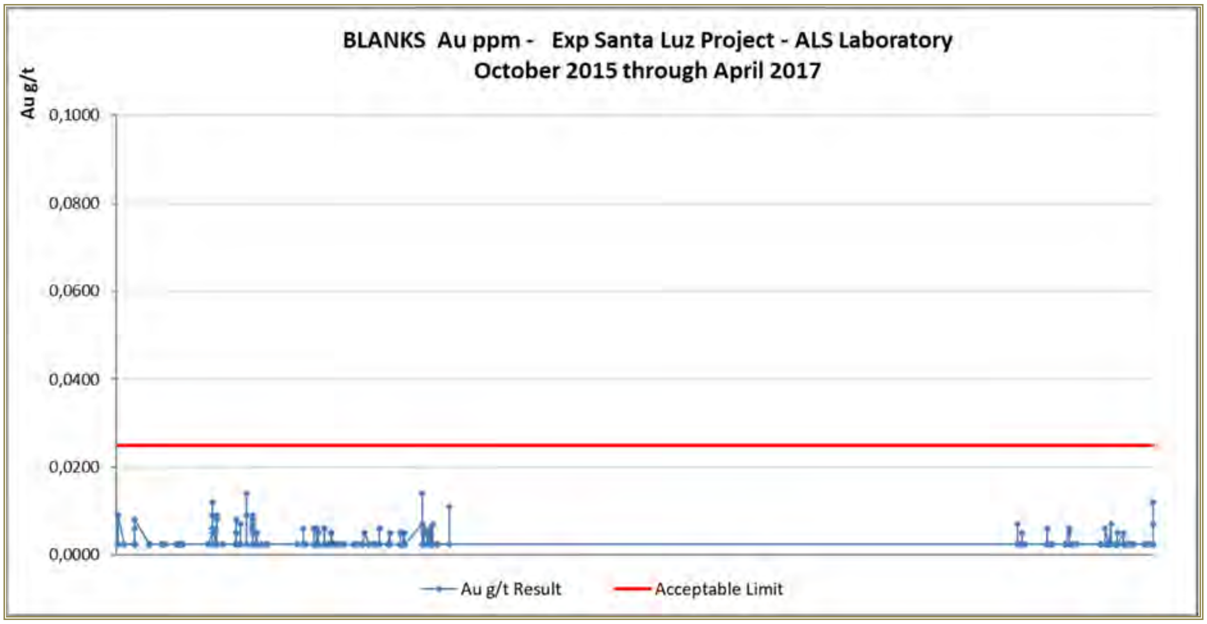


Figure 11-2: Blank Performance for Gold 2016–2017

11.8 Duplicates

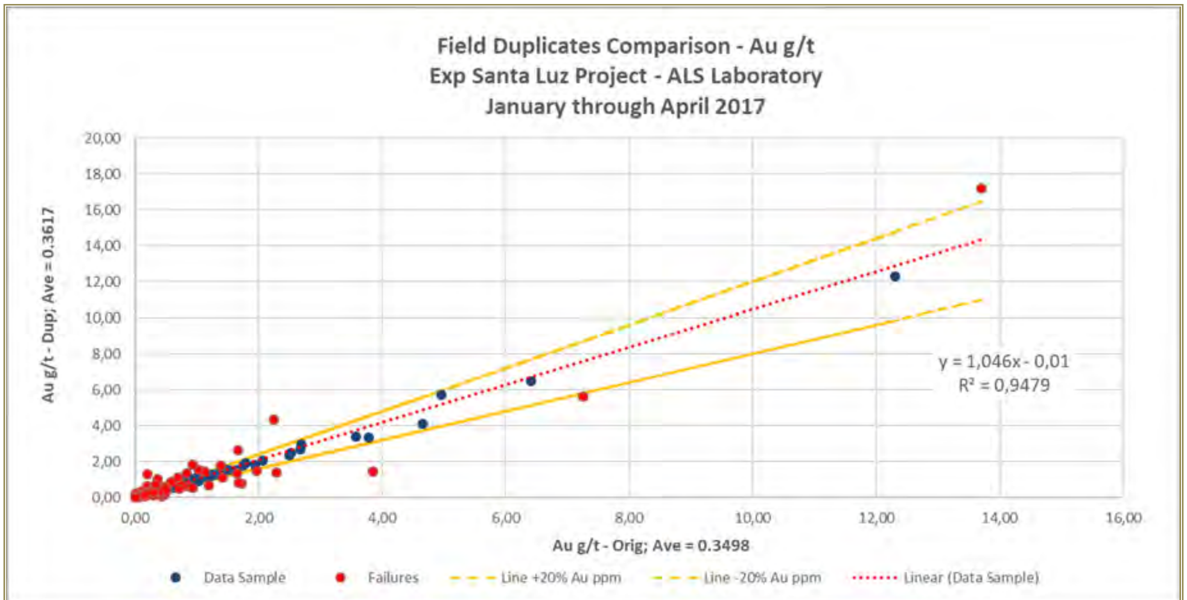
Both field duplicates and preparation duplicates were prepared for insertion into the sample stream. Field duplicates were taken by preparing two quarter-split samples from one sample interval and sending both to ALS labs for analysis. Preparation duplicates were prepared by producing two pulps from the drill core sample after it has been crushed to <2 mm. Field duplicate assays show moderate variability between the two quarter-split samples, even considering the low gold content which represents 63.89% of all data (144 pairs analyzed at 92 pairs with Au grade <0.10 g/t). More than a third of the pairs show <10% variation in returned gold grade, and 85% of all samples demonstrate 50% variation or less.

The fire assay duplicates (43 pairs analyzed in 2017), generated from pulp below -150 mesh, showed very minor variation in the sample pairs, better reproducibility, and low failures rate (11.63%) when compared to field (40.28%) and preparation duplicates (33.73%). No systematic biases were observed with average results, and the comparison of original and duplicates showed very good precision.

These results demonstrate the amount of natural heterogeneity on the scale of quarter-split core, and indicate that while the splitting and sampling are generally reliable, there is some natural heterogeneity on the scale of a quartered drill core sample. This indicates that the sample preparation method is doing an excellent job of homogenizing the samples.

Field duplicates were prepared for insertion into the sample stream only for the 2017 drilling program. No field duplicate was generated at the drilling program from October 2015 through May 2016. The same procedure was adopted for the fire assay duplicates, where some ore pulps were returned to ALS CHEMEX to reanalysis.

The 2017 drilling program processed 144 pairs of field duplicates, where 92 pairs (63.89%) represented a gold grade <0.10 g/t, which showed a greater number of failures between the pairs and moderate dispersion (SD average = 41.59%). The average grades of original vs. duplicates samples were reproducible, as shown in Figure 11-3. The relative differences (RD, %) showed a higher concentration of pairs with high dispersion and failures in the range <1.0 g/t Au; however, there is moderate variation in the ranges near the cut-off grade.



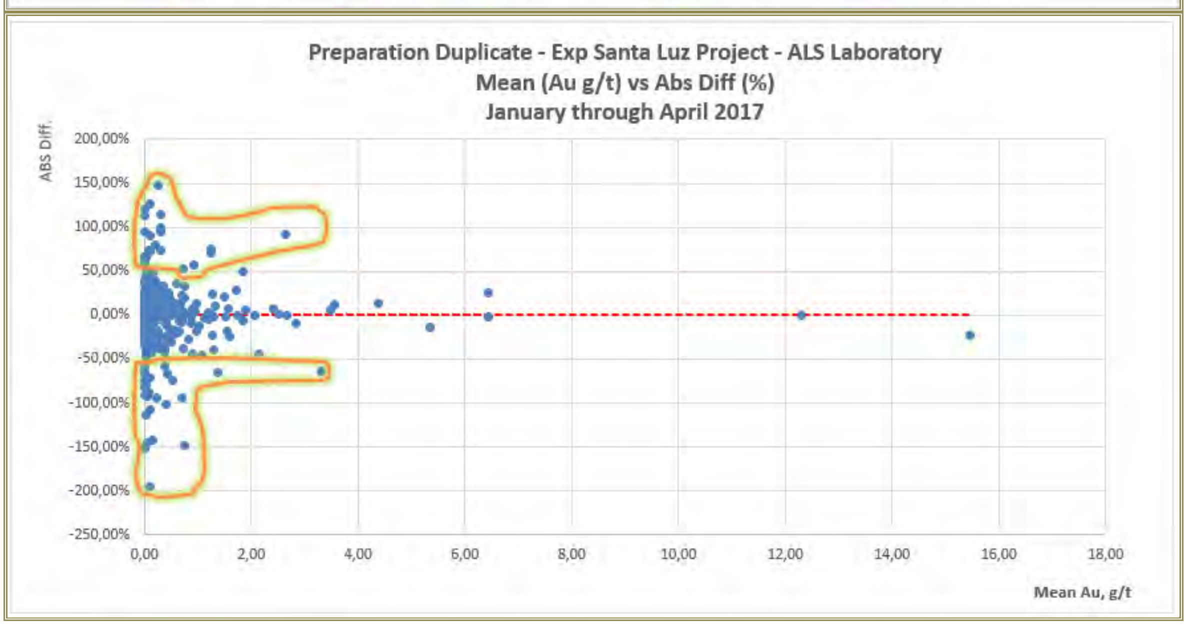


Figure 11-3: Field Duplicates Performance for Gold (2017)

Brio drilling programs at the property from October 2015 through April 2017 processed 510 pairs of field duplicates, where 317 pairs (62.16%) yielded gold grades <0.10 g/t, which showed a greater number of failures (33.73%) between pairs, and moderate dispersion (SD average = 36.06%). The average grades of original vs. duplicates samples were reproducible, as shown in Figure 11-4. The relative differences (RD, %) show a similar behaviour to fields duplicates: more failure rates and variabilities for low grades with relative differences in ±50%, yet showing better precision for grades >1.50g/t Au.

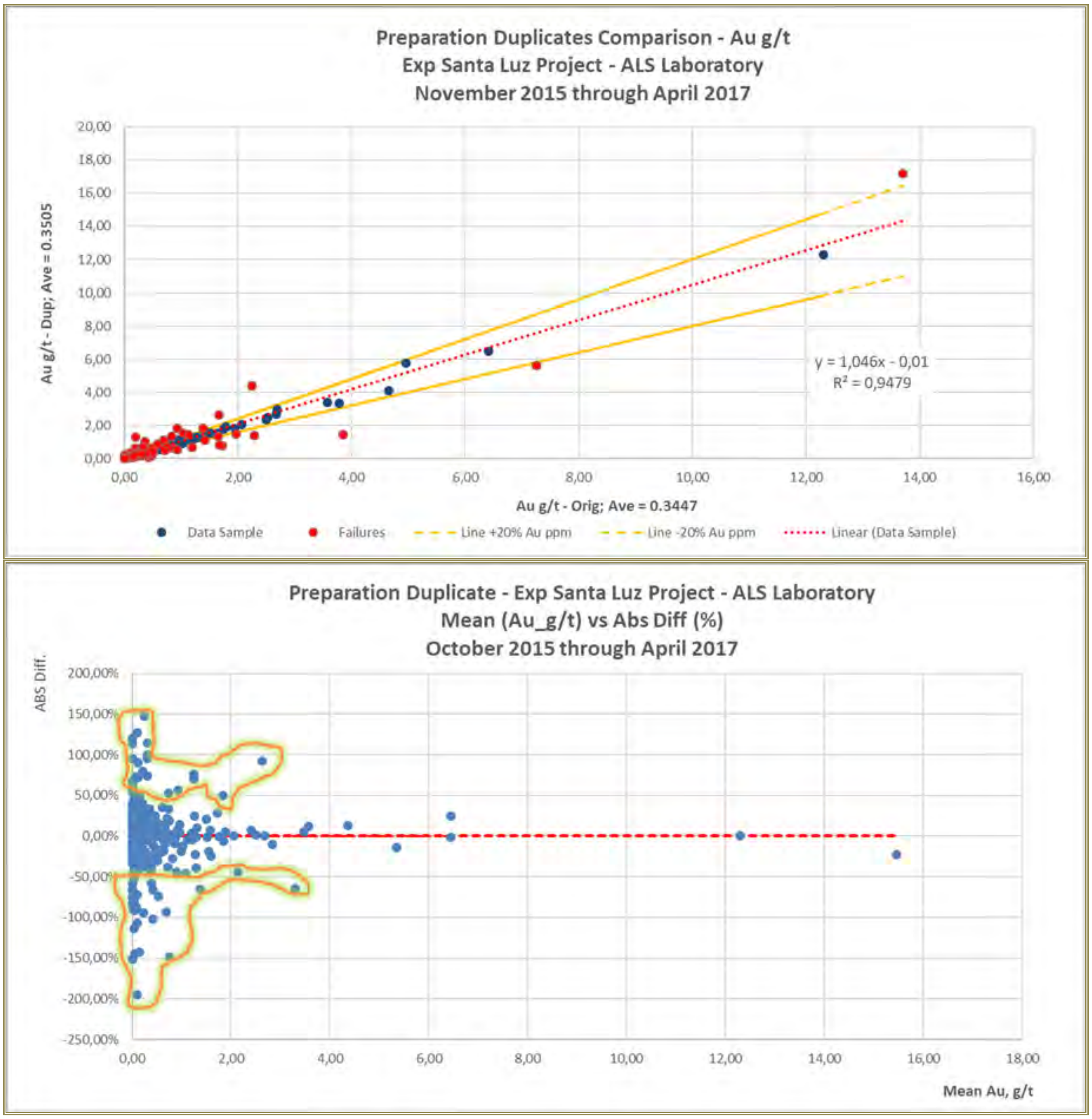


Figure 11-4: Field Duplicates Performance for Gold (2015–2017)

As mentioned above, there is better precision and less dispersion with very good accuracy for fire assay duplicates. Due to the greater liberation level of metal content from samples after pulverization at 95% finer than 150 mesh, it is possible to get a very good homogeneity between the sample pairs.

Only 43 pairs were analyzed in 2017; this protocol was not implemented by the exploration team during the 2015–2016 drilling program. A great number of pairs with low grade gold (27) was observed, representing 62.79% (Au <0.10 g/t). However, there is excellent precision and correlation (R^2 = 0.9991), a low failure rate (11.63%), and dispersion (SD average = 14.64%) compared with the other duplicate types.

The average results (original vs. duplicate) show excellent correlation, 0.58 g/t Au compared with 0.58 g/t Au, respectively (Figure 11-5). No systematic biases were evinced in the period.

Thompson´s Howarth criteria, which estimate the precision (P_c, %) for a specific metal content (C_{Au}, g/t), clearly illustrate the best precision as follows: FAAS_DUP (Pc = ±2.06% to 1.01 g/t Au); PREP_DUP (Pc = ±12.49% to 1.02 g/t Au), and FIELD_DUP (Pc = ±14.72% to 1.04 g/t Au), respectively, respectively (Figure 11-6).

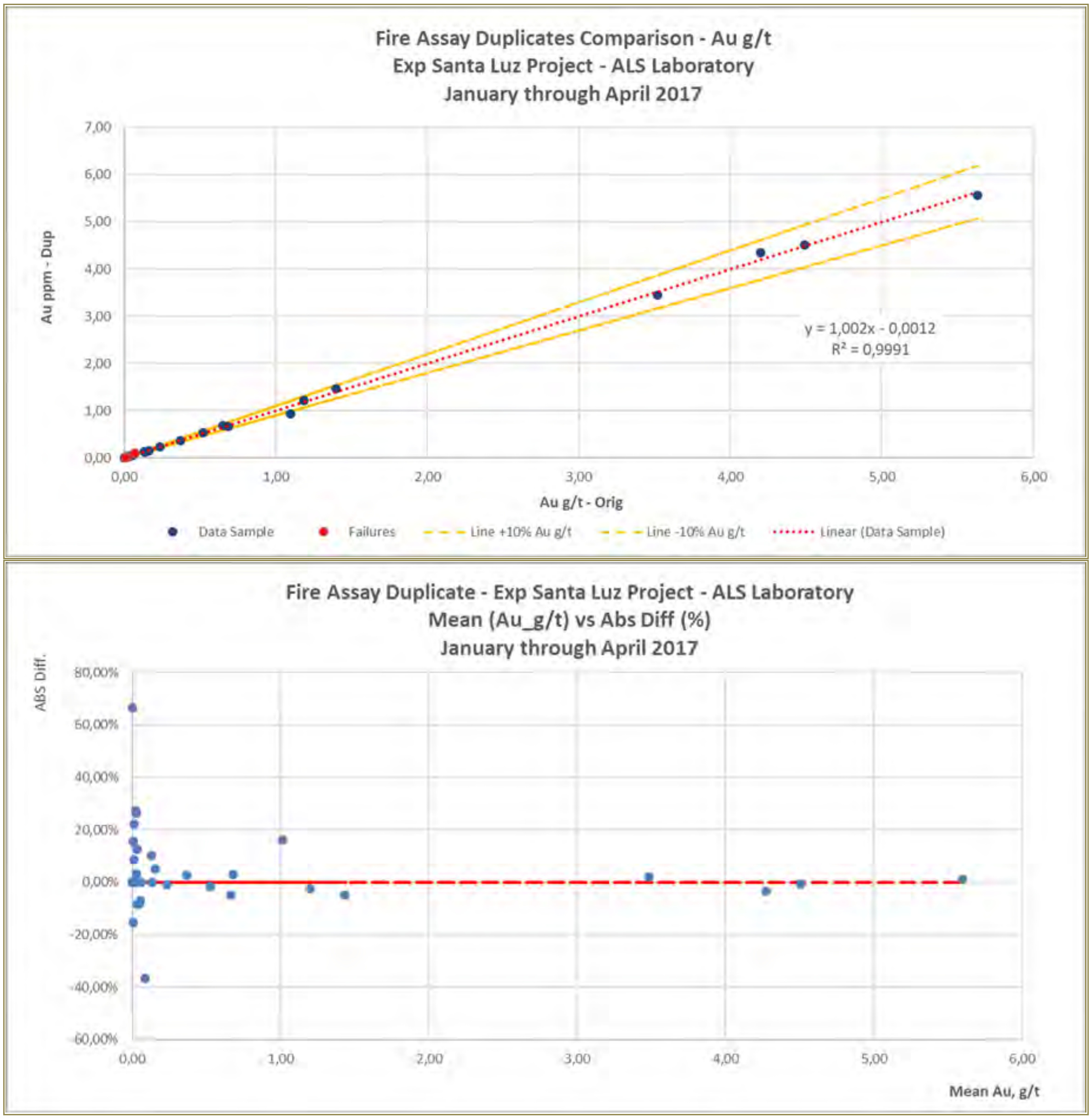


Figure 11-5: Fire Assay Duplicates Performance for Gold (2017)

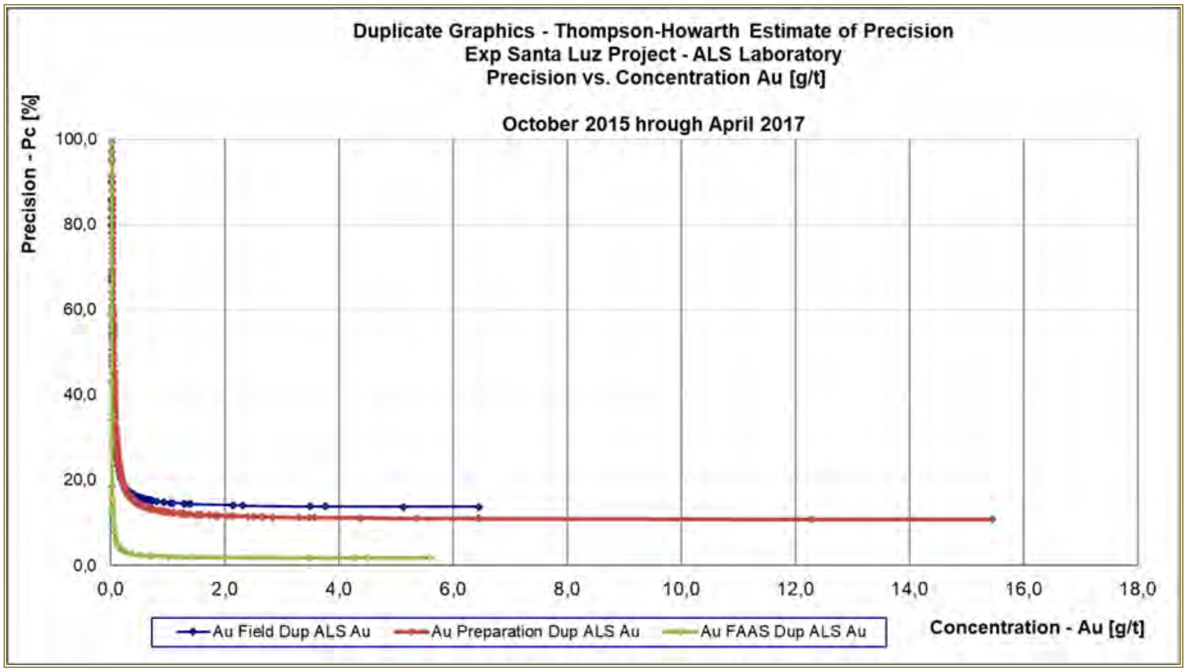


Figure 11-6: Thompson Howarth Criteria—Duplicates Performance for Gold (2015–2017)

11.9 Check Assays

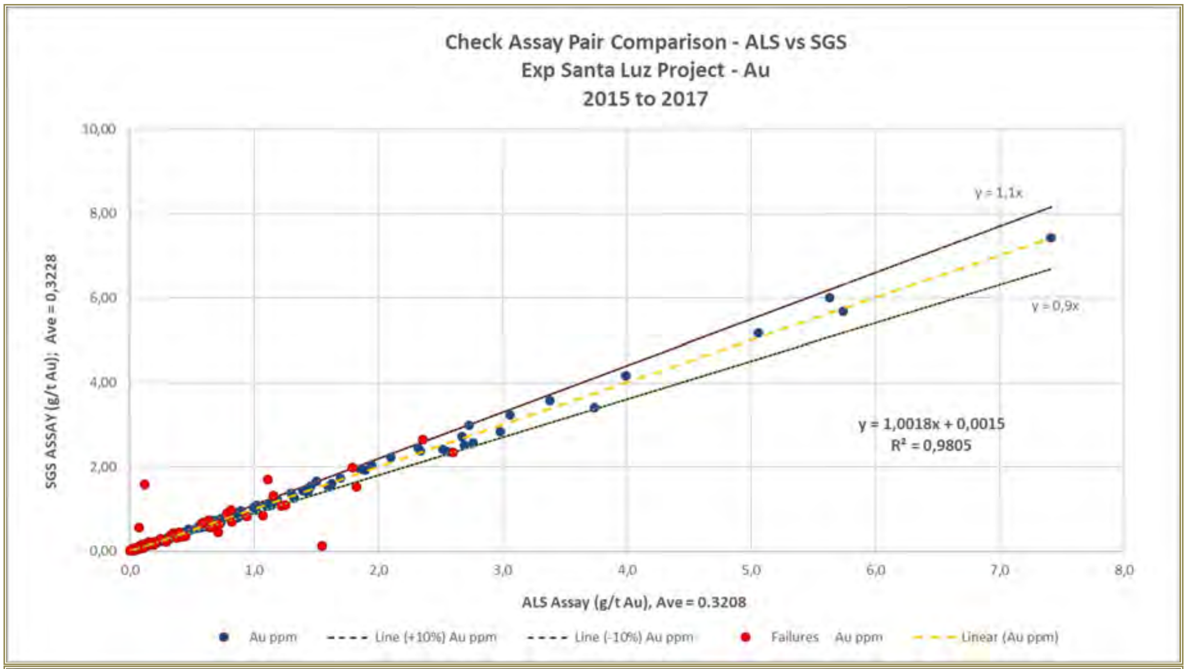
The primary laboratory for the Project in 2015 and 2017 was ALS Chemex Ltd. (ALS) in Vespasiano, Minas Gerais, with analysis at ALS Lima, Perú. Approximately 8.27% of all pulp samples were reanalyzed for gold at the secondary laboratory, SGS Geosol Laboratório Ltda. (SGS) in Vespasiano, Mina Gerais. Figure 11-7 and Figure 11-8 illustrate the check assay's performance.

The gold fire assay results show good agreement (R^2 = 0.9805), with 48.17% of failures (277 pairs) associated with low grade ranges (62.09% from 575 pairs analyzed). Both sets of analyses averaged 0.32 g/t Au. Variation in sample pairs is partially attributable to the precision of the fire assay method at the gold grades being compared.

On the chronological order of the pairs (Figure 11-7), it is observed that great dispersion is associated with low-grades samples (83.30% <0.50 g/t Au).

However, on the increasing order of the pairs (Figure 11-8), it is noted that there is a positive and systematic bias (Average, +20.05%—Moving Average, Red continuous line in the graphic) between the range from 0.01 g/t to 0.03 g/t Au, with significant dispersion for these pairs. For grades greater than 0.10 g/t Au, the biases are random, ±10% variance, with some pair outliers above 10% absolute relative difference.

The check assay results indicate good accuracy and precision in the gold analyses performed at the ALS lab.



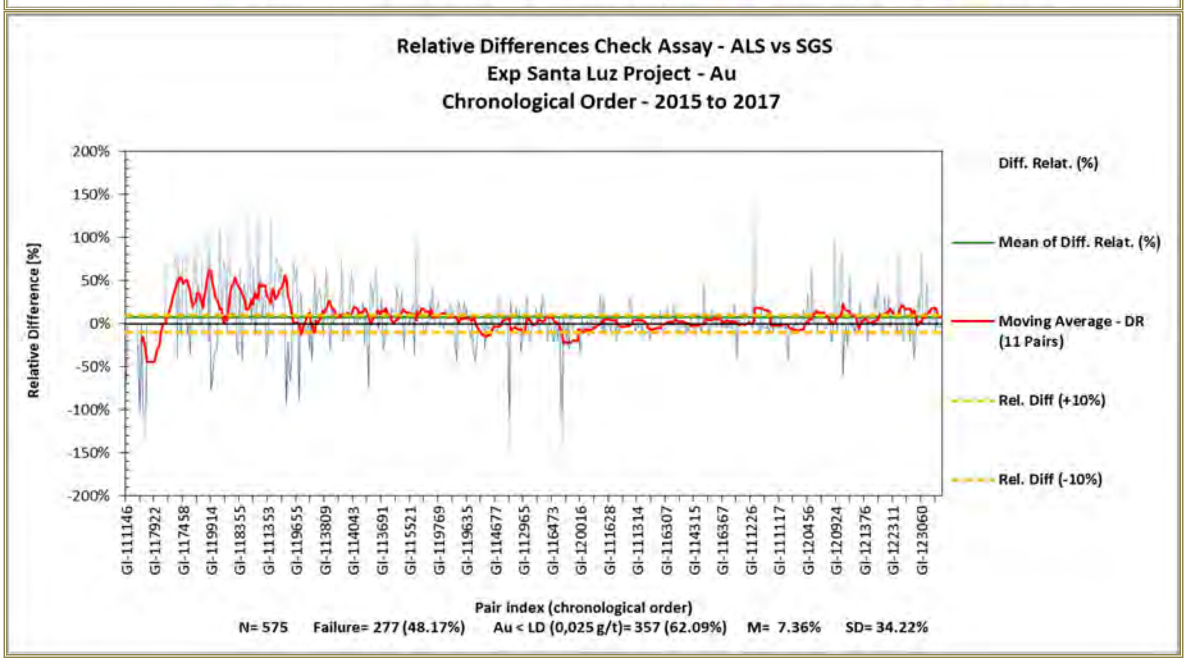


Figure 11-7: Comparison and Chronological Order for Check Assay Results (2015–2017)

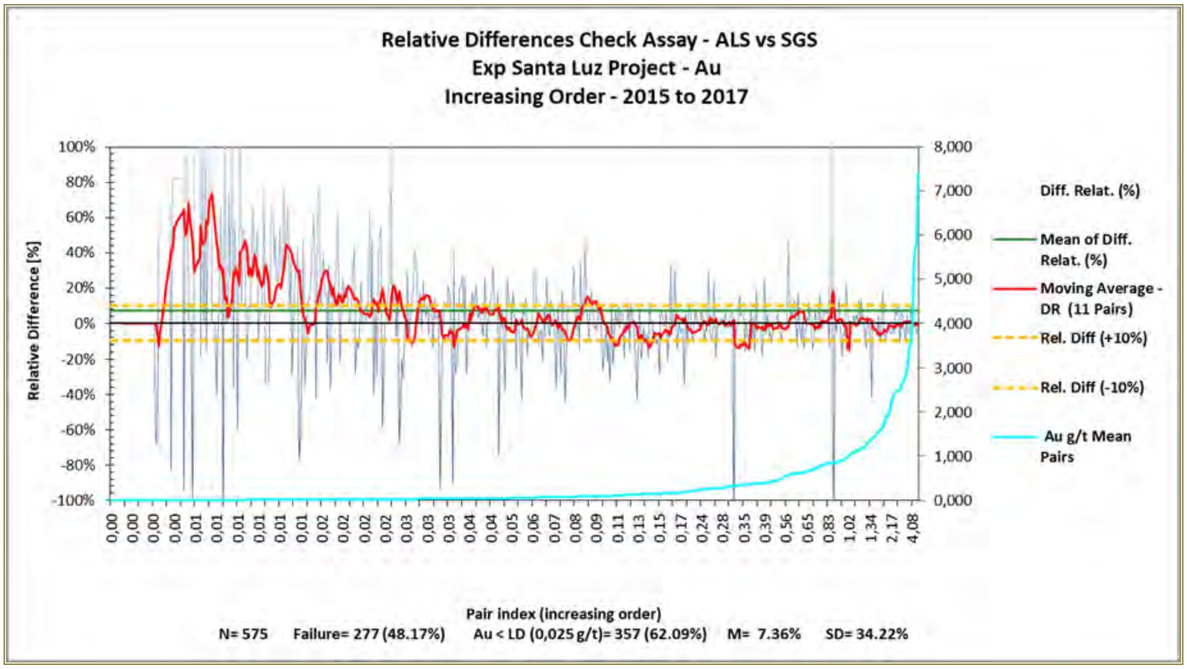


Figure 11-8: Increasing Order for Check Assay Results (2015–2017)

A QA/QC protocol for drill hole samples using standard geologic practices in accordance with industry guidelines was used at Santa Luz. The results verified the accuracy and precision of the geochemical analyses, and Santa Luz Project personnel believe that the drill results are acceptable to be used for Mineral Resource and Mineral Reserve estimation.

External consultants, such as RPA, recommended that the number of CRMs be limited to four in any drill program: one approximating the cut-off grade, two proximal to the average grade, and one representative of high-grade material at the Project site.

The results of the field duplicate analysis are consistent with the natural variability often seen in orogenic gold deposits; however, Equinox gold should continue to monitor results of high-grade duplicate samples and consider submitting larger half-core samples in place of quarter core samples.

11.10 Security

The Santa Luz mine site is surrounded by a security fence, with full-time security personnel controlling access at the gatehouse. At the drill site, samples were under the control of Santa Luz and drilling company employees. Sample handling procedures at the drill rig were as described in Section 10.5. Drilling company personnel delivered samples daily to Santa Luz personnel at the mine site sample processing facility. Only Santa Luz personnel and drilling contractor employees were authorized to be at the drill sites and in the sample processing facility. Core was stored within two purpose-built core sheds on site, both of which are locked at night.

After logging and sampling, the samples were prepared for shipment to ALS Chemex in Vespasiano, Minas Gerais. The samples were placed in large plastic shipping sacks, accompanied by documentation of the batch and samples, loaded onto a truck owned and driven by a locally-based transport company, and driven to the laboratory. Samples were under constant supervision during transport.

ALS Chemex placed large emphasis on confidentiality and data security. Industry standard chain-of-custody and work-order forms were used in sample transfers. Appropriate steps were taken to protect the integrity of samples at all processing stages. After analyses were completed, data were sent securely via electronic transmission to Santa Luz site personnel, accompanied by signed assay certificates. After sample analyses were completed, pulp and reject materials were returned to the Santa Luz site by the sample shipping contractor and stored in the core sheds.

Through the evidence of compliance with the good market practices of mining projects, as well as regulatory standards, sample preparation, analysis, and the security and confidentiality protocols (as designed and implemented), were deemed to be adequate and generally complied to industry standards, and were suitable for use in a Mineral Resource estimate.

12 DATA VERIFICATION

12.1 Audit of Drill Hole Database

In 2018 RPA conducted a series of verification tests on the drill hole database provided for Santa Luz. These tests included a search for missing information and tables, unique location of drill hole collars, and overlapping sample or lithology intervals. Empty tables were limited to lithology, alteration, and geotechnical results. No database issues were identified.

12.1.1 Assay Certificates

RPA compared 2% of assays within the complete Santa Luz drill hole database to assay certificates, including 24% of the C1 assay database. Santa Luz personnel provided certificates, not the original assay laboratory results. No major discrepancies or limitations were found.

12.2 Drill Core Review

The core from several drill holes was reviewed during RPA's 2018 site visit to confirm logging and sampling practices. Acceptable practices were noted.

Based on the 2018 analysis, RPA is of the opinion that the Santa Luz database complies with industry standards with no major discrepancies or limitations found, and the data are adequate for the purposes of Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

The ore in the Santa Luz deposit consists of essentially two types: carbonaceous and dacitic. The carbonaceous ore comprises approximately 52% of the deposit and the dacitic ore approximately 48% of the deposit. The carbonaceous ore exhibits strong adsorption of gold from solution (preg-robbing) when subjected to conventional cyanide leaching. The dacitic ore leaches well when subjected to conventional cyanide leaching, with little or no preg-robbing characteristics, except for some high-sulphide dacitic ore in the Antas 3 satellite deposit, which has significant sulphide-encapsulated gold. Metallurgical testwork has been primarily focused on the carbonaceous ore and pilot plant processing of blended carbonaceous and dacitic ores. Principal metallurgical parameters derived from the testwork are shown in Table 13-1.

Table 13-1: Principal Metallurgical Parameters

Parameter	Unit	Value
Strategy: Process Carbonaceous—Dacite blend		
Comminution		
Grind Size	80% passing, µm	75
Bond Mill Work Index	kWh/t	19
Abrasion Index		0.5
Reagent Consumptions		
Lime as CaO	kg/t ore	1.00
Cyanide as Na CN	kg/t ore	0.75
Kerosene	kg/t ore	2.00
Leaching		
Slurry Density	% solids	40
Conditioning Times		
Pre-Aeration	hours	2
Kerosene Conditioning	hours	4
Leach Time	hours	20
Recovery	%	84
Flotation (Antas 3 dacitic-high-sulphide ore)		
Regrind Size	80% passing, µm	45.0
Recovery	%	84.0
Concentrate Grade	g/t Au	80.0

Based on early testing (from 2006 through 2010), a plant was built and placed in operation in mid-2013. The plant process consisted of gravity recovery followed by flotation and CIL of the flotation concentrate with kerosene blanking of the organic carbon. The plant did not operate well and was shut down in September 2014 after 14 months of operation. The reasons for the failure were partly related to process difficulties and partly to mechanical problems.

Metallurgical testwork was re-initiated in 2014 and continued through 2017, with further optimization testwork in 2019–2020. The testwork has, from the start in 2006 to 2017, consisted of extensive mineralogical, diagnostic, comminution, gravity separation, flotation concentration, leaching of flotation concentrate, calcination of flotation concentrate, whole ore CIL, and whole ore RIL testing. Leach testwork in 2016-2017 and 2019-2020 has been carried out at the Santa Luz pilot plant to overcome the limitations of using batch tests to estimate plant recovery and gold loadings when processing preg-robbing ores.

Testing results are summarized below:

Mineralogical: The principal minerals of the deposit are quartz and feldspar. Carbonaceous ore contains approximately 1% TOC and dacitic ore approximately 0.2% TOC. Both carbonaceous and dacitic ore contain approximately 2% sulphides, approximately 80% to 90% of which is pyrite and 10% to 20% of which is arsenopyrite.

Diagnostic: Approximately 14% of the gold in both the carbonaceous and dacitic ore is locked in sulphides and 3% is locked in quartz; the remaining 80% is leachable with adequate blanking of the organic carbon. For the dacitic-high-sulphide ore in the Antas 3 deposit, approximately 50% of the gold is locked in sulphides and 4% is locked in quartz.

Comminution: Both carbonaceous and dacitic ores are very hard, with a Bond Work Index of about 19 kWh/t, though they are amenable to semi-autogenous grinding (SAG) milling. Both ores are also very abrasive with an Abrasion Index of approximately 0.5.

Gravity Separation: Results indicate that approximately 35% of the gold can be recovered into approximately 1.5 weight-percent (wt%) of the ore at a grade of approximately 35 g/t Au, and that the concentrate is amenable to intensive cyanide leaching.

Flotation: Flotation of carbonaceous and dacitic ores results in approximately 85% recovery of the gold into approximately 12.5% of the ore at a grade of approximately 10 g/t Au.

Leaching of Flotation Concentrate: CIL of flotation concentrate preceded by kerosene blanking, extracts approximately 85% of the gold. Combined recovery of flotation with leaching of the flotation concentrate is approximately 72%.

Calcination of Flotation Concentrate: Calcining of flotation concentrate at 600°C to 750°C removes practically all organic carbon and sulphur, and approximately 20% of the arsenic. Cyanide leaching of the calcined flotation concentrate extracts about 95% of the gold, to give around 81% overall recovery. However, calcination is considered impractical because of excessive elutriation of fines, the necessity of recovering arsenic fume, and the necessity of converting sulphur dioxide to sulphuric acid.

Whole Ore CIL and RIL of Carbonaceous Ore: Success of leaching of carbonaceous ore is strongly dependent on blanking process efficacy, depends on the quantity of kerosene used and the mixing procedures. Gold recovery also depends on the ability of the adsorbent to retain its activity in the presence of residual kerosene. Bottle-roll tests indicate that by using 2 kg/t of illuminant kerosene, together with good kerosene mixing and adequate conditioning time, between 66% and 91% of the gold can be extracted. Based on both bottle-roll and pilot-plant tests, average recovery of carbonaceous ore is 82.9%.

Whole Ore CIL and RIL of Dacitic Ore: Most dacitic ore leaches well with bottle-roll tests extracting between 78% and 92% of the gold using 0.5 kg/t kerosene with good kerosene mixing. Based on both bottle-roll and pilot-plant tests that excluded dacitic-high-sulphide samples, average recovery of dacitic ore from the C1 deposit is 84.7%. Dacitic-high-sulphide ore from the satellite Antas 3 deposit containing more than 1,500 g/t As is refractory to cyanide leaching, averaging 36.7% recovery. Dacitic ore from the satellite Antas 3 deposit containing <1,500 g/t As, cyanide leaches reasonably well, averaging 80.6% recovery. Some dacitic ore from the C1 deposit can also be somewhat high-sulphide; however, there is no direct link between recovery and arsenic content for the C1 ore.

High-TOC Carbonaceous Ore and Blending Low and High Recovery Ores: Carbonaceous ores containing more than 2% TOC benefit from adding 4 kg/t kerosene, rather than the standard 2 kg/t kerosene. Blending of low-recovery carbonaceous ore with higher recovery dacitic ore is a viable option, with overall recovery similar to the weighted recovery of each fraction. Grinding the ores finer than 80% passing (P_{80}) 75 µm is not beneficial.

Santa Luz Pilot-Plant Findings:

- Pilot plant testwork shows that for realistic gold loadings on the carbon or resin adsorbent, RIL gives higher gold recovery than CIL, even if a mop-up stage is used ahead of CIL.
- Conditioning of the ore prior to leaching is critical: using two hours of pre-aeration and four hours of kerosene conditioning stabilizes and improves recovery, it also results in cleaner loaded resin.
- Due to the presence of kerosene, excessive frothing occurs with air addition to the leach tanks when using resin, leading to the conclusion that it will be necessary to use oxygen rather than air in the industrial plant.
- Gold recovery in the first four hours of leaching is approximately 75%, with an incremental recovery of only approximately 0.5% in the last four hours of a 24-hour leach time at a loaded carbon gold grade of 500 to 900 g/t.

Santa Luz Pilot Plant RIL Optimization: Stage 17 optimization testwork in conjunction with Mintek established that 84% Au recovery could be achieved using 1.56 kg/t kerosene with blended carbonaceous/dacitic ore at gold loadings of up to 2,000 g/t with around 10,500 g/t Cu.

Gold Recovery from Resin: Mintek testwork on loaded resin from Stage 17 identified:

- Improved base-metal stripping by acid washing and recycling 60°C acid for two hours
- Potential to further reduce base-metal concentration in recycled 60°C acid thiourea eluate by diverting first portion of strip solution that is high in copper and low in gold. Much of the copper precipitates if the solution is allowed to cool. Solution can be reused in the following elution cycle after filtering off the copper-rich precipitate.
- Minimal base metals plating onto cathode along with gold at copper concentrations up to at least 1,000 mg/L.

Detoxification: The cyanide concentration of leached slurry is reduced to <50 mg/L weak-acid dissociable (WAD) cyanide (CN) within 60 minutes using the SO_2/air process, and the soluble arsenic concentration post cyanide detoxification is <0.1 mg/L, negating the need for arsenic precipitation.

13.2 General and Sample Locations

The testwork that has been done over the years, including the time of testing, the test laboratory, the location of the sample, the sample type, the sample grade, and the type of testing, is summarized in Table 13-2, Table 13-3 and Table 13-4.

Table 13-2: Leach and Diagnostic Testwork Conducted, 2005—2015

Period	Laboratory	Sample Location	Sample Type	Sample Grade (g/t) Au	Ag	Cu	As	(%) S	TOC	Test Type: Mineralogy	Comminution	Gravity	Diagnostic Leaching	Flotation	Calcine of Flot. Conc. + CIL of Calcine	Bottle-Roll: CIL of Flot. Conc.	CIL of Whole Ore
2005-2006	Univ. São Paulo	C1	carbonaceous, drill core	2.3	3	1,150	-	0.5	-	X	-	-	-	-	-	-	X
			dacite, drill core	5.0	1	160	-	0.8	-	X	-	-	-	-	-	-	X
		C1	shallow-depth ore	1.7	-	-	-	-	-	-	-	-	-	-	-	-	X
			intermediate-depth ore	2.2	-	-	-	-	-	-	-	-	-	-	-	-	X
			deep ore	2.8	-	-	-	-	-	-	-	-	-	-	-	-	X
2006	Knelson Research	C1	carbonaceous	1.4	-	-	-	-	-	-	-	X	-	-	-	-	-
2006	HDA Services	C1			-	-	-	-	-	-	X	-	-	-	-	-	-
2007	Amdel	C1	fresh rock, 50% carb; 50% dac. RC cuttings	2.0	-	-	1,200	1.5	0.8	-	-	-	X	X	-	X	X
		C1	semi-oxidized, drill core	3.3	-	-	820	0.8	0.5	-	X	-	X	X	-	-	-
		Antas 3	RC cuttings	1.2	-	-	150	1.2	1.5	-		-			-	-	-
		Mansinha	drill core	2.3	-	-	1,240	0.8	0.1	-	X	-	X	X	-	-	-
		Heap leach tail	-10 mm bulk	1.0	-	-	480	0.1	0.1	-	X	-	X	X	-	-	-
2008	Amdel	C1	drill core	2.3	-	-	-	-	-	-	-	X	-	X	-	X	-
		Mansinha	-5 mm crushed	2.5	-	-	-	-	-	-	-	X	-	X	-	-	-
		Antas	-5 mm crushed	2.2	-	-	-	-	-	-	-	X	-	X	-	-	-
2009-2010	Funmineral Pilot Plant	Blend 1A	30% carb; 70% dac, RC cuttings	1.9	-	-	-	-	-	-	-	X	-	X	-	X	-
		Blend 2A	50% carb; 50% dac, RC cuttings	1.8	-	-	-	-	-	-	-	X	-	X	-	X	-
		Blend 3A	70% carb; 30% dac, RC cuttings	0.6	-	-	-	-	-	-	-		-	X	-	X	-
		Blend 1B	30% carb; 70% dac, drill core	1.6	-	-	-	-	-	-	-	X	-	X	-	X	-
		Blend 3B	70% carb; 30% dac, drill core	1.3	-	-	-	-	-	-	-	X	-	X	-	X	-
2014	Hazen		Flotation concentrate	26.1	8	-	4,690	-	5.8	-	-	-	-	-	X	-	-
2015	Univ. São Paulo	C1	High S, part of 17-t carb. sample	4.0	-	110	620	1.0	1.2	X	X	X	X	X	-	-	-
	SGS, Belo Horizonte	C1	Medium S, part of 17-t carb. sample	1.7	-	160	890	0.5	0.7	X	X		X	X	-	-	-
	FL Smidth Knelson	C1	Low S, part of 17-t carb. sample	1.2	-	130	630	0.3	0.7	X	X	X	X	X	-	-	-
	HDA Servicios	C1	Dacite	2.1	-	40	630	0.1	0.1	-	X	-	-		-	-	X
2015	Funmineral Pilot Plant	C1	High S, part of 17-t carb. sample	4.0	0	130	530	1.0	1.2	-	-	-	-	X	-	X	X
		C1	Medium S, part of 17-t carb. sample	1.9	0	160	800	0.5	0.7	-	-	-	-	X	-	X	X
		C1	Low S, part of 17-t carb. sample	1.3	0	160	560	0.3	0.6	-	-	-	-	-	-	-	X
2015	Gekko Systems	C1	High S, part of 17-t carb. sample	4.2	-	-	-	-	-	-	-	-	-	-	-	-	X
		C1	Low S, part of 17-t carb. sample	1.3	-	-	-	-	-	-	-	-	-	-	-	-	X

Table 13-3: Leach and Diagnostic Testwork Conducted (2016–2017)

				Sample Grade						Test Type					
				(g/t)				(%)				Bottle-Roll			
Period	**Laboratory**	**Sample Location**	**Sample Type**	**Au**	**Ag**	**Cu**	**As**	**S**	**TOC**	**Comminution**	**Diagnostic Leaching**	**CIL of Whole Ore**	**RIL of Whole Ore**	**CN Detox. As Precip.**	**Flotation & Flot. Conc. Processing**
2016	Hazen and RDI	C1	High S, part of 17-t carbonaceous sample	3.8	2	100	500	0.3	1.7	-	-	X		-	-
		C1MP028	Drill core (carbonaceous)	1.9	0		400		1.2	-	-	X	X	-	-
		C1MP031	Drill core (carbonaceous)	1.3	0	100		1.1	1.4	-	-	X	X	-	-
2017	Hazen	C1MP028	Drill core (carbonaceous)	-	-	-	-	-	-	X	-	-	-	-	-
		C1MP028	Drill core (carbonaceous)	-	-	-	-	-	-	X	-	-	-	-	-
		C1MP027a	Drill core (dacite) waste rock	-	-	-	-	-	-	X	-	-	-	-	-
		C1MP027a	Drill core (dacite) waste rock	-	-	-	-	-	-	X	-	-	-	-	-
		C1MP027a	Drill core (dacite, semi-high-sulphide)	1.4	-	-	6,980		-	-	-	X	X	X	
		A3MP010a	Drill core (dacite, high-sulphide)	2.3	-	-	4,080		-	-	-	-	-	-	X
2016–2017	CSIRO	C1MP029	Drill core (carbonaceous)	1.3	1	100	400		1.5	-	X	X	X	-	-
2016–2017	SGS & Testwork Desenvolvimento de Processo	C1MP023	Drill core (carbonaceous)	3.7	<3	173	610	2.1	1.0	-	-	X	X	-	-
		C1MP034	Drill core (carbonaceous)	1.3	<3	114	340	1.4	0.7	-	-	X	X	-	-
		C1MP033	Drill core (carbonaceous)	1.7	<3	96	350	1.7	1.8	-	-	X	X	-	-
		C1MP006	Drill core (carbonaceous)	1.1	<3	229	1,210	0.9	1.3	-	-	X	X	-	-
		C1MP032	Drill core (carbonaceous)	1.3	<3	101	210	1.5	1.4	-	-	X	X	-	-
		C1MP013	Drill core (carbonaceous)	2.9	<3	106	1,980	2.2	0.7	-	-	X	X	-	-
		C1MP016	Drill core (carbonaceous)	2.1	<3	132	1,440	1.3	1.2	-	-	X	X	-	-
		C1MP040	Drill core (carbonaceous)	3.0	<3	141	340	1.7	1.8	-	-	X	X	-	-
		C1MP029a	Drill core (carbonaceous)	3.3	<5	64	810	2.8	0.9	X	X	X	X	-	-

Table 13-3, continues…

Table 13-3 (cont'd)

Period	Laboratory	Sample Location	Sample Type	Sample Grade						Diagnostic Leaching	Bottle-Roll		Pilot-Plant	
				(g/t)			(%)				CIL of Whole Ore	RIL of Whole Ore	CIL of Whole Ore	RIL of Whole Ore
				Au	Ag	Cu	As	S	TOC					
2016–2017	SGS & Testwork Desenvolvimento de Processo	C1MP036	Drill core (dacite)	2.1	<3	57	<500	0.5	0.10	-	X	X	-	-
		C1MP023	Drill core (dacite)	2.4	<3	43	1,430	2.5	0.40	-	X	X	-	-
		C1MP011	Drill core (dacite)	1.2	<3	82	1,200	0.1	0.10	-	X	X	-	-
		C1MP021	Drill core (dacite)	0.7	<3	90	1,090	0.7	0.10	-	X	X	-	-
		C1MP028	Drill core (dacite)	1.4	<5	33	7,800	0.6	0.20	-	X	X	-	-
		C1MP041	Drill core (dacite)	0.8	<5	18	1,530	0.2	0.00	-	X	X	-	-
		C1MP030	Drill core (dacite)	1.1	<5	36	<500	0.9	0.40	-	X	X	-	-
		C1MP040	Drill core (dacite)	2.6	<5	33	790	1.7	0.30	-	X	X	-	-
		C1MP002	Drill core (dacite)	1.2	<5	81	1,000	0.7	0.20	-	X	X	-	-
		C1MP007	Drill core (dacite)	0.7	<5	48	820	0.2	0.30	-	X	X	-	-
		A3MP024	Drill core (dacite, non-high-sulphide)	2.3	<5	89	880	0.0	0.10	-	X	X	-	-
		A3MP019	Drill core (dacite, non-high-sulphide)	1.4	<5	23	880	0.8	0.30	-	X	X	-	-
		A3MP021	Drill core (dacite, non-high-sulphide)	2.3	<5	29	910	0.4	0.20	-	X	X	-	-
		A3MP003	Drill core (dacite, non-high-sulphide)	1.3	<5	38	<500	0.4	0.40	-	X	X	-	-
		A3MP023	Drill core (dacite, non-high-sulphide)	1.0	<5	<10	820	0.3	0.10	-	X	X	-	-
		A3MP005	Drill core (dacite, semi-high-sulphide)	2.18	<5	<10	2,900	0.8	0.06	-	X	X	-	-
		A3MP009	Drill core (dacite, semi-high-sulphide)	1.88	<5	22	1,100	0.4	0.37	-	X	X	-	-
		A3MP011	Drill core (dacite, semi-high-sulphide)	1.63	<5	17	2,120	0.4	0.09	-	X	X	-	-
		A3MP010	Drill core (dacite, high-sulphide)	1.74	<5	13	4,040	0.2	0.08	X	X	X	-	-
		A3MP018	Drill core (dacite, high-sulphide)	4.00	<5	<10	6,200	1.0	0.08	X	X	X	-	-
		A3MP008-A	Drill core (dacite, high-sulphide)	2.27	<5	<10	4,000	0.5	0.05	-	X	X	-	-
		A3MP008-B	Drill core (dacite, high-sulphide)	2.83	<5	84	5,400	1.0	0.05	-	X	X	-	-
		A3MP008-C	Drill core (dacite, high-sulphide)	1.59	<5	18	2,400	0.4	<0.05	-	X	X	-	-

Table 13-4: Santa Luz Pilot Plant and Associated Testwork Conducted (2016–2020)

Period	Laboratory	Sample Location	Sample Type	Sample Grade (g/t)				Sample Grade (%)		Diagnostic Leaching	Bottle-Roll		Pilot-Plant	
				Au	Ag	Cu	As	S	TOC		CIL of Whole Ore	RIL of Whole Ore	CIL of Whole Ore	RIL of Whole Ore
2016	Santa Luz Pilot Plant	C1 (Stage 1)	Ore stockpile, carbonaceous, low TOC	1.45	-	-	-	0.45	0.63	-	-	-	X	-
		C1 (Stage 2)	Ore stockpile, carbonaceous, low TOC	1.63	-	-	-	0.48	0.77	-	-	-	X	-
		C1 (Stage 3)	Ore stockpile, carbonaceous, high TOC	1.96	-	-	-	0.69	1.35	-	-	-	X	-
		C1 (Stage 4)	Ore stockpile, carbonaceous, high TOC	1.70	-	-	-	0.46	1.35	-	-	-	X	-
2017		C1 (Stage 5)	Ore stockpile carbonaceous, high TOC	1.96	-	-	630	0.81	1.37	-	-	-	X	-
		C1 (Stage 6)	Ore stockpile carbonaceous, low TOC	1.20	-	-	470	0.80	0.90	-	-	-	-	X
		C1 (Stage 7)	Ore stockpile dacite	1.10	-	-	540	0.01	0.15	-	-	-	-	X
		C1 (Stage 8)	Ore stockpile carbonaceous, high TOC	2.36	-	-	-	0.93	1.32	-	-	-	-	X
		C1 (Stage 9)	Ore stockpile carbonaceous, high TOC	2.62	-	-	-	0.96	1.31	-	-	-	-	X
		C1 (Stage 10)	Ore stockpile carbonaceous, high TOC	2.51	-	-	500	0.96	1.29	-	-	-	-	X
		C1 (Stage 11)	Ore stockpile carbonaceous/dacite blend	1.49	-	-	-	0.64	0.66	-	-	-	-	X
		C1 (Stage 12)	Ore stockpile carbonaceous/dacite blend	1.47	-	-	-	0.67	0.65	-	-	-	-	X
		C1 (Stage 13)	Ore stockpile carbonaceous/dacite blend	2.18	-	-	-	0.84	1.01	-	-	-	-	X
		C1 (Stage 14)	Ore stockpile carbonaceous/dacite blend	1.23	-	-	-	0.82	0.89	-	-	-	-	X
		C1 (Stage 15)	Ore stockpile carbonaceous/dacite blend	2.01	<1	<500	-	-	-	-	X	X	-	
2019		C1 (Stage 16)	Ore stockpile carbonaceous/dacite blend	1.93	-	-	-	-	-	-	-	-	-	X
	Mintek Stage 1	C1	Ore stockpile carbonaceous/dacite blend	2.01	<1	-	-	-		-	-	-	-	
		C1 (Stage 17)	Ore stockpile carbonaceous/dacite blend	1.96	-	-	-	-		-	-	-	-	X
	Mintek Stage 2		Stage 17 loaded resin		-	-	-	-		-	-	-	-	

The following is a brief descriptions of the laboratories that have conducted testwork:

- University of São Paulo: The LCT laboratory, a department of the University of São Paulo that provides metallurgical research services.
- Knelson Research: The research services division of the company manufacturing Knelson gravity concentrators; it has since been acquired by the FL Smidth company.
- HDA Servicios: An organization led by a University of São Paulo metallurgy professor Homero Delboni, Jr., an expert in comminution.
- Amdel: The Western Australia branch of Amdel Mineral Laboratories, formerly Independent Metallurgical Laboratories Pty Ltd, providing metallurgical testwork services.
- Funmineral Pilot Plant: The Fundo de Fomento a Mineração, a Brazilian government metallurgical testwork organization based in Goiânia, a city in the state of Goiâs; the facility includes a pilot plant that was used by Yamana in two separate campaigns, one in 2009–2010 and one in 2015.
- SGS, Belo Horizonte: SGS Geosol Laboratories Ltda, a testwork laboratory located on the outskirts of Belo Horizonte, a branch of the SGS company.
- FL Smidth Knelson: The Knelson division of FL Smidth, a major international process equipment supplier; FL Smidth acquired the Knelson company in 2011.
- Gekko Systems: The Western Australia division of the Gekko company that provides research services and ore processing equipment; the work was done in conjunction with Curtin University in Western Australia.
- Hazen: The Denver-based employee-owned metallurgical testwork laboratory.
- RDI: The Denver-based metallurgical laboratory led by Deepak Mulhotra.
- CSIRO: The Western Australia-based Commonwealth Scientific and Industrial Research Organization (CSIRO) Mineral Laboratories, a branch of the government organization providing research services.
- Testwork Desolvimento de Processo: A metallurgical testwork laboratory in Belo Horizonte, the director of which is Walter de Moura.
- SGS, Lakefield: SGS testwork laboratory located in Lakefield, Ontario, Canada, a branch of the SGS company.
- Santa Luz Pilot Plant: The pilot plant at the Santa Luz Mine.
- Mintek: The semi-government test laboratory in South Africa that originally developed the Minix gold selective resin selected for gold recovery.

Regarding sample location, most of the samples for the earlier testwork are designated by deposit location; those for more recent tests are designated by drill hole. Figure 13-1 shows the deposit locations and the location of the drill holes tested.

The samples used for the Santa Luz pilot plant tests were obtained from stockpiles of ore generated by earlier mining.

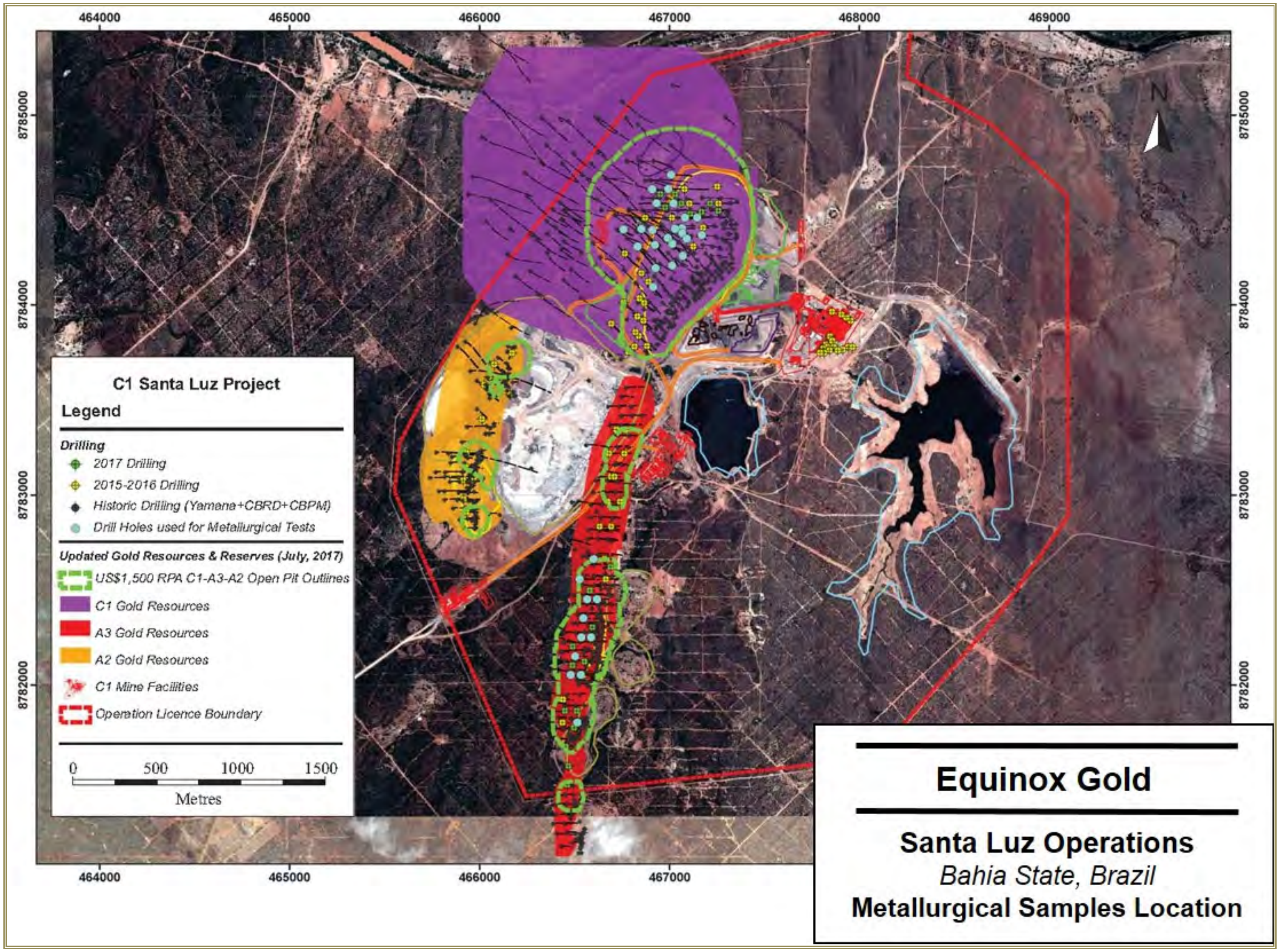


Figure 13-1: Metallurgical Sample Locations

13.3 Mineralogy

Mineralogical investigations have resulted in the following findings regarding the carbonaceous and dacitic ore:

Carbonaceous Ore

- Major minerals are quartz, plagioclase feldspar (albite), and muscovite.
- Minor minerals are dolomite, magnetite, pyrite, arsenopyrite, pyrrhotite, galena, sphalerite, chalcopyrite, antimonite, and graphitic carbon.
- On average, pyrite comprises approximately 2 wt% of the ore occurring as crystals ranging from 5 µm to 90 µm.
- On average, arsenopyrite comprises approximately 0.2 wt% of the ore, occurring primarily as inclusions in pyrite.
- Organic carbon, ranging from amorphous to graphitic, comprises approximately 1 wt% of the ore and occurs as very fine dispersions of <10 µm.
- Copper occurs as chalcopyrite, either as separate grains or in association with pyrite or arsenopyrite.
- Gold occurs as native gold with particle sizes typically ranging from 1 µm to 25 µm, mainly associated with pyrite and arsenopyrite. Some of the gold is attached to iron oxides on the borders of pyrite grains.

Dacitic Ore

- Major minerals in the dacite ore are the same as carbonaceous ore, i.e., quartz, plagioclase feldspar (albite), and muscovite.
- Minor minerals are dolomite, magnetite, hematite, goethite, pyrite, arsenopyrite, and possibly tennantite.
- Pyrite comprises approximately the same fraction as in the carbonaceous ore, i.e., approximately 2.0 wt%.
- Arsenopyrite content and crystal form in the leachable ore, and the high-sulphide (refractory) ore is different:
  - That which is in the leachable ore is approximately 0.4 wt%, approximately double that of the carbonaceous ore, and is poorly crystalized.
  - That which is in the high-sulphide (refractory) ore is 0.6 wt% or more and occurs as well-formed needle-like crystals.
- Graphitic carbon is much less than in the carbonaceous ore, comprising <0.2 wt% of the ore.

- Gold occurs as native gold, the size and association of which in the leachable and high-sulphide ore is significantly different:
  - That which is in the leachable ore is approximately the same as that in the carbonaceous ore, occurring as particles <20 µm in diameter, and commonly occurring as small vein fillings in pyrite or the interface between sulphides and iron oxide and quartz.
  - That which is in the high-sulphide ore is much finer, mostly approximately 2 µm or less in diameter, and is principally locked in pyrite and arsenopyrite.

13.4 Diagnostic Leach Testing

Results of diagnostic leach testing are summarized in Table 13-5. The results are in accord with the mineralogical assessment and cyanide leaching tests.

Amdel's 2007 testing on C1 ore showed that approximately 14% of the gold is locked in sulphides, and 4% in silica. University of São Paulo's 2015 testing of gold mineral associations of the ore coarse fraction generally confirms the earlier Amdel results and mineralogical assessments. CSIRO's and Testing Desinvolvimento de Processo's 2016 to 2017 testing of the carbonaceous ore shows that approximately 10% to 18% of the gold is locked in sulphides and 5% in silica.

CSIRO's and Testing Desinvolvimento de Processo's 2016 to 2017 testing of the Antas 3-deposit high-sulphide dacite ore shows that approximately 50% of the gold is locked in sulphides and 5% in silica.

Table 13-5: Diagnostic Leach Test Results and Gold Association Mineralogy

Date	Laboratory	Sample Location	Sample Type	Cumulative Gold Distribution (%)						From Cum. Distrib. Data on Left	
				Gravity @ 75 µm	CIL @ 1,000 mg/L NaCN	Aceto-nitrile elution	Coarse Sulphide Locked	Fine Sulphide Locked	Silica Locked	Sulphide Locked	Silica Locked
2007	Amdel	C1	Fresh rock, 50% carb; 50% dacite RC cuttings	15	82	82	92	96	100	14	4
		C1	Semi-oxidized, drill core	16	81	84	96	98	100	14	2
		Mansinha	Drill core	38	91	93	99	99	100	6	1
		Heap leach tailings	-10 mm bulk	12	73	79	95	96	100	17	4

Date	Laboratory	Sample Location	Sample Type	Gold Associations (%) in -600+37 µm fraction of heavy fraction of heavy-liquid separation (approximately 50% of gold in approximately 5 weight %)									
				Exposed Gold	Pyrite	Quartz	Arseno-Pyrite	Siderite	Goethite	Other Carbonates	Other Silicates	Other	Total
2015	Univ. São Paulo	C1	High S, part of 17-t carbonaceous ore sample	32.6	35.4	8.0	4.8	4.0	3.8	3.7	5.3	2.4	100.0
		C1	Medium S, part of 17-t carbonaceous ore sample	20.3	26.7	4.9	6.9	11.6	21.0	5.6	0.0	3.0	100.0
		C1	Low S, part of 17-t carbonaceous ore sample	11.3	29.9	5.7	1.1	6.9	32.9	0.0	8.9	3.3	100.0

Date	laboratory	Sample Location	Sample Type	Amenable to CIL	Preg-Robbed	Sulphide Locked	Silica Locked
2016-2017	**CSIRO & Testwork Desolvimento de Processo**	**C1MP029**	**Drill core (carbonaceous)**	**76**		**18**	**6**
		C1MP029a	Drill core (carbonaceous)	78	2	13	7
		A3MP010	Drill core (dacite, high-sulphide)	39	2	54	5
		A3MP018	Drill core (dacite, high-sulphide)	41	5	51	3

13.5 Comminution

Results of comminution testing are summarized in Table 13-6.

Tests by the different laboratories gave similar results. The Bond Ball Mill Work Index is approximately 19 kWh/t with little variation, indicating the ore is exceptionally hard. Fortuitously, the Bond Rod Mill Work Index is like the Ball Mill Work Index, indicating the ore is amenable to SAG milling. The JKTech SAG mill parameters, with the Axb values averaging approximately 40, also indicate that the ore is amenable to SAG milling, though the ore is relatively tough, with a moderate resistance to impact breakage.

The Abrasion Index of the ore is approximately 0.5 indicating the ore is very abrasive. With such a high Abrasion Index, the frequency of changing crusher liners will be much higher than normal. There will also be much higher-than-normal wear of metal pump parts and metal pipes.

Table 13-6: Comminution Test Results

Date	Laboratory	Sample Location	Sample Type	Bond Work Indexes (kWh/t)		Abrasion Index	JK Tech SAG Milling Parameters				SG (g/cm³)
				Rod Mill	Ball Mill		A	b	Axb	t_a	
2006	HDA Services	C1	Sample 1	-	-	-	55.5	0.735	40.8	0.408	-
			Sample 2	-	-	-	51.0	0.703	35.9	0.359	-
			Sample 3	-	-	-	53.4	0.687	37.2	0.372	-
			Sample 4	-	-	-	53.5	0.810	43.3	0.433	-
2007	Amdel	C1	Semi-oxidized, drill core	17.2	16.7	0.21	-	-	-	-	-
		Mansinha	Drill core	20.9	15.2	-	-	-	-	-	-
		Heap leach tailings	-10 mm bulk	-	15.4	-	-	-	-	-	-
2015	HDA Services	C1	High S, part of 17 t carb. sample	-	19.0	0.57	57.0	0.75	42.65	0.31	2.65
		C1	High S, part of 17 t carb. sample	-	21.2	0.52	58.7	0.74	43.67	0.49	2.66
		C1	Medium S, part of 17 t carb. sample	-	19.8	0.44	57.3	0.82	47.34	0.72	2.55
		C1	Low S, part of 17 t carb. sample	-	16.6	0.24	55.3	0.92	50.86	0.48	2.55
		C1	Dacite	-	17.1	0.55	-	-	-	0.29	2.59
		C1	Dacite	-	18.0	-	-	-	-	-	-
2016	SGS	C1MP029a	Drill core (carbonaceous)	-	19.7	-	-	-	-	-	-
2017	Hazen	C1MP028	Drill core (carbonaceous)	16.0	18.7	0.54	68.7	0.61	41.90	0.40	2.76
		C1MP028	Drill core (carbonaceous)	16.5	18.9	0.55	61.9	0.73	45.20	0.42	2.75
		C1MP027a	Waste rock	18.3	19.4	0.31	62.5	0.62	38.80	0.36	2.81
		C1MP027a	Waste rock	19.0	19.7	0.14	62.1	0.54	33.50	0.31	2.80

13.6 Gravity Concentration

Results of gravity concentration testing are summarized in Table 13-7.

Knelson Research's 2005 studies grinding a sample of C1 carbonaceous ore to progressively finer size fractions, and ultimately to 80% passing 80 µm, showed that it was possible to recover 34% of the gold in 1.4 wt% of the ore. Grinding to approximately 80% passing 240 µm resulted in recovery of approximately 23% of the gold into 1 wt% of the ore.

Amdel conducted gravity concentration tests in 2008. These tests were of ore ground to three different size fractions (80% passing 1,000 µm, 600 µm, and 150 µm) followed by gravity separation. The tests closest to that of the Knelson Research tests, those ground to 150 µm (approximately twice as coarse as those tested by Knelson Research) indicated that it is possible to recover approximately 20% of the gold in approximately 3% of the ore. Amdel tested two additional ore types from satellite deposits, Mansinha and Antas, both of which indicated better recovery than that of ore from C1; recovery from Antas was approximately double that of C1.

In 2009 to 2010, gravity concentration tests of flotation concentrate were conducted in conjunction with the pilot plant testing at Santa Luz. These tests showed that it was possible to recover approximately 70% of the gold in the flotation concentrate into a gravity concentrate. Given that flotation recovers approximately 85% of the gold, overall gold recovery of flotation followed by gravity separation can be expected to be approximately 60%.

In 2015, FL Smidth conducted whole ore gravity concentration tests of two carbonaceous samples from C1 and essentially confirmed the Knelson results a decade earlier. The grade of the high-sulphur sample tested was three to four times the grade of the low-sulphur sample; consequently, the grade of the gravity concentrate from the high-sulphur sample was significantly higher than that of the low-sulphur sample. FL Smidth conducted a further test on dacite ore in 2016. For each sample, tests were accompanied by intensive cyanidation leach tests on the concentrate.

Considering the planned plant operations with primary grinding to approximately 80% passing 850 µm, the Knelson tests indicate that it is possible to recover approximately 10% of the gold into 0.5 wt% of the ore, while the Amdel tests indicate that it is possible to recover approximately 10% of the gold into 4 wt% of the ore. Although the results are not in close agreement, the results do indicate that gold and gold-bearing pyrite are not well liberated at 80% passing 850 µm. The results need to be weighed against typical plant scale mass recoveries of 0.05% by weight.

When the ore is ground to the secondary grind size of approximately 80% passing 75 µm, the Knelson tests indicate that it is possible to recover approximately 35% of the gold into approximately 1.3 wt% of the ore. The results indicate that gold and gold-bearing pyrite are considerably better liberated at 80% passing 75 µm than at 80% passing 850 µm, which is expected.

Based on the gravity separation testwork and intensive cyanidation gold recovery from the concentrates, it appears that it may well be worthwhile installing gravity recovery ahead of the leach. The intent is to remove any free gold prior to leach to minimize opportunities for dissolved gold to be preg-robbed.

Table 13-7: Gravity Concentration Test Results

Date	Laboratory	Sample Location	Sample Type	Stage	Size (P_{80} µm)	Mass (%)	Grade (g/t)	Grav. Rec. (%)	4 h Intv. CN Rec. (%)
2006	Knelson Research	C1	Carbonaceous ore	1	430	0.46	-	9.7	-
				2	243	0.49	-	13.0	-
				3	81	0.43	-	11.5	-
				Overall		**1.38**	**-**	**34.2**	**-**
2008	Amdel	C1	Drill core	-	1,000	4.57	-	6.5	-
				-	600	3.60	-	10.1	-
				-	150	3.32	-	16.8	-
		Mansinha	-5 mm crushed	-	1,000	4.01	-	10.4	-
				-	600	3.51	-	15.6	-
				-	150	3.19	-	21.1	-
		Antas	-5 mm crushed	-	1,000	4.26	-	16.7	-
				-	600	3.33	-	43.9	-
				-	150	3.01	-	38.8	-
2009-2010	Pilot Plant	Blend 1A	flotation conc. (25% D + 75% C)	-	-	-	-	48.8	-
		Blend 2A	flotation conc. (50% D + 50% C)	-	-	-	-	79.8	-
		Blend 1B	flotation conc.	-	-	-	-	78.2	-
		Blend 3B	flotation conc. (75% D + 25%C)	-	-	-	-	69.1	-
2015	FL Smidth Knelson	C1	High TOC, high S, part of 9 tonne carbonaceous sample 6.8 g/t Au, 0.85% S, 1.17% TOC	1	1,239	0.40	148	9.1	-
				2	240	0.40	288	16.7	-
				3	77	0.40	206	13.4	-
				Overall		**1.30**	**213**	**39.2**	**95.2**
		C1	Low S, part of 2 tonne carbonaceous sample 1.3 g/t Au, 0.37% S, 0.67% TOC	1	1,265	0.40	20	5.8	-
				2	229	0.40	48	14.3	-
				3	62	0.40	55	14.7	-
				Overall	-	**1.10**	**41**	**34.8**	**92.3**
2016	FL Smidth Knelson	C1	Dacite ore, part of 2 tonne sample 2 g/t Au, 0.86% S, 0.11% TOC	1	1,055	0.4	52	11.4	-
				2	256	0.4	81	18.0	-
				3	73	0.4	89	18.5	-
				Overall		**1.3**	**74**	**47.9**	**96.8**

13.7 Flotation

Four different laboratories conducted flotation testing from 2005 to 2015. The results are reasonably consistent, with the only exception being the tests of heap-leach tailings, which is as expected given that the heap-leach ore was primarily oxide material. Excluding the heap-leach tailings, gold recovery ranges from 71% to 93%, the weight percent from 7% to 15%, and the concentrate grade from 8 g/t Au to 40 g/t Au. With ore of average grade, it can be expected that flotation will recover approximately 85% of the gold into 12.5 wt% of the ore with a concentrate grade of 10 g/t Au.

Results of flotation testing are summarized in Table 13-8.

Table 13-8: Flotation Test Results

Date	Laboratory	Sample Location	Sample Type	Mass Pull (wt%)	Au Grade (g/t)	Au Recovery (%)
2007	Amdel	C1	fresh rock, 50% carb; 50% dacite RC cuttings	7.1	21.4	82.9
		C1	semi-oxidized, drill core	6.7	40.4	85.3
		Mansinha	drill core	7.0	23.6	84.8
		Heap leach tailings	-10 mm bulk	4.4	13.5	55.4
2008	Amdel	C1	drill core	11.2	16.1	92.5
		Mansinha	-5 mm crushed	13.6	12.5	90.1
		Antas	-5 mm crushed	16.0	13.1	92.1
2009-2010	Pilot Plant	Blend 1A	30% carb; 70% dacite RC cuttings	-	-	82.1
		Blend 2A	50% carb; 50% dacite RC cuttings	-	-	90.3
		Blend 3A	70% carb; 30% dacite RC cuttings	-	-	71.0
		Blend 1B	30% carb; 70% dacite drill core	-	-	91.5
		Blend 3B	70% carb; 30% dacite drill core	-	-	86.6
2015	Funmineral	C1	High S, part of 17-t carbonaceous sample	12.5	31.2	87.0
		C1	Medium S, part of 17-t carbonaceous sample	14.7	8.3	76.2

13.8 CIL of Flotation Concentrate

The same four laboratories conducted CIL testing of flotation concentrates as those that conducted the flotation tests. The flotation tests included kerosene blanking of the concentrate prior to CIL, with kerosene addition rates ranging between 2 g/t and 20 g/t. All the CIL tests were bottle-roll tests, except those conducted by Funmineral in 2015. Bottle-roll tests mask the effect of excess kerosene, because the ratio of carbon to gold-extracted used in the tests is much higher than in industrial operations, allowing the excess carbon to adsorb the excess kerosene in addition to the dissolved gold. The pilot-plant tests resulted in gold recoveries approximately 10% lower than that of the bottle-roll tests.

Excluding two strongly divergent bottle roll test results, gold recoveries of flotation concentrates ranged between 74% and 90%. Considering all the results, it can be expected that CIL of flotation concentrates preceded by kerosene blanking and mop-up of excess kerosene will recover approximately 85% of the gold.

Results of CIL testing of flotation concentrate are summarized in Table 13-9.

Table 13-9: CIL of Flotation Concentrate Test Results

Date	Laboratory	Sample Location	Sample Type	Test Type	Calculated Au Head Grade (g/t)	Kerosene Addition (kg/t)	Kerosene Mop-Up	Pre-Oxidation	$Pb(NO_3)_2$ (kg/t)	NaCN Conc. (g/L solution)	Carbon Conc. (g/L slurry)	Reagent (kg/t)		Au Recovery (%)
												CaO	NaCN	
2007	Amdel	C1	Fresh rock, 50% carb; 50% dac. RC cuttings	bottle roll	13.5	2	-	-	-	-	-	0.2	6.8	80.8
		C1	Semi-oxidized, drill core	bottle roll	14.3	20	-	-	-	-	-	0.2	6.1	88.4
		Antas 3	RC cuttings	bottle roll	29.0	2	-	-	-	-	-	1.4	3.4	89.3
		Mansinha	Drill core	bottle roll	9.4	2	-	-	-	-	-	0.7	5.2	53.4
2008	Amdel	C1	Drill core	bot. roll, bulk leach	8.5	10	included	-	-	-	-	0.4	2.5	88.0
		C1	Drill core	bot. roll, baseline leach	9.9	10	included	-	-	-	-	1.0	4.1	89.6
		C1	Drill core	bot. roll w/pre-oxidation	9.4	10	included	12 h @ pH 9	-	-	-	1.2	3.9	88.9
		C1	Drill core	bot. roll, baseline leach	9.5	5	included	-	-	-	-	1.0	4.1	86.9
		C1	Drill core	bot. roll w/$Pb(NO_3)_2$	9.7	10	included	-	48	-	-	0.9	3.6	89.8
		C1	Drill core	bot. roll w/pre-ox + $Pb(NO_3)_2$	10.6	10	included	12 h @ pH 9	57	-	-	1.2	4.0	90.4
2009-2010	Funmineral	Blend 1A	30% carb; 70% dac., RC cuttings	bottle roll	-	4	included	-	-	-	-	-	-	93.3
		Blend 2A	50% carb; 50% dac., RC cuttings	bottle roll	-	4	included	-	-	-	-	-	-	92.9
		Blend 3A	70% carb; 30% dac., RC cuttings	bottle roll	-	4	included	-	-	-	-	-	-	73.1
		Blend 1B	30% carb; 70% dac., drill core	bottle roll	-	4	included	-	-	-	-	-	-	78.2
		Blend 3B	70% carb; 30% dac., drill core	bottle roll	-	4	included	-	-	-	-	-	-	69.1
2015	Funmineral	C1	High S, part of 17 t carb. sample	pilot plant w/O_2 sparging	15.1	2	-	-	-	1.5	30	-	-	73.9
		C1	Medium S, part of 17 t carb. sample	pilot plant w/O_2 sparging	4.4	2	-	-	-	2	30	-	-	78.6

13.9 Calcining of Flotation Concentrate and CIL of Calcine

Calcining of flotation concentrate at any of the temperatures tested, between 600°C and 750°C, results in removal of all the sulphur. However, the proportion of organic carbon removed depends on the temperature: at 600°C the calcine assays approximately 2% organic carbon, at 750°C the calcine assays 0% organic carbon. Little of the arsenic is removed at any temperature.

CIL of the calcine consistently results in approximately 95% recovery with slightly higher recovery at higher calcine temperature.

Though the test data indicate good results, the very fine character of the concentrate results in excessive elutriation of fine material making calcine unlikely to be industrially applicable. In addition, though little arsenic is removed by calcination, the arsenic fume in calcine gas would need to be captured, and the sulphur dioxide in the calcine gas would need to be converted to sulphuric acid. The complexity and cost of the calcine gas systems add to the impracticality of calcination of the flotation concentrate.

Results of tests of calcining the flotation concentrate followed by CIL of the calcine are summarized in Table 13-10.

Table 13-10: Calcining of Flotation Concentrate and CIL of Calcine Tests

Date	Laboratory	Sample Type	Analysis of Flotation Concentrate prior to Calcination				
			Au (g/t)	Ag (g/t)	S^{2-} (%)	C_{org} (%)	As (g/t)
2014	Hazen	Flotation concentrate	26	8	9	6	4,700

Temp. (°C)	Furnace Diameter (inch)	Calcine Analysis S^{2-} (%)			CIL of Calcine		
					Reagent Concentration		Au Recovery (%)
			C_{org} (%)	As (g/t)	CaO (kg/t)	NaCN (kg/t)	
600	4	0	1.95	3,770	18	4	96
600	4	0	1.78	3,650	17	4	94
600	4	-	-	-	16	3	95
650	13	0	0.04	6,360	22	4	96
750	13	0	0.02	6,690	29	0.5	97
750	8	0	0	5,240	20	3	99
650	8	0	0	5,190	18	2	99

13.10 CIL of Whole Ore (Bottle-Roll Tests and Pilot-Plant Tests by Funmineral)

Results of CIL bottle-roll testing of whole ore and pilot-plant testing by Funmineral of whole ore up to 2015 are summarized in Table 13-11.

CIL of whole ore was first tested in 2005 and has been subject to periodic further testing up to the present. Some limited RIL testing was also conducted in 2005 at the University of São Paulo, and gave similar results to CIL.

Table 13-11: CIL of Whole Ore Tests

Date	Laboratory	Sample Location	Sample Type	Test Type	Grind Size (P_{80} µm)	Kerosene Addition (kg/t)	NaCN Concentration (g/L soln.)	Carbon Conc. (g/L slurry)	Reagent Concentration (kg/t)		Au Recovery (%)
									CaO	NaCN	
2005–2006	Univ. São Paulo	C1	Carbonaceous, drill core	Bottle roll	74	0	-	-	-	3.9	67
		C1	Dacite, drill core	Bottle roll	74	0	-	-	-	3.0	92
		C1	Shallow-depth ore	Bottle roll	74	1.0	1.50	30	-	0.7	85
		C1	Intermediate-depth ore	Bottle roll	74	4.0	1.50	30	-	2.0	77
		C1	Deep ore	Bottle roll	74	4.0	1.50	30	-	1.4	79
2007	Amdel	C1	Fresh rock, 50% carb; 50% dac. RC cuttings	Beaker w/air sparging	-	2.0	0.25	30	0.4	1.5	89
		C1	Fresh rock, 50% carb; 50% dac. RC cuttings	Beaker w/O_2 sparging	-	2.0	0.25	30	0.4	1.9	87
2015	Funmineral	C1	High S (1.17% TOC), part of 17 t carb. sample	Bottle roll	-	2.0	4.00	30	-	-	87
		C1	Low S (0.67% TOC), part of 17 t carb. sample	Bottle roll	-	2.0	4.00	30	-	-	84
		C1	High S (1.17% TOC), part of 17 t carb. sample	Pilot plant w/O_2 sparging	-	2.0	2.00	30	-	-	67
		C1	Medium S (0.71% TOC), part of 17 t carb. sample	Pilot plant w/O_2 sparging	-	2.0	1.50	30	-	-	78
		C1	Low S (0.67% TOC), part of 17 t carb. sample	Pilot plant w/O_2 sparging	-	0.5	1.50	30	-	-	78
2015	SGS	C1	Dacite	Bottle roll	74	0.0	-	-	-	-	85
		C1	Dacite	Bottle roll	74	0.5	-	-	-	-	92
2015	Gekko Systems	C1	High S, part of 17 t carbonaceous sample	Beaker w/O_2 sparging	75	0.5	0.50	30	-	-	72
		C1	High S, part of 17 t carbonaceous sample	Bottle roll	75	0.5	0.50	30	-	-	53
		C1	Low S, part of 17 t carbonaceous sample	Beaker w/O_2 sparging	75	0.5	0.50	30	--	-	76
		C1	Low S, part of 17 t carbonaceous sample	Bottle roll	75	0.5	0.50	30		--	52

The grind size of almost all the tests was approximately 80% passing 75 µm, which is the grind size of almost all CIL plants, and is dictated primarily by the need to have the ore particle size substantially finer than the carbon. Kerosene blanking was found to be essential for processing carbonaceous ore; however, dacitic ore could be leached without kerosene blanking or with very little kerosene. Kerosene requirement for carbonaceous ore was shown to be 2 kg/t, whereas 0.5 kg/t appeared to be more than adequate for dacitic ore. Leach time for most tests was 24 hours, which appeared to be more than adequate.

Most of the tests were bottle roll tests, but Funmineral also conducted pilot plant tests. Recovery in the pilot plant tests was generally 5% to 10% lower than the bottle roll tests due to issues with the operation of the pilot plant equipment and lack of a kerosene conditioning stage ahead of CIL.

Poor recovery in the Gekko Systems tests was probably related to insufficient kerosene, though using oxygen with low kerosene addition resulted in approximately 20% higher recovery, indicating that use of oxygen may be beneficial.

Both CIL of whole ore and of flotation concentration followed by CIL appeared to be equally viable processes at the time of the decision to proceed with building of the existing plant; however, the decision was made to base the plant on flotation concentration followed by CIL of the flotation concentrate, apparently on the supposition that it would be environmentally more acceptable.

13.11 Whole Ore CIL and RIL (Bottle-Roll Tests Under Identical Leach Conditions)

Results of RIL bottle roll testing of whole ore, together with results of CIL testing under identical conditions (where available), are summarized in Table 13-12.

All tests in this category were conducted in 2016 using four different laboratories: Hazen, RDI, CSIRO, and SGS Belo Horizonte. The kerosene requirement for the carbonaceous ore and dacitic ore are the same as that established by prior tests reported in Table 13-12, namely, 2 kg/t and 0.5 kg/t, respectively.

RIL and CIL tests under identical conditions give similar results. There is no consistent difference between CIL and RIL gold recovery at the low gold loadings achieved in bottle roll tests.

Average gold recovery of carbonaceous ore ranges between 75% and 91%, and that of dacitic ore ranges between 79% and 84%. Reagent consumptions in industrial operation can be expected to be approximately 1 kg/t for lime (as CaO) and approximately 0.75 kg/t for cyanide (as NaCN).

High-sulphide dacitic ore from the Antas 3 satellite deposit is refractory: ore containing up to 1,500 g/t As results in gold recoveries of approximately 70%; some of those containing more than 1,500 g/t As results in gold recoveries as low as 35%. Some dacitic ore from the C1 deposit can also be semi-high-sulphide; however, the recovery is not directly related to arsenic content.

Table 13-12: CIL and RIL Bottle Roll Whole Ore Test Results

Date	Laboratory	Sample Location	Sample Type	Test Type	Grind Size (P_{80} µm)	Kerosene Addition (kg/t)	Kerosene Addition Point	NaCN Concentration (g/L solution)	Resin Concentration (mL/L slurry)	Reagent Concentration (kg/t)		RIL Au Recovery (%)	CIL Au Recovery (%)
										CaO	NaCN		
2016	Hazen	C1	High S, part of 17-t carb. Sample 1.7% TOC	Bottle roll	76	2.0	post grind	1.0	30	1.3	1.1	85	-
		C1MP028	Drill core (carbonaceous) 1.24% TOC	Bottle roll	70	2.0	with grind	1.0	30	1.3	0.9	78	82
		C1MP031	Drill core (carbonaceous) 1.42% TOC	Bottle roll	71	2.0	with grind	1.0	30	1.4	1.0	80	79
2017		C1MP027a	Drill core (dacite, semi-high-sulphide)	Bottle roll	~75	2.0	with grind	0.5	30	0.9	0.4	58	63
2016	RDI	C1	High S, part of 17-t carb. Sample 1.7% TOC	Bottle roll	74	2.0	post grind	1.0	30	2.8	0.7	91	-
		C1MP028	Drill core (carbonaceous) 1.24% TOC	Bottle roll	74	10.0	with grind	2.0	30	3.6	0.8	80	-
		C1MP031	Drill core (carbonaceous) 1.42% TOC	Bottle roll	74	2.0	with grind	1.0	30	7.9	1.0	77	84
2016	CSIRO	C1MP029	Drill core (carbonaceous) 1.47% TOC	Bottle roll	75	2.0	with grind	1.0	30	8.2	0.7	80	79
2016	SGS	C1MP023, -034, -033, -006, -032, -013, -016, -040	Drill core (carbonaceous) 0.8-2.65 TOC	Bottle roll	75	2.0	post grind	2.0	30			49[a]	80
		C1MP029a	Drill core (carbonaceous) 1.3% TOC	Bottle roll	75	2.0	with grind	2.0	32			75	75
		C1MP036, -023, -011, -021	Drill core (dacite) 0.13-0.5% TOC	Bottle roll	75	2.0	post grind	2.0	32			79	84
		C1MP028, -037, -041, -030, -040, -002, -007	Drill core (dacite)	Bottle roll	75	0.5	with grind	0.5	30	0.8	0.6	84	82
		A3MP024, -019, -021, -003, -023	Drill core (dacite, non-high-sulphide)	Bottle roll	75	0.5	with grind	0.5	30	0.8	0.6	84	82
		A3MP005, -009, -011	Drill core (dacite, semi-high-sulphide)	Bottle roll	75	0.5	with grind	0.5	30	0.8	0.6	70	69
		A3MP018, -010, -008	Drill core (dacite, high-sulphide)	Bottle roll	75	0.5	with grind	0.5	30	0.8	0.6	37	34

Note: [a] Less kerosene mixing time than CIL
Where available, CIL recoveries under identical test conditions.

13.12 Effect of Kerosene Quantity, Finer Grinding, and Blending on Low-Recovery Ores

Table 13-13 summarizes results of 2016–2017 bottle-roll tests at SGS to assess the effect of kerosene quantity and finer grinding.

The tests show that for dacitic ores additional kerosene offers no benefit, as it would result only in greater fouling of the CIL carbon. However, for carbonaceous ores, especially those with high TOC content, additional kerosene can increase recovery by more effective blanking of the carbonaceous material to minimize preg-robbing.

The tests show no advantage in grinding the ore finer than 80% passing 75 µm.

Table 13-14 summarizes results of 2016–2017 bottle-roll tests at SGS to assess the effect of blending different ores.

In general, the tests show blending of different ores is acceptable, and results in an overall blend recovery similar to what would be expected from the weighted average of the blend components. However, the SGS test results suggest that this may not always be true of blending very poor-recovery dacitic ore and good-recovery dacitic ore. Given that diagnostic leaching testwork showed that the low recovery from the high arsenic Antas 3 samples was due to sulphide encapsulation, it is likely that the lower recoveries with these dacite blends may be related to inadequate oxygen addition.

Table 13-13: Kerosene Effect with Finer Grinding and Blending on Low Recovery Ores

Date	Laboratory	Sample Location	Sample Type and Conclusions	Test Type	Recovery @ P_{80} of 75 µm (%)				Recovery @ P_{80} of 45 µm (%)		
					@ 0.5 kg/t kerosene	@ 2 kg/t kerosene	@ 4 kg/t kerosene	@ 8 kg/t kerosene	@ 0.5 kg/t kerosene	@ 2 kg/t kerosene	@ 4 kg/t kerosene
2016-2017	SGS	C1MP037	Drill core (dacite, high-sulphide) 2,570 g/t As	Bottle roll	41.4	34.1	39.3	-	39.8	39.3	-
		A3MP010	Drill core (dacite, high-sulphide) 4,040 g/t As	Bottle roll	35.9	30.3	-	-	31.9	-	-
		A3MP018	Drill core (dacite, high-sulphide) 6,190 g/t As	Bottle roll	26.9	-	-	-	25.0	-	-
		A3MP008-A	Drill core (dacite, high-sulphide) 3,980 g/t As	Bottle roll	33.4	21.0	23.5	-	19.8	22.9	-
		A3MP008-B	Drill core (dacite, high-sulphide) 5,410 g/t As	Bottle roll	36.4	29.3	-	-	32.1	-	-
		A3MP008-C	Drill core (dacite, high-sulphide) 2,390 g/t As	Bottle roll	34.4	50.3	-	-		-	-
		Average	**Additional kerosene, no effect, finer grind no effect**		**34.7**	**33.0**	**31.4**	**-**	**29.7**	**31.1**	**-**
		C1MP007	Drill core (dacite) 820 g/t As	Bottle roll	87.6	83.3	-	-	-	-	-
		C1MP033	Drill core (carbonaceous) 3.1% TOC	Bottle roll	-	53.3	74.1	75.7	-	56.7	55.3
		C1MP006	Drill core (carbonaceous) 2.8% TOC	Bottle roll	-	62.4	75.3	-	-	65.8	
		C1MP020	Drill core (carbonaceous) 0.4% TOC	Bottle roll	-	90.4	92.0	92.2	-	92.1	88.5
		C1MP001	Drill core (carbonaceous) 0.3% TOC	Bottle roll	-	84.7	84.7	-	-	-	-
		C1MP029a	Drill core (carbonaceous) 0.9 % TOC	Bottle roll	-	73.6	80.7	81.0	-	79.5	80.7
		Average	**Additional kerosene advantageous, finer grind no effect**		-	**72.9**	**81.4**	**83.0**	-	**73.5**	**74.8**

Table 13-14 Effect of Blending on Low Recovery Ores

Date	Laboratory	Sample Location	Sample Type and Conclusions	Test Type	Grind Size (P_{80} µm)	Recovery (%)	
						Calculated Weighted	Tested Actual
2016-2017	SGS	C1MP037 & C1MP007	50% sulphidic & 50% normal dacite blend	Bottle roll	75	64.5	63.7
		A3MP010 & C1MP007	50% sulphidic & 50% normal dacite blend	Bottle roll	75	61.7	57.5
		A3MP018 & C1MP007	50% sulphidic & 50% normal dacite blend	Bottle roll	75	57.2	42.4
		A3MP008-A & C1MP007	50% sulphidic & 50% normal dacite blend	Bottle roll	75	60.5	44.0
		A3MP008-B & C1MP007	50% sulphidic & 50% normal dacite blend	Bottle roll	75	62.0	49.1
		A3MP008-C & C1MP007	50% sulphidic & 50% normal dacite blend	Bottle roll	75	61.0	53.0
		Average	**Blending sulphidic & normal dacite disadvantageous**			**61.2**	**51.6**
		C1MP037 & C1MP001	50% sulphidic dacite & 50% normal carbonaceous	Bottle roll	75	63.1	69.8
		A3MP010 & C1MP001	50% sulphidic dacite & 50% normal carbonaceous	Bottle roll	75	60.3	59.2
		A3MP018 & C1MP001	50% sulphidic dacite & 50% normal carbonaceous	Bottle roll	75	55.8	50.5
		A3MP008-A & C1MP001	50% sulphidic dacite & 50% normal carbonaceous	Bottle roll	75	59.1	49.7
		Average	**Blending sulphidic dacite & normal carbonaceous, no effect**			**59.6**	**57.3**
		C1MP037 & C1MP029a	30% sulphidic dacite & 70% normal carbonaceous	Bottle roll	75	63.9	75.3
		C1MP037 & C1MP029a	50% sulphidic dacite & 50% normal carbonaceous	Bottle roll	75	57.5	76.5
		C1MP037 & C1MP029a	70% sulphidic dacite & 30% normal carbonaceous	Bottle roll	75	51.1	70.4
		Average	**Blending sulphidic dacite & normal carbonaceous advantageous**			**57.5**	**74.1**
		C1MP033 & C1MP029a	50% sulphidic & 50% normal carbonaceous	Bottle roll	75	63.5	64.4
		C1MP006 & C1MPo29a	50% sulphidic & 50% normal carbonaceous	Bottle roll	75	68.0	69.8
		Average	**Blending sulphidic & normal carbonaceous, no effect**			**65.8**	**67.1**

13.13 Initial Whole Ore CIL (Santa Luz Pilot-Plant Tests)

Results of initial Stage 1-4 Santa Luz pilot plant CIL tests on whole ore are shown in Table 13-15.

Table 13-15: Initial CIL Tests of Whole Ore, Stages 1–4

Parameter	Stage 1	Stage 2 A	Stage 2 B	Stage 2 C	Stage 3 A	Stage 3 B	Stage 4 A	Stage 4 B
Start Date	6-Oct-16	12-Oct-16	16-Oct-16	20-Oct-16	1-Nov-16	10-Nov-16	21-Nov-16	26-Nov-16
End Date	11-Oct-16	15-Oct-16	17-Oct-16	28-Oct-16	7-Nov-16	17-Nov-16	25-Nov-16	4-Dec-16
Sample Location	C1	C1			C1			C1
Sample Type	Carbonaceous	Carbonaceous			Carbonaceous			Carbonaceous
Sample Head Grade								
Gold (g/t)	1.45	1.63			1.96			1.70
TOC (%)	0.63	0.77			1.35			0.80
S^{2-} (%)	0.45	0.48			0.69			0.46
Reagent Conditioning Time (hours)	0 (added to mill)	2	2	2	2	2	1	1
Leach Time (hours)	24	24	24	24	24	24	24	24
Kerosene Addition (kg/t ore)	2	2	2	4	2	2	2	5
Cyanide Addition (mg CN/L solution)	600	600	1,200	600	600	600	600	600
Gold Recovery (%) Based on Head and Tailings Assays	79.5	81.5	81.1	82.4	76.3	53.1	67.1	80.6

Note: Performed at the Santa Luz Pilot Plant.

These initial pilot plant tests incorporated a dedicated kerosene mop-up carbon adsorption circuit to remove excess kerosene following kerosene conditioning prior to CIL leaching.

In the Stage 1 test, the kerosene was added to the grinding mill to emulate bench-scale tests; however, the results were not particularly encouraging without a kerosene conditioning stage, and it was decided to add the kerosene post grinding for subsequent tests.

In the Stage 2 tests, the ore was conditioned with kerosene for two hours prior to the mop-up circuit. The cyanide addition was doubled in Test 2B with no effect, and the kerosene addition was doubled in Test 2C resulting in marginally better recovery.

In the Stage 3 tests, the conditions were the same as for the Stage 2A tests, but the ore was a high-TOC sample, approximately twice the TOC content of the ore for the Tests 1 and 2. Recoveries were poor and erratic.

In the Stage 4 tests the kerosene conditioning time was dropped to 1 hour and the ore changed to ore of approximately the same TOC as that used in Tests 1 and 2. The results of Test 4A were disappointing, and it was decided to dramatically increase the quantity of kerosene for Test 4B to 5 kg/t. The recovery improved; however, it was no better than that observed in Test 2.

13.14 Whole Ore CIL and RIL of Whole Ore (Santa Luz Pilot-Plant Tests)

Results of CIL and RIL whole ore Stage 5, 6, 7, and 10 tests at Santa Luz pilot-plant are summarized in Table 13-16.

Table 13-16: Detailed Results from Stage 5, 6, 7, and 10 Whole Ore Tests at Santa Luz Pilot Plant

Parameter	Stage 5	Stage 6	Stage 7	Stage 10
Start Date	18-Dec 16	8-Feb 17	29-Mar 17	21-May 17
End Date	5-Feb 17	26-Mar 17	9-Apr 17	4-Jun 17
Sample Location	C1	C1	C1	C1
Sample Type	Ore stockpile carbonaceous high TOC	Ore stockpile carbonaceous low TOC	Ore stockpile dacite	Ore stockpile carbonaceous high TOC
Sample Head Grade				
Gold (g/t)	1.96	1.20	1.10	2.51
TOC (%)	1.37	0.90	0.15	1.29
S^{2-} (%)	0.71	0.80	0.01	0.96
As (g/t)	628	468	541	500
Test Type	CIL	RIL	RIL	RIL
Feed Rate (kg/h)				
Line 1	1.6	1.6	1.6	1.6
Line 2	-	3.2	3.2	3.2
Grind Size (P_{80} µm)	83.5	82.0	82.5	82.5
Slurry Density (% solids)	39	39	39	39
Conditioning Time (hours)				
Pre-aeration	0	4	4	2
Kerosene Conditioning	2	4	4	4
Kerosene Addition (kg/t ore)	2	2	0.30	2
Kerosene Addition Point	post grind	post grind	post grind	post grind
Kerosene Mop-Up Time (hours)	4	0	0	0
Reagent Addition (kg/t ore)				
Lime (as $Ca(OH)_2$)	1.66	1.70	1.09	
Cyanide (as CN)	0.76	0.80	0.52	
Leach pH	10.50–11.00	10.50–11.00	10.20–10.80	10.20–10.50
Cyanide Concentration (g/t CN)				
Leach tank 1	600	649	421	531
Tailings	250	492	325	357
Leach Time (hours)	24	24	24	20
Adsorbent Concentration per Tank				
Carbon (g/L)	25,15,15,12,10,13			
Resin (mL/L)		60,23,15,10,10,10	60,23,15,10,10,10	45,25,15,10,10
Loaded Adsorbent Movement (g/d)				
Line 1	51.5	67	67	67
Line 2		134	134	134
Av. Loaded Adsorbent Parameters				
Gold (g/t)	550	420	520	1,000

Parameter	Stage 5	Stage 6	Stage 7	Stage 10
Copper (g/t)		13,000	3,250	12,000
Iron (g/t)		200	335	230
Activity (%)[a]	31	60	70	67
Tailings Solids Grade (g/t Au)	0.500	0.200	0.152	0.383
Tailings Solids Grade (g/t As)	610	459	537	480
Tailings Solution Grade (mg/L)				
Gold	0	0	0	0
Copper	40	45	0.13	30
Iron	6	-	-	14
Arsenic	0.10	0.16	0.31	0.10
Gold Recovery (%)				
Based on Head and Tailings Assays	75.7%	83.3%	86.2%	84.7%
Based on Adsorbent and Tailings	66.8%	78.7%	85.6%	82.9%

Notes: [a] The reported adsorbent (carbon or resin) activity is the percentage of gold adsorbed by one gram of adsorbent in one hour from one litre of solution containing 10 mg/L gold in a cyanide solution.

The Stage 5 CIL tests were conducted on high-TOC carbonaceous ore. The amount of kerosene added varied widely, from 2 kg/t to 8 kg/t, and the circuit included a mop-up system to remove excess kerosene prior to leaching. With the circuit operating consistently with 2 kg/t kerosene addition, gold recovery was only 75.7% at a loaded carbon gold grade of 550 g/t. This was disappointing, as it was lower than the 80.8% recovery that was achieved in the control CIL and RIL bottle roll tests and the grade of the sample was relatively high, at 2 g/t Au. However, the final gold loading on the carbon was much lower in the bottle roll tests, showing the limitation of trying to use bottle roll tests to determine gold recovery under plant conditions for high-preg-robbing ores. Despite the mop-up system, CIL carbon was not able to cope with 2 kg/t kerosene addition to maintain enough activity to achieve the target gold recovery at realistic carbon loadings. As a result, the focus of pilot plant testwork was shifted to RIL. Table 13-16 shows that after further development, by Stage 10 RIL operation with 2 kg/t kerosene addition was able to achieve target gold recoveries around 84% with resin gold loadings around 1,000 g/t on high TOC ore similar to that used in Stage 5.

Following the Stage 5 tests and prior to the start of the Stage 6 tests, an additional processing line (PP 02) was added with double the throughput rate of the initial processing line (PP 01). Stage 6 tests were RIL tests conducted on low-TOC carbonaceous ore with identical conditions on both processing lines, which tested whether or not the lines behaved similarly. The amount of kerosene added was 2 kg/t, the same as that at the end of the Stage 5 tests. When the tests were initiated, there was found to be excessive froth formation in the leach tanks; to obviate the problem, air addition to the leach tanks was stopped. Initial results were disappointing, and it was surmised that the kerosene conditioning time of one hour was inadequate. (The prior Stage 5 CIL tests had effectively included additional kerosene conditioning time in the mop-up circuit.) There was also concern that insufficient air addition to the leach tanks could result in lower recovery, though the oxygen content of the leach tanks was only marginally reduced. To alleviate the concerns regarding insufficient oxygenation and kerosene conditioning of the slurry prior to leaching, two 4-hour-retention-time tanks were installed ahead of the circuit, one for pre-aeration and one for kerosene conditioning. Results with the new conditioning tanks improved significantly, and stabilized, resulting in a gold recovery of 83% on ore grading 1.2 g/t Au. A serendipitous effect of the pre-conditioning system was an increase in the activity of the loaded resin, and minimal coating of the resin with kerosene-carbon-sulphide film compared to prior tests (i.e., more effective loading of kerosene onto carbonaceous matter leaving less emulsified free kerosene)

Stage 7 RIL tests were conducted on dacitic ore, again with identical conditions on both processing lines. The conditions were the same as the prior Stage 6 tests except the kerosene addition was dropped to 0.3 kg/t since the ore processed was dacitic rather than carbonaceous. Gold recovery stabilized at 86% at a gold loading of 520 g/t on ore grading 1.1 g/t Au. Gold recovery consistently for both Stages 6 and 7 was approximately 0.5% lower on PP 02 than on PP 01; the reason for the difference was not apparent.

In reviewing the data from the pilot-plant tests up to Stage 7, it was evident that at the adsorbent gold loadings tested, the incremental gold recovery was approximately 1% (~US$1.5 million/a) for the four hours of leaching from 16 hours to 20 hours and approximately 0.5% for the 4 hours of leaching from 20 hours to 24 hours. Economic assessment indicated that the incremental recovery from 16 hours to 20 hours may only be marginally economic and that from 20 hours to 24 hours may be uneconomic. To test the effect of leach time, Stage 7A tests were conducted with 24 hours leaching on PP 01 and 16 hours leaching on PP 02. The Stage 7A tests confirmed the projected difference in recovery; however, rather than radically reducing the leach time to 16 hours, it was decided to compromise with 20 hours of leach time. Accordingly, subsequent tests were conducted using 24 hours leach time on PP 01, and 20 hours leach time on PP 02.

Stages 8, 9, and 10 were all run on high-TOC carbonaceous ore and examined the effect of variation of leach and conditioning times. Results of these tests are summarized in Table 13-17. The first of these tests, Stage 7A, was to check on the effect of reducing the leach time to 16 hours on dacitic ore and subsequently to 20 hours on high-TOC carbonaceous ore. After deducting the tendency of Line PP 01 to give 0.6% higher recovery than Line PP 02, recovery with 16 hours leach time dropped 1.1% and recovery with 20 hours leach time dropped 0.5%. The next set of tests, Stages 8, 9, and 10 was to validate the 20 hours leach time and to assess pre-aeration and kerosene conditioning times using high-TOC carbonaceous ore. Using four hours of pre-aeration time and two hours of kerosene conditioning time resulted in 1.0% lower recovery; using two hours of pre-aeration time and four hours of kerosene conditioning time resulted in no reduction in recovery, though it was ascertained that wear of the Line PP 01 mixers during this test prevented the obtaining of meaningful comparison of the two lines.

Stages 11–14 were all run on blended C1 carbonaceous and dacitic ores from stockpiles to assess plant performance if carbonaceous and dacitic ores were blended rather than separately treated. Adding 2 kg/t kerosene to the blended ore allowed a higher effective kerosene dose to the carbonaceous matter, while capping kerosene addition at 2 kg/t.

Table 13-17: Leach and Conditioning Times Effects for RIL of Whole Ore, Stages 7–14

Parameter	Stage 7	Stage 7A	Stage 8	Stage 9	Stage 10	Stage 11	Stage 12	Stage 13	Stage 14
Start Date	29-Mar 17	11-Apr 17	20-Apr 17	4-May 17	21-May 17	10-Jun 17	26-Jun 17	19-Jul 17	8-Aug 17
End Date	9-Apr 17	17-Apr 17	2-May 17	15-May 17	4-Jun 17	20-Jun 17	16-Jul 17	5-Aug 17	20-Aug 17
Sample Location	C1	C1	C1	C1	C1	C1	C1	C1	C1
Sample Type	Ore stockpile dacite	Ore stockpile dacite	Ore stockpile carbonaceous high-TOC	Ore stockpile carbonaceous high -TOC	Ore stockpile carbonaceous high -TOC	Ore stockpile, blend of 44% dacite, 23% low-TOC carbonaceous, 33% high-TOC carbonaceous	Ore stockpile, blend of 44% dacite, 23% low-TOC carbonaceous, 33% high-TOC carbonaceous	Ore stockpile, blend of 15% dacite, 9% low-TOC carbonaceous 76% high-TOC carbonaceous	Ore stockpile, blend of carbonaceous and dacite
Sample Head Grade									
Gold (g/t)	1.10	1.20	2.36	2.62	2.51	1.49	1.47	2.18	1.23
TOC (%)	0.15	0.15	1.32	1.31	1.29	0.66	0.67	1.01	0.88
S^{2-} (%)	0.01	0.02	0.93	0.96	0.96	0.64	0.65	0.84	0.81
Reagent Conditioning Time (hours)									
Line 1, Pre-Aeration	4	4	4	4	4	4	0	2	2
Line 1, Kerosene Conditioning	4	4	4	4	4	4	6	4	4
Line 2, Pre-Aeration	4	4	4	4	2	2	2	2	2
Line 2, Kerosene Conditioning	4	4	4	2	4	4	4	4	4
Leach Time (hours)									
Line 1	24	24	24	24	24	24	24	24	24
Line 2	24	16	20	20	20	20	20	20	20
Kerosene Addition (kg/t ore)									
Line 1	0.3	0.3	2.0	2.0	2.0	1.15	1.15, then 1.5	2.0	2.0
Line 2	0.3	0.3	2.0	2.0	2.0	1.15	1.15, then 1.5	2.0	2.0
Cyanide Addition (mg CN/L solution)									
Line 1	600	600	600	600	600	500	400, then 500	500	250
Line 2	600	600	600	600	600	500	400, then 250	250	250

Parameter	Stage 7	Stage 7A	Stage 8	Stage 9	Stage 10	Stage 11	Stage 12	Stage 13	Stage 14	
Resin Form										
Line 1	fresh	fresh	fresh	fresh	fresh	fresh	eluted 1 cycle	eluted 1 cycle	MTA 9920[a]	MTA 9930[a]
Line 2	fresh	fresh	fresh	fresh	fresh	eluted 1 cycle	eluted 2, then 1 cycle[b]	eluted 1 cycle[c]		
Gold Recovery (%) based on Head and Tailings Assays										
Line 1	86.2%	85.9%	84.7%	84.7%	84.6%[d]	84.8%	83.8%	83.8%	73.1%	
Line 2	85.6%	84.2%	83.6%	83.1%	86.4%	84.7%	83.1%	84.0%		80.20%
Difference (Line 1–Line 2)	0.6%	1.7%	1.1%	1.6%	−1.8%	0.1%	0.7%	-0.2%	--7.1%	
Difference (Line 1–Line 2) after Deducting Effect of 0.6% Higher Recovery in Line 1 than Line 2		1.1%	0.5%	1.0%						
Average Gold Loading on Resin (g/t)	410	530	730	890	1000	580	630	1070	350	520
Average Copper Loading on Resin (g/t)	2,600	4,000	17,700	12,000	12,000	13,500	11,000	13,000	2,000	14,000

Notes: [a] Purogold resins, fresh using 20 mL resin/L slurry.
[b] Ran out of 2 cycle resin so switched to 1 cycle resin.
[c] 20 mL resin/L slurry.
[d] Agitators in Line 1 were becoming worn out and not mixing properly resulting in lower recovery in Line 1 than Line 2.

Profiles of leaching circuit were obtained once for Stages 5, 6, 7, and 10. Results of these profiles are shown in Table 13-18 for single line operation and in Table 13-19 for parallel line operation.

The adsorbent (carbon or resin) activity value provided in the tables is the percentage of gold adsorbed by 1 g of adsorbent in one hour from 1 L of solution containing 10 mg/L gold in a cyanide solution. Since these profiles represent the status of the system at an instant in time, they are not truly representative of longer-term values, but they do show how the values of the various parameters change from tank to tank. The activity values in Table 13-18 show that the loaded resin (Stage 6 onwards) retained more activity than loaded carbon (Stage 5). The most critical of all the parameters is the gold extraction from the solids. The results show that almost all the leaching occurs in the first tank and that the incremental leaching in the last two tanks is minimal, amounting to approximately 1.0% in Tank 5 and 0.5% in Tank 6 for the adsorbent loadings in these tests.

Table 13-18: Whole Ore CIL and RIL Leach Tank Profiles (Performed at the Santa Luz Pilot Plant)

Date		Tank						
		Conditioning	TQ 01	TQ 02	TQ 03	TQ 04	TQ 05	TQ 06
8 Dec 16	***Parameter, Stage 5, CIL, Carb. High-TOC***							
	Au–Solids (g/t)	1.95	0.51	0.42	0.42	0.38	0.38	0.35
	Au–Solution (mg/L)	<0,001	0.014	<0,001	<0,001	<0,001	<0,001	<0,001
	Au–Carbon (g/t)	2.33	224.17	73.32	16.77	9.53	5.44	2.58
	Activity, Carbon (%)	28.44	29.54	33.27	34.30	37.53	39.86	45.45
	Ca–Carbon (%)	0.073	0.140	0.115	0.130	0.133	0.108	0.105
	Conc. Carbon in Leach (g/L)	30.00	25.00	15.00	15.00	12.00	10.00	13.00
	Conc. Cyanide (mg/L)	<0,10	596	559	406	398	299	226
	Conc. Oxygen (mg/L)	2.17	8.59	7.92	8.53	8.58	8.17	8.68
	% Solids	42	37	37	38	37	37	35
	As–Solution (mg/L)	<0,01	0.10	0.04	0.01	0.09	0.09	0.10
	Fe–Solution (mg/L)	0.29	3.71	3.66	4.53	4.52	5.38	5.97
	Cu–Solution (mg/L)	<0,001	37.76	38.23	38.35	38.99	39.67	40.41
	TOC	1.46	1.46	1.49	1.44	1.45	1.39	1.46
	pH	10.41	10.84	10.77	10.68	10.63	10.45	10.35
16 Mar 17	***Parameter, Stage 6, RIL, Carb. Low-TOC***							
	Au–Solids (g/t)	1.184	0.305	0.265	0.249	0.243	0.229	0.225
	Au–Solution (mg/L)	<0,001	0.016	<0.001	<0.001	<0.001	<0.001	<0.001
	As–Solids (mg/L)		431	483	470	432	428	435
	Au–Resin (g/t)		252.53	159.17	79.23	58.61	49.98	29.40
	Cu–Resin (g/t)		7,258	7,992	7,870	9,183	8,168	8,928
	Fe–Resin (g/t)		439	143	250	183	167	640
	Activity, Resin (%)		46.72	43.24	45.22	61.57	64.21	71.93
	Ca–Resin (%)		0.043	0.048	0.058	0.048	0.023	0.018
	Conc. Resin (g/L)		60.00	23.00	15.00	10.00	10.00	10.00
	Conc. Cyanide (mg/L)		537	562	557	557	546	531
	Conc. Oxygen (mg/L)	7.89	6.82	6.76	7.08	7.14	6.90	7.20
	% Solids	44.08	41.06	41.68	40.23	39.85	40.19	39.49
	As–Solution (mg/L)		0.07	0.07	0.08	0.06	0.05	0.09
	Fe–Solution (mg/L)	<0,01	0.35	0.48	1.12	1.52	2.33	3.29
	Cu–Solution (mg/L)	0.07	24.99	28.05	30.38	34.84	36.22	34.54
	S%	0.80	0.77	0.78	0.77	0.76	0.76	0.76
	TOC (%)	0.86	0.79	0.82	0.82	0.82	0.85	0.88
	pH	10.04	10.82	10.84	10.73	10.81	10.91	10.80

Date		Tank						
		Conditioning	TQ 01	TQ 02	TQ 03	TQ 04	TQ 05	TQ 06
4 Mar 17	***Parameter, Stage 7, RIL, Dacite***							
	Au–Solids (g/t)	1.052	0.259	0.204	0.191	0.172	0.149	0.148
	Au–Solution (mg/L)	0.003	0.010	0.003	0.003	0.004	0.007	0.008
	As–Solids (mg/L)	532	556	557	565	546	561	512
	Au–Resin (g/t)	-	444.586	231.298	91.787	47.904	29.115	15.195
	Cu–Resin (g/t)	-	3123	1397	159	295	200	100
	Fe–Resin (g/t)	-	391	422	369	709	579	390
	Activity, Resin (%)	-	74.56	75.73	76.42	77.80	78.76	82.09
	Ca–Resin (%)	-	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
	Conc. Resin (g/L)	-	60	23	15	10	10	10
	Conc. Cyanide (mg/L)	-	398.97	312.24	322.65	291.42	235.91	239.38
	Conc. Oxygen (mg/L)	8.26	7.33	7.59	7.70	7.66	7.85	7.63
	% Solids	42.88	40.29	39.58	39.14	39.79	39.25	40.91
	As–Solution (mg/L)	0.08	0.17	0.17	0.30	0.23	0.13	0.28
	Fe–Solution (mg/L)	0.63	0.76	1.08	1.49	0.57	7.80	2.23
	Cu–Solution (mg/L)	0.00	0.15	0.05	0.02	0.36	0.05	0.05
	S%	<0,01	<0,01	0.02	<0,01	<0,01	0.02	<0,01
	TOC (%)	0.16	0.16	0.15	0.17	0.16	0.16	0.16
	pH	10.15	11.02	11.25	11.00	11.13	10.98	10.99

Table 13-19: Whole Ore CIL and RIL Leach Tank Profiles (Performed at the Santa Luz Pilot Plant) with Parallel Lines

Date		Tank, Line 1							Tank, Line 2					
		Conditioning	**TQ 01**	**TQ 02**	**TQ 03**	**TQ 04**	**TQ 05**	**TQ 06**	**Conditioning**	**TQ 01**	**TQ 02**	**TQ 03**	**TQ 04**	**TQ 05**
31 May 17	***Parameter, Stage 10, RIL, Carb. High-TOC***													
	Au–Solids (g/t)	2.646	0.567	0.469	0.419	0.403	0.381	0.401*	2.441	0.555	0.448	0.440	0.408	0.383
	Au–Solution (mg/L)	-	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	-	<0.001	<0.001	<0.001	<0.001	<0.001
	As–Solids (mg/L)	515	525	530	542	552	516	558	484	491	501	455	478	463
	Au–Resin (g/t)	-	857.216	413.895	159.337	111.870	34.385	62.683	-	1,064.01	695.19	242.07	144.65	68.72
	Cu–Resin (g/t)	-	11,748	11,012	13,526	14,529	16,999	15,824	-	12,044	11,088	11,979	13,716	15,794
	Fe–Resin (g/t)	-	285	183	249	240	276	156	-	208	297	412	122	107
	Activity, Resin (%)	-	68.48	70.03	71.78	71.29	71.03	72.33	-	60.48	58.43	63.52	64.43	69.41
	Ca–Resin (%)	-	0.01	0.01	0.02	0.01	0.04	0.02	-	<0,01	<0,01	<0,01	<0,01	<0,01
	Conc. Resin (g/L)	-	45	20	15	10	10	10	-	45	20	15	10	10
	Conc. Cyanide (mg/L)	-	548.15	468.36	392.03	364.28	353.87	315.71	-	530.81	482.24	392.03	381.63	357.34
	Conc. Oxygen (mg/L)	6.21	7.06	7.37	7.58	7.65	7.53	7.48	6.49	7.27	7.59	7.38	7.63	7.58
	% Solids	43.62	40.04	38.85	40.56	40.62	38.92	40.90	44.83	38.80	39.14	39.58	37.87	37.78
	As–Solution (mg/L)	-	0.06	0.09	0.06	0.08	0.15	0.23	-	0.07	0.05	0.03	0.06	0.13
	Fe–Solution (mg/L)	-	8.40	10.79	11.14	11.92	12.41	14.58	-	9.32	12.31	12.52	14.01	14.45
	Cu–Solution (mg/L)	-	39.00	33.48	32.50	32.26	31.80	28.56	-	36.90	35.94	34.16	33.42	33.08
	S%	0.92	0.95	0.98	1.00	0.96	0.96	1.00	0.92	0.94	0.92	0.92	0.91	0.90
	TOC (%)	1.33	1.31	1.31	1.26	1.29	1.29	1.25	1.29	1.24	1.25	1.26	1.29	1.28
	pH	8.65	10.24	10.28	10.43	10.53	10.62	10.68	8.99	10.21	10.27	10.43	10.46	10.50

Stage	Recovery Summary for All Four Stages	Tank, Line 1							Tank, Line 2					
		Conditioning	**TQ 01**	**TQ 02**	**TQ 03**	**TQ 04**	**TQ 05**	**TQ 06**	**Conditioning**	**TQ 01**	**TQ 02**	**TQ 03**	**TQ 04**	**TQ 05**
Stage 5	Cumulative Recovery	-	74.0%	78.4%	78.7%	80.6%	80.7%	82.2%	-	-	-	-	-	-
	Incremental Recovery per Tank	-	-	4.4%	0.3%	2.0%	0.1%	1.5%	-	-	-	-	-	-
Stage 6	Cumulative Recovery	-	74.2%	77.6%	79.0%	79.5%	80.7%	81.0%	-	-	-	-	-	-
	Incremental Recovery per Tank	-	-	3.4%	1.4%	0.5%	1.2%	0.3%	-	-	-	-	-	-
Stage 7	Cumulative Recovery	-	75.4%	80.6%	81.8%	83.7%	85.8%	85.9%	-	-	-	-	-	-
	Incremental Recovery per Tank	-	-	5.2%	1.2%	1.8%	2.2%	0.1%	-	-	-	-	-	-
Stage 10	Cumulative Recovery	-	77.0%	81.0%	83.0%	83.6%	84.5%	-	-	77.3%	81.7%	82.0%	83.3%	84.3%
	Incremental Recovery per Tank	-	-	4.0%	2.0%	0.6%	0.9%	-	-	-	4.4%	0.3%	1.3%	1.0%
Average Incremental Recovery per Tank		-	-	-	-	-		-	-	-	-	-	-	-
From test data		-	75.2%	4.2%	1.2%	1.2%	1.1%	0.6%	-	-	-	-	-	-
Rounded generic values		-	75.0%	4.5%	1.5%	1.5%	1.0%	0.5%	-	-	-	-	-	-

Bottle-roll tests were conducted in parallel with the pilot-plant tests. Comparing the bottle-roll tests with the pilot-plant tests shows the following phenomena:

- Carbonaceous ore (preg-robbing): approximately 4% higher recovery in bottle-roll tests compared to pilot-plant tests
- Dacitic ore: similar recovery in bottle-roll tests compared to pilot-plant tests.

The ratio of resin to ore in bottle-roll tests compared to the pilot plant is as follows:

- Bottle-roll tests: 45 mL wet resin/t ore processed
- Pilot-plant tests: 5 mL wet resin/t ore processed.

As a consequence of the above values the loaded resins in bottle-roll tests have a much lower metal loading than that in pilot plant tests, and consequently higher residual activity (faster kinetics) . The tests show that the higher activity (faster kinetics) allows the bottle roll resin to better counter the strong preg-robbing propensity of carbonaceous ore. With dacitic ore, the preg-robbing propensity of the ore is relatively weak and thus there is no difference between bottle-roll and pilot-plant recoveries. The results show the limitation of trying to use bottle roll tests to estimate gold recovery from preg-robbing ores under plant conditions.

The residual activity of the loaded resin can also be affected by species that have not been fully stripped in previous cycles of use. Fresh carbon or resin was used in all the tests up to Stage 10. For Stage 11 fresh resin was used in Line PP 01 and eluted resin in Line PP 02. For Stages 12 and 13 eluted resin was used on both lines.

For Stage 14 two Purogold resins were tested: MTA 9920 (a mixed weak-base resin that can be eluted with alkaline cyanide) and MTA 9930 (a mixed weak-base/strong-base resin that must be eluted with acidic thiourea, similar to the Dow Ambersep XZ-91419 resin). The MTA 9920 did not provide good gold extraction, but had a low copper loading. The MTA 9930 resin performed similarly to the Dow resin, but with slightly lower gold extraction and approximately the same high copper loading as the Dow resin. The Purogold resins were found to be not as durable as the Dow resin, with the MTA 9920 resin having a higher tendency to break than the MTA 9930 resin.

13.15 Summary of Recovery Data

A summary of bottle roll test recovery data is shown in Table 13-20.

Table 13-20 was generated to provide values for the ore reserve calculations. The recovery values are for the following ore categories:

- Carbonaceous ore
- C1 Dacitic ore
- Antas 3 Dacitic ore:
    - high-sulphide (refractory)
    - non-high-sulphide.

Table 13-20: Relationship Between Ore Grade, Recovery, TOC, and Arsenic in Bottle Roll Tests

Laboratory	Sample	Au (g/t)	Rec. (%)	As (g/t)	TOC (%)	Recovery Weighting	Recovery X Weight
Carbonaceous Ore							
Hazen	C1 bulk	3.71	85		1.71	1	85
	C1MP028	1.89	78		1.24	1	78
	C1MP031	1.28	80		1.42	1	80
RDI	C1 bulk	3.84	91		1.71	1	91
	C1MP028	1.89	80		1.24	1	80
	C1MP031	1.16	77		1.42	1	77
CSIRO	C1MP029	1.27	80		1.47	1	80
SGS	C1MP023	3.52	82.1		0.97	1	82.1
	C1MP034	1.93	80.9		1.86	1	80.9
	C1MP034	1.42	66.2		2.61	1	66.2
	C1MP034	1.13	80.7		0.83	1	80.7
	C1MP033	1.62	78.4		1.87	1	78.4
	C1MP006	1.05	76.8		1.47	1	76.8
	C1MP032	1.13	79.8		1.50	1	79.8
	C1MP013	2.29	89.1		0.76	1	89.1
	C1MP016	2.24	82.8		1.42	1	82.8
	C1MP040	2.81	80.3		1.82	1	80.3
SGS	C1MP029a	3.28	74.6		1.29	1	74.6
Pilot Plant	Stage 6 (low TOC)	1.20	83.3		0.90	5	416.5
	Stage 8 (high TOC)	2.36	84.7		1.32	5	423.5
	Stage 9 (high TOC)	2.31	84.7		1.31	5	423.5
	Stage 10 (high TOC)	2.51	84.6		1.29	5	423
	Stage 11 (blend)	1.49	84.8		0.66	5	424
	Stage 12 (blend)	1.47	83.8		0.65	5	419
	Stage 13 (blend)	2.18	83.8		1.01	5	419
Weighted Average Recovery			**82.9**			53	4,391.2
Dacitic Ore C1							
Hazen	C1MP027a	1.43	78.2[a]	6,890			
SGS	C1MP015	2.36	80.6[a]	7,470		1	78.2
	C1MP027	1.03	80.4[a]	2,100		1	80.6
	C1MP022	0.83	89.8[a]	1,260		1	80.4
	C1MP025	0.74	91.6[a]	<500		1	89.8
	C1MP030	1.10	86.1[a]	<500		1	91.6
SGS	C1MP036	1.12	78.5[a]	<500		1	86.1
	C1MP023	2.42	86.8[a]	1,430		1	78.5
	C1MP011	1.31	84.0[a]	1,200		1	86.8
	C1MP021	0.77	84.0[1]	1,090		1	84

Laboratory	Sample	Au (g/t)	Rec. (%)	As (g/t)	TOC (%)	Recovery Weighting	Recovery X Weight
SGS	C1MP028	1.43	86.8	7,800		1	86.8
	C1MP037	0.80	38.1	2,660			
	C1MP041	0.77	80.4	1,530		1	80.4
	C1MP030	1.06	86.1	<500		1	86.1
	C1MP040	2.60	88.2	790		1	88.2
	C1MP002	1.16	78.0	1,000		1	78
	C1MP007	0.75	86.7	810		1	86.7
Pilot Plant	Stage 7	1.10	86.2	540		5	431
Weighted Average Recovery Excluding High-As High-Sulphide Ore Samples			**84.7**			**20**	**1,693.2**
Dacitic Ore Antas 3							
SGS	A3MP009	1.42	84.1[a]	1,400			
	A3MP010	1.67	33.2[a]	2,770			
	A3MP018	1.50	41.6[a]	1,770			
	A3MP021	1.71	90.9[a]	690			
SGS	A3MP010	1.74	33.2	4,040			
	A3MP005	2.18	69.8	2,900			
	A3MP009	1.88	75.7	1,100			
	A3MP011	1.63	63.4	2,120			
	A3MP024	2.28	88.4	890			
	A3MP018	4.00	25.8	6,190			
	A3MP019	1.36	78.6	880			
	A3MP021	2.32	88.6	910			
	A3MP003	1.25	87.8	<500			
	A3MP008-A	2.27	44.1	3,980			
	A3MP008-B	2.83	40.6	5,410			
	A3MP008-C	1.59	38.3	2,390			
	A3MP023	0.98	78.8	820			
Average Recovery, High-sulphide Ore			**36.7**				
Average Recovery, Non-High-sulphide Ore			**80.6**				

Note: [a] Values are CIL data

All the recovery values used for the carbonaceous ore are those from RIL tests. Due to the limited amount of data on dacitic ore, the recovery values used for dacitic ore include those from CIL tests that were done prior to determination of appropriate RIL test procedures, in addition to RIL tests. Results of CIL tests are similar to those of RIL tests when using appropriate RIL test procedures; accordingly, the use of CIL data in addition to RIL data for determining dacitic ore is considered acceptable. Pilot plant testwork, which comprises seven sets of data, has been given a weighting of five times that of bottle-roll test data since this testing is more closely related to expected plant operating results than the bottle-roll tests.

Plotting of the recovery values against the gold ore grade, against TOC for carbonaceous ore, and against arsenic for dacitic ores within each ore category, shows no clear correlation, consequently the average recovery values of the different ore types is considered to give a reasonable estimate of the expected value.

The dacitic ore recoveries show correlation with arsenic content, particularly in the Antas 3 deposit; that for C1 is not clear, with some high arsenic ores giving good recoveries and others not. Fortuitously, the high-sulphide ore in Antas 3 is isolated in one contiguous zone, making it easy to avoid. The plan is to provide blast hole assays for both TOC and arsenic, in addition to gold; consequently, it is expected that it will be possible to separately process high-sulphide, high-arsenic ores from C1.

13.16 Initial Santa Luz Resin Elution Tests

Santa Luz laboratory carried out preliminary acid washing tests on loaded resin using approximately 100 g/L sulphuric acid, followed by batch elution with approximately 76 g/L thiourea in approximately 100 g/L sulphuric acid solution. However, the tests did not include electrowinning. Hence elution efficiency in the batch tests reported in Table 13-21, with once-through eluant solution, is expected to be higher than in a corresponding Zadra type of process, with electrowinning and recycled eluate.

For loaded resin that contains close to 1,000 g/t Au, the principal contaminants are copper, at approximately 14,000 g/t, and zinc, at approximately 1,500 g/t. The other contaminants—iron, silver, and nickel—are all <600 g/t. Clearly, copper is, by far, the largest contaminant.

Although the plan for the plant is to conduct the elution at close to 60 °C, duplicate tests were done at ambient temperature and at 50°C. The initial set of tests included an acid-washing step prior to elution, as is planned for the plant. These initial tests indicate similar results, independent of whether the elution is done at ambient temperature or at 50°C. The results indicate that acid washing does remove approximately 30% of the copper, 50% of the iron, and most of the nickel and zinc. Subsequent tests were conducted to assess elution at ambient temperature and at 50 °C without prior acid washing.

The results in Table 13-21 indicate that, even with the once-through eluant solution, acid washing is required ahead of elution to get the gold loading on the eluted resin down to the target value approximately 50 g/t. While not an issue in the tests with once-through solutions, the limited efficacy of base-metal stripping in the acid wash increases the base-metal load in the elution strip. With a Zadra type of elution with recycled eluent this could inhibit base-metal stripping if base-metal concentrations build up in the eluate.

Table 13-21: Loaded Resin Acid Washing and Elution

Metal	Loaded Resin Assay (g/t/dry resin)	Acid-Washing		Elution		Combined Acid-Washing plus Elution Metal Removal (%)
		Acid-Washed Resin Assay (g/t/dry resin)	Metal Removal (%)	Eluted Resin Assay (g/t/dry resin)	Metal Removal (%)	
Ambient Temperature Acid wash followed by Ambient Temperature Elution						
Gold	909	903	1	51	94	94
Copper	14,038	9,849	30	304	97	98
Iron	241	124	49	30	75	87
Silver	239	216	10	11	95	95
Zinc	1,275	44	97	36	18	97
Nickel	567	329	42	0	100	100
Ambient Temperature Acid wash followed by 50°C Elution						
Gold	909	903	1	53	94	94
Copper	14,038	9,917	29	234	98	98
Iron	241	149	38	50	67	79
Silver	239	223	7	8	96	97
Zinc	1,275	3[a]		16		99
Nickel	567	68	88	0	100	100
Ambient Temperature Elution without Acid Wash						
Gold	1,025			221	78	
Copper	12,214			492	96	
Iron	162			61	62	
Silver	277			35	87	
50°C Elution without Acid Wash						
Gold	1,025			106	90	
Copper	12,214			317	97	
Iron	162			42	74	
Silver	277			23	92	

Note: [a] Probable erroneous assay

13.17 Optimization of RIL Process

The Mintek test laboratory in South Africa was contracted to partner with the Santa Luz pilot plant to optimise RIL process operating conditions. Mintek is the organization that originally developed the Minix (Ambersep XZ-91419) gold selective resin and assisted in its first commercial application for gold recovery from the preg-robbing carbonaceous ore at Penjom in Malaysia. Samples of leach slurry from the Santa Luz were sent to Mintek for the first phase of testwork. Based on the results of their laboratory testwork, Mintek then provided new operating targets for Santa Luz pilot plant operation. Under Mintek's direction the Santa Luz pilot plant was then operated to evaluate gold recovery performance under conditions proposed by Mintek (Stage 16). Results from the pilot plant testwork are summarized in Table 13-22 and discussed in the paragraphs below. Stage 17 of pilot plant operation used the optimized conditions identified in Stage 16 (higher gold loading on resin, five tanks and 1.5 kg/t kerosene) to generate loaded resin for the second phase of Mintek testwork to optimise gold recovery from the resin.

Table 13-22: Results from Santa Luz Pilot Plant RIL Optimization Tests

Parameter	Stage 16				
	Phase 1	Phase 2	Phase 3	Phase 4	Stage 17
Start Date	25-Feb-19	10-Mar-19	27-Mar-19	4-Apr-19	12-Nov-19
End Date	9-Mar-19	26-Mar-19	2-Apr-19	9-Apr-19	29-Dec-19
Sample Location	C1	C1	C1	C1	C1
Sample Type	Blend	Blend	Blend	Blend	Blend
Sample Head Grade					
Gold (g/t)	2.18	1.86	1.80	1.77	1.96
TOC (%)	0.70	0.67	0.67	0.69	
Kerosene Addition (kg/t ore)	7.8	1.56	1.56	1.56	1.56
Reagent Conditioning Time (hours)	4	4	4	4	4
No of Leach tanks	8	8	8	5	5
Leach Time (hours)	32	32	32	20	20
Cyanide Addition (kg/t)	0.53	0.53	0.53	0.53	
Resin Concentration (mL/L pulp)	16.4	16.4	4.2	13.4	
Resin Flow Rate (mL/d)	205	205	210	168	
Resin Inventory (mL)	3280	3280	840	1667	
Total Resin Residence Time (h)	384	384	96	240	
Av. Gold Recovery (%) based on Head and Tail Assays	**89.4**	**86.5**	**85.9**	**85.0**	**84.2**
Av Gold on Loaded Resin (g/t)	2,200	1,470	1,080	1,550	1,900
Av. Copper on Loaded Resin (g/t)	8,880	10,350	11,260	10,480	10,380

Mintek suggested increasing:

- Gold loading on the resin (increase resin residence time) to reduce copper loading, as gold is able to displace some of the copper
- The number of tanks from 5 to 8 (32 h residence time instead of 20 h)
- Kerosene addition from approximately 2kg/t to approximately 8 kg/t.

Comparing gold and copper loadings in Table 13-22 with earlier results in Table 13-17 shows that increasing resin residence time is able to increase gold loading and reduce copper loading on the resin.

Comparing Stage 16 Phase 1 and Phase 2 results in Table 13-22 shows that increasing kerosene addition from 1.56 kg/t to 7.8 kg/t is able to improve gold recovery by 2.9%; however, the additional recovery does not pay for the cost of the additional kerosene. Thus, Mintek's suggestion to increase kerosene addition to 7.8 kg/t could not be justified.

Comparing Stage 2 results with Stage 4 (base case) results shows that it was possible to increase gold recovery by 1.5% by increasing residence time by 12 h (3 more tanks) with an additional 22% resin in each tank. The extra leach tanks and higher resin concentration in Phases 1 and 2 almost double the in-circuit resin inventory; that will cause more gold lock up and higher resin wear in the circuit. Hence, it is likely that increasing the RIL residence time to 32 h (3 extra tanks) will not be cost effective.

Figure 13-2 compares the incremental gold recovery using tank profile data from Stage 17 (roughly 1,900 g/t Au on loaded resin) and Stage 10 (previously discussed, approximately 1,000 g/t Au on loaded resin). While the data are noisy, the trend line shows that the increasing the gold grade on the loaded resin has pushed the recovery process down the train. Hence the 1% incremental recovery criterion previously used to justify increasing RIL duration from 16 h to 20 h may justify increasing leach duration to 24 h at the higher loaded resin target of approximately 2,000 g/t, though addition of gravity treatment ahead of whole-ore leaching may make this unnecessary.

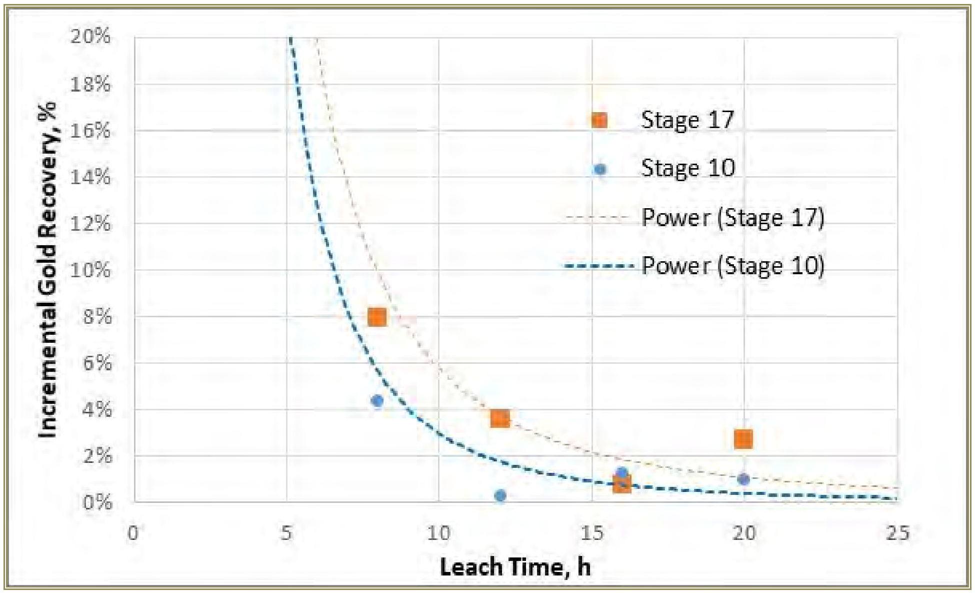


Figure 13-2 Comparison of Incremental Recovery by Tank for Stage 10 and Stage 17

13.18 Optimization of Acid Washing and Elution Process

In their first phase of testwork Mintek flagged that the proposed acid wash/elution procedure based on that used at the Penjom Mine needed to be optimized for the higher base-metal loadings on the Santa Luz resin. In particular, base-metal stripping ahead of elution needed to be improved to reduce the base-metal load in the recycled eluate that could inhibit effective base-metal stripping in elution (minimal base-metal removal by electrowinning). They suggested that options to improve base-metal removal ahead of elution could include:

- Increased volume of acid used for acid wash
- Carrying out acid wash at higher temperature
- Using cyanide strip to remove base metals ahead of acid thiourea to strip the gold.

About 6.5 L of loaded Dow XZ-91419 resin from the Stage 17 operation of the Santa Luz pilot plant was sent to Mintek for their Phase 2 testwork to optimise the gold recovery from the resin—acid washing, and elution with simultaneous electrowinning. Loaded resin composition is shown in Table 13-23:.

Table 13-23: Resin Acid Wash and Elution Results

Resin	Au (g/t)	Ag (g/t)	Cu (g/t)	Ni (g/t)	Zn (g/t)	Fe (g/t)	Co (g/t)	Mn (g/t)
Loaded	1,590	311	10,486	1,525	2,605	376	202	3,810
Acid washed	1,555	265	10,601	815	221	382	232	0
Eluted	57	0	404	25	164	417	218	0

The Mintek testwork showed that base-metal stripping in the 100 g/L sulphuric acid wash could be improved by increasing the temperature to 60°C and increasing the volume of acid to 0.8 bed volumes (BV) to 1 BV. However, this still left most of the copper on the resin. Fortuitously it was observed that the first 0.6 BV of eluate had a high copper concentration and low gold concentration. Hence between hot acid washing and diverting the first BV of eluate, most of the copper could be kept out of the recycled eluent. It was also observed that if the initial diverted high-copper solution was allowed to cool to ambient temperature, around 50% of the copper (and 25% of the silver) precipitated out and could be filtered off. This would allow the filtrate to be returned to the eluent tank to be used in make-up of the next batch of acid thiourea solution.

Alternatively, use of 40 g/L caustic with 25 g/L sodium cyanide wash at 60 °C could further improve base-metal stripping ahead of acid thiourea elution. However, this approach was thought impractical due to the large volume of strip solution required, elution of part of the gold and silver along with the base metals, and increased osmotic shock on the resin going from concentrated alkaline solution to concentrated acid solution.

No base-metal removal from solution by electroplating onto the cathode was observed during the simultaneous elution/electrowinning process for copper concentrations approaching 1,000 mg/L, while gold was plated out to around 10 mg/L (94% Au plated out) in 100 minutes. In the second cycle of eluate use, gold solution concentration fell to 10 mg/L in 80 minutes before plating stopped falling to about 90% Au recovery at 100 minutes.

Optimized acid wash and elution procedure:

- Acid wash at 60 °C with 2 h recycle of 0.8BV to 1 BV of 100 g/L sulphuric acid solution
- Water wash with 0.5 BV at 60°C
- Split elution with 76 g/L thiourea in 60 g/L sulphuric acid solution at 60°C:
  - First 0.6 BV of eluate diverted to copper recovery
  - Second stage with 2 BV of eluent, solution passed through electrowinning before recycle to elution column for 6 h
- Water wash with 0.5 BV before returning resin to RIL circuit.

The volumes and compositions of the solution streams produced during the Mintek confirmatory test under optimized acid wash and elution conditions are shown in Table 13-24. The tests achieved 96% Au elution efficiency and 84% recovery by electrowinning.

Table 13-24: Volume and Composition of Solutions Produced during Confirmatory Elution Test

Solution	Volume, BV	Solution Concentrations (mg/L)								H_2SO_4, g/L
		Au	Ag	Cu	Ni	Zn	Fe	Co	Mn	
Acid Eluate	0.8	<0.08	0.22	72	364	730	3	<2	1399	65
Acidic Wash Water	0.5	<0.08	0.19	66	160	409	16	<2	664	37.5
Combine Acid Effluent	1.3	<0.08	0.21	70	285	606	8	<2	1116	54.4
Thiourea Filtrate	0.5	1	20	1018	271	92	11	<2	74	32
Final Spent Electrolyte	2	26	34	731	28	5	47	<2	5	63
Thiourea Wash Waters	0.5	3	20	279	8	15	12	<2	<2	21

Notes: Detection limit 0.08 mg/L for Au, 0.1 mg/L for Ag and 2 mg/L for Cu, Ni, Zn, Fe, Co & Mn

13.19 Cyanide Detoxification and Arsenic Precipitation Tests

Cyanide detoxification and arsenic precipitation testing was done on a high-arsenic-content dacitic ore from the C1 deposit. This sample was used because of the expectation that the high arsenic content of the ore (approximately 6,000 mg/t) would provide a relatively high arsenic content in the leach solution. The results of the testing are shown in Table 13-25.

Table 13-25: Cyanide Detoxification and Arsenic Precipitation Test Results

Parameter Time	Unit (minutes)	Cyanide Detoxification								Arsenic Precipitation (minutes)		
		Feed Solution	After Cu Added	(minutes) 10	33	45	60	90	Total	0	90	Total
Reagent Additions												
$CuSO_4.5H_2O$	kg/t ore	-	0.40	-	-			-	0.40	-	-	-
$Na_2S_2O_5$[a]	kg/t ore	-	-	0.13		0.05	0.06	-	0.24	-	-	-
NaOH	kg/t ore	-	-				0.04	-	0.04	-	-	-
O_2[b]	m^3/t ore	-	-	10.00	23.00	12.00	1.005	-	60.00	-	-	-
$Fe_2(SO_4)_3$	kg/t ore	-	-	-	-	-	-	-	-	0.14	-	0.14
Measured Values			-	-	-	-	-	-	-	-	-	-
Cu	mg/L	<0.05	-	-	-	-	-	-	-	-	-	-
Fe	mg/L	<0.05	-	-	-	-	-	-	-	-	-	-
Zn	mg/L	<0.02	-	-	-	-	-	-	-	-	-	-
pH		-	8.40	8.30	8.60	8.50	8.40	8.50	-	-	8.5	-
Dissolved O_2	mg/L	-	6.20	-	-	20.40	-	5.10	-	-	-	-
Cyanide (as CN)		-	-	-	-	-	-	-	-	-	-	-
Free	mg/L	143	-	-	-	-	-	-	-	-	-	-
WAD	mg/L	132	82.00	-	-	54.00	45.00	-	-	-	-	-
Total	mg/L	-	-	-	-	-	36.00	-	-	-	-	-
Arsenic (as As)	mg/L	0.08	-	-	-	-		-	-	0.08	<0.05	-

Notes: [a] $Na_2S_2O_5$ added continuously at a rate of 0.013 kg/t ore from 0 to 10 minutes. Flow rate was too high, so no $Na_2S_2O_5$ added from 10 to 33 minutes. Then $Na_2S_2O_5$ added at a rate of 0.004 kg/t ore from 33 to 60 minutes. No $Na_2S_2O_5$ added after 60 minutes.
[b] O_2 added continuously at a rate of 1 m^3/t ore from 0 to 60 minutes.

As shown in Table 13-25, the cyanide-detoxification and arsenic-precipitation tests were conducted as sequential batch tests over periods of 90 minutes each. The tests show that cyanide detoxification and arsenic precipitation occur as expected and provide an indication of probable reagent consumptions.

Because the cyanide-detoxification test is a batch test, the copper addition is approximately ten times as much as that used in continuous industrial operations. In a continuous operation, only a small amount of copper is required since the copper acts as a catalyst to convert free cyanide to copper cyanide; the copper is then released and recycled to the solution as the copper cyanide is oxidized to cyanate. Copper sulphate solution is slightly acidic, and the amount added was sufficient to drop the solution pH to the requisite 8.0 to 8.5 range without acid addition. In continuous operation, with a much-reduced copper sulphate addition, the pH will decrease as a result of the following side reaction:

$$SO_2 + H_2O + \frac{1}{2} O_2 \rightarrow H_2SO_4$$

As a consequence, NaOH (or other alkaline source) will be required to maintain the target pH value.

The cyanide-detoxification test shows that the requisite weak-acid dissociable cyanide (CN_{WAD}) concentration of <50 mg/L was obtained within 60 minutes; accordingly, a retention time of 60 minutes is adequate. Reagent requirements for cyanide detoxification were shown to be as follows:

- Sodium metabisulphite ($Na_2S_2O_5$)................ 0.24 kg/t ore
- Sodium hydroxide (NaOH) .... 0.04 kg/t ore
- Oxygen (O_2) .... 60 m^3/t ore.

The test shows the discharge target, <50 mg/L CN_{WAD}, could be achieved with a low consumption of sodium metabisulphite (using a factor of 1.2 mg SO_2/mg CN instead of the more conventional 4 mg SO_2/mg CN used in a prior test). Based on the data from other mining operations, the actual SO_2 consumption is expected to be closer to 1.0 kg/t ore if a CN_{WAD} cyanide level of 10 mg/L is targeted.

The amount of dissolved arsenic in the product of cyanide detoxification was found to be negligible at 0.08 mg/L. As such, arsenic precipitation is not required; nevertheless, an arsenic precipitation test was conducted. The only reagent added for arsenic precipitation is ferric sulphate—$Fe_2(SO_4)_3$; the amount added was 0.14 kg/t ore, which resulted in a dissolved arsenic concentration of <0.05 mg/L after 90 minutes.

13.20 Antas 3 Dacitic-High-Sulphide Ore Tests

To determine the feasibility of processing dacitic-high-sulphide ore in the Antas 3 deposit, which does not leach well, a sample of this ore was obtained and sent to Hazen Research in Denver. The results of the flotation tests are presented in Table 13-26.

Table 13-26: Antas 3 High Sulphide Dacite Flotation Test Results

	Unit	Test 1	Test 2	Test 3	Test 4	Test 5	Test 6	Test 7
Sample Size	kg	1	1	1	1	10	10	10
Grind								
For Rougher Flotation	80% passing, µm	75	75	75	75	224	55	60
For Cleaner Flotation	80% passing, µm				~50[a]	9	22	42[a]
Gold Recovery								
Rougher	%	95.3	95.6	95.4	96.7	89.5	90.5	
Cleaner	%				89.8	72.3	81.9	
Concentrate Grade								
Rougher	g/t Au	21.5	20.4	31.8	29.9	35	54.2	
Cleaner	g/t Au				81.5	90.2	105	

Note: [a] No regrind of rougher concentrate

The initial flotation tests (Tests 1–4) using 1 kg samples focused on making a flotation concentrate. With a primary grind of 80% passing 75 µm and no regrinding of the rougher concentrate, it was possible to recover 90% of the gold into a cleaner concentrate of 80 g/t Au. A subsequent flotation test (Test 5), using a 10 kg sample with a primary grind of 80% passing 224 µm, and a regrind of the rougher concentrate to 80% passing 9 µm, resulted in 72% Au recovery into a cleaner concentrate with a concentrate grade of 90 g/t Au. In the next set of tests (Tests 6 and 7), the rougher flotation concentrates of two 10 kg samples were combined and split in two, with one split being reground to 80% passing 22 µm and the other split, which was 80% passing 42 µm, not reground. Cleaner flotation of both rougher concentrate splits gave practically identical results, with overall gold recovery of 82% and a concentrate grade of 105 g/t Au.

RIL leaching of the whole ore and of flotation cleaner concentrate resulted in poor extraction with overall gold recoveries of 20% or less as shown in Table 13-27. RIL leaching of roasted flotation rougher concentrate and of flotation cleaner concentrate gave overall gold recoveries of 80% and 71% respectively. An attempt to oxidize a sample of flotation cleaner concentrate at ambient pressure with oxygen addition and close to boiling temperature did not appear to have any effect. It had been planned to include an autoclave oxidation test followed by RIL leaching, but insufficient sample was available and the testwork was terminated.

Table 13-27: Antas 3 High Sulphide Dacite RIL Test Results

RIL Leaching	CN Concentrate (g NaCN/L)	RIL Recovery (%)	Overall Recovery (%)
Whole Ore			
P_{80} 74 µm Grind	0.5	20	18
P_{80} 39 µm Grind	0.5	23	21
Roasted Rougher Concentrate	5	88	80
Cleaner Concentrate			
No Oxidation	5	14	11
Roasted	5	86	71

14 MINERAL RESOURCES ESTIMATE

The Mineral Resources were estimated from a block model constructed by Santa Luz Project personnel in March 2017 using the results from drilling conducted by Brio during 2016. RPA subsequently audited the model and found that it was reasonably prepared and provided a good representation of the geologic data.

The Santa Luz Project consists of six deposits: C1, Antas 2, Antas 3, Mansinha South, Mansinha North, and Mari. The largest of these are C1, Antas 3, and Antas 2, which are the only deposits used in the Mineral Resource and Mineral Reserve estimates.

In late 2016, a drill program was completed with the primary purpose of better defining the Mineral Resource in support of the Open Pit Feasibility Study. The program comprised a total of 4,036.71 m of diamond drilling. Compared to the previous Mineral Resource estimate reported in the 2016 PFS (RPA, 2016), overall contained metal inclusive of Mineral Reserves has decreased by approximately 2% as a result of the better definition of zones of high-grade mineralization.

SLR has summarized the Mineral Resource estimate in Table 14-1, based on the end-of-2016 topographic surface, which has not changed. The Mineral Resources in Table 14-1 are exclusive of Mineral Reserves as of June 30, 2020. This Mineral Resource estimate conforms to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM Definition Standards [2014]). No additional drilling or resource estimation work has been completed at Santa Luz since March 2017.

Table 14-1: Summary of Mineral Resource Estimate (Exclusive of Reserves)—June 30, 2020

Category of Mineral Resource	Tonnes ('000s)	Gold Grade (g/t)	Contained Gold (oz)
Measured—Open Pit	9,986	1.22	390,306
Measured—Underground	121	1.94	7,561
Indicated—Open Pit	562	0.99	17,924
Indicated—Underground	5,913	2.55	484,066
Total Measured & Indicated	**16,582**	**1.69**	**899,857**
Inferred—Open Pit	694	1.29	28,748
Inferred—Underground	6,560	2.19	461,367
Total Inferred	**7,254**	**2.09**	**490,115**

Notes: **1.** CIM Definition Standards (2014) were followed for Mineral Resource estimates.
2. Open Pit Mineral Resources are reported at a cut-off grade of 0.50 g/t Au.
3. Underground Mineral Resources are reported at a cut-off grade of 1.5 g/t Au.
4. Mineral Resources are exclusive of Mineral Reserves.
5. No minimum thickness was used in the resource estimation.
6. Mineral Resources are estimated using a gold price of US$1,500/oz and constrained by a pit shell.
7. Totals may not add due to rounding.

SLR and Equinox are not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that would materially affect the Mineral Resource estimate. Mineral resources are not Mineral Reserves, and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into a Mineral Reserve upon application of economic factors.

14.1 Resource Database

Santa Luz Project personnel prepared the resource model as of March 2017, and all relevant files were transferred to RPA for an independent review and audit. RPA was supplied with the drill hole database for Santa Luz in Excel format. The database included collar, downhole survey, assay, alteration, density, lithology, and geotechnical tables, and was complete as of March 31, 2017. RPA was also supplied with Santa Luz's lithology and grade envelopes developed in Leapfrog v. 4.2, and the block model for the geology and grades in CSV format, completed during June 2017.

The drill hole database was initially compiled into an MS Access drill hole database. The relevant contents of the database were downloaded to Excel files for transfer to RPA. The drill hole database used for the Santa Luz resource modelling included historical drilling by CVRD and subsequently Yamana. Historical drilling was predominantly core drilling, with limited RC holes. The historical RC drill holes were mostly shallow, and to a large extent were above the current pit topography (mined out). In addition, the database was updated with the new drilling conducted by Brio during 2016. The drilling included both core and RC drilling, as discussed previously in the drilling and sampling sections, and as summarized in Table 14-2. Diamond drill and RC holes are sampled at a nominal 1 m length.

Table 14-2: Summary of Mineral Resource Databases

Database Item	C1	Antas 3	Antas 2	Stockpiles
Collars (No.)	284	202	43	446
Total Length (m)	72,651	27,331	7,474	4,166
Survey (No.)	16,866	3,309	1,146	446
Stratigraphy (No.)	46,341	12,680	10,424	-
Assay Values (No.)	29,709	18,335	5,554	1,446
Assay Length (m)	29,709	18,335	6,061	4,166
Total Block Model Composites	**10,046**	**1,396**	**1,095**	**-**

It was noted that core recovery was determined for most of the drill holes, and observed to have been generally within acceptable limits. Measured values ranged from a high of 100% to a low of 84%. The average recovery is observed to be equal to 95% for the entire data set. The lowest recovery values are observed to have been from near-surface oxidized material, most of which has been removed by subsequent mining activity.

The 2016 Santa Luz drilling was reviewed and validated by Santa Luz Project personnel before it was used in grade modelling. RPA subsequently conducted its own independent review of the drill hole database.

Based on the above, and the verification steps undertaken by RPA (Section 12), SLR is of the opinion that the drill hole database is sufficiently reliable and appropriate for use in the estimation of Mineral Resources.

14.2 Geological Interpretation and 3-D Solids

As a first step in the construction of the Mineral Resource block model, Santa Luz Project personnel developed 3-D wireframes for the deposit using Leapfrog v. 4.2 software. Wireframes were prepared to delineate the principal geological and structural domains. These, in turn, were used in conjunction with the drill hole sample gold grades to delineate wireframes of the mineralized zones. The mineralized zones were then used for control during the subsequent grade modelling process.

Lithology solids and grade shells were modelled separately for all three deposits. The primary reason for modelling lithology is that different lithologies have contrasting TOC values, which impacts gold recovery. There are many lithology types logged; therefore, to improve the continuity of lithology solids, the on-site Santa Luz staff grouped related lithologies into seven and eight primary geological domains for the C1 and Antas 3 deposits, respectively (i.e., soil, volcanic tuff, diorite, dacite leachable, dacite high-sulphide, andesite, saprolite, and carbonaceous sediments/breccia) and assigned a block model rock code to each lithology as shown in Table 14-3. Figure 14-1 illustrates the lithology model at C1.

The mineralized zone wireframes or envelopes were defined at nominal drill hole sample cut-off grades of 0.1 g/t Au, with no minimum thickness applied. Interpretations were snapped to breccia zone intercepts in drill holes, which improved the continuity of the interpreted grade shell. This process caused the inclusion of some waste dilution within the grade shell. The 0.10 g/t Au cut-off grade was chosen based on a population break inside the mineralized lithology as illustrated in Figure 14-2 for C1. Figure 14-3 shows the 0.10 g/t Au grade shell for C1.

SLR inspected the wireframe models and notes that they appear to honour the logged lithology and represent reasonable interpretations of the geology and grade distribution that are consistent with field observations. The boundaries of the deposit appear to be constrained to a reasonable distance from the nearest drill holes.

Table 14-3: Geologic and Block Model Lithology Codes

Block Model Code	C1	Antas 3	Antas 2
1	Soil	Saprolite	Andesite
2	Volcanic Tuff Top	Volcanic Metatuffs	Carbonaceous Sediment/Breccia
3	Diorite	Carbonaceous Breccia	Diorite
4	Dacite Leachable	Metatuffs Breccia	Soil
5	Carbonaceous Sediment/Breccia	Andesitic Breccia	Volcanic Tuff
6	Volcanic Tuff	Dacite Leachable	
7	Volcanic Tuff Base	Dacite High Sulphide	
8		Diorite	
100			Waste

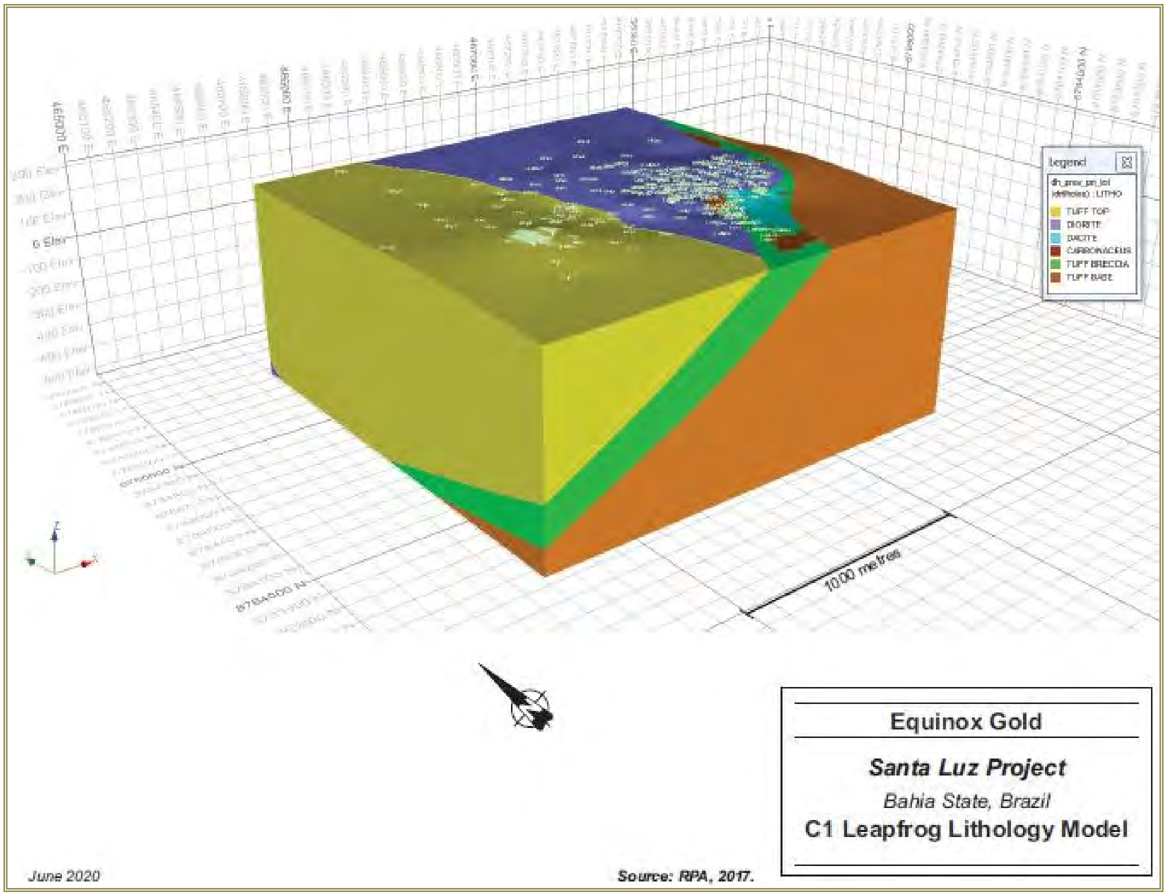
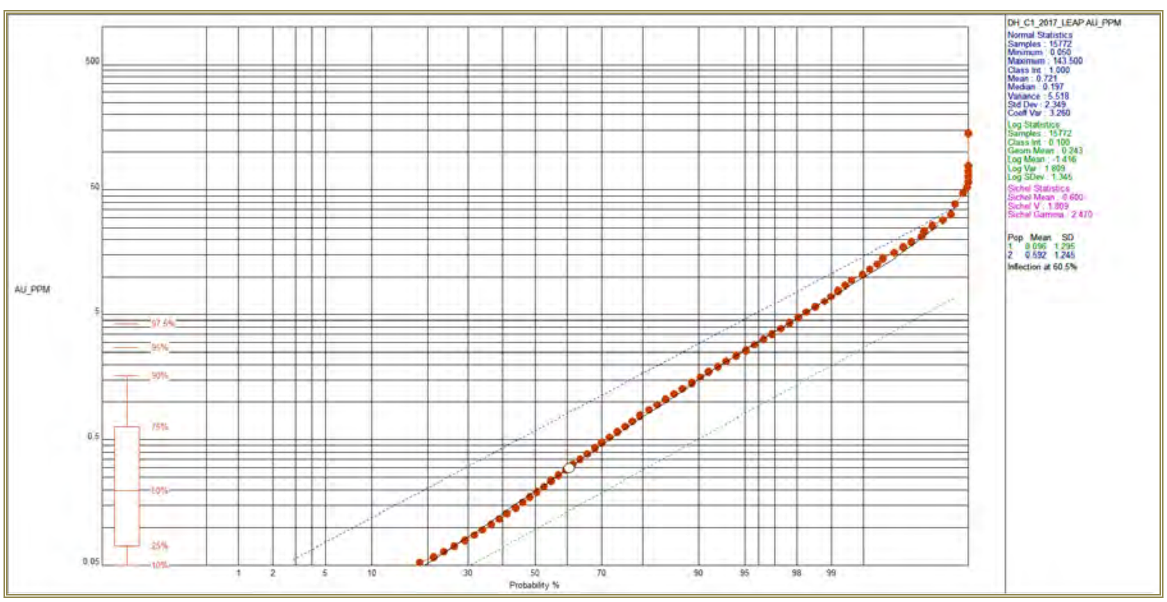


Figure 14-1: C1 Leapfrog Lithology Model



Source: SLR, 2018

Figure 14-2: C1 Grade Shell Modelling Cut-Off Curve (0.1 g/t Au)

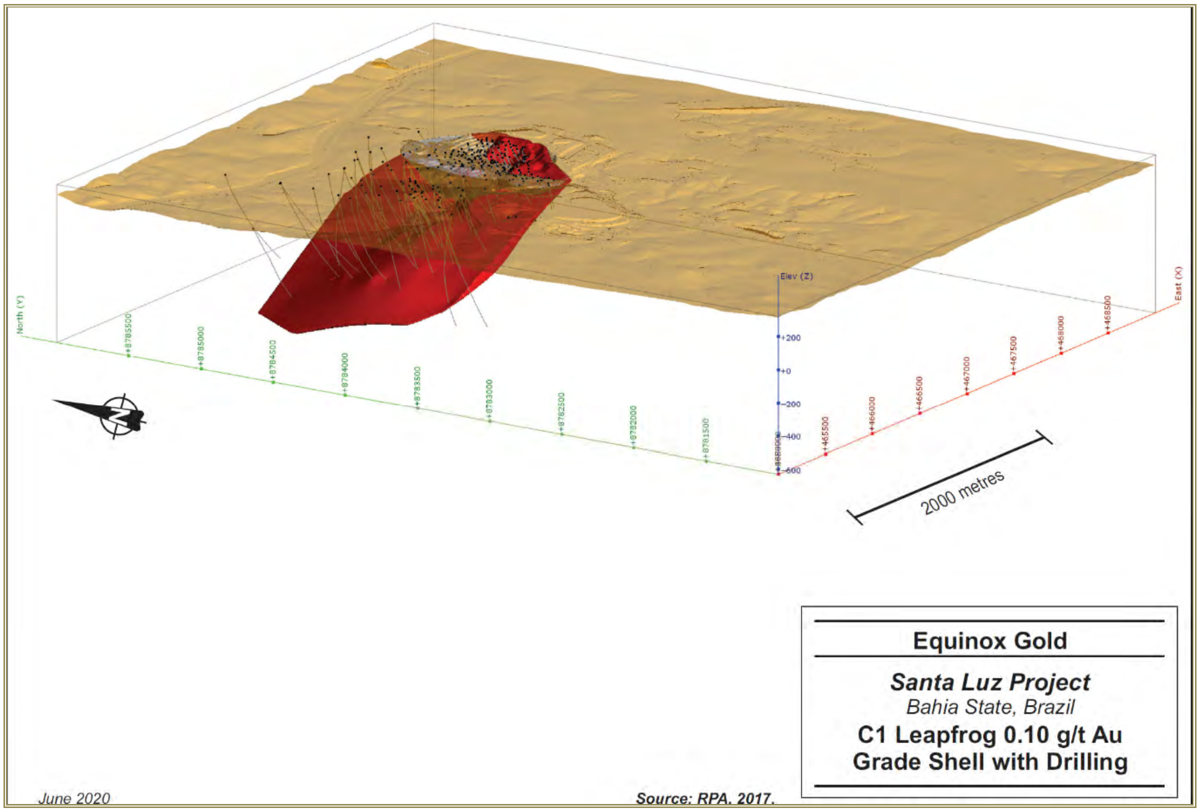
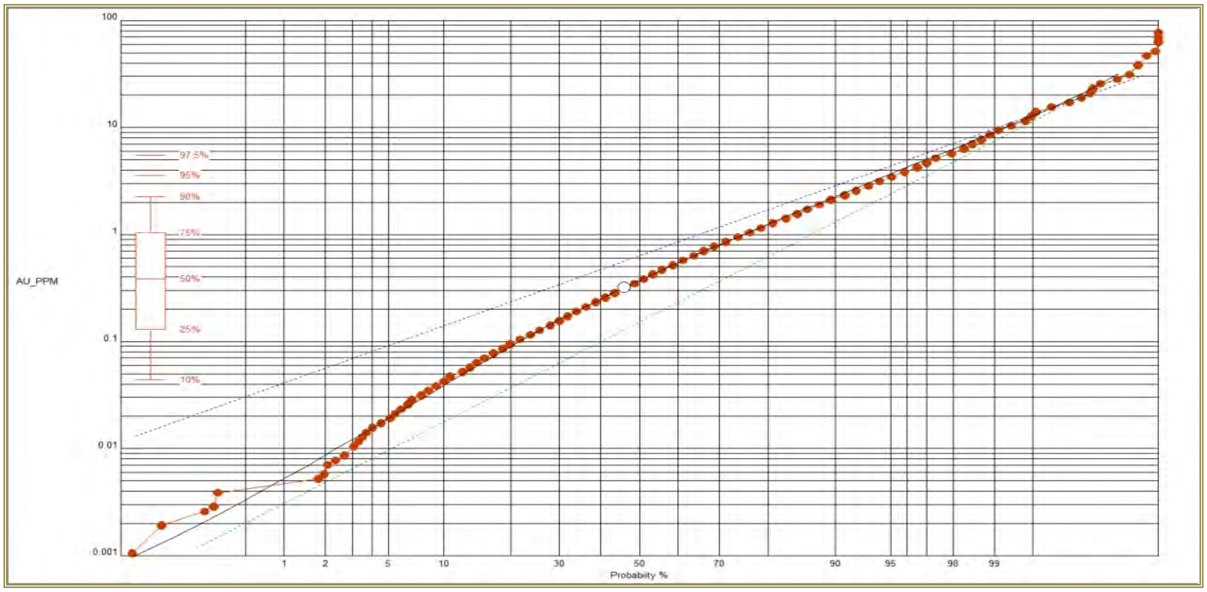


Figure 14-3: C1 Leapfrog 0.10 g/t Au Grade Shell with Drilling

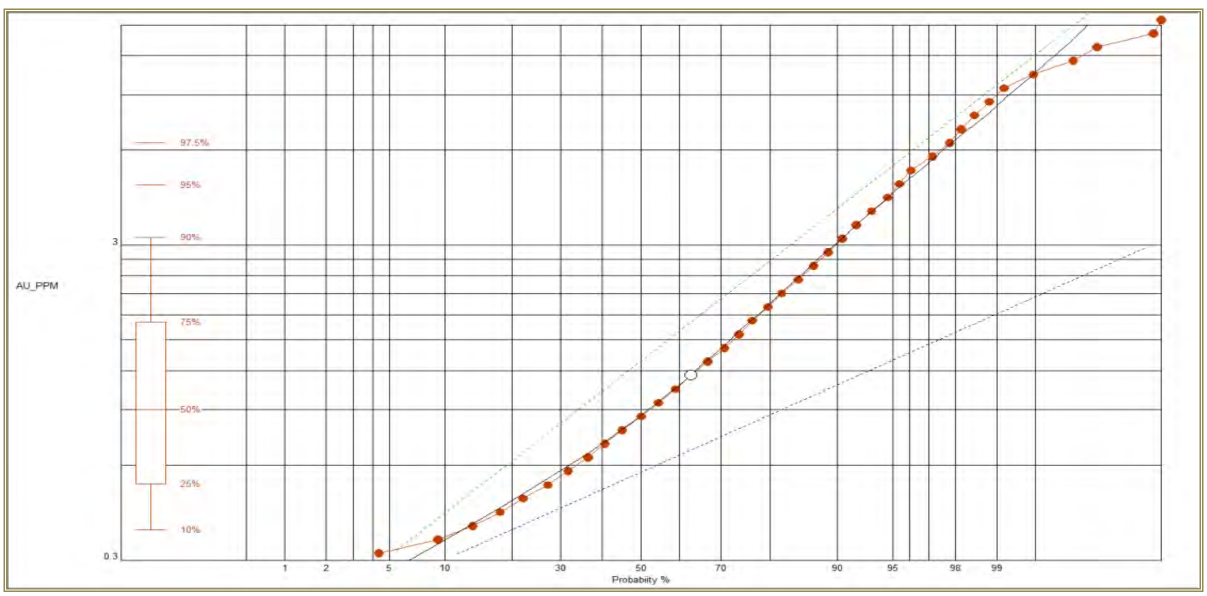
To facilitate the interpolation of gold grades within the 0.1 g/t Au grade shell at C1, multiple indicator kriging was used to define three separate internal grade zones or solids (Figure 14-4 and Figure 14-5):

- Low Grade (0.1 to 0.3 g/t Au)
- Medium Grade (≥0.3 to 0.6 g/t Au)
- High Grade (≥0.6 g/t Au).



Source: Equinox, 2018

Figure 14-4: C1 Probability Plot Cut-off 0.30 g/t Au



Source: Equinox, 2018

Figure 14-5: C1 Probability Plot Cut-Off 0.60 g/t Au

The strategy was to flag samples that are above or below a cut-off value with values of 1 and 0, respectively. The first estimation step separated the values using a high cut-off grade of 0.6 g/t Au. The blocks with 55% probability of being high grade were flagged with high grade indicator (IND2=2). After that, the lower grade material (below 0.6 g/t Au) was divided into two different zones separated by a cut-off grade of 0.3 g/t Au. Material below 0.3 g/t Au was considered low grade, and material between 0.3 g/t Au and 0.6 g/t Au was considered medium grade. The blocks with 55% probability of being medium grade were flagged IND2=1, while the remaining, lowest-grade blocks were flagged

IND2=0. The zones from the indicator kriging show good continuity. Figure 14-6 shows that the zones are reasonably representative of the grades of the contained drill hole samples. Figure 14-7 is a probability plot of gold grades for each zone, which shows a good contrast between the zones. The indicator zones were used only to estimate gold. All other elements were estimated in the 0.10 g/t Au grade shell.

Multiple indicator kriging was not used at Antas 2 and Antas 3 because it was possible to model discrete seams, thus avoiding excessive dilution. Mineralization structures at both Antas 2 (Figure 14-8) and Antas 3 (Figure 14-9 and Figure 14-10) feature a high degree of heterogeneity across strike and below grade. Antas 3 is defined as six discreet seams. Threshold samples were included within the mineralization wireframes where necessary to ensure continuity.

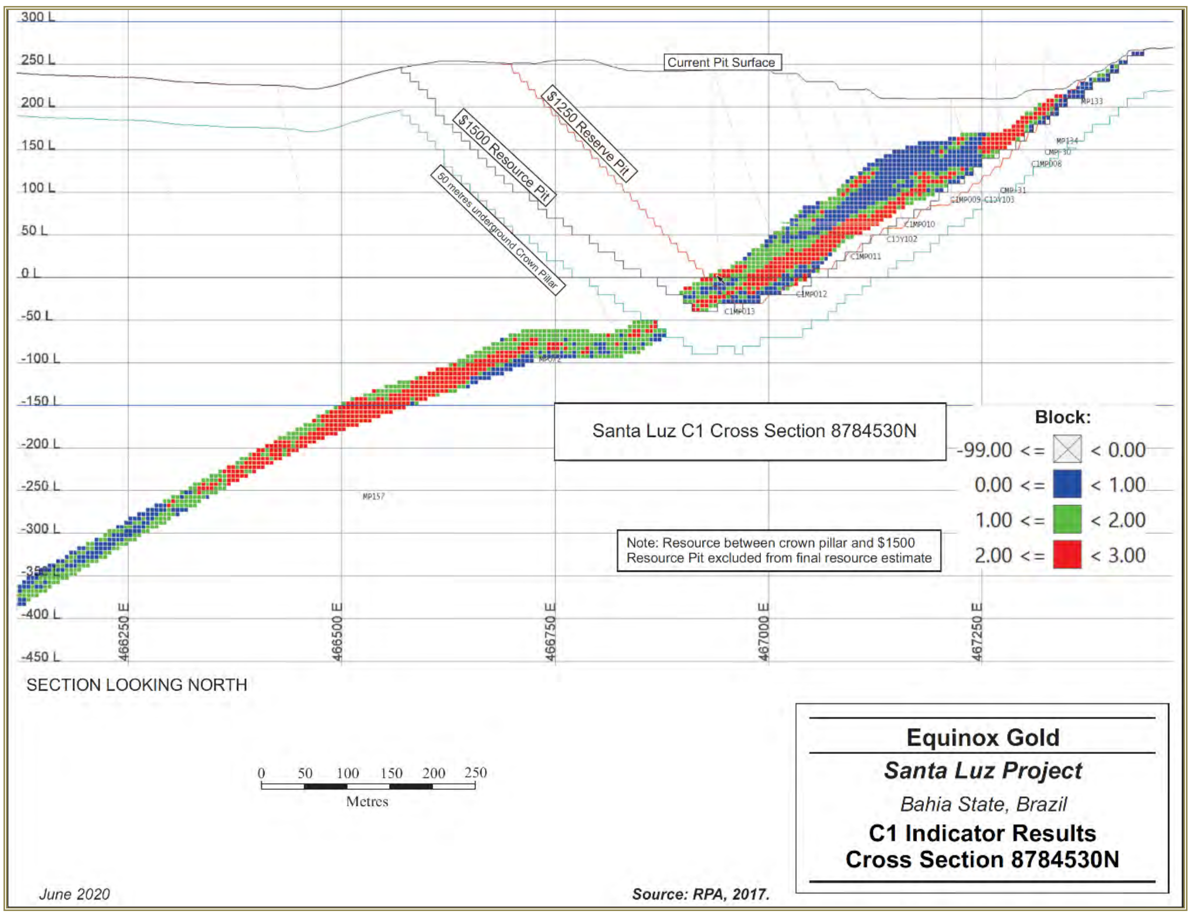
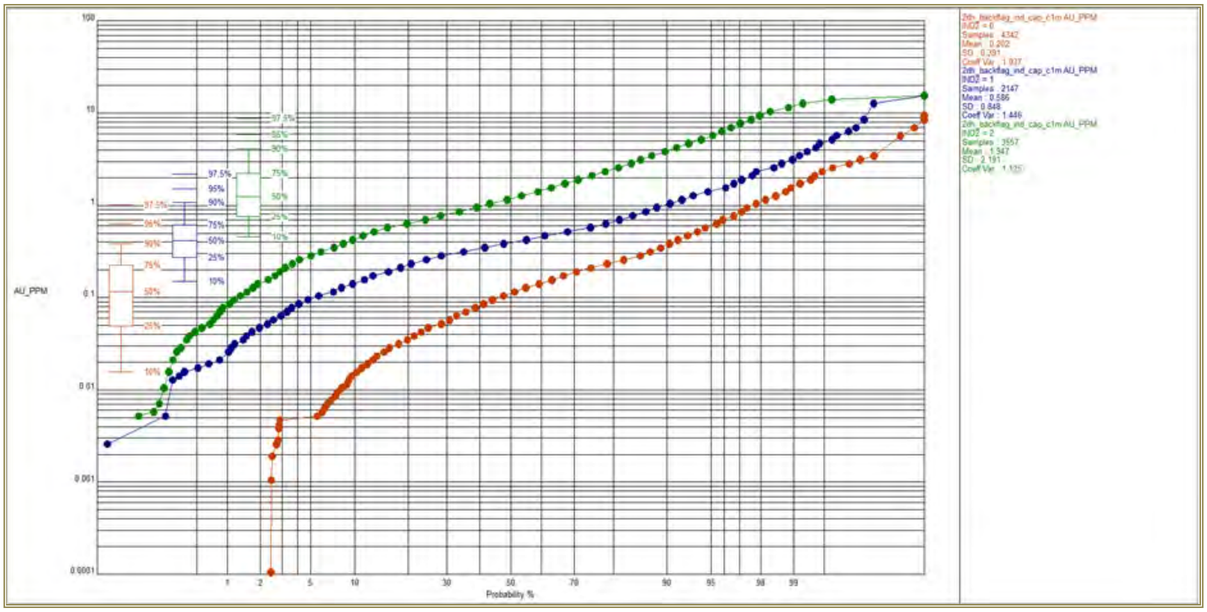


Figure 14-6: C1 Indicator Results Cross Section 8784530N



Source: Equinox, 2018

Figure 14-7: C1 Probability Plot of Gold for Three Indicator Zones

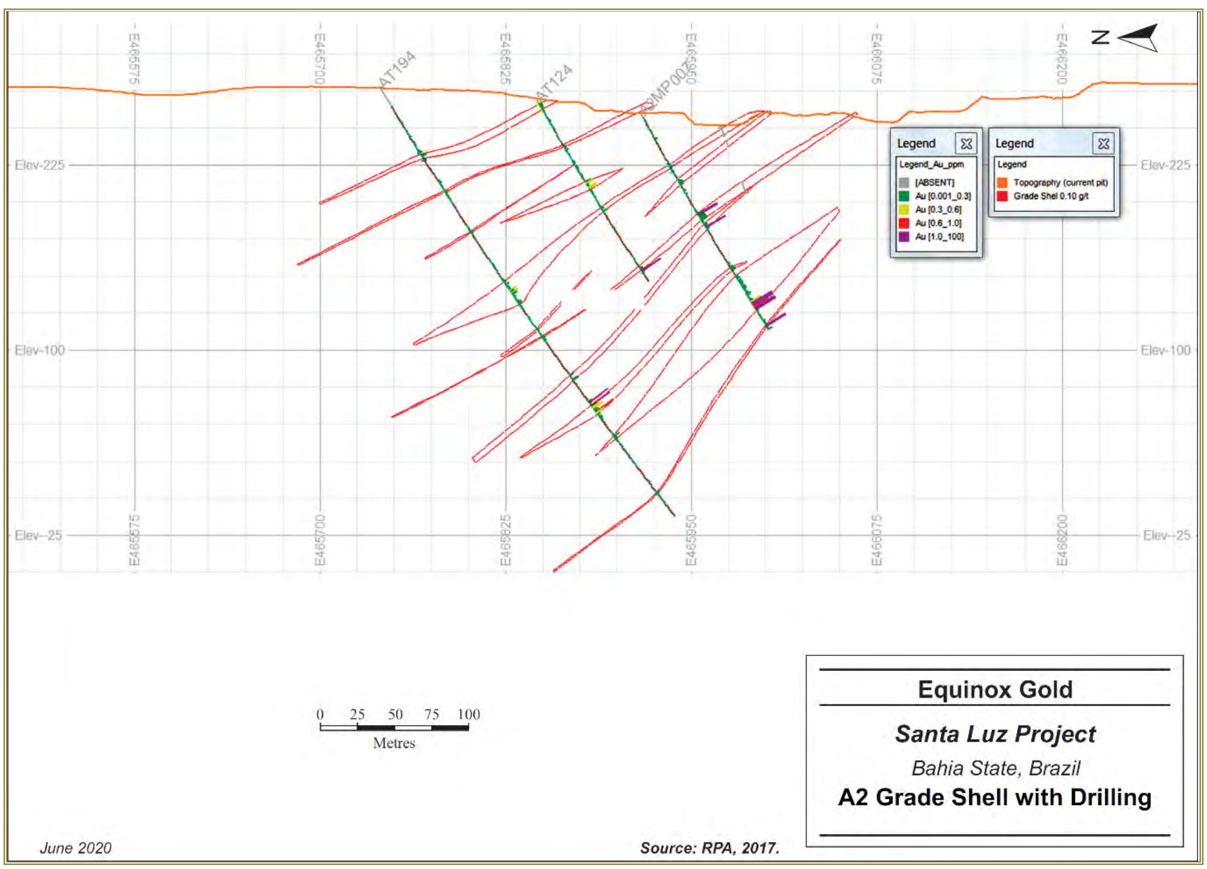


Figure 14-8: Antas 2 Grade Shell with Drilling

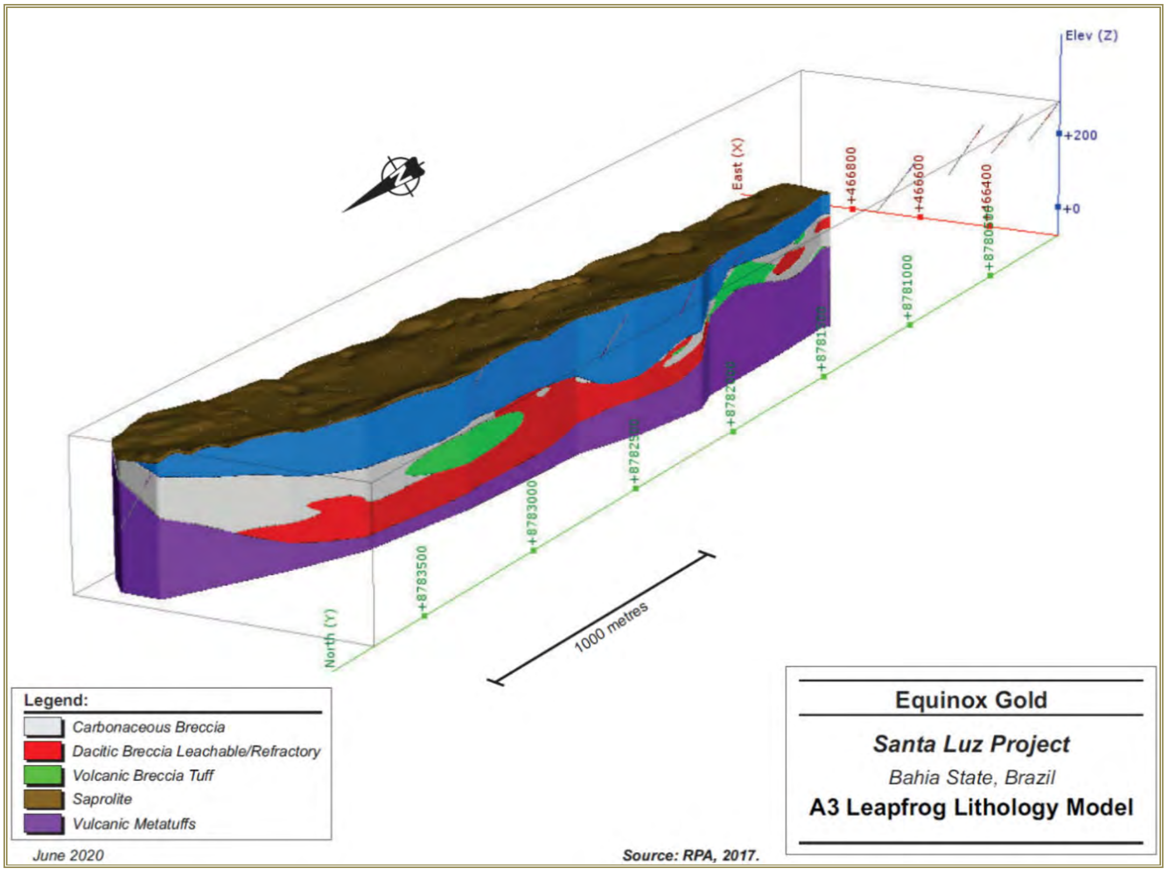


Figure 14-9: Antas 3 Leapfrog Lithology Model

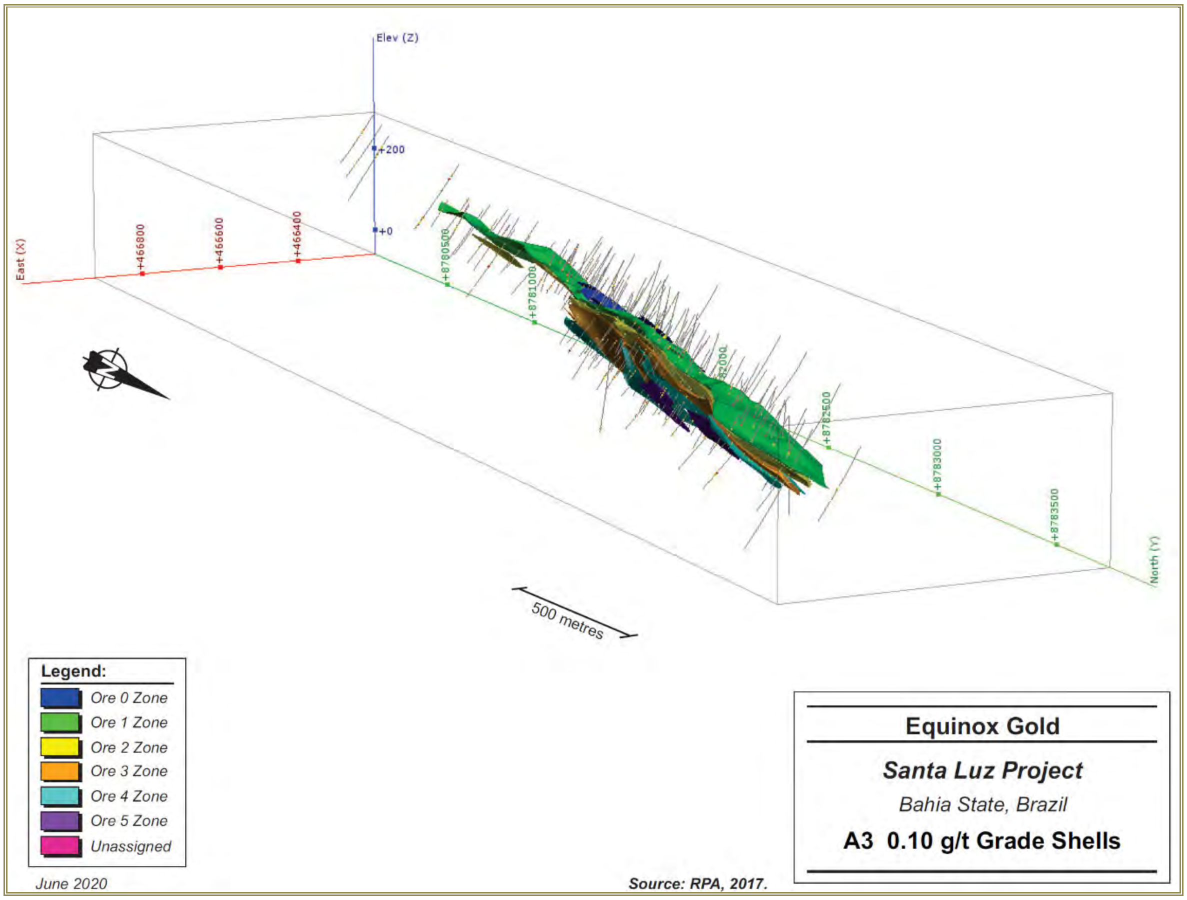


Figure 14-10: Antas 3 0.10 g/t Grade Shells

14.3 Statistical Analysis

Gold assays located inside the wireframe models were tagged with domain identifiers and exported for statistical analysis. A summary of the basic statistics of the gold assays for all drill holes is provided by mineralized domain in Table 14-4.

Table 14-4: Descriptive Statistics of Gold and TOC Assays

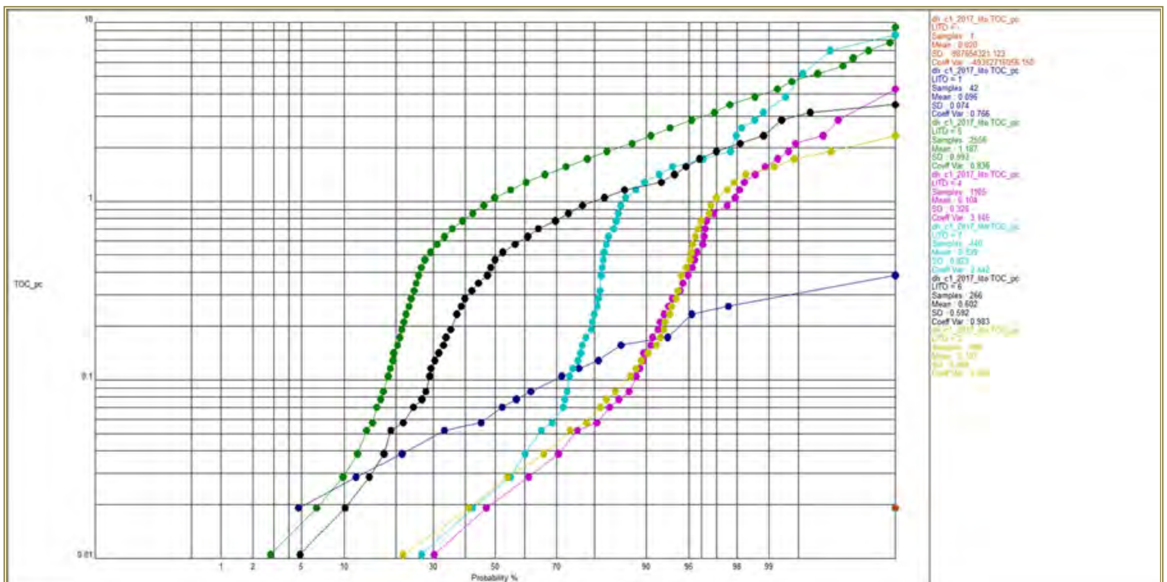
Zone	Grade	Count	Minimum	Maximum	Mean	Variance	SD	CV
Santa Luz Assay (Au)								
Antas 2	au_ppm	6,291	0.00	171.00	0.19	9.81	3.13	16.35
Antas 3	au_ppm	19,056	0.00	95.00	0.27	1.39	1.18	4.35
C1	au_ppm	37,162	0.00	143.50	0.32	2.35	1.53	4.75
Santa Luz Assay (TOC)								
Antas 2	toc_pc	6,291	0.00	13.40	0.20	0.55	0.74	3.76
Antas 3	toc_pc	19,056	0.00	7.38	0.08	0.11	0.33	4.27
C1	toc_pc	37,162	0.00	11.40	0.10	0.18	0.43	4.35

Notes: SD = standard deviation; CV = coefficient of variation

14.3.1 Capping High Grade Values

Where the assay distribution is skewed positively, or approaches log normal, erratic high-grade assay values can have a disproportionate effect on the average grade of a deposit. One method of treating these outliers to reduce their influence on the average grade is to cut or cap them at a specific grade level. In the absence of production data to calibrate the cutting level, inspection of the assay distribution can be used to estimate a *first pass* cutting level.

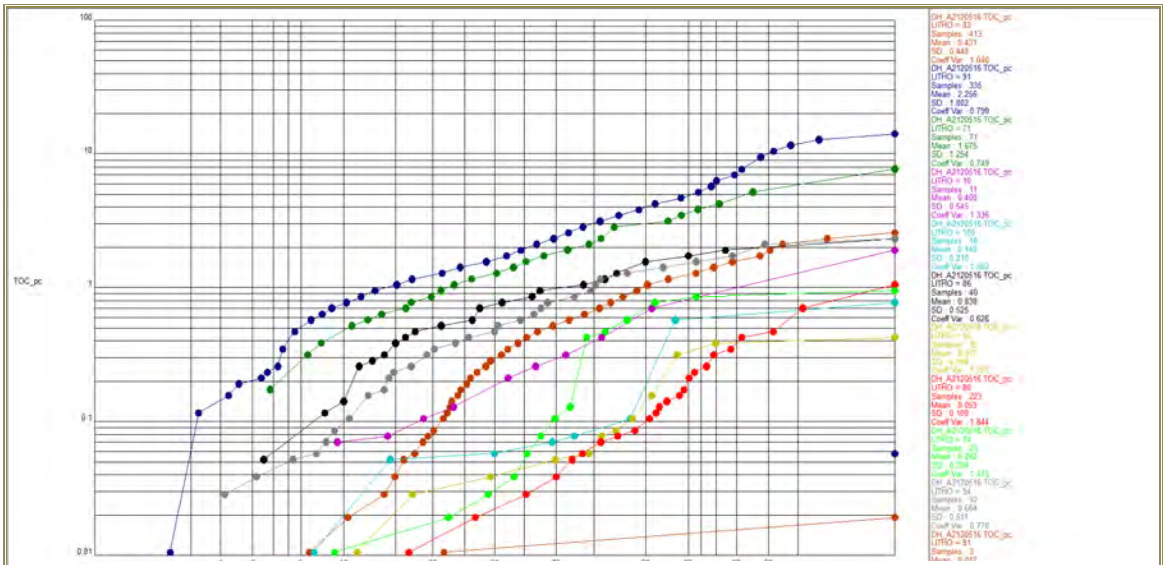
For each of the deposits cumulative frequency plots for TOC were calculated for the specific lithology (Figure 14-11 to Figure 14-13). Similarly, cumulative frequency plots for gold were made for the different lithology groups (Figure 14-14 to Figure 14-16). There is similar gold behaviour across different lithologies at C1 and Antas 3, indicating that gold content is not strongly controlled by lithology and that grade zones can cross lithologic boundaries. For Antas 2, the high-grade gold content is more related to the carbonaceous rock than is observed at C1 and Antas 3. Statistical analysis also shows that TOC values at Antas 2 are higher than at C1 and Antas 3.



	TOC_pc					
	SOIL	BC/SED CARBONACEOUS	DACITE	VOLCANIC TUFF BAS	VOLCANIC TUFF	DIORITE
CODE	1	5	4	7	6	3
Number	71	2684	1119	453	276	923
Minimum	0.02	0.01	0.01	0.01	0.01	0.01
Maximum	0.39	11.4	3.966	8.672	3.563	2.43
Mean	0.107	1.179	0.102	0.33	0.604	0.105
Median	0.085	1.087	0.03	0.03	0.51	0.03
Std Dev	0.076	0.956	0.292	0.803	0.58	0.279
Variance	0.006	0.913	0.085	0.645	0.337	0.078
Std Error	0.001	0	0	0.002	0.002	0
Coeff Var	0.71	0.811	2.862	2.436	0.961	2.653

Source: Equinox, 2018

Figure 14-11: C1 TOC Probability Plot by Lithology

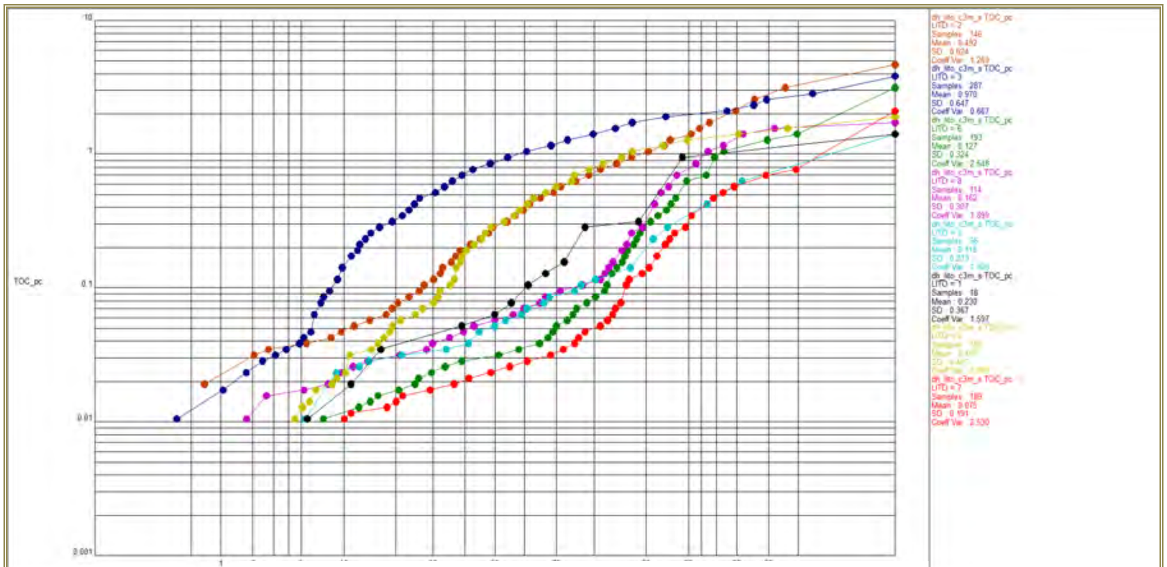


LITO	83	91	71	10	105	86	85	80	74	94	81
Number	413	335	71	11	16	40	33	223	23	92	3
Minimum	0.01	0.01	0.17	0.07	0.01	0.05	0.01	0.01	0.01	0.03	0.01
Maximum	2.62	13.4	7.64	1.95	0.78	2.23	0.42	1.07	0.95	2.43	0.02
Mean	0.431	2.256	1.675	0.408	0.14	0.838	0.077	0.059	0.202	0.664	0.017
Median	0.31	1.705	1.315	0.215	0.06	0.76	0.045	0.03	0.035	0.48	0.015
Std Dev	0.448	1.802	1.254	0.545	0.21	0.525	0.098	0.109	0.298	0.511	0.006
Variance	0.201	3.246	1.572	0.297	0.044	0.276	0.01	0.012	0.089	0.261	0
Std Error	0.001	0.005	0.018	0.05	0.013	0.013	0.003	0	0.013	0.006	0.002
Coeff Var	1.04	0.799	0.749	1.336	1.502	0.626	1.272	1.844	1.475	0.77	0.346

Source: Equinox, 2018

Note: Description of LITHO codes can be found in Table 10-3.

Figure 14-12: Antas 2 TOC Probability Plot by Lithology



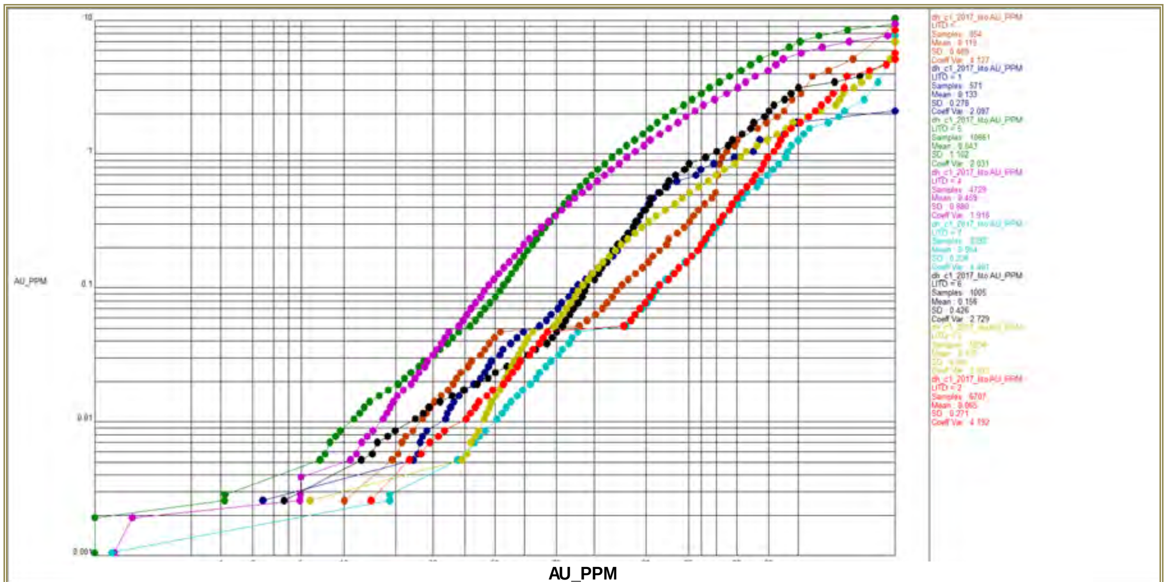
TOC

SEAMS	0	1	2	3	4	5
Number	8	175	85	68	35	5
Minimum	0.26	0.013	0.01	0.01	0.01	0.03
Maximum	2.187	2.247	2.519	2.029	4.03	2
Mean	0.932	0.369	0.466	0.164	0.563	0.535
Median	0.527	0.202	0.05	0.031	0.08	0.079
Std Dev	0.696	0.415	0.624	0.325	1.001	0.84
Variance	0.485	0.172	0.39	0.106	1.001	0.705
Std Error	0.087	0.002	0.007	0.005	0.029	0.168
Coeff Var	0.748	1.124	1.339	1.978	1.778	1.57

Source: Equinox, 2018

Notes: Description of LITO codes can be found in Table 14-3.
Summarized TOC statistics of composite data separated by grade shell seam.

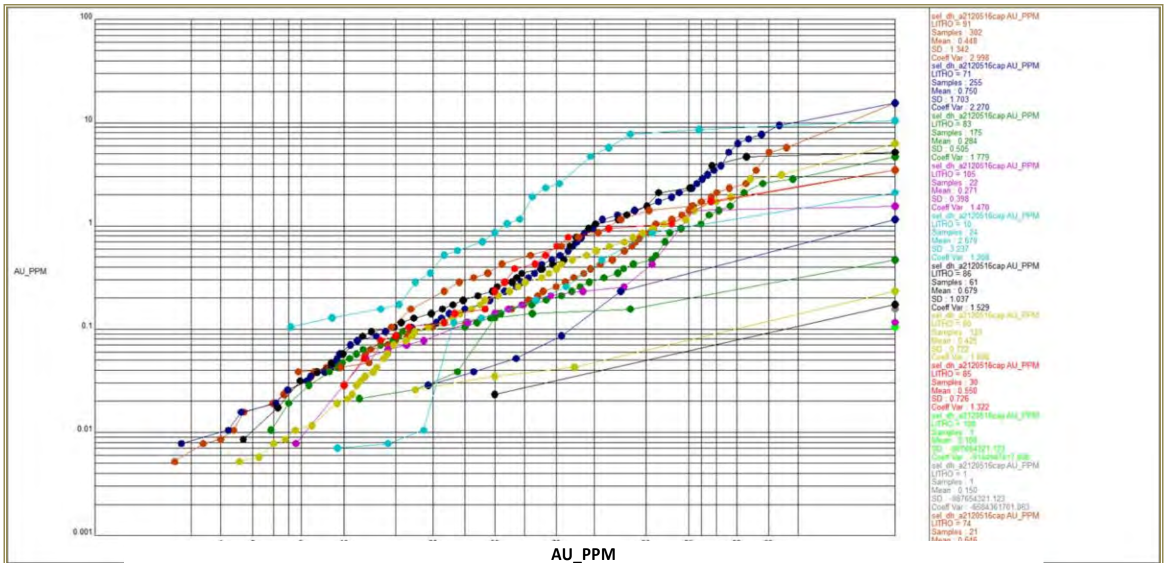
Figure 14-13: Antas 3 TOC Probability Plot by Lithology



	SOIL	BC/SED CARBONACEOUS	DACITE	VOLCANIC TUFF BAS	VOLCANIC TUFF	DIORITE	VOLCANIC TUFF TOP
CODE	1	5	4	7	6	3	2
Number	571	10731	4756	3392	1005	5234	6712
Minimum	0.003	0	0	0	0.003	0.003	0.003
Maximum	2.19	74.762	143.5	7.939	5.305	7.094	25.8
Mean	0.133	0.663	0.581	0.054	0.156	0.116	0.077
Median	0.03	0.091	0.127	0.011	0.024	0.016	0.019
Std Dev	0.278	2.142	2.634	0.236	0.426	0.305	0.549
Variance	0.077	4.589	6.937	0.056	0.182	0.093	0.301
Std Error	0	0	0.001	0	0	0	0
Coeff Var	2.097	3.233	4.534	4.401	2.729	2.633	7.109

Source: Equinox, 2018

Figure 14-14: C1 Au Probability Plot vs. Lithology

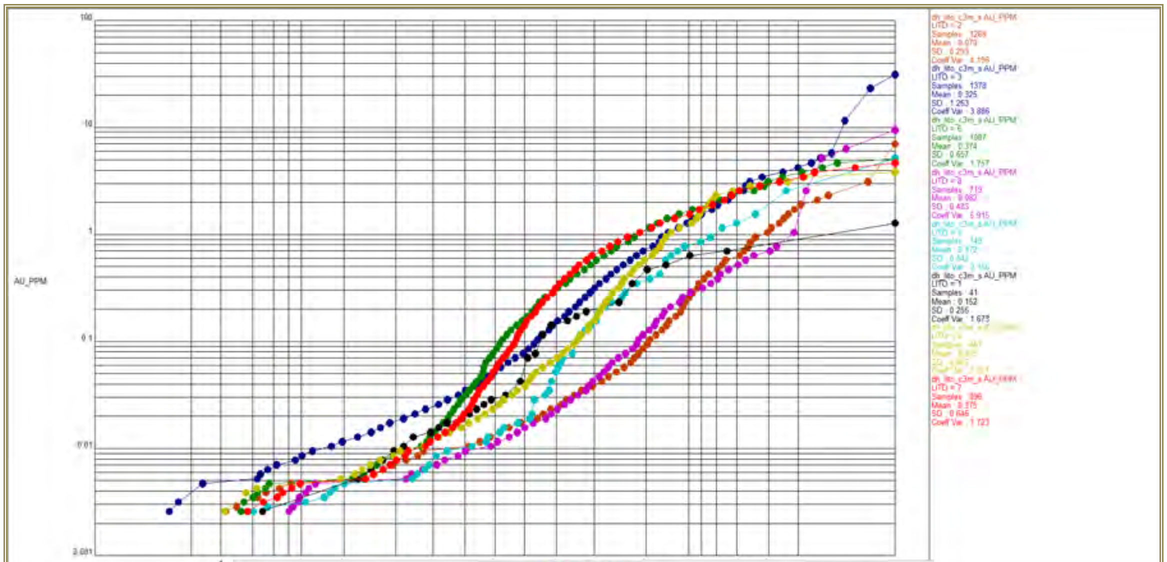


LITO	83	91	71	10	105	86	85	80	74	94	81
Number	1429	1537	633	39	160	137	311	886	36	105	29
Minimum	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.005	0.003	0.003
Maximum	4.48	17.8	171	9.96	1.525	5.09	3.64	6.41	3.37	0.492	0.089
Mean	0.059	0.12	0.605	1.662	0.052	0.314	0.076	0.082	0.392	0.033	0.014
Median	0.02	0.033	0.06	0.222	0.01	0.048	0.02	0.01	0.102	0.019	0.006
Std Dev	0.197	0.701	6.954	2.836	0.171	0.763	0.275	0.317	0.649	0.054	0.02
Variance	0.039	0.491	48.356	8.043	0.029	0.583	0.076	0.1	0.421	0.003	0
Std Error	0	0	0.011	0.073	0.001	0.006	0.001	0	0.018	0.001	0.001
Coeff Var	3.365	5.85	11.5	1.706	3.299	2.433	3.636	3.866	1.656	1.64	1.412

Source: Equinox, 2018

Note: Description of LITHO codes can be found in Table 10-3.

Figure 14-15: Antas 2 Au Probability Plot vs. Lithology



AU_PPM

SEAMS	0	1	2	3	4	5
Number	42	677	320	238	103	16
Minimum	0.014	0.036	0.035	0.07	0.045	0.14
Maximum	1.799	8.742	3.829	3.85	2.48	0.929
Mean	0.461	0.926	0.867	0.871	0.585	0.46
Median	0.331	0.62	0.6	0.592	0.386	0.378
Std Dev	0.415	0.921	0.763	0.741	0.539	0.255
Variance	0.173	0.848	0.582	0.549	0.291	0.065
Std Error	0.01	0.001	0.002	0.003	0.005	0.016
Coeff Var	0.901	0.955	0.881	0.851	0.922	0.555

Source: Equinox (2018)

Notes: Description of LITHO codes can be found in Table 10-3.
Summarized Au statistics of composite data separated by grade shell seam.

Figure 14-16: Antas 3 Au Probability Plot vs. Lithology

The influence of outliers was assessed for each domain, and involved:

- Reviewing the grade histograms and the point at which the high-grade tailings of the distribution begins to break up
- Reviewing the global influence after applying various top cuts
- Reviewing the spatial location of outliers.

Based on the evaluation of the outliers found in the upper end of the sample population distribution for each of the grade zones, the grades were capped at an appropriate level. The highest-grade samples in the distribution were considered outliers, which are deemed unrepresentative of the population distribution. The capping parameters are shown in Table 14-5.

Table 14-5: Range of Capping Values by Deposit

Deposit	Wireframe	Cap Grade (g/t Au)	No. of Samples Capped	Ag (ppm)	As (ppm)	Cu (ppm)	S (%)	TOC (%)
Antas 2	-	15	4	10	2,000	600	7	5
Antas 3	Ore 0	3	4	4	7,000	200	2	7
	Ore 1	10	6					
	Ore 2	5	12					
	Ore 3	6	4					
	Ore 4	3	4					
	Ore 5	2	1					
C1	-	15	45	10	6,000	500	4	5

14.3.2 Treatment of Samples Below Detection Limit

Samples that assayed below the detection limit were treated to better handle the influence of these samples in the subsequent compositing process. Negative or zero gold grades (below detection) were generally assigned a value of 0.0025 g/t Au, which is one half of the laboratory detection limit (i.e., 0.005 g/t Au). In SLR's view, this is a reasonable approach.

14.3.3 Compositing

The composite lengths used for grade interpolation were chosen considering the predominant sample length, the minimum mining width, the style of mineralization, the width of domains, and the continuity of grade. The raw assay data contains samples having irregular sample lengths. Sample lengths range from 10 cm to 3.0 m within the wireframe models; however, 91% of the samples were taken at 1 m intervals (Figure 14-17). Given this distribution, and considering the width of mineralization, the samples were composited to 1 m lengths broken at domain boundaries for C1 and Antas 2 deposits, and 3 m lengths for Antas 3 deposit. Assays within the wireframe domains were composited starting at the first mineralized wireframe boundary from the collar and resetting at each new wireframe boundary. Assays were capped prior to compositing. Composites <0.5 m, located at the bottom of the mineralized intercept, were removed from the database. These composites were then used in the subsequent grade modelling in the block model. SLR considers that the compositing approach is reasonable. Table 14-6 shows the capped composite statistics by domain.

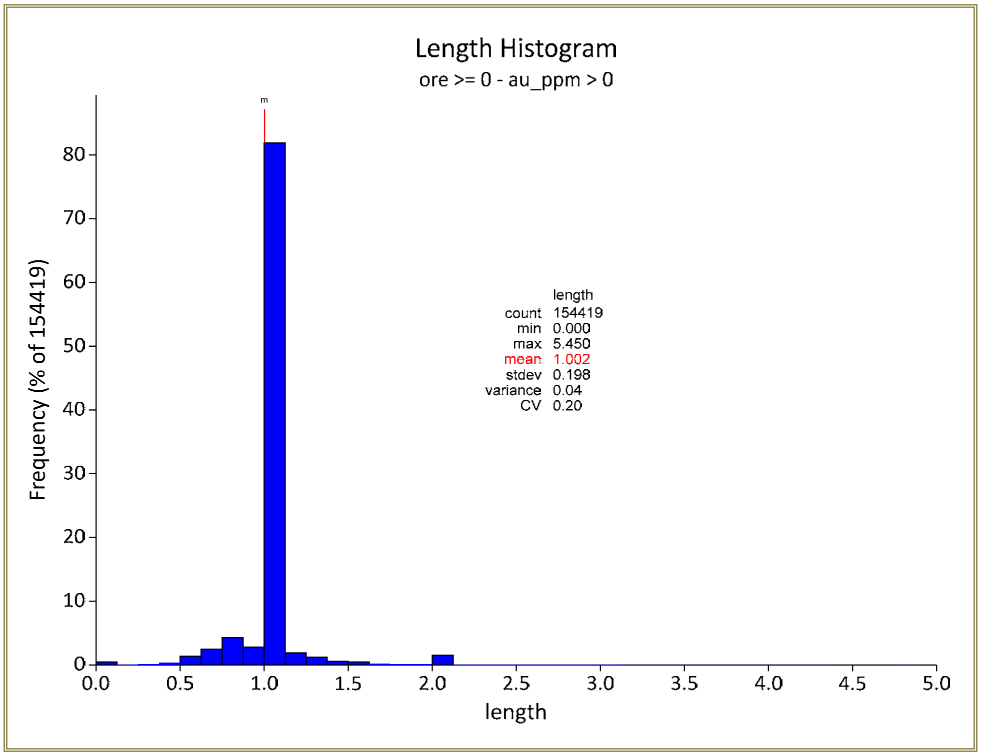


Figure 14-17: Histogram of Sample Lengths

Table 14-6: Basic Statistics of Capped Gold and TOC Composites

Zone	Grade	Count	Minimum	Maximum	Mean	Variance	SD	CV
Santa Luz Assay (Au)								
Antas 2	au_ppm	1,095	0.00	15.00	0.53	1.45	1.20	2.29
Antas 3	au_ppm	1,396	0.014	8.74	0.86	0.68	0.82	0.96
C1	au_ppm	10,046	0.00	15.00	0.90	2.54	1.59	1.77
Santa Luz Assay (TOC)								
Antas 2	toc_pc	1,095	0.00	9.89	0.37	0.99	0.99	2.66
Antas 3	toc_pc	1,396	0.00	4.03	0.10	0.11	0.34	3.24
C1	toc_pc	10,046	0.00	9.71	0.19	0.34	0.58	3.11

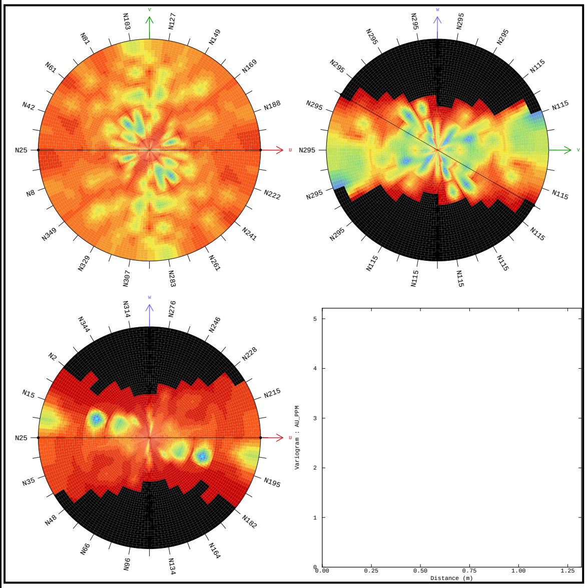
Notes: SD = standard deviation; CV = coefficient of variation.

14.3.4 Continuity Analysis

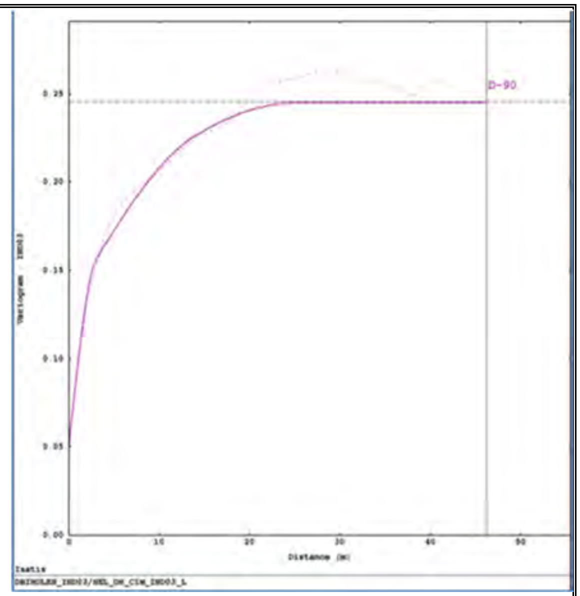
Geostatistical analyses were carried out using Isatis (Version 2013.3) mining software to evaluate the grade variability changes with distance, as well as to quantify nugget effects. Variography is used to describe the spatial continuity of a variable—gold—and the best structure was defined using a correlogram. In this document, the term variogram will be used to denote a generic spatial measure and will be used interchangeably with correlogram. Downhole correlograms were developed for the 1 m and 3 m composites flagged in C1 and Antas 3 and are shown in Figure 14-18 through Figure 14-21 for each grade zone. For Antas 2, specific variography has not been undertaken because the current drill hole spacing and number of samples are not adequate to generate meaningful variograms.

The correlograms were evaluated to determine the optimum range and directions of grade continuity, which were found to be consistent with the structural orientation of the mineralized zones. RPA visually inspected the correlograms and notes that they appear to have been generated and interpreted correctly, and are reasonable. The observed distribution of grades within the model is reasonable, which suggests that the models were configured appropriately. In SLR's opinion, the search ranges are reasonable for the style of mineralization and are generally consistent with the correlogram results.

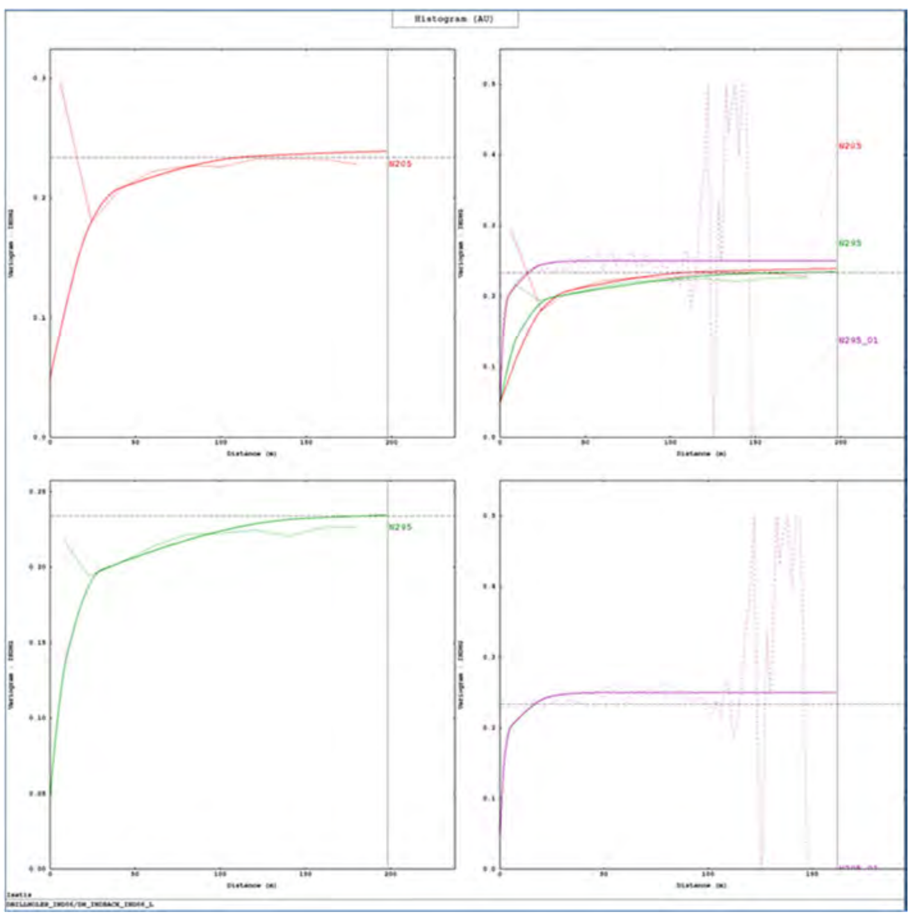
For C1, the major and semi-major search directions are typically within a plane oriented with an azimuth of 205° and dipping 30°, which is the direction of maximum continuity of the mineralization. For Antas 2 and Antas 3, the major and semi-major search directions are typically within a plane oriented with the major axis oriented at 190° to 200° azimuth parallel to the dominant northern trend of the zones. The semi-major axis was oriented down dip, normal to the major axis, and the minor axis was oriented across strike for an overall search ellipse oriented toward the northeast with a plunge of 0° and dip of 40°, which is the direction of maximum continuity of the mineralization (Table 14-7 and Table 14-8).



Varmap of Au—Anisotropy Test C1



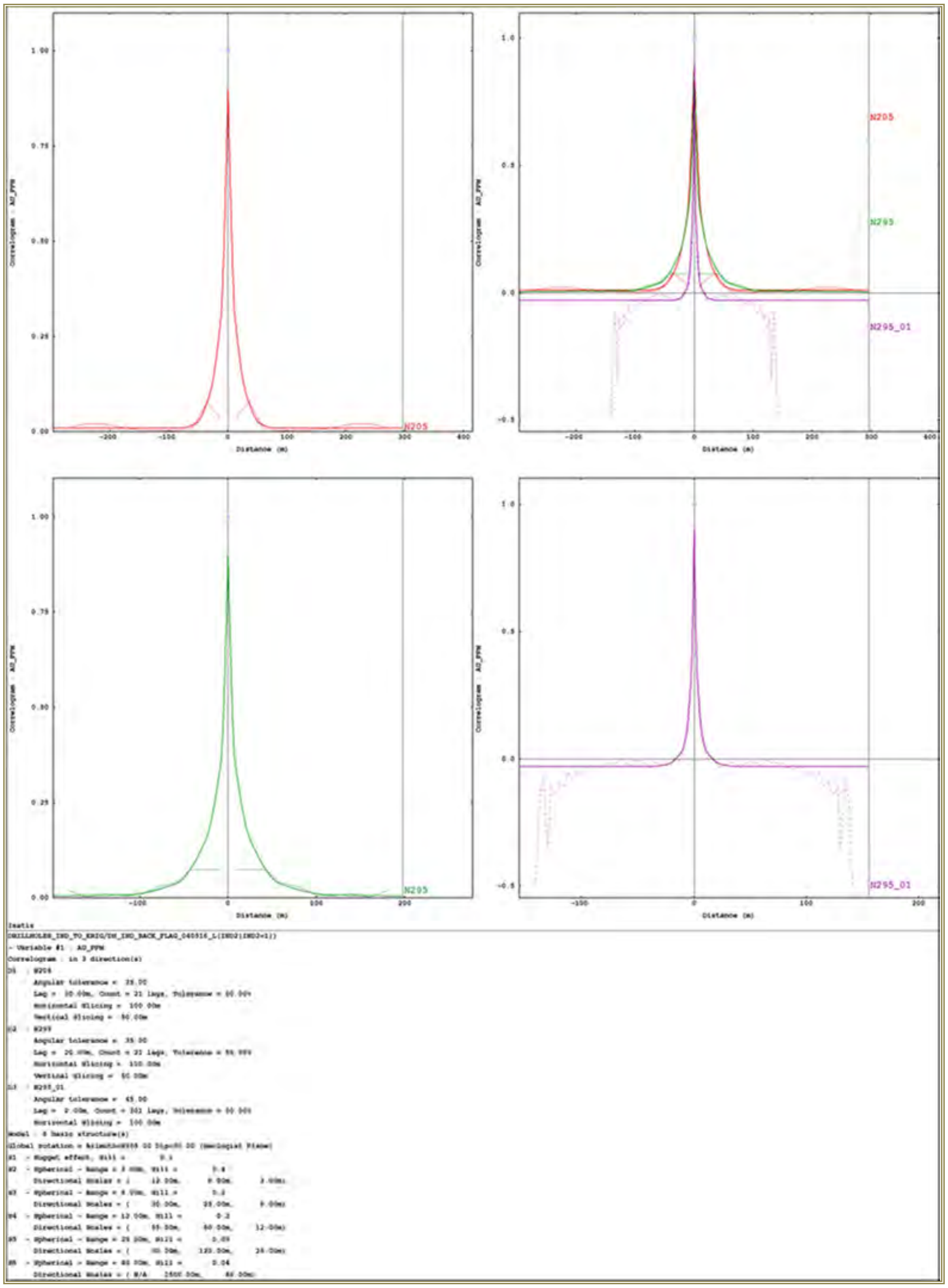
Varmap of Au—Nugget Test C1



Variogram of Indicator Variable C1

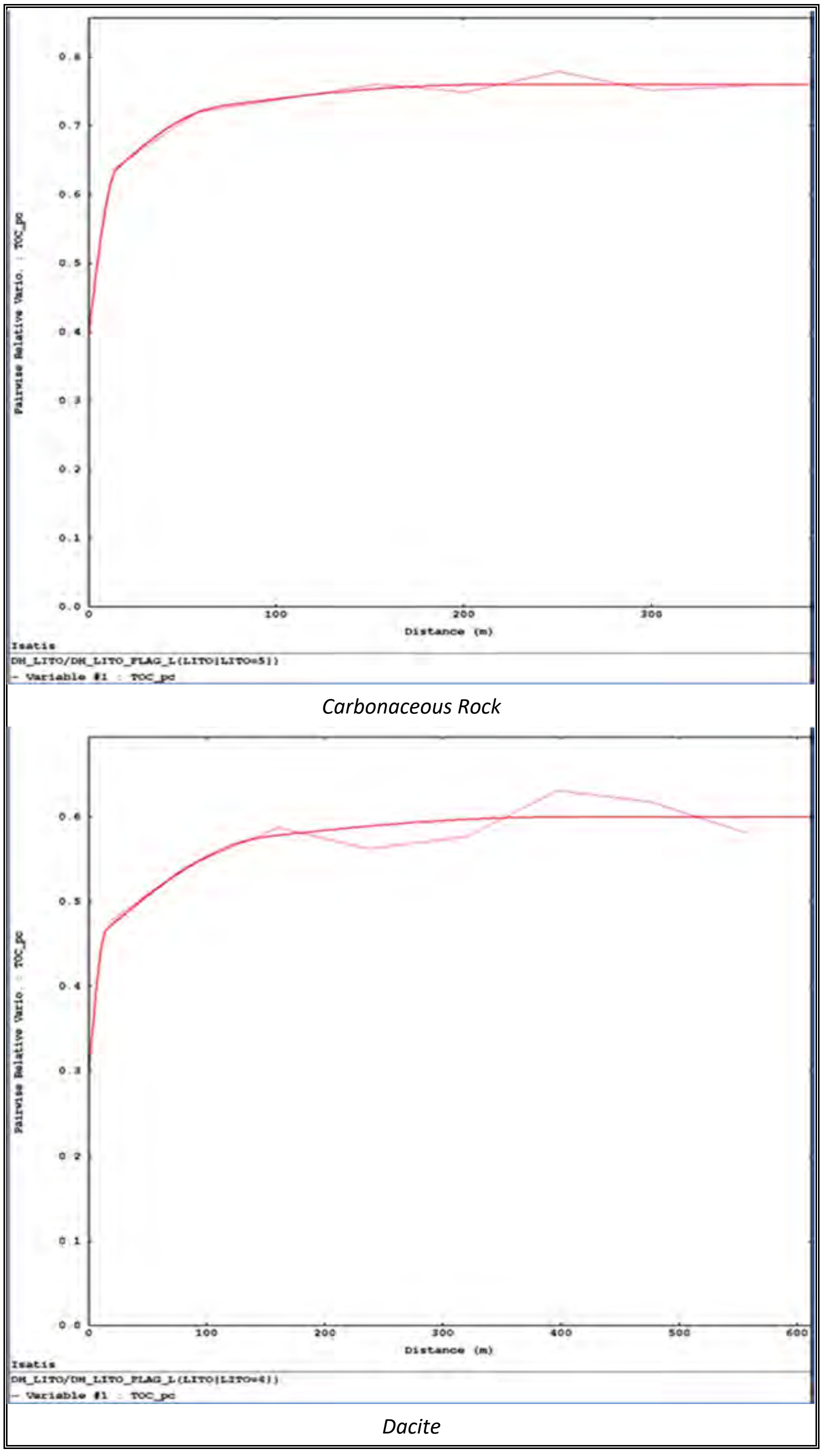
Source: RPA, 2018

Figure 14-18: Variogram Model at C1



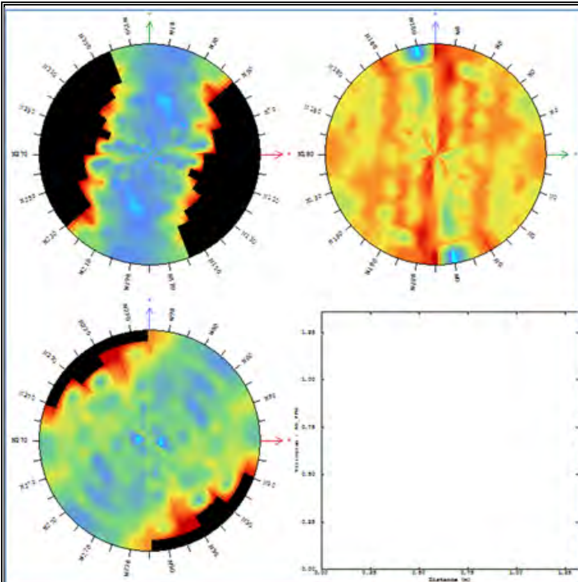
Source: RPA, 2018

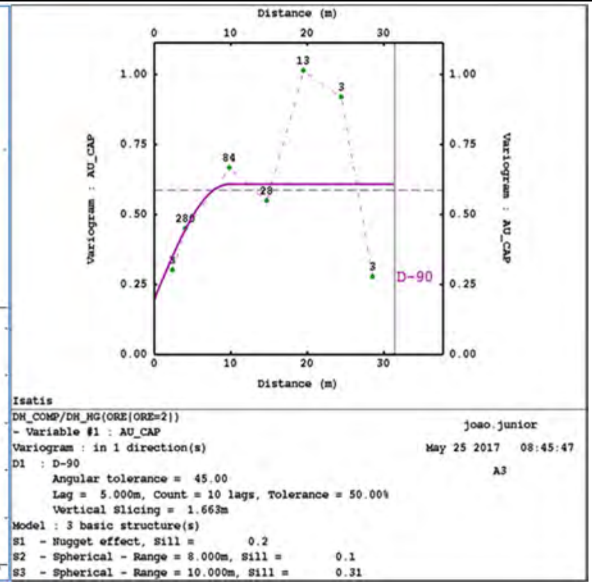
Figure 14-19: Correlogram of Au at C1



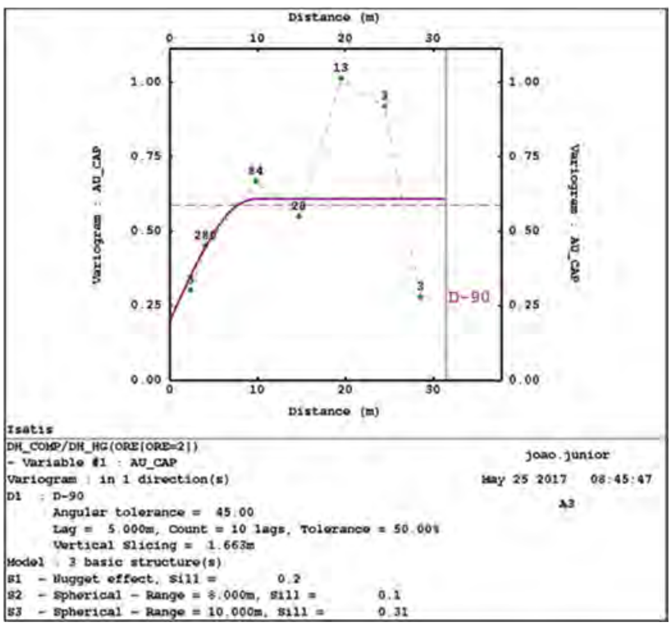
Source: RPA, 2018

Figure 14-20: Variogram of TOC—Carbonaceous Rock and Dacite At C1





Varmap of Au—anisotropy test Antas 3 *Varmap of Au—nugget test Antas 3*



Variogram of Au Antas 3 (Ore 2)

Source: RPA, 2018

Figure 14-21: Variogram Model at Antas 3

Table 14-7: C1 and Antas 3 Variogram and Correlogram Parameters

					Structure 1				Structure 2				Structure 3				Structure 4			
Seam	**Variable**	**Azimuth**	**Dip**	**Nugget**	**Sill**	**Maj (m)**	**Semi (m)**	**Min (m)**	**Sill**	**Maj (m)**	**Semi (m)**	**Min (m)**	**Sill**	**Maj (m)**	**Semi (m)**	**Min (m)**	**Sill**	**Maj (m)**	**Semi (m)**	**Min (m)**
C1 Deposit																				
	Au	205	30	0.1	0.4	12	8	3	0.2	40	25	8	0.2	70	60	15	0.09	20	200	25
	TOC	205	30	0.1	0.165	25	25	25	0.097	165	165	165								
	Ag	205	30	0.4	0.4	10	10	10	0.2	20	20	20								
	As	205	30	0.4	0.25	10	10	10	0.3	70	70	70								
	Cu	205	30	0.3	0.35	14	14	14	0.35	20	20	20								
	S	205	30	0.3	0.15	10	10	10	0.55	25	25	25								
	Indicator (0.6 g/t)	205	30	0.05	0.04	12	8	3	0.085	40	25	8	0.045	100	100	30	0.015	20	200	40
	Au-Indicator	205	30	0.1	0.4	12	8	3	0.2	30	30	8	0.2	60	50	12	0.09	75	140	20
Antes 3 Deposit																				
ORE 1	Au	0	0	0.3	0.32	20	20	20	0.34	40	40	40								
ORE 2	Au	0	0	0.2	0.27	50	50	50	0.1	100	100	100								
ORE 3	Au	0	0	0.2	0.5	30	30	30	0.29	60	60	60	0.05	80	80	80				
Total	**TOC**	**0**	**0**	**0.05**	**0.19**	**10**	**10**	**10**	**0.08**	**65**	**65**	**65**	**0.095**	**90**	**90**	**90**				

Table 14-8: Block Estimate Search Strategy by Deposit

Variable	Estimation Type	Azimuth	Dip	Major Axis	Semi Major	Minor Axis	Min. No. of Samples
C1 Deposit Search Parameters							
Indicator	OK	205	30	120	120	40	4
Au	OK	205	30	150	150	40	4
TOC	OK	205	30	220	220	100	4
Ag	OK	205	30	70	70	40	4
As	OK	205	30	70	70	40	4
Cu	OK	205	30	70	70	40	4
S	OK	205	30	70	70	40	4
Variable	**No. of Horizontal Angular Sectors (Split Vertically)**	**Optimum No. of Samples per Sector**	**Min. Distance Between Two Selected Samples**	**Max. No. of Consecutive Empty Sector**	**Optimum No. of Samples per Hole**	**Max. No. of Samples per Hole**	**Nugget**
Indicator	4	2	1	4	2	2	
Au	4	2	1	4	2	2	
TOC	4	2	1	4	2	2	
Ag	4	2	1	4	2	2	
As	4	2	1	4	2	2	
Cu	4	2	1	4	2	2	
S	4	2	1	4	2	2	
Variable	**Estimation Type**	**Azimuth**	**Dip**	**Major Axis**	**Semi Major**	**Minor Axis**	**Min. No. of Samples**
Antes 2 Deposit Search Parameters							
Au	IPD	200	30	150	150	40	4
TOC	IPD	200	30	150	150	40	4
Ag	IPD	200	30	150	150	40	4
As	IPD	200	30	150	150	40	4
Cu	IPD	200	30	150	150	40	4
S	IPD	200	30	150	150	40	4
Variable	**No. of Horizontal Angular Sectors (Split Vertically)**	**Optimum No. of Samples per Sector**	**Min. Distance Between Two Selected Samples**	**Max. No. of Consecutive Empty Sector**	**Optimum No. of Samples per Hole**	**Max. No. of Samples per Hole**	**Nugget**
Au	4	2	1	4	2	2	
TOC	4	2	1	4	2	2	
Ag	4	2	1	4	2	2	
As	4	2	1	4	2	2	
Cu	4	2	1	4	2	2	
S	4	2	1	4	2	2	
Variable	**Estimation Type**	**Azimuth**	**Dip**	**Major Axis**	**Semi Major**	**Minor Axis**	**Min. No. of Samples**
Antes 3 Deposit Search Parameters							
ORE 0	IPD	190	30	150	150	50	3
ORE 1	OK	190	30	150	150	50	3
ORE 2	OK	190	30	150	150	50	3
ORE 3	OK	190	30	150	150	50	3
ORE 4	IPD	190	30	150	150	50	3
ORE 5	IPD	190	30	150	150	50	3
TOC	OK	190	30	150	150	50	4
Ag	IPD	190	30	150	150	50	4
As	IPD	190	30	150	150	50	4
Cu	IPD	190	30	150	150	50	4
S	IPD	190	30	150	150	50	4
Variable	**No. of Horizontal Angular Sectors (Split Vertically)**	**Optimum No. of Samples per Sector**	**Min. Distance Between Two Selected Samples**	**Max. No. of Consecutive Empty Sector**	**Optimum No. of Samples per Hole**	**Max. No. of Samples per Hole**	**Nugget**
ORE 0	4	2	1	4	2	2	
ORE 1	4	2	1	4	2	2	
ORE 2	4	2	1	4	2	2	
ORE 3	4	2	1	4	2	2	
ORE 4	4	2	1	4	2	2	
ORE 5	4	2	1	4	2	2	
TOC	4	2	1	4	2	2	
Ag	4	2	1	4	2	2	
As	4	2	1	4	2	2	
Cu	4	2	1	4	2	2	
S	4	2	1	4	2	2	

14.4 Bulk Density

Bulk density determinations are made by the Santa Luz project personnel on an ongoing basis on core specimens using a water immersion method. Prior to analysis, the samples are dried and weighed in air. Those samples for which porosity and permeability are judged significantly high are coated with paraffin before analysis.

Bulk density was determined for each of the major host lithologies for both oxide and sulphide zones. Average density by zone used in the estimation ranges from 1.80 g/cm^3 to 2.75 g/cm^3 with an overall average of 2.60 g/cm^3, which in SLR's opinion is reasonable for the rock types observed at site and typical for the host rock types of the mineralization (Table 14-9).

Table 14-9: Bulk Density Values by Deposit

Deposit	Block Rock Code	Lithology	Density (g/cm^3)
C1	1	Soil	1.80
	2	Volcanic Tuff Top	2.69
	3	Diorite	2.75
	4	Dacite	2.72
	5	Carbonaceous Sediment/Breccia	2.69
	6	Volcanic Tuff	2.69
	7	Volcanic Tuff Base	2.69
Antas 2	1	Soil	1.80
	2	Carbonaceous Sediment/Breccia	2.70
	3	Diorite	2.70
	4	Andesite	2.70
	5	Volcanic Tuff	2.70
	100	Waste	2.70
Antas 3	1	Soil	1.80
	2	Volcanic Tuff Top	2.67
	3	Dacite	2.69
	4	Carbonaceous Sediment/Breccia	2.70
	5	Volcanic Tuff	2.70
	6	Diorite Base	2.75

14.5 Block Model

For each deposit Santa Luz Project personnel constructed a block model representing the geology and grades of each deposit in Isatis (Version 2013.3) mining software. The block size was selected to be consistent with the selective mining unit (SMU) size for the planned open pit mining and used a whole-block approach. Estimated grade values used a regular parent cell of 5 m x 5 m x 5 m for the C1 and Antas 2 deposits, and a regular parent cell size of 3 m x 3 m x 3 m for the Antas 3 deposit. A summary of the block model extents is provided in Table 14-10.

Table 14-10: Block Model Parameters

Deposit	Origin			Block Model Length (m)			Parent Block Dimension (m)		
	X	Y	Z	X	Y	Z	X	Y	Z
C1	465,800	8,783,600	-650	1,750	1,770	1,000	5	5	5
Antas 2	465,750	8,782,700	-100	700	1,275	905	5	5	5
Antas 3	466,000	8,779,800	-100	1,200	4,401	402	3	3	3

14.6 Interpolation Parameters

Block model representations were constructed for the lithology and grade zones at each of the deposits, along with grade estimations for gold. Estimation of the grades was controlled by the grade zones. Search ellipsoid geometry of the major, semi-major, and minor axis was oriented into the structural plane of the mineralization, as indicated by the correlogram ranges for each domain. The interpolation strategy involved setting up search parameters in three nested estimation runs for each domain. Each subsequent pass employed larger search ellipse dimensions and the same number of composite restrictions.

For C1, the search was assisted by the use of a dynamic function, which allowed the search ellipsoid to stay subparallel to the orientation of the mineralized zone trend (Figure 14-22). First-, second-, and third-pass search ellipses maintained a 1:1:3 anisotropic ratio. A sector approach was used, with the search ellipsoid divided into four sectors. A nearest neighbour (NN) block model was also prepared for comparison purposes. Search parameters are listed in Table 14-8 for the Santa Luz C1, Antas 2, and Antas 3 deposits.

To reduce the influence of very high-grade composites, grades greater than a designated threshold level for the C1 high grade indicator zone (IND2=2 above 0.6 Au g/t) were restricted to shorter search ellipse dimensions. The threshold grade level of 12 g/t Au was chosen from the basic statistics, variography, and from visual inspection of the apparent continuity of very high grades within each domain, which indicated the need to limit their influence to approximately half the distance of the drill hole spacing

Grade interpolation was carried out using ordinary kriging (OK) on mineralized domains in C1, and the Ore-1, Ore-2, and Ore-3 seams at Antas 3. All other domains that were lacking variogram models due to insufficient samples, including Antas 2, were estimated using inverse distance squared (ID^2).

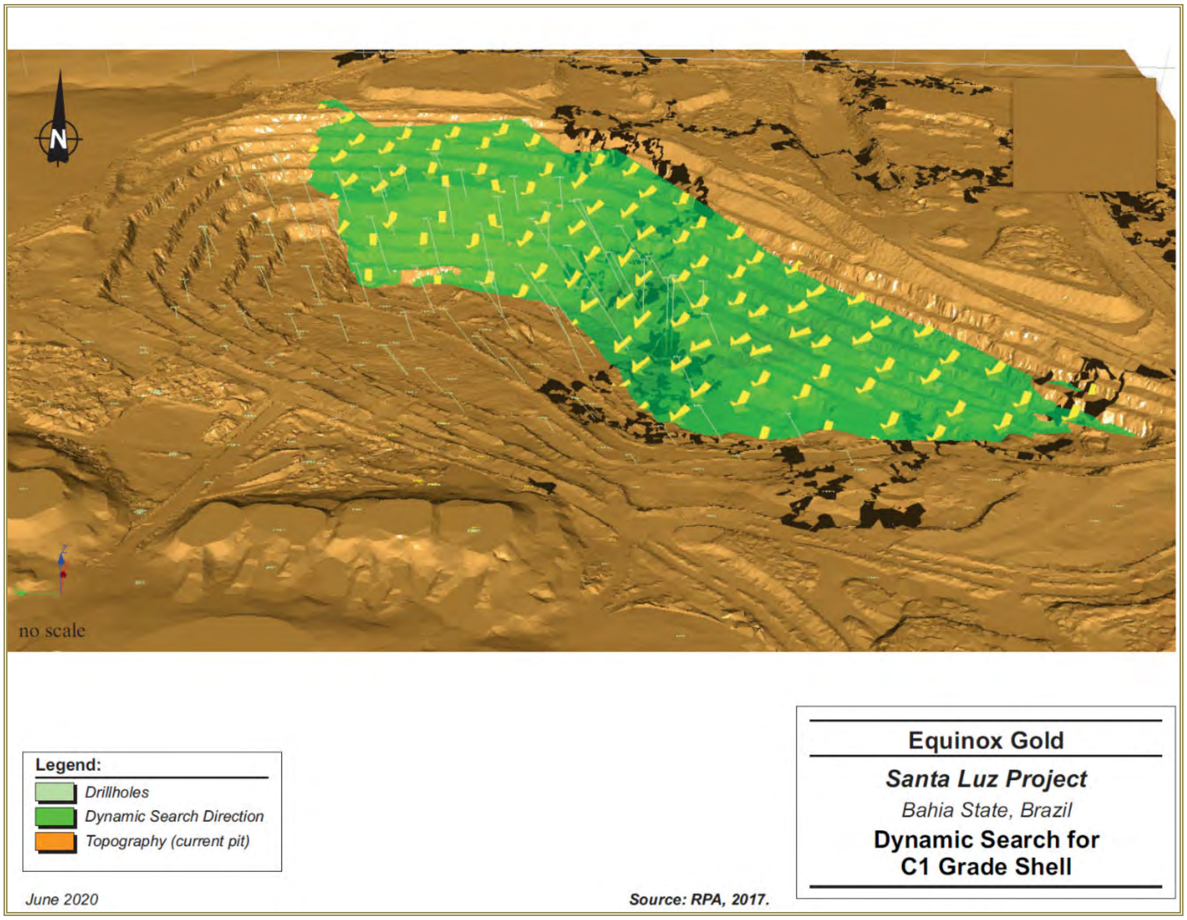


Figure 14-22: Dynamic Search for C1 Grade Shell

14.7 Block Model Validation

SLR validated the block model using the following methods:

- Visual inspection of block versus composite grades on plan, vertical cross section, and long section
- Swath plots of composite grades versus OK and NN grades in the X, Y, and Z directions
- Volumetric comparison of blocks versus wireframes
- Statistical comparison of block grades with assay and declustered composite grades.

14.7.1 Visual Comparison

The block model was visually validated by the examination of the drill hole composite grades compared with the estimated block grades on sections and plans. SLR found grade continuity to be reasonable, and confirmed that the block grades were reasonably consistent with local drill hole composite grades. Figure 14-23 through Figure 14-27 show cross sections and plan views for C1 and Antas 3.

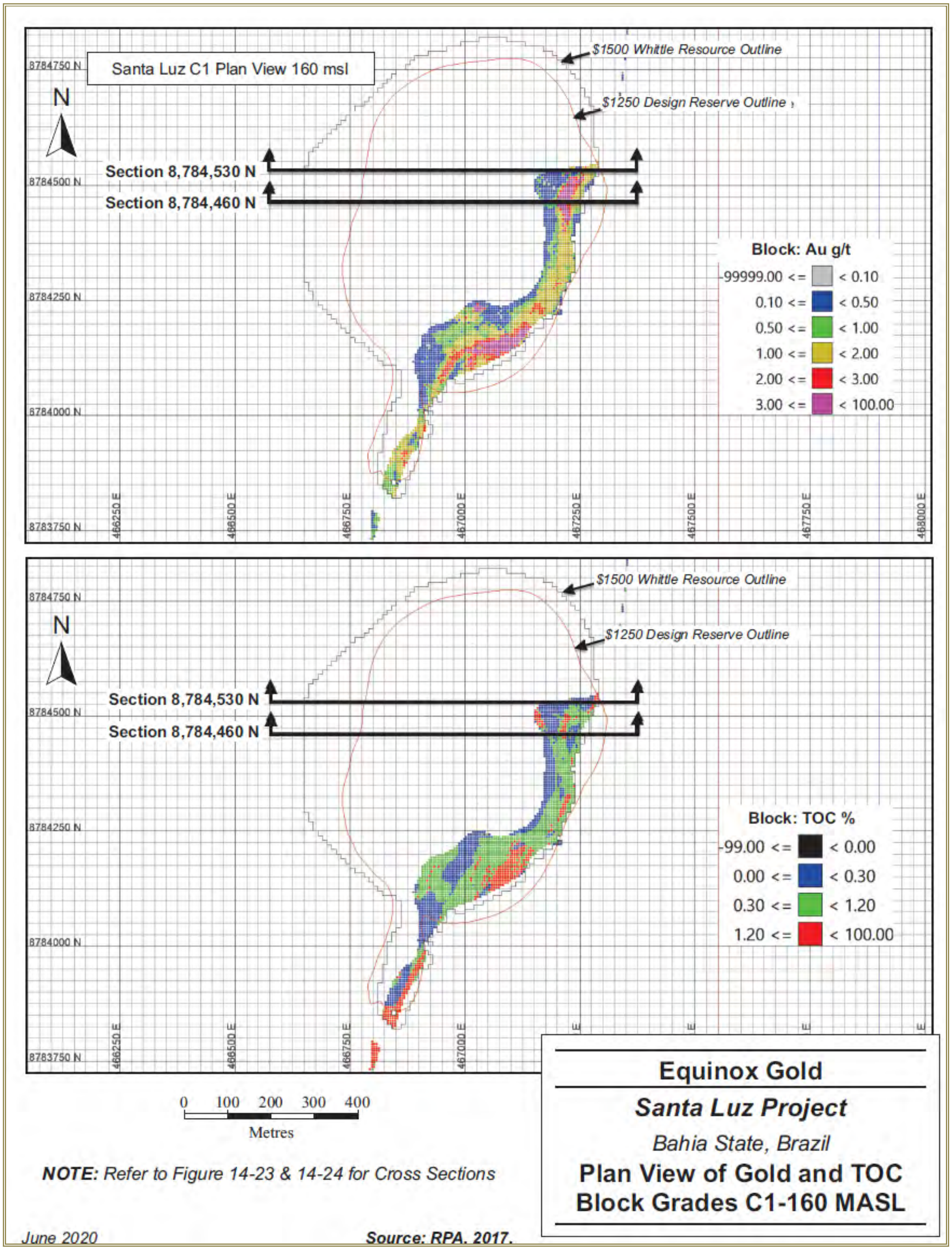


Figure 14-23: Plan View of Gold and TOC Block Grades C1—160 mASL

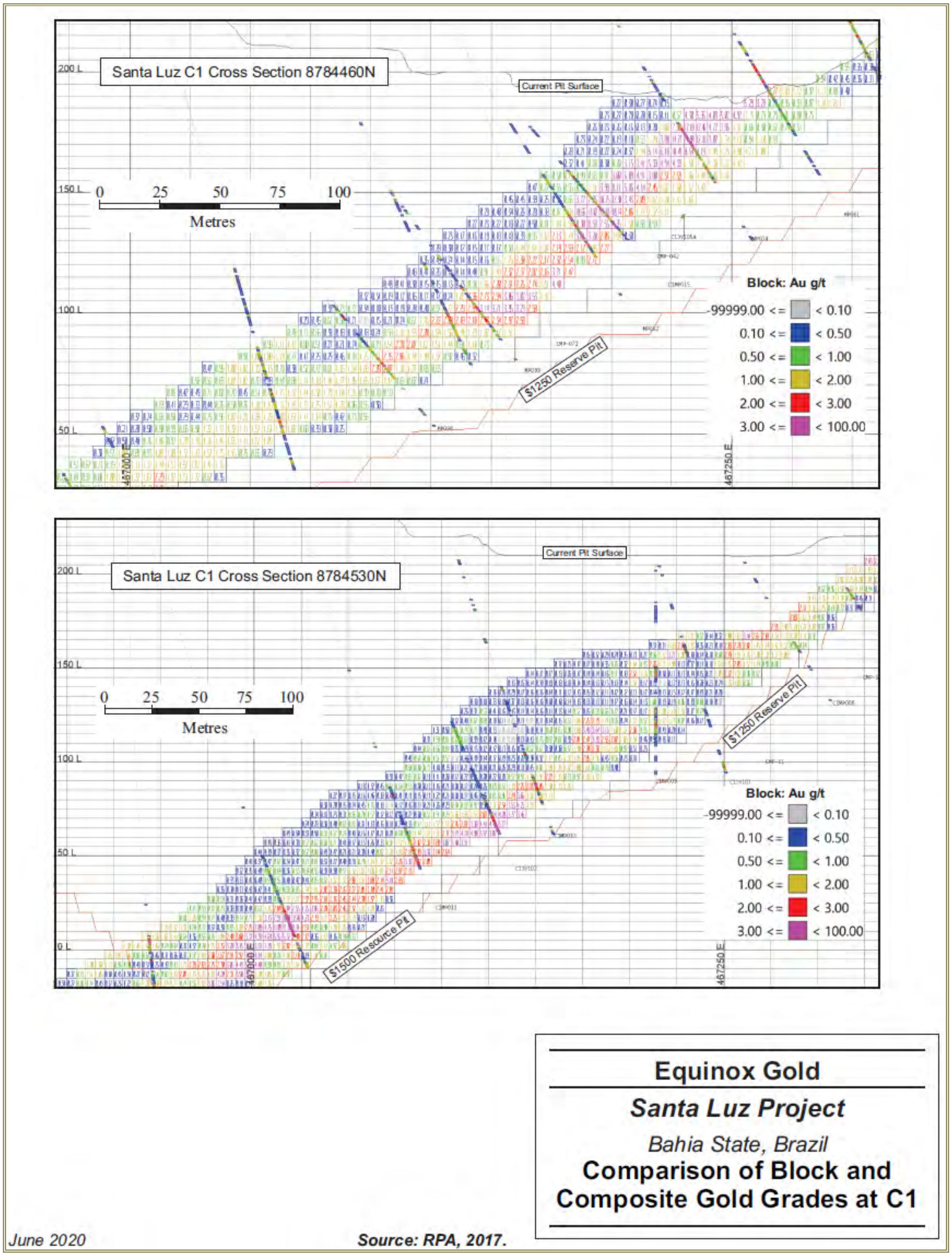


Figure 14-24: Comparison of Block and Composite Gold Grades at C1

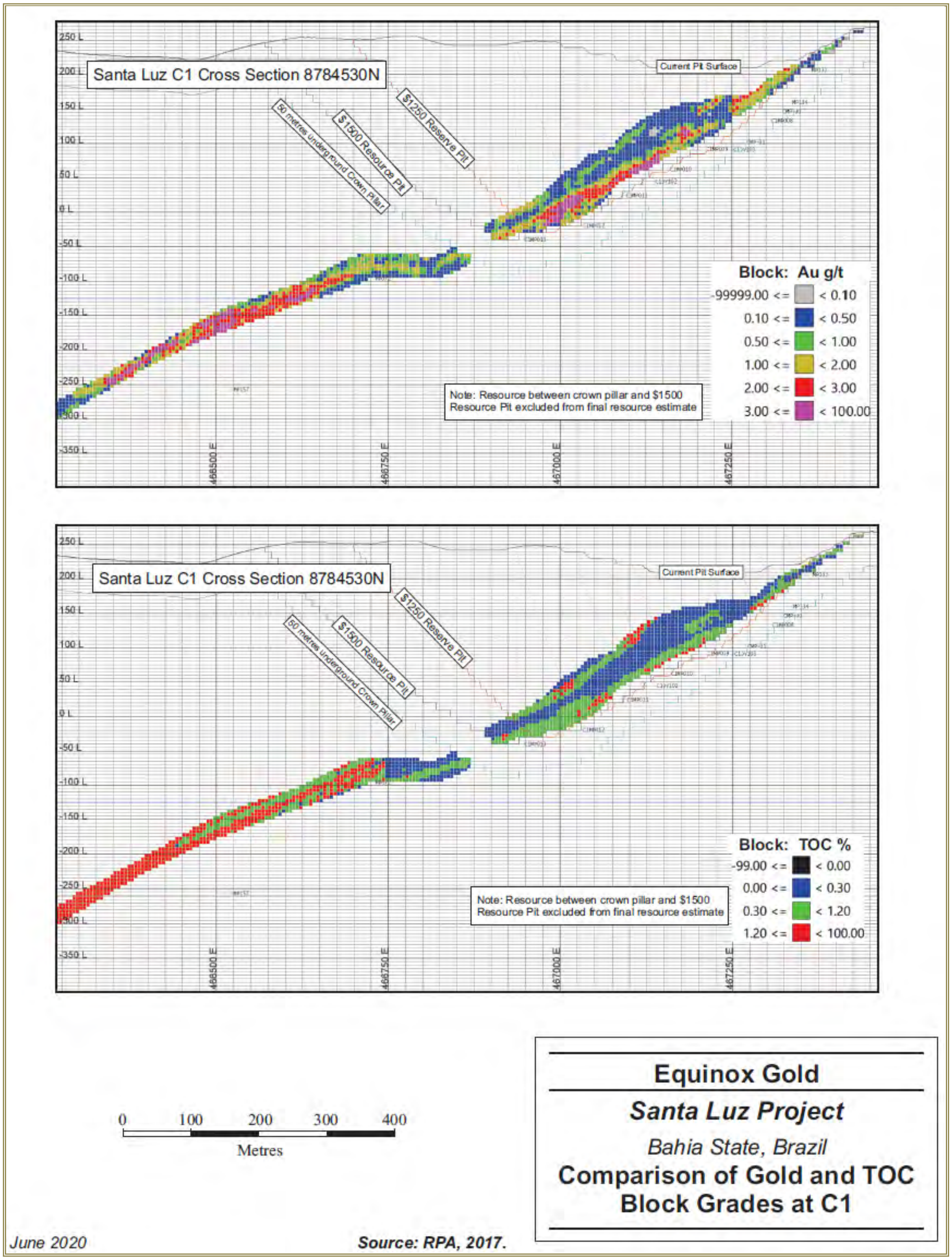


Figure 14-25: Comparison of Gold and TOC Block Grades at C1

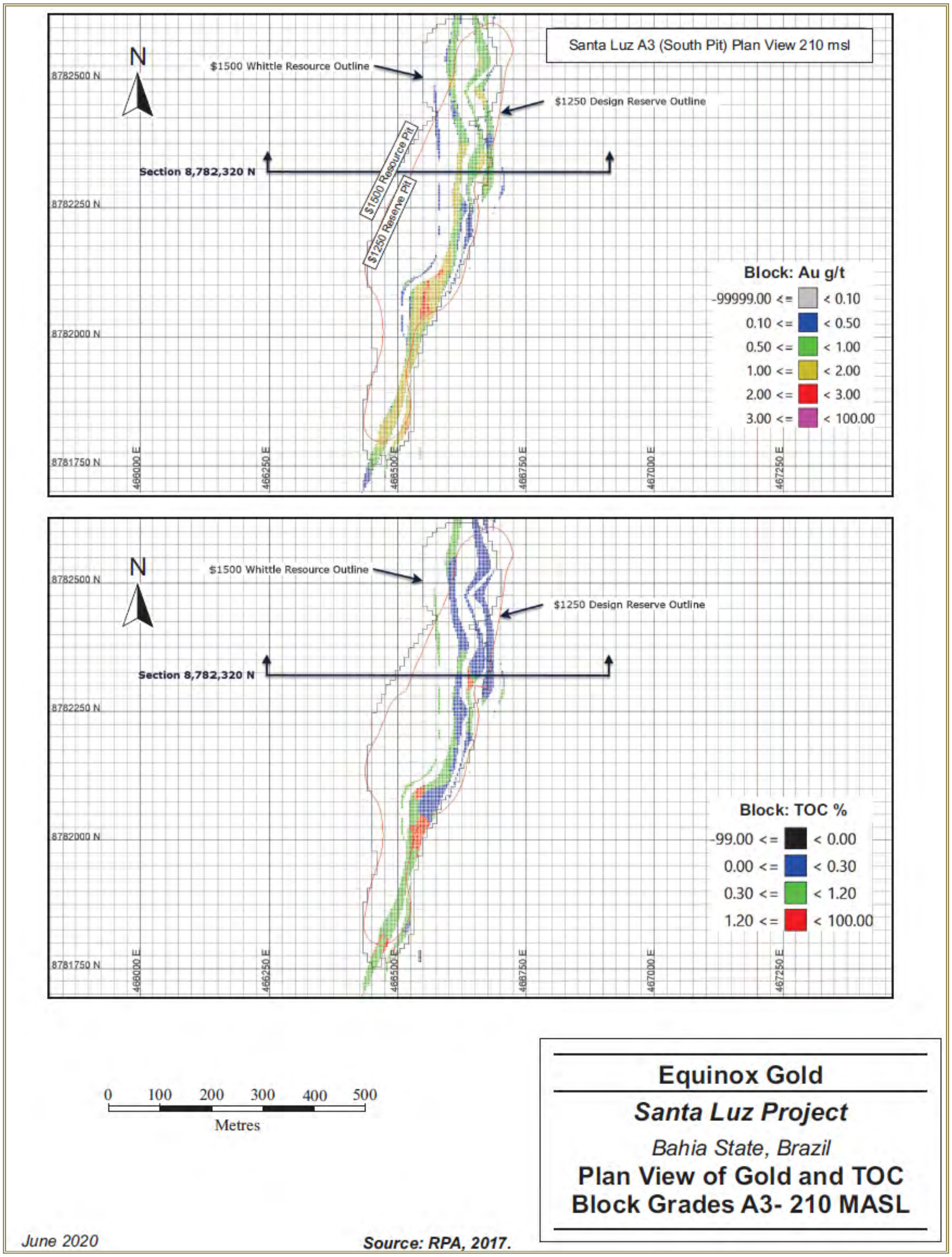


Figure 14-26: Plan View of Gold and TOC Block Grades Antas 3—210 mASL

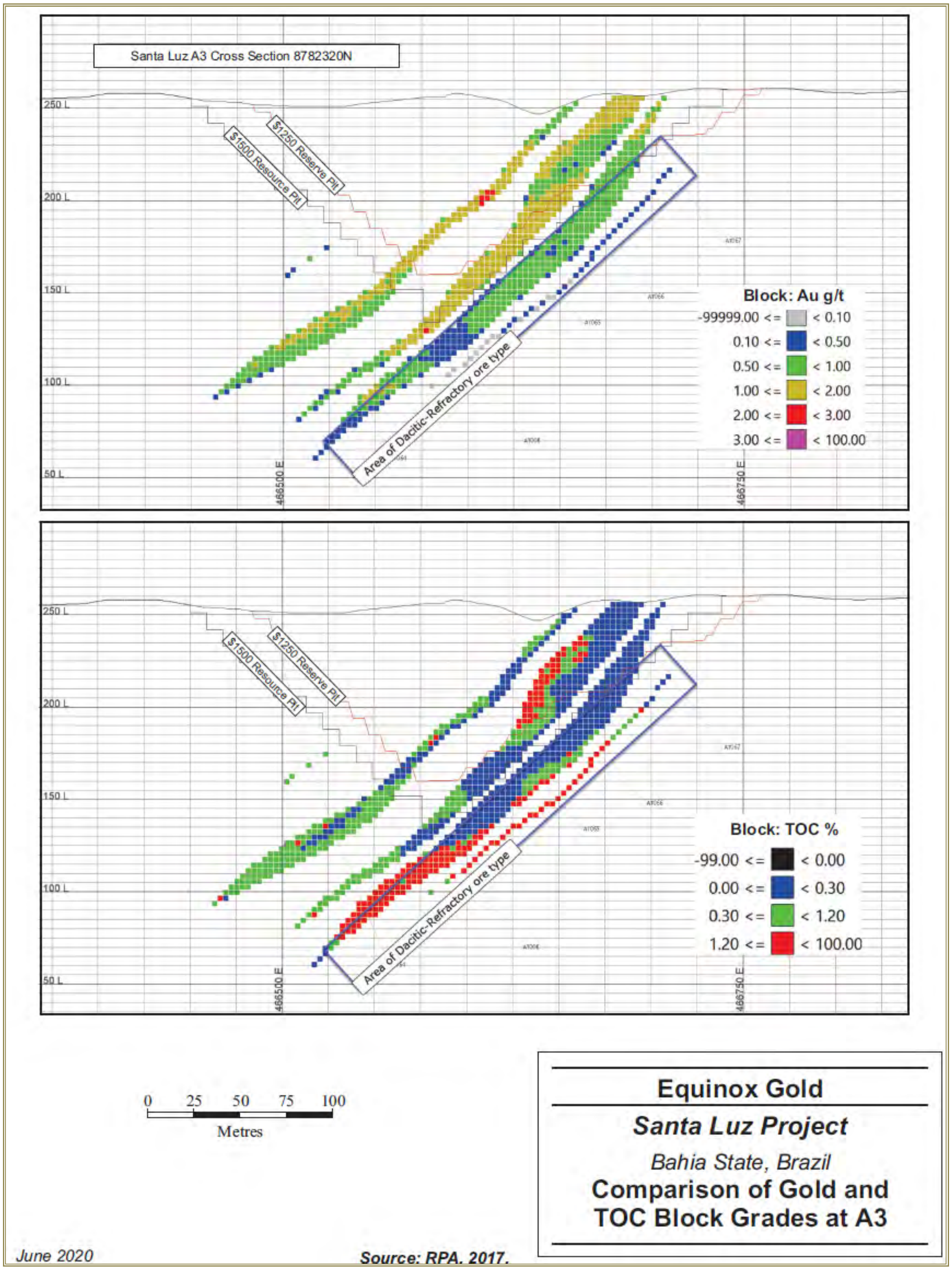
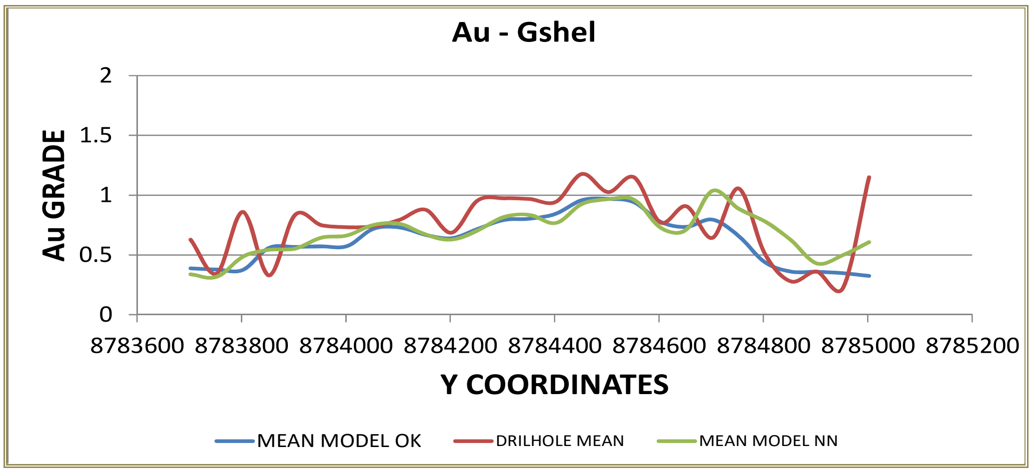


Figure 14-27: Comparison of Gold and TOC Block Grades at Antas 3

14.7.2 Swath Plots

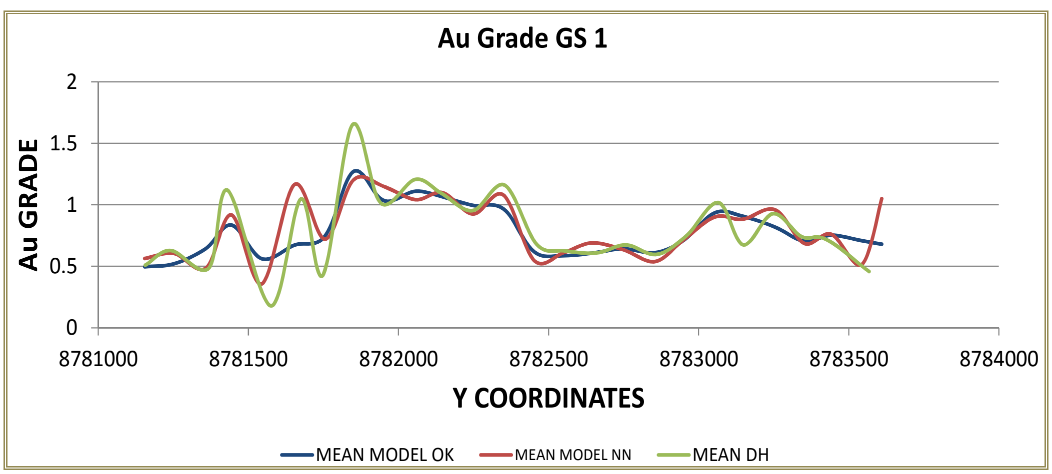
The gold grade block model was also evaluated on a sectional basis using swath plots, as shown in Figure 14-28 and Figure 14-29. These were generated for easterly increments (north–south sections) to show the composite mean compared to the OK gold grade model, as well as with the NN check model. An NN estimate is considered an unbiased grade check, representing the highest grades that would be expected globally in the block model. The OK estimate, being a moving-averages estimate, will tend to be more smoothed. Some local variability between the NN and OK grades would be expected. The swath plots show reasonably good correlation among the OK grades, the NN grades, and the composite grades.



Source: Equinox, 2018

Note: NN = Nearest Neighbour

Figure 14-28: Drift Analysis of C1 Grade Shell Block Estimates vs. NN Composite



Source: Equinox, 2018

Notes: NN = Nearest Neighbour; Variable: (Ore-1)

Figure 14-29: Drift Analysis of Antas 3 Grade Shell Block Estimates vs. NN Composite

14.7.3 Volume Comparison

Wireframe volumes within the Mineral Reserve pit were compared to block volumes for each domain at the Santa Luz deposits. This comparison is summarized in Table 14-11 and results show that there is good agreement between the total wireframe volumes and block model volumes.

Table 14-11: Volume Comparison

Deposit	Zone	Wireframe Volume (m^3)	Block Model Volume (m^3)	% Difference
Antas 2	grade_shell_Antas2_C.00t	5,897,842	5,897,842	0.00
Antas 3	0_0c.00t	156,358	156,358	0.00
	0_1c.00t	5,002,580	5,002,580	0.00
	0_2c.00t	1,909,786	1,909,786	0.00
	0_3c.00t	1,461,436	1,461,436	0.00
	0_4c.00t	673,812	673,812	0.00
	0_5c.00t	72,556	72,556	0.00
C1	LOW_GRADE–SEL_LOW_TOPO.00t	58,027,133	62,279,491	7.33
Grand Total		**73,201,504**	**77,453,862**	**5.81**

14.7.4 Statistical Comparison

A statistical comparison of the estimated block grades with the 1 m and 3 m composites is shown in Table 14-12. Declustered composites are shown to reduce the effects of the drilling bias towards the mineralized areas, and to compare more directly with the model blocks, which are declustered by the kriging process. The block results compare well with the composites, indicating a reasonable overall representation of the gold grades in the block model.

Table 14-12: Block Gold Grades vs. Composite Grades

Descriptive Statistics	Antas 2		Antas 3		C1	
	Comp.	Block	Comp.	Block	Comp.	Block
Au						
Number of Samples	1,095	7,467	1,396	470,184	10,046	463,737
Min	0.000	0.047	0.014	0.000	0.000	0.000
Max	15.00	12.05	8.74	4.77	15.00	15.00
Mean	0.53	0.90	0.86	0.59	0.90	1.00
Variance	1.45	1.00	0.68	0.18	2.54	0.73
SD	1.20	1.00	0.82	0.43	1.59	1.00
CV	2.29	1.11	0.96	0.73	1.77	1.20
TOC						
Number of Samples	1,095	21,766	1,396	470,184	10,046	463,861
Min	0.000	0.010	0.000	0.010	0.000	0.000
Max	9.89	4.32	4.03	2.87	9.71	5.41
Mean	0.37	0.66	0.10	0.46	0.19	0.81
Variance	0.99	0.46	0.11	0.21	0.34	0.40
SD	0.99	0.68	0.34	0.46	0.58	0.64
CV	2.66	1.02	3.24	1.00	3.11	0.79

Notes: SD = standard deviation; CV + coefficient of variation

14.8 Cut-off Grade

CIM Definition Standards (CIM, 2014) specifies that to satisfy the definition of Mineral Resources, there must be "reasonable prospects for eventual economic extraction." This is most commonly understood to imply that a cut-off grade should be applied to the resource model that reflects some generally acceptable assumptions concerning metal prices, metallurgical recoveries, costs, and other operational constraints. SLR prepared a preliminary open pit shell to constrain the block model for resource reporting purposes. The preliminary pit shell was generated using Whittle software.

SLR notes that the price assumptions for gold are different for Mineral Resource and Mineral Reserve pit shells. The Mineral Resource pit shell uses US$1,500/oz as the gold price for Whittle pit optimization and a 0.50 g/t Au cut-off grade. The Mineral Resource for the underground portion used a 1.50 g/t Au cut-off grade. The cost and gold recovery assumptions are the same as for the Mineral Reserve estimate. The Mineral Resource pit optimization inputs are shown in Table 14-13.

Table 14-13: Pit Shell Optimization Factors as of June 30, 2020

Item	Resource Pit Assumptions
C1	
Gold Price	US$1,500/oz Au for geologic resource
Payable	100.00%
Mining Operating Cost	US$2.09/t of waste moved US$2.40/t of ore moved Both costs +0.03 US$/t/a and +0.0072 US$/t/10-m bench
Process Op. + G&A	US$16.65/t feed—dacite-leachable and carbonaceous
Gold Sales Cost	US$10.00/oz—dacite-leachable and carbonaceous
Metallurgic Recovery	84%—dacite-leachable 84%—carbonaceous
Hanging Wall Pit Slope	27.5°—oxide; 48.5 degrees—fresh rock
Footwall Pit Slope	29.2°—oxide; 40.7 degrees—fresh rock
Royalties and Taxes	CBPM = 4.5% gross revenue royalty
Haulage Charge	US$0.00/t of ore hauled
	(Measured + Indicated + Inferred)
Block Model	2855_geo_mod_c1_2017_ind_rev3_rpa_reg_translate.bmf (Measured + Indicated + Inferred)
Antes 3	
Gold Price	US$1,500/oz Au for geologic resource
Payable	100.00%
Mining Operating Cost	US$2.09/t of waste moved US$2.40/t of ore moved Both costs +0.03 US$/t/a and +0.0072 US$/t/ 10-m bench
Process Op. Cost + G&A	US$16.65/t feed—dacite-leachable and carbonaceous
	US$11.92/t feed—dacite-high-sulphide
Gold Sales Cost	US$10.00/oz—dacite-leachable and carbonaceous US$177.00/oz—dacite-high-sulphide
Metallurgic Recovery	84%—dacite-leachable 84%—dacite-high-sulphide 84%—carbonaceous
Hanging Wall Pit Slope	27.5°—oxide; 48.5 degrees—fresh rock
Footwall Pit Slope	29.2°—oxide; 40.7 degrees—fresh rock
Royalties and Taxes	CBPM = 4.5% gross revenue royalty
Haulage Charge	US$0.00/t of ore hauled
Block Model	2855_geo_mod_a3_final_nn_rpa_reg_translate.bmf (Measured + Indicated + Inferred)
Antes 2	

Item	Resource Pit Assumptions
Gold Price	US$1,500/oz Au for geologic resource
Payable	99.90%
Mining Operating Cost	US$1.94/t of ore or waste moved at 0.007 US$/t/5 m bench
Process Operating Cost	US$14.40/t feed—Dacite (oxide & sulphide) US$17.18/t feed—Carbonaceous (oxide & sulphide)
G&A	US$2.8/t feed
Gold Sales Cost	US$3.25/oz
Metallurgic Recovery	90%—dacite (oxide & sulphide)—TOC <0.3% 81%—carbonaceous (oxide & sulphide)—TOC >= 0.3% & TOC <1.2 78%—carbonaceous (oxide & sulphide)—TOC >= 1.2
Hanging Wall Pit Slope	45°
Footwall Pit Slope	38°
Royalties and Taxes	CBPM = 4% gross revenue royalty / Not CBPM = 2% gross revenue royalty
Haulage Charge	US$0.00/t of ore hauled
Block Model	mgs_litho_a2_2016_rev2_dm_rpa.bmf (Measured + Indicated + Inferred Resources)

Metal prices used for Mineral Reserves are based on consensus, long-term forecasts from banks, financial institutions, and other sources. For Mineral Resources, metal prices used are higher than those for Mineral Reserves. The foreign exchange rate used in this analysis has changed and it represents an economic benefit to the Project.

Table 14-14 and Figure 14-30 and Figure 14-31 show the sensitivity of the Santa Luz block models for C1 and Antas 3 to various cut-off grades. The tonnages and grades shown are for comparative purposes only and are not to be considered as Mineral Resources for the purpose of this Mineral Resource estimate.

Table 14-14: C1 and Antas 3 Sensitivity to Cut-Off Grade

Cut-off Au (g/t)	C1		Antas 3	
	Tonnes	Mean Au (g/t)	Tonnes	Mean Au (g/t)
0	156,419,031	0.71	34,213,829	0.59
0.1	150,120,456	0.74	30,541,010	0.66
0.2	118,651,825	0.89	27,205,137	0.72
0.3	94,041,950	1.06	24,142,389	0.78
0.4	77,774,358	1.21	21,308,585	0.84
0.5	64,911,473	1.36	18,264,501	0.90
0.6	56,337,075	1.48	15,003,129	0.98
0.7	49,713,194	1.60	11,776,775	1.07
0.8	44,252,024	1.70	9,080,778	1.16
0.9	39,706,084	1.80	7,099,413	1.25
1.0	35,661,478	1.89	5,513,029	1.33
1.1	32,192,469	1.98	4,189,378	1.42
1.2	28,900,769	2.08	3,151,077	1.51
1.3	25,958,363	2.17	2,265,763	1.61
1.4	22,783,021	2.29	1,615,280	1.71
1.5	20,174,798	2.40	1,168,347	1.82
1.6	17,701,423	2.51	856,066	1.91
1.7	15,168,620	2.66	627,606	2.01
1.8	13,379,081	2.78	454,315	2.11
1.9	11,679,338	2.92	328,685	2.21
2.0	10,304,895	3.04	229,271	2.33
2.1	9,159,458	3.17	163,020	2.44
2.2	8,146,605	3.30	115,652	2.56
2.3	7,320,581	3.41	83,704	2.67
2.4	6,566,259	3.54	64,908	2.77
2.5	5,887,018	3.66	47,527	2.88
2.6	5,315,415	3.78	36,206	2.99
2.7	4,788,721	3.90	26,893	3.11
2.8	4,275,228	4.04	20,432	3.22
2.9	3,860,715	4.17	16,018	3.32
3.0	3,492,265	4.30	12,335	3.44

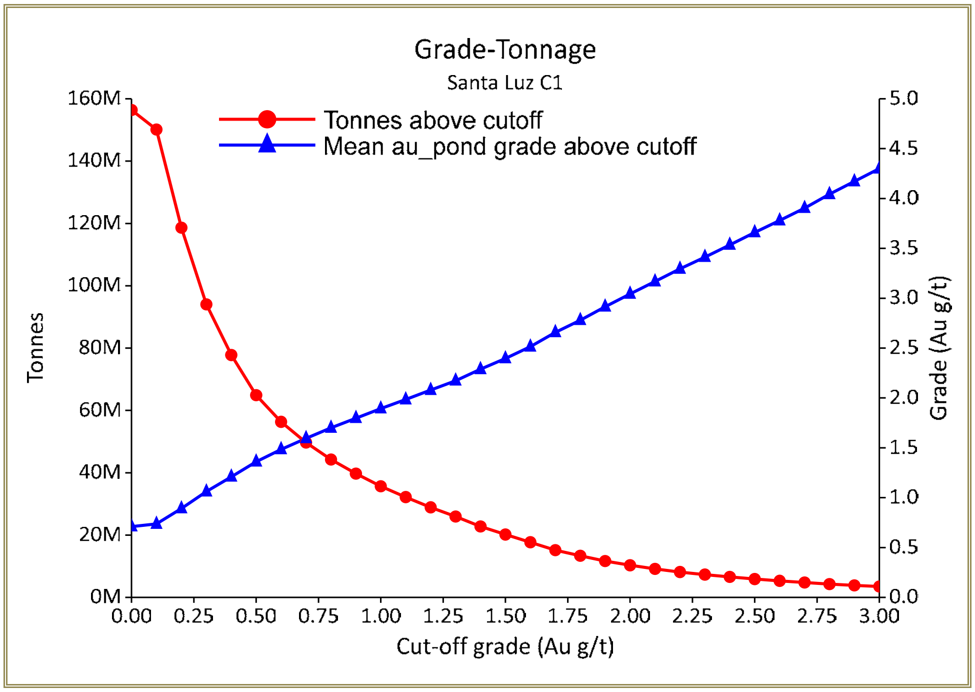


Figure 14-30: C1 Mineral Resource Tonnes and Grade at Various Cut-Off Grades

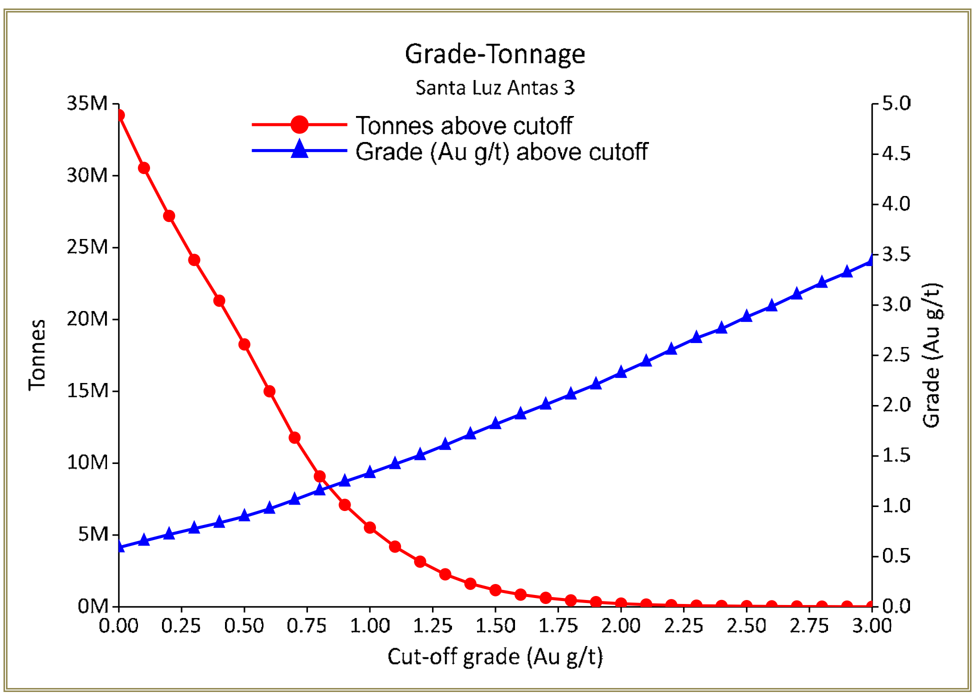


Figure 14-31: Antas 3 Mineral Resource Tonnes and Grade at Various Cut-Off Grades

14.9 Classification

Definitions for resource categories used in this report are consistent with the CIM Definition Standards (CIM, 2014) adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction." Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the "economically mineable part of a *Measured* and/or *Indicated Mineral Resource*" demonstrated by studies at Pre-Feasibility or Feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories.

In SLR's opinion, the Mineral Resources are classified according to the nomenclature specified in CIM (2014). A range of criteria were used in determining an appropriate Mineral Resource classification including:

- Representativeness, quality, and positional accuracy of samples
- Geological continuity and reasonableness of the interpreted mineralized model
- Geostatistical spatial continuity and estimation quality
- Scale of mining and associated level of risk.

Mineral Resources were classified as Measured, Indicated, or Inferred based on the number of samples and drill holes used to estimate a block within a certain range and apparent continuity of mineralization. This distance was based on ranges interpreted from gold correlograms (Table 14-15). After the classification was completed, a manual review and smoothing algorithm was applied to the blocks to smooth the boundaries between categories and eliminate any inconsistencies. Figure 14-32 is an example of the smoothing algorithm being applied to C1 and Antas 3.

Table 14-15: Classification by Deposit

Classification	Deposit	Minimum Samples	Minimum Drill Holes	Search Range (m)		
				X	Y	Z
Measured	C1	8	4	50	50	15
	Antas 2	8	4	50	50	15
	Antas 3	8	4	50	50	15
Indicated	C1	8	4	100	100	30
	Antas 2	8	4	100	100	30
	Antas 3	8	4	100	100	30

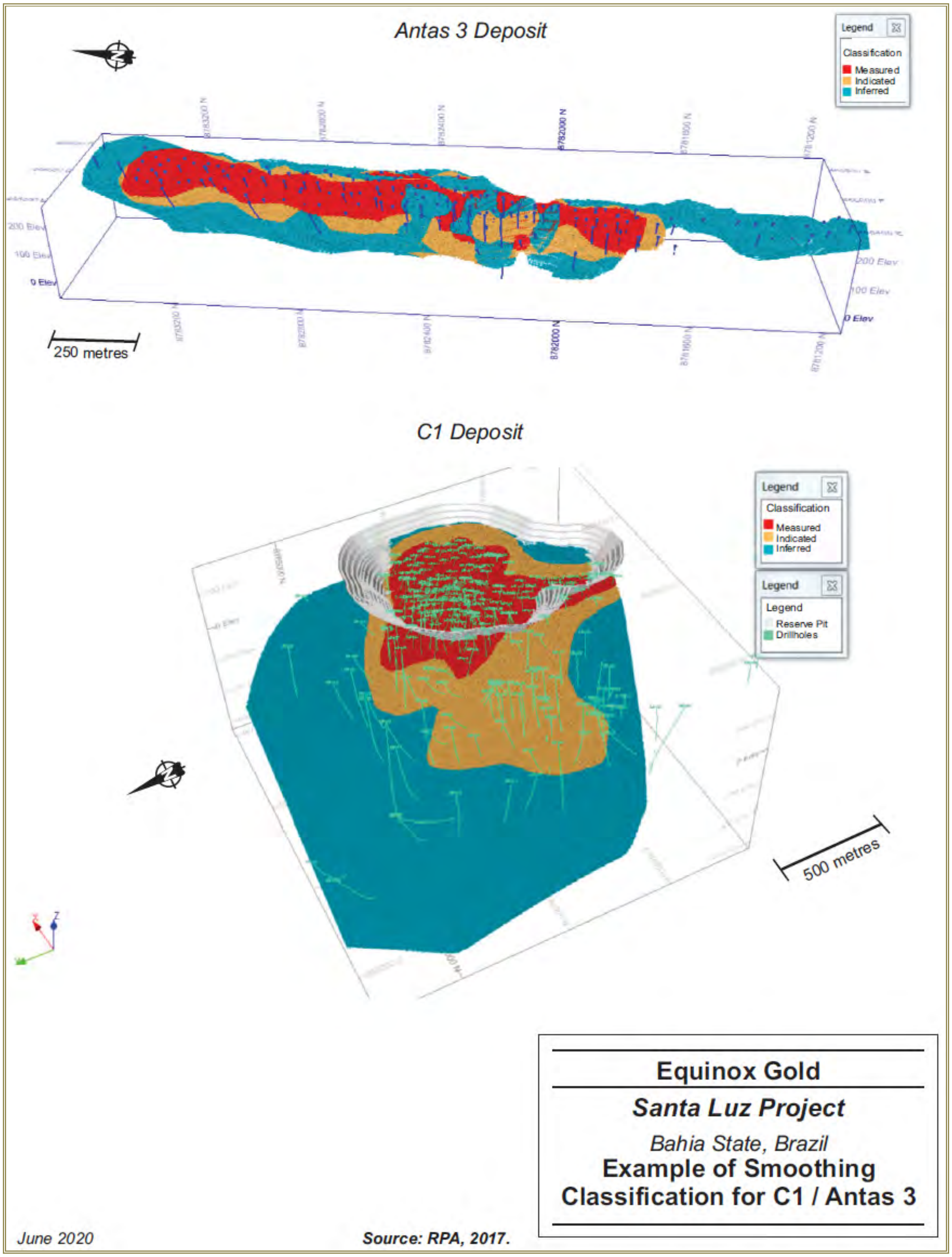


Figure 14-32: Example of Smoothing Classification for C1 and Antas 3

In SLR's opinion, the classification appears to be reasonable, and appropriate for the style of mineralization and deposit type. It is likely that definition drilling will upgrade a portion of the Inferred Mineral Resources to Measured and Indicated Mineral Resources. This is the expectation at depth, where drill hole spacing is currently relatively wide.

14.10 Mineral Resource Reporting

A summary of the Santa Luz Mineral Resources by material type is presented in Table 14-16. A cut-off grade of 1.50 g/t Au was applied to the underground Mineral Resources, and a cut-off grade of 0.50 g/t Au was applied to the open pit Mineral Resources. The Mineral Resources have been depleted for production to March 31, 2015, and are exclusive of Mineral Reserves. SLR has used a gold price of US$1,500/oz. Note that there has been no production on the Project since September 2014.

In SLR's opinion, the assumptions, parameters, and methodology used for the Santa Luz Mineral Resource estimates are appropriate for the style of mineralization and mining methods.

SLR recommends the use of a minimum thickness of 5 m for future resource estimation to remove narrow intersections that may not be economic when *diluted out*. The high grades from narrow intersections will influence block grades in the interpolation process.

Table 14-16: Mineral Resource Summary by Material Type—June 30, 2020

Classification	Type	Tonnes ('000s)	Gold Grade (g/t)	Contained Gold (oz)	TOC (%)
Open Pit Resources					
Measured	Dacite-Leach	3,304	1.06	112,802	0.08
	Dacite-High Sulphide	773	1.02	25,275	0.07
	Carbonaceous	5,909	1.33	252,229	1.01
Total Measured		**9,986**	**1.22**	**390,306**	**0.63**
Indicated	Dacite-Leach	112	0.99	3,590	0.11
	Dacite-High Sulphide	65	1.04	2,165	0.12
	Carbonaceous	385	0.98	12,169	0.96
Total Indicated		**562**	**0.99**	**17,924**	**0.69**
Measured + Indicated	Dacite-Leach	3,416	1.06	116,392	0.08
	Dacite-High Sulphide	838	1.02	27,440	0.07
	Carbonaceous	6,294	1.31	264,398	1.00
Total Measured + Indicated	**Open Pit**	**10,548**	**1.20**	**408,230**	**0.63**
Inferred	Dacite-Leach	153	1.1	5,440	0.13
	Dacite-High Sulphide	11	0.7	248	0.11
	Carbonaceous	529	1.35	23,060	0.98
Inferred Total	**Open Pit**	**694**	**1.29**	**28,748**	**0.78**
Underground Resources					
Measured	Dacite-Leach	79	1.91	4,827	0.10
	Dacite-High Sulphide	0	0	0	0.00
	Carbonaceous	42	2.01	2,733	0.71
Total Measured		**121**	**1.94**	**7,561**	**0.32**

Classification	Type	Tonnes ('000s)	Gold Grade (g/t)	Contained Gold (oz)	TOC (%)
Indicated	Dacite-Leach	695	2.35	52,549	0.10
	Dacite-High Sulphide	0	0	0	0.00
	Carbonaceous	5,218	2.57	431,518	1.37
Total Indicated		**5,913**	**2.55**	**484,066**	**1.22**
Measured + Indicated	Dacite-Leach	774	2.31	57,376	0.10
	Dacite-High Sulphide	0	0	0	0.00
	Carbonaceous	5,260	2.57	434,251	1.37
Total Measured + Indicated	**Underground**	**6,034**	**2.53**	**491,627**	**1.21**
Inferred	Dacite-Leach	890	1.9	54,375	0.09
	Dacite-High Sulphide	0	0	0	0.00
	Carbonaceous	5,670	2.23	406,992	1.29
Total Inferred	**Underground**	**6,560**	**2.19**	**461,367**	**1.13**
Grand Total Measured + Indicated		**16,582**	**1.69**	**899,857**	**0.84**
Grand Total Inferred		**7,254**	**2.10**	**490,115**	**1.09**

Notes:
1. CIM Definition Standards (2014) definitions were followed for Mineral Resource estimates.
2. Open Pit Mineral Resources are reported at a cut-off grade of 0.50 g/t Au.
3. Underground Mineral Resources are reported at a cut-off grade of 1.50 g/t Au.
4. Mineral Resources are exclusive of Mineral Reserves.
5. No minimum thickness was used in the resource estimation.
6. Mineral Resources are estimated using a short-term gold price of US$1,500/oz and constrained by a pit shell.
7. Totals may not add due to rounding.

15 MINERAL RESERVE ESTIMATE

Santa Luz Project personnel have generated the Mineral Reserve estimates for the C1 and Antas 3 deposits and reviewed the stockpile estimates. During May 2020 a number of checks to verify the procedures and numerical calculations used in the estimation of the Mineral Reserves were carried out and the Equinox QP visited the Santa Luz site in June 2020.

The open pit Mineral Reserves as estimated are summarized in Table 15-1, using a gold price of US$1,350/oz Au with a pit design based on selected pits shells and an overall metal recovery of 84% for all types of ore. Mineral Reserves are estimated only for C1, Antas 3, and stockpiles; Antas 2 has not been delineated enough to classify it as a Mineral Reserve. The Mineral Reserve estimates for the C1 and Antas 3 were run using a long-term gold price of US$1,350/oz Au.

Equinox's QP is of the opinion that the Measured and Indicated Mineral Resources within the final pit designs for Santa Luz can be classified as Proven and Probable Mineral Reserves.

Table 15-1: Santa Luz Mineral Reserves—June 30, 2020

Category of Mineral Reserve	Tonnes ('000s)	Gold Grade (g/t)	Contained Gold (oz)
Proven—Open Pit	21,578	1.39	966,106
Probable—Open Pit	1,170	1.28	48,202
Probable—Stockpile	2,191	0.86	60,634
Total Proven & Probable	**24,939**	**1.34**	**1,074,941**

Notes: **1.** CIM Definition Standards (2014) were followed for Mineral Reserve estimates.
2. Mineral Reserves were generated by Equinox based on the June 30, 2020 mining surface.
3. Mineral Reserves are quoted at cut-off grades of 0.52 g/t Au for dacite-leachable and carbonaceous ore, for the C1 deposit; and for the Antas 3 deposit, cut-off grades of 0.54 g/t Au for dacite-leachable and carbonaceous ore and 0.45 g/t Au for dacite-high-sulphide.
4. C1 and Antas 3 use a 10 m bench height (flitch height 5 m benches).
5. Process recovery of 84% for all types of ore.
6. Mineral Reserves were run using a long-term gold price of US$1,350/oz.
7. Totals may not add due to rounding.

Table 15-2 lists the current Mineral Reserves by material type. Table 15-3 and Figure 15-1 provide details of the Mineral Reserves in stockpiles including gold grade and TOC.

Table 15-2: Mineral Reserve Summary by Material Type—June 30, 2020

Classification	Type	Tonnes ('000s)	Gold Grade (g/t)	Ounces	TOC (%)
Open Pit Reserves					
Proven	Dacite-Leach	8,257	1.31	347,407	0.09
	Dacite-High-Sulphide	1,522	1.00	48,817	0.05
	Carbonaceous	11,799	1.50	569,881	0.96
Total Proven		**21,578**	**1.39**	**966,106**	**0.56**
Probable	Dacite-Leach	547	1.43	25,112	0.12
	Dacite-High-Sulphide	178	0.64	3,657	0.07
	Carbonaceous	445	1.36	19,432	0.90
Total Probable		**1,170**	**1.28**	**48,202**	**0.41**
Proven + Probable	Dacite-Leach	8,804	1.32	372,520	0.09
	Dacite-High-Sulphide	1,700	0.96	52,474	0.05
	Carbonaceous	12,244	1.50	589,313	0.96
Total Proven + Probable		**22,748**	**1.39**	**1,014,307**	**0.55**
Stockpile Reserves					
Probable	Dacite	1,481	0.78	37,062	0.15
	Carbonaceous	709	1.03	23,572	0.69
Stockpile Total		**2,191**	**0.86**	**60,634**	0.33
Total Reserves		**24,939**	**1.34**	**1,074,941**	**0.53**

Notes: **1.** CIM Definition Standards (2014) were followed for Mineral Reserve estimates.
2. Mineral Reserves were generated by Equinox based on the June 30, 2020 mining surface.
3. Mineral Reserves are quoted at cut-off grades of 0.52 g/t Au for dacite-leachable and carbonaceous ore, for the C1 deposit; and for the Antas 3 deposit, cut-off grades of 0.54 g/t Au for dacite-leachable and carbonaceous ore and 0.45 g/t Au for dacite-high-sulphide.
4. C1 and Antas 3 use a 10 m bench height (flitch height 5 m benches).
5. Process recovery of 84% for all types of ore.
6. Mineral Reserves were run using a long-term gold price of US$1,350/oz.
7. Totals may not add due to rounding.

Table 15-3: Mineral Reserve Summary of Stockpiles—June 30, 2020

Stockpile Name	Type	Tonnes (t '000s)	Old Grade (g/t)	TOC (%)	Contained Metal (oz Au)
01_DAC	Dacite	333	0.93	0.25	9,922
01_CARB	Low Carbon	187	0.95	0.46	5,754
Subtotal Stockpile 1		**521**	**0.94**	**0.32**	**15,676**
02_DAC	Heap Leach	452	0.56	0.08	8,149
03_DAC	Heap Leach	106	0.66	0.10	2,227
04_DAC	Heap Leach	351	0.74	0.12	8,304
Subtotal Stockpile 2-3-4		**908**	**0.64**	**0.10**	**18,680**
05_DAC	Dacite	73	0.90	0.28	2,096
05_CARB	Low Carbon	58	1.11	0.49	2,063
Subtotal Stockpile 5		**131**	**0.99**	**0.37**	**4,159**
06_CARB	Low Carbon	418	1.06	0.84	14,217
07_DAC	Dacite	167	1.19	0.21	6,364
07_CARB	Low Carbon	46	1.04	0.54	1,538
Subtotal Stockpile 6-7		**631**	**1.090**	**0.65**	**22,119**
Total Dacite		1,481	0.78	0.15	37,062
Total Low Carbon		709	1.03	0.69	23,572
Grand Total		**2,191**	**0.86**	**0.33**	**60,634**

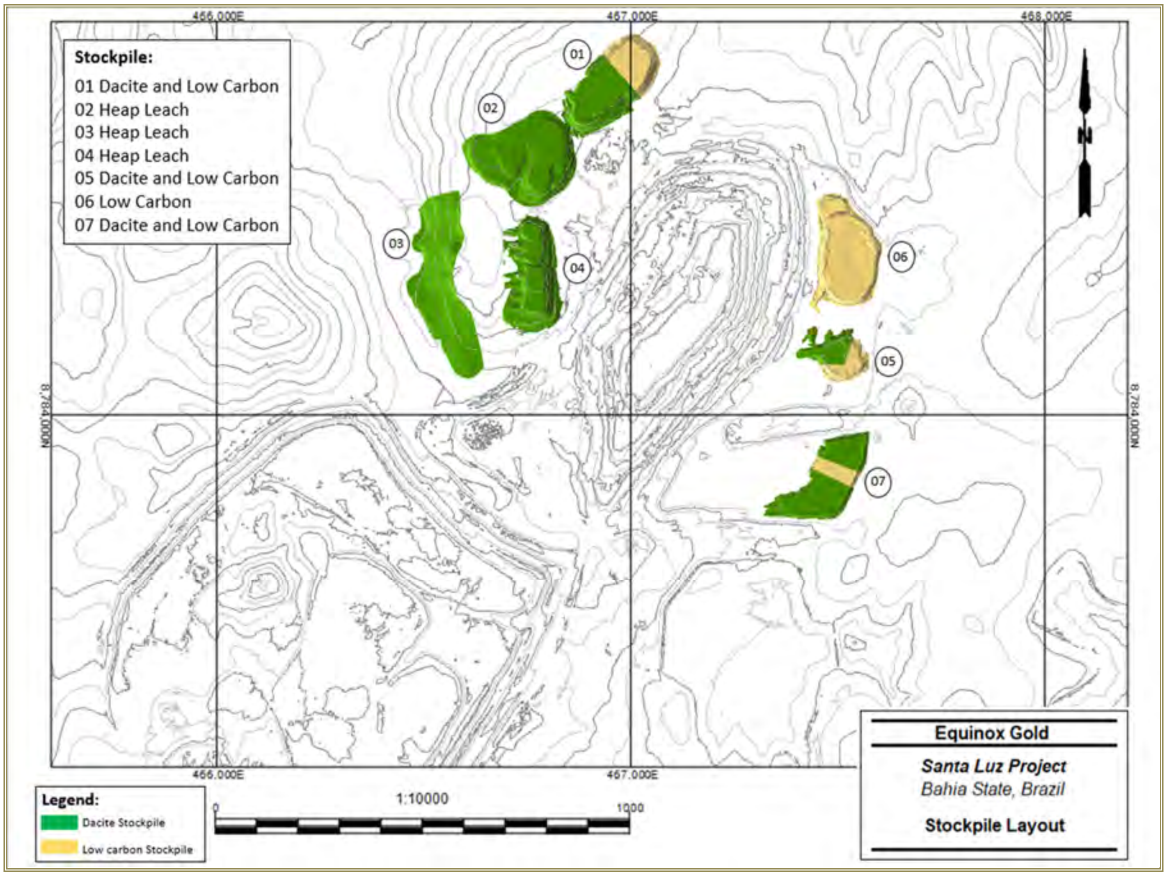


Figure 15-1: Stockpile Layout

15.1 Production Reconciliation

A balance of the 2013 and 2014 production was carried out by RPA in 2018. It was verified that the depleted tonnage from the 2016 block model was approximately 19% less than production summary reports.

Reasons for this difference were carefully discussed with Santa Luz Project personnel and can be summarized in the following points:

- Poor sampling from RC drilling campaign impacted the Mineral Resource estimate.
- Different cut-off grade criteria applied for long-term Mineral Reserves and short-term plan.
- Part of material assigned as waste in the 2016 block model was mined as ore based on operations factors applied in the period.

Based on this analysis, it is considered that there is a reasonable understanding about the reconciliation numbers. It is recommended that the issues raised in this discussion should be used as improvements to be implemented in the short-term plans and grade control once the mining operations resume.

15.2 Dilution

The internal dilution for C1 was estimated comparing the original block model grade (5 m x 5 m x 5 m blocks) and the re-blocked grade (10 m x 10 m x 10 m blocks). Antas 3 internal dilution was estimated comparing the original block model grade (3 m x 3 m x 3 m blocks) and the re-blocked grade (9 m x 9 m x 9 m blocks). Table 15-4 presents the open pit internal dilution included in the block model grade.

Table 15-4: Current Open Pit Mining Dilution

Zone Name	Block Model Internal Dilution %
C1	4.2%
Antas 3 North	3.9%
Antas 3 South	4.9%

The mine plan was based on 10 m x 10 m x 10 m block size for C1 and 9 m x 9 m x 9 m block size for Antas 3, and both block models included internal dilution. No additional dilution was applied to the mineralized blocks in the model. Mining in 10 m bench height increments (as double benches of 5 m each) and selective loading equipment were considered to minimize the waste inclusion and to maximize ore extraction.

15.3 Extraction

An open pit mining extraction factor of 97% for C1 and 95% for Antes 3 was applied in NPV Scheduler pit optimization.

15.4 Cut-off Grade

Mineral Reserves included in the Open Pit Feasibility Study were based on open pit mine designs. Various cut-off grades were used to generate the open pit production schedule based on 2017 parameters for C1 and Antes 3. An updated set of cut-off grades was calculated based on US$1,350/oz Au price and R$4.00:US$1.00 exchange rate, with no significant change from the 2018 cut-off grades.

The economic parameters used to calculate the cut-off grade at a US$1,350/oz Au price, and a comparison of 2018 and 2020 cut-off grade calculations are presented in Table 15-5. The cut-off grades applied to the Mineral Reserve estimate and production schedule were:

- For C1, 0.54 g/t Au for dacite-leachable and carbonaceous ore
- For Antas 3, 0.54 g/t Au for dacite-leachable and carbonaceous ore and 0.45 g/t Au for dacite-high-sulphide.

Due to block sizes, 5% was considered for operational dilution for C1 and 10% for Antas 3.

Table 15-5: Cut-Off Grade Parameters

Parameter	2018 Both	2020	
		C1	Antas 3
Gold Price (US$/oz)	1,250	1,350	1,350
Payable (%)	99.50	100	100
Gold Sales Cost (US$/oz)—Dacite-leachable	3.25	10.00	10.00
Gold Sales Cost (US$/oz)—Dacite-High-Sulphide	3.25	-	177.00
Gold Sales Cost (US$/oz)—Carbonaceous	3.25	10.00	10.00
Royalties (Gross Revenue) (%)	2.0	4.5	4.5
Exchange Rate (R$:US$)	3.5	4.0	4.0
Mine			
Premium Cost for Ore (US$)	-	0.31	0.31
Operational Dilution (%)	-	5.0	10.0
Process + G&A (US$)			
Dacite-Leachable	17.94	16.65	16.65
Dacite-High-Sulphide	12.73	-	11.92
Carbonaceous	19.72	16.65	16.65
Metallurgic Recovery			
Dacite-Leachable (%)	86	84	84
Dacite-High-Sulphide (%)	84	-	84
Carbonaceous (%)	84	84	84
Cut-Off Grade (g/t)			
Dacite-Leachable	0.534	0.515	0.540
Dacite-High-Sulphide	0.388	-	0.448
Carbonaceous	0.601	0.515	0.540

Two factors may have a material impact on the Mineral Reserve estimate: gold price and metallurgical recovery. A sensitivity analysis to gold is discussed in Section 22.

15.5 Pit Shell Selection

The open pit optimization for the Mineral Reserves was generated using NPV Scheduler software. The open pit optimization parameters are discussed in Section 16.1.2.

The selected pit shells for the final pit design were based on US$1,161/oz Au for C1 at a revenue factor (RF) of 0.86 as presented in Table 15-6, and US$1,296/oz Au for Antas 3 at an RF of 0.96 as presented in Table 15-7. Pits for C1 and Antes 3 were selected to avoid the discounted cash flow decrease associated with a significant incremental increase of the stripping ratio at a higher RF.

Table 15-6: C1 Pit Optimization Results

Revenue Factor (RF)	Gold Price (x RF) (US$/oz)	Total Rock Mined Tonnes ('000s)	Waste Mined Tonnes ('000s)	Ore Mined		Stripping Ratio (w:o)	Undiscounted Cash Flow Excluding Project CAPEX 5% (US$ M)	Discounted Cash Flow Excluding Project CAPEX 5% (US$ M)
				Tonnes ('000s)	(g/t)			
0.06	81	0	0	0	7.94	0.0	0.0	0.0
0.08	108	21	8	13	7.44	0.6	3.0	3.0
0.10	135	102	48	54	5.86	0.9	9.4	9.4
0.12	162	140	53	87	5.63	0.6	14.6	14.5
0.14	189	414	222	192	5.15	1.2	28.9	28.9
0.16	216	853	531	321	4.93	1.7	45.6	45.5
0.18	243	1,026	570	456	4.39	1.2	57.0	56.8
0.20	270	1,177	612	565	4.14	1.1	66.0	65.6
0.22	297	1,455	740	715	3.82	1.0	76.1	75.5
0.24	324	2,773	1,736	1,037	3.61	1.7	101.6	100.6
0.26	351	3,690	2,328	1,363	3.35	1.7	121.2	119.6
0.28	378	3,810	2,352	1,458	3.26	1.6	125.6	123.8
0.30	405	4,256	2,636	1,619	3.16	1.6	134.1	131.9
0.32	432	5,209	3,090	2,119	2.82	1.5	152.3	148.9
0.34	459	5,819	3,457	2,361	2.73	1.5	162.5	158.5
0.36	486	7,399	4,399	2,999	2.53	1.5	186.5	180.7
0.38	513	8,405	5,057	3,349	2.45	1.5	199.5	192.7
0.40	540	8,901	5,192	3,709	2.36	1.4	209.9	202.0
0.42	567	9,522	5,536	3,987	2.30	1.4	218.0	209.1
0.44	594	10,313	6,063	4,250	2.26	1.4	225.9	216.1
0.46	621	11,313	6,726	4,588	2.20	1.5	235.0	224.0
0.48	648	12,659	7,441	5,218	2.10	1.4	249.6	236.6
0.50	675	14,749	8,984	5,765	2.04	1.6	263.5	248.6
0.52	702	15,511	9,353	6,158	1.99	1.5	271.2	255.0
0.54	729	16,761	10,282	6,479	1.96	1.6	278.4	261.1
0.56	756	17,757	10,987	6,770	1.94	1.6	284.0	265.8
0.58	783	18,498	11,597	6,901	1.93	1.7	286.8	268.0
0.60	810	19,257	12,087	7,170	1.90	1.7	291.0	271.3
0.62	837	19,821	12,467	7,354	1.88	1.7	293.7	273.4
0.64	864	21,126	13,526	7,600	1.86	1.8	297.8	276.5
0.66	891	27,099	18,374	8,725	1.79	2.1	314.8	289.5
0.68	918	27,947	18,993	8,954	1.77	2.1	317.5	291.4
0.70	945	29,053	19,882	9,171	1.76	2.2	320.2	293.3
0.72	972	32,201	22,508	9,693	1.73	2.3	326.5	297.7
0.74	999	42,951	32,096	10,855	1.71	3.0	342.8	307.9
0.76	1,026	46,190	34,938	11,252	1.70	3.1	347.4	310.6
0.78	1,053	46,488	35,145	11,343	1.70	3.1	348.1	310.9
0.80	1,080	77,781	63,017	14,765	1.63	4.3	381.2	327.0
0.82	1,107	81,047	65,839	15,208	1.61	4.3	384.8	328.4
0.84	1,134	83,603	67,970	15,634	1.60	4.3	387.4	328.9
0.86	1,161	91,601	75,074	16,528	1.58	4.5	393.2	328.7
0.88	1,188	103,986	86,524	17,462	1.58	5.0	399.5	328.0
0.90	1,215	110,350	91,977	18,373	1.56	5.0	403.2	327.5
0.92	1,242	110,976	92,485	18,490	1.55	5.0	403.6	327.3
0.94	1,269	119,347	100,221	19,125	1.55	5.2	405.7	325.3
0.96	1,296	126,710	106,813	19,897	1.54	5.4	407.4	322.3
0.98	1,323	128,372	108,293	20,079	1.53	5.4	407.6	321.6
1.00	1,350	129,427	109,261	20,166	1.53	5.4	407.6	321.0

Table 15-7: Antas 3 Pit Optimization Results

Revenue Factor (RF)	Gold Price (x RF) (US$/oz)	Total Rock Mined Tonnes ('000s)	Waste Mined Tonnes ('000s)	Ore Mined		Stripping Ratio (w:o)	Undiscounted Cash Flow Excluding Project CAPEX 5% (US$ M)	Discounted Cash Flow Excluding Project CAPEX 5% (US$ M)
				Tonnes ('000s)	(g/t)			
0.28	378	0	0	0	2.30	0.0	0.0	0.0
0.30	405	1	0	1	2.22	0.2	0.0	0.0
0.32	432	2	0	1	2.08	0.2	0.1	0.1
0.34	459	31	8	23	1.71	0.3	0.8	0.8
0.36	486	44	13	31	1.72	0.4	1.1	1.1
0.38	513	59	20	39	1.73	0.5	1.4	1.4
0.40	540	194	84	111	1.56	0.8	3.3	3.3
0.42	567	268	130	137	1.58	0.9	4.1	4.1
0.44	594	291	144	147	1.58	1.0	4.4	4.4
0.46	621	343	167	176	1.54	1.0	5.0	5.0
0.48	648	962	655	307	1.63	2.1	9.0	9.0
0.50	675	1,053	717	336	1.62	2.1	9.7	9.6
0.52	702	1,411	988	423	1.61	2.3	11.9	11.8
0.54	729	1,427	995	431	1.60	2.3	12.0	12.0
0.56	756	1,537	1,070	467	1.58	2.3	12.7	12.7
0.58	783	2,245	1,614	631	1.54	2.6	15.9	15.8
0.60	810	2,521	1,819	701	1.51	2.6	17.1	17.0
0.62	837	3,225	2,348	877	1.47	2.7	20.0	19.8
0.64	864	3,440	2,514	926	1.46	2.7	20.8	20.5
0.66	891	3,915	2,874	1,040	1.44	2.8	22.4	22.1
0.68	918	4,311	3,176	1,135	1.42	2.8	23.6	23.3
0.70	945	4,765	3,537	1,228	1.40	2.9	24.8	24.4
0.72	972	6,016	4,567	1,449	1.38	3.2	27.4	27.0
0.74	999	7,347	5,565	1,783	1.32	3.1	30.4	29.8
0.76	1,026	8,340	6,361	1,979	1.31	3.2	32.2	31.4
0.78	1,053	10,683	8,277	2,406	1.28	3.4	35.9	34.9
0.80	1,080	16,539	12,599	3,940	1.15	3.2	45.2	43.0
0.82	1,107	18,279	14,049	4,230	1.15	3.3	47.2	44.7
0.84	1,134	18,630	14,299	4,331	1.15	3.3	47.7	45.1
0.86	1,161	24,132	18,514	5,617	1.10	3.3	53.5	49.4
0.88	1,188	25,868	19,835	6,033	1.09	3.3	55.0	50.5
0.90	1,215	26,469	20,318	6,151	1.09	3.3	55.4	50.8
0.92	1,242	27,354	21,045	6,309	1.08	3.3	55.9	51.0
0.94	1,269	28,088	21,618	6,470	1.08	3.3	56.2	51.2
0.96	1,296	29,746	23,000	6,745	1.07	3.4	56.6	51.3
0.98	1,323	30,626	23,729	6,897	1.07	3.4	56.7	51.3
1.00	1,350	32,879	25,605	7,274	1.06	3.5	56.9	51.1

16 MINING METHODS

Santa Luz will be an off-highway truck and excavator open pit mining operation. The ore and waste rock will be drilled and blasted, loaded onto trucks, and then hauled to the crusher area, stockpile areas or waste dumps. Ancillary activities will include, but not be limited to: road maintenance; road-dust control; site dewatering; waste dump and stockpile maintenance; grade control; and TSF heavy equipment support (as needed).

16.1 Mine Design

Prior to updating the Mineral Reserves estimates, designs, and production schedule, Santa Luz Project personnel carried out a trade-off study to determine the optimal loader/truck combination using the 2018 FS mine plan. Table 16-1 shows the studied scenarios.

Table 16-1: Mining Equipment Scenarios

Scenarios	Mined	Excavator	Truck
Small Equipment	Ore/Waste	6 m^3	40-t
Medium Equipment	Ore/Waste	12 m^3	100-t
Mixed Equipment	Ore	6 m^3	40-t
	Waste	12 m^3	100-t

Results shown that 6 m^3 excavators associated with 8x4 (wheel and axle) on-highway (40-t) trucks are impractical due to number of necessary mining faces to mine in narrow pit shells such as presented by the Santa Luz deposits.

Since Santa Luz is a gold mine with a highly variable gold grade, and composed of narrow orebodies, mining selectivity is important to avoid excess ore dilution, which can increase processing costs. The mixed-fleet scenario was also discarded due to safety concerns having to do with small and large equipment working together. In order to improve ore selectivity, the decision was made to mine waste with 12 m^3 class excavators and to mine ore with 8 m^3 to 9 m^3 class excavators, both associated with 100-t trucks.

Santa Luz Project personnel developed the LOM plan based on the 2016 Santa Luz Mineral Resource estimate. The new LOM schedule was generated in the Deswik scheduling software, Deswik.Sched.

16.1.1 Block Model Statistics and Model Parameters

Two separate mine block models, C1 and Antas 3, were prepared in Deswik.Sched and used for open pit optimization. The block size used in the C1 mine plan block model is 10 m x 10 m x 10 m and the block size in Antas 3 is 9 m x 9 m x 9 m. The list of relevant variables for mine planning is presented in Table 16-2.

Table 16-2: C1 and Antas 3 Block Model Statistics

Name	Description	Unit	Value (Ranges Found in Block Models)
		C1	
Au_pond	Gold Grade	g/t	0.002—15.00
TOC	Total Organic Carbon	%	0.005—4.702
Ok_Ag	Silver Grade	ppm	0.100—6.232
Ok_Cu	Copper Grade	ppm	8.114—296.715
Ok_As	Arsenic Grade	ppm	27.771—4,239
Ok_S	Sulphur Grade	%	0.038—3.059
Density	Density	t/m^3	1.800—2.750
Class	Resource Classification	g/t	Measured (1); Indicated (2); Inferred (3)
Lito	Lithology Code		Lithology Code
		Antas 3	
Au_pond	Gold Grade	g/t	0.001–4.976
TOC	Total Organic Carbon	%	0.010–3.040
Ok_Ag	Silver Grade	ppm	0.250–1.440
Ok_Cu	Copper Grade	ppm	4.007–135.015
Ok_As	Arsenic Grade	ppm	89.553—4,761
Ok_S	Sulphur Grade	%	0.037–1.745
Density	Density	t/m^3	1.8—2.750
Class	Resource Classification	g/t	Measured (1); Indicated (2); Inferred (3)
Lito	Lithology Code		Lithology Code

16.1.2 Open Pit Optimization

Open pit optimization was conducted on the potentially mineable Mineral Resources to determine the potential pit limits, using US$1,350/oz Au. NPV Scheduler software was used for open pit optimization. Only blocks classified as Measured and Indicated Resources were included in the Mineral Reserve pit optimization process for C1 and Antes 3.

It was determined that C1 and Antes 3 can be classified as Mineral Reserves from the evaluations completed.

The open pit optimization parameters used for the Open Pit Feasibility Study are listed in Table 16-3. These parameters were used in the generation of NPV Scheduler pit shells for Mineral Reserves and may differ somewhat from the final economic parameters used in the economic model.

Table 16-3: C1 and Antas 3 Open Pit Feasibility Study Optimization Parameters

Pit Optimization Parameter	Unit	Santa Luz Values
C1 Deswik.Sched Block Size	m	5 x 5 x 5
C1 NPV Scheduler Block Size	m	10 x 10 x 10
Antas 3 Deswik.Sched Block Size	m	3 x 3 x 3
Antas 3 NPV Scheduler Block Size	m	9 x 9 x 9
C1 and Antas 3 Footwall Fresh Rock Pit Slope	°	40.7
C1 and Antas 3 Hanging Wall Fresh Rock Pit Slope	°	48.5
C1 and Antas 3 Footwall Oxide Pit Slope	°	29.2
C1 and Antas 3 Hanging Wall Oxide Pit Slope	°	27.5
Gold Price	US$/oz	1,350
Payable Gold	%	100.00
Gold Selling Cost Dacite-Leachable	US$/oz	10.00
Gold Selling Cost Dacite-High-Sulphide	US$/oz	177.00
Gold Selling Cost Carbonaceous	US$/oz	10.00
Gold Recovery Dacite-Leachable	%	84
Gold Recovery Dacite-High-Sulphide	%	84
Gold Recovery Carbonaceous	%	84
Royalty (Gross Revenue)	%	4.5
Costs		
Ore Mining Cost	US$/t	2.40
Waste Mining Cost	US$/t	2.09
Mining Cost—Incremental	US$/t/5-m bench	0.007
Process + G&A Cost Dacite-Leachable	US$/t	16.65
Process + G&A Cost Dacite-High Sulphide	US$/t	11.92
Process + G&A Cost Carbonaceous	US$/t	16.65

Note: Costs based on R$4.00:US$1.00 exchange rate.

16.1.3 Open Pit Design

The pit shells used for the design and production schedule are based on a price of US$1,161/oz Au for C1 and US$1,296/oz Au for Antes 3, totalling 24.9 Mt of Proven and Probable Mineral Reserves, and include mining of 106.7 Mt of waste. The overall waste to ore stripping ratio is 4.3:1, which includes the 2.2 Mt of stockpile treated as ore, or 4.7:1 excluding the stockpiles.

Figure 16-1 shows the optimized pits layout for the C1 and Antes 3 deposits. Figure 16-2 shows typical cross sections for the two deposits, the US$1,350 mathematical pit shells, the optimized pit designs and block model grade shells.

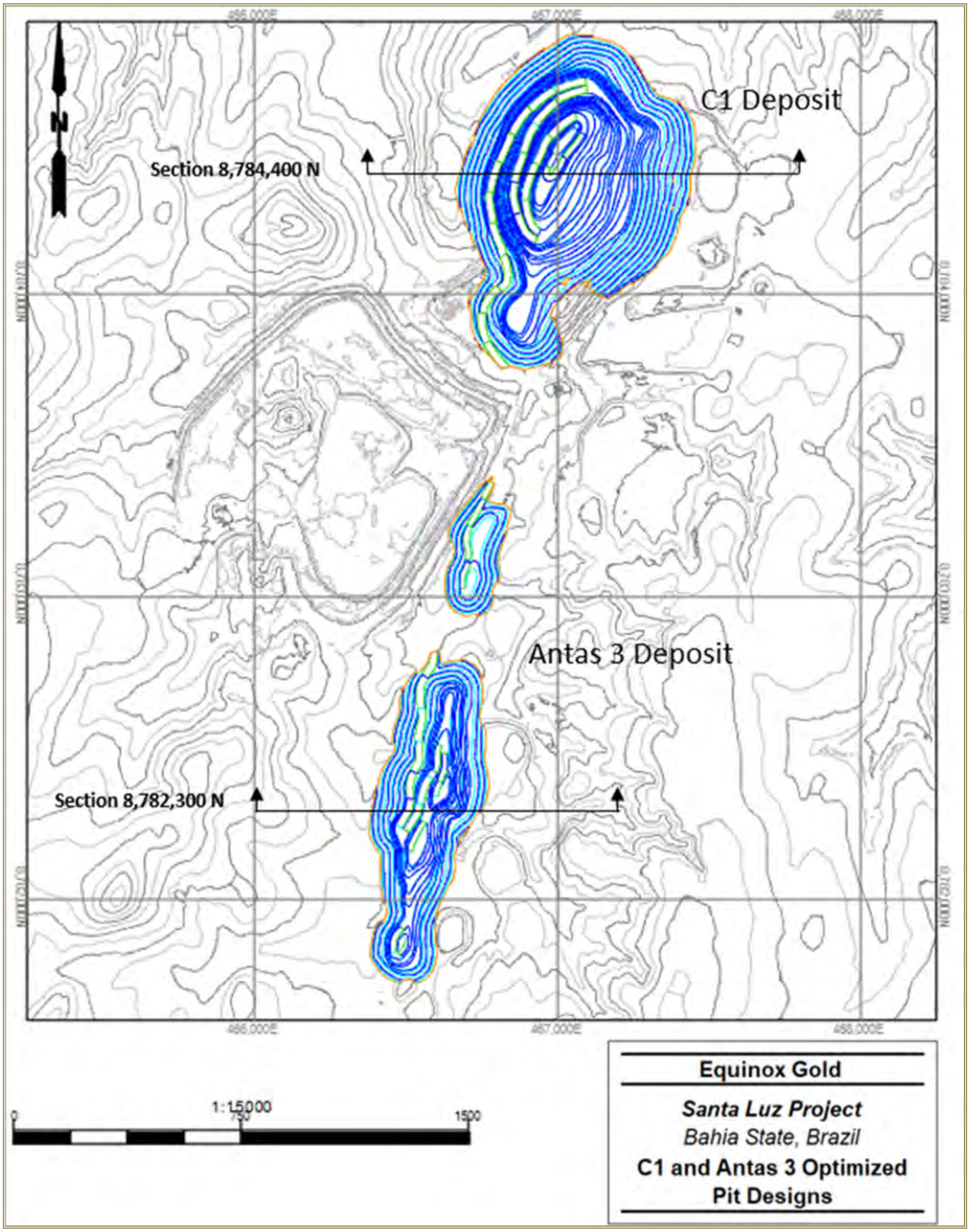


Figure 16-1: C1 and Antas 3 Optimized Pit Designs

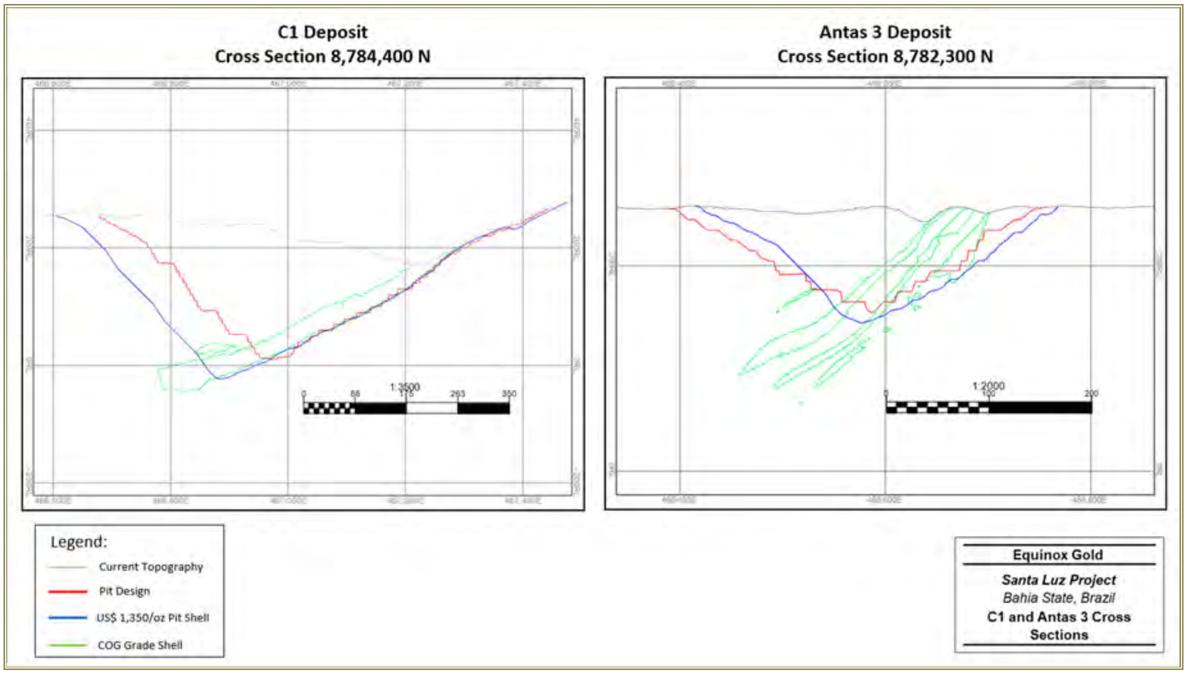


Figure 16-2: C1 and Antas 3 Cross Sections

The mineralization classification used for the Open Pit Feasibility Study Production Schedule is summarized in Table 16-4. Table 16-5 summarizes the pit design parameters used.

Table 16-4: Feasibility Study Design Pit Mineral Reserve by Classification Summary

Open Pit Feasibility Study Reserve by Classification	Tonnes ('000s)	Gold Grade (g/t Au)
C1		
Proven	16,192	1.51
Probable	672	1.56
Waste	81,275	-
Stripping Ratio	4.8	-
Antes 3		
Proven	5,386	1.05
Probable	498	0.90
Waste	25,447	-
Stripping Ratio	4.3	-
Total Santa Luz (no stockpiles)		
Proven	21,578	1.39
Probable	1,170	1.28
Total	**22,748**	**1.39**
Stockpiles		
Proven	-	-
Probable	2,191	0.86

Open Pit Feasibility Study Reserve by Classification	Tonnes ('000s)	Gold Grade (g/t Au)
Total Santa Luz (with stockpiles)		
Proven	21,578	1.39
Probable	3,361	1.01
Total	**24,939**	**1.34**
Total Santa Luz Waste	106,723	-
Stripping Ratio (Includes Stockpiles)	4.3	-

Note: Numbers may not add due to rounding.

Table 16-5: C1 and Antas 3 Optimized Open Pit Design Parameters

Pit Dimensions	Unit	C1 Pit	Antas 3 Pit
Pit Length	m	1,122	1,079
Pit Width	m	740	357
Surface Area	m^2	567,387	278,408
Maximum Pit Depth	m	232	120
Pit Bottom Elevation	mASL	5	140
Pit Exit Elevation	mASL	237	260
Average Ramp Grade	%	10	10
Ramp Width Double-Lane	m	25	25
Ramp Width Double-Lane	m	18.5	12.5
Overall Footwall Slope	degrees	31	42
Overall Hanging Wall Slope	degrees	41	32
Mining Bench Height	m	10	10

16.1.4 Geomechanics

Geotechnical domains and recommended inter-ramp pit slope angles were originally designed by Itasca Consulting Group (Itasca) in 2006.

In 2014, VogBR Recursos Hídricos e Geotecnia Ltda (VogBR) reviewed the pit slope stability of the current pit. The VogBR 2014 pit slope recommendations were considered for the final pit optimization in RPA's 2015 Technical Report.

In May 2016, Santa Luz Project personnel made a pit slope design recommendation. The geotechnical analysis included: a review of the existing data (Itasca, 2006; VogBR, 2014); structural modelling characterization; geotechnical modelling; rock mass strength; kinematic analysis; and empirical and numerical modelling analysis of overall pit slope angles.

The new pit slopes are comparable to the pit slopes recommended by VogBR (2014). The pit designs follow these recommendations and are based on a total bench height of 10 m.

A bid process is in place to perform geotechnical drill holes for a geotechnical study to determine the optimum pit slopes for the C1 pit based on the US$1,350/oz Au pit geometry. The study should be concluded by mid-2021.

It is recommended that a hydrogeology study be completed to determine the open pit dewatering parameters. Consideration should be given to the establishment of the overburden dewatering parameters needed for the design of surface diversions and drainage systems.

Ramp Design

The dimensions of the loaders, haul trucks, and excavators were evaluated to define the minimum mining width and ramp design. Ramp road grades were limited to 10% or less. Road widths were designed to be approximately 25 m wide from toe to crest for the C1 and Antes 3 deposits, with exception to single lanes used for mining the last benches.

Minimum Mining Width

A minimum mining width of 20 m was used for the pit designs. This width ensures safe loading and hauling.

16.1.5 Waste Dump Design

The rock waste dumps were designed to contain the capacity of material that will be excavated over the LOM. The parameters that were used for the rock waste dump designs are summarized in Table 16-6.

Table 16-6: Santa Luz Waste Dump Design Parameters

Waste Dump Design	Unit	West	South
Road Grade	%	6	6
Minimum Road Width	m	30	30
Catch Bench Width	m	10	10
Loose Density	t/m^3	2.09	2.09
Overall Slope	degrees	22	22
Lift Slope	degrees	32	32
Lift Height	m	10	10
Capacity	Mm^3	44.7	15.0

The design is considered adequate for the waste dumps to ensure long-term stability based on Petrus's (2013) overall slope analysis.

To confirm that the waste rock dump locations do not restrict access to potential mineralization, it is recommended that condemnation drilling be performed at the locations of the latest rock dump design extents, and a detailed trade-off study be performed to determine the optimal waste dump locations.

16.2 Mine Schedule

Mining operations will consist of: waste rock stripping and overburden removal; drilling and blasting; and loading and hauling of ore and waste. Ancillary activities will include: road maintenance; site dewatering; waste dump and stockpile maintenance; and grade control.

Equinox is undertaking a bid process to have a mining contractor for drilling, loading, and hauling, as well as for haul road maintenance. Separate bid processes of contractors, one for blasting and one for grade control drilling, are also to start. It is anticipated that the bid process and contractor selections will be completed by year-end 2020.

16.2.1 Production Schedule

The LOM production schedule was the basis for cash flow analysis.

Temperature, precipitation, topographical relief, and elevation are not expected to adversely affect mining operations at Santa Luz. The Project is located in a temperate region of Brazil, where there is moderate precipitation. Topography at the Project site is gentle, and it is at a nominal elevation of 250 mASL.

Project open pits may require dewatering. It is recommended that further investigation into dewatering requirements is pursued.

Pre-Production

The C1 deposit was mined in 2013 and 2014. Pre-stripping required for the mine restart will include 6.9 Mt of material from the C1 deposit and 1.2 Mt of stockpile movement, for a total of 8.1 Mt of material movement, to take place during the pre-production period.

LOM Production Schedule

Processing rates are estimated at approximately 7,400 t/d ore, or 2.7 Mt/a. Mining rates vary according to ore availability, as a 50%–50% blend of dacite–carbonaceous ores is essential to plant performance, and was an operational condition for the development of the LOM schedule. The ore blend can be achieved by using existing stockpiles. The LOM schedule was also developed to postpone lower-grade ores and target higher-grade ores to the process plant. A detailed mine production schedule was prepared that outlines the quantities of ore and waste rock that will be mined from the C1 and Antes 3 open pits.

The mine production schedule in Table 16-7 shows a summary of the tonnages and grades used in the Open Pit Feasibility Study on a yearly basis, with mining occurring over a period of approximately nine years, including a one-year period of mining during pre-production (note that processing activities from stockpiles extend another year beyond when the mining activities cease). The production schedule was developed on a monthly basis for pre-production and Years 1 and 2, and on a quarterly basis for the remaining mine life, with a maximum sinking rate of six benches per year. It includes four pushbacks designed in C1, two in Antes 3 South, and one in Antes 3 North.

The pushback designs include access ramps and are based on the pit shell economic sequence defined by pit optimization using a low gold price for the first pushbacks, and gradually increasing the gold price, in order to select adequate pushback widths and adequate ore supply for the processing plant. As a result, material located near the surface, with a low stripping ratio, will be mined in the first pushbacks, followed by material with higher waste-to-ore ratios, mined over the remaining LOM.

Table 16-7: Santa Luz LOM Production Schedule and Stockpile Balance

	Unit	Total	Years										
			0	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Mined													
Dacite Leach	t '000s	8,803		304	667	1,051	1,101	1,258	1,374	1,201	1,630	217	0
	Au g/t	1.32		1.60	1.81	1.28	1.45	1.42	1.09	0.95	1.33	1.61	0.00
Dacite-High-Sulphide	t '000s	1,700		0	0	0	33	99	283	777	393	113	0
	Au g/t	0.96		0.00	0.00	0.00	0.96	0.76	0.72	0.96	1.10	1.22	0.00
Carbonaceous	t '000s	12,244		522	859	1,759	1,480	1,656	2,303	1,210	2,029	427	0
	Au g/t	1.50		1.44	1.69	1.92	1.62	1.30	1.37	1.06	1.49	1.76	0.00
Total Ore Mined	t '000s	22,747		825	1,526	2,809	2,614	3,014	3,961	3,188	4,052	757	0
	Au g/t	1.39		1.50	1.75	1.68	1.54	1.33	1.23	1.00	1.39	1.64	0.00
Stockpile Balance													
Initial Stockpile (DAC)	t '000s			1,481	1,448	765	466	250	258	565	1,193	1,867	847
	Au g/t			0.78	0.73	0.60	0.56	0.59	0.69	0.75	0.86	0.86	0.63
Ore Mined (DAC)	t '000s			182	220	144	108	315	669	825	754	108	0
	Au g/t			1.30	0.66	0.55	0.64	0.64	0.74	0.90	0.83	0.82	0.00
Milled Ore (DAC)	t '000s			216	902	444	324	307	361	196	81	1,128	847
	Au g/t			1.53	0.82	0.64	0.56	0.55	0.69	0.70	0.68	1.03	0.63
Final Stockpile (DAC)	t '000s		1,481	1,448	765	466	250	258	565	1,193	1,867	847	0
	Au g/t		0.78	0.73	0.60	0.56	0.59	0.69	0.75	0.86	0.86	0.63	0.00
Initial Stockpile (CARB)	t '000s			709	893	402	811	941	1,247	2,200	2,060	2,739	1,816
	Au g/t			1.03	0.88	0.48	0.69	0.45	0.52	0.60	0.49	0.54	0.46
Ore Mined (CARB)	t '000s			249	325	737	508	548	1,180	283	715	89	0
	Au g/t			0.75	0.78	0.82	0.74	0.71	0.81	0.65	0.73	0.75	0.00
Milled Ore (CARB)	t '000s			65	816	328	378	242	227	423	36	1,012	1,816
	Au g/t			2.04	1.04	0.71	1.36	0.69	1.29	1.17	1.10	0.71	0.46
Final Stockpile (CARB)	t '000s		709	893	402	811	941	1,247	2,200	2,060	2,739	1,816	0
	Au g/t		1.03	0.88	0.48	0.69	0.45	0.52	0.60	0.49	0.54	0.46	0.00
Processed													
Dacite Leach	t '000s	10,285		338	1,350	1,350	1,350	1,350	1,350	1,350	1,350	497	0
	Au g/t	1.24		2.00	1.32	1.09	1.31	1.36	1.07	0.92	1.47	1.08	0.00
Dacite-High-Sulphide	t '000s	1,700		0	0	0	0	0	0	0	0	853	847
	Au g/t	0.96		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.20	0.72
Carbonaceous	t '000s	12,953		338	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,816
	Au g/t	1.47		1.80	1.73	1.95	1.83	1.46	1.76	1.11	1.87	1.04	0.68
Total Ore Processed	t '000s	24,938		675	2,700	2,700	2,700	2,700	2,700	2,700	2,700	2,700	2,663
	Au g/t	1.34		1.90	1.53	1.52	1.57	1.41	1.41	1.02	1.67	1.10	0.69
Recovery	%	84%		84%	84%	84%	84%	84%	84%	84%	84%	84%	84%
Recovered Gold	Au oz '000s	903		35	111	111	114	103	103	74	122	80	50

Notes: 1. Numbers may not add due to rounding.
2. Dacite ore in stockpile balance includes high-sulphide ore.
3. Initial stockpile is shown in Table 15-3.

16.2.2 Mine Equipment

Equipment will be provided by a mining contractor. An explosives contractor will provide all the blasting equipment, including all bulk agent loading trucks.

Table 16-8 presents the type and size of equipment that will be provided by the mining contractor, but a check of the equipment will be performed once the mining contractor bid process is finalized by year-end 2020.

Table 16-8: Mining Contractor Equipment Type and Size

Type	Item
Operations	
Excavator for Waste	12 m^3 bucket size
Excavator for Ore	8-9 m^3 bucket size
Haul Truck	90 t capacity
Stockpile Movement	
Excavator	5-7 m^3 bucket size
Haul Truck	40 t capacity
Support	
Front-End Loader	CAT 990 (or equivalent)
Grader	CAT 140k (or equivalent)
Bulldozer	D9 (or equivalent)
Water Truck	20 m^3 capacity
Blast Hole Drill	3.5" for pre-split; 4.5" for production

16.2.3 Labour Force

Mining operations for the Santa Luz Project will be 365 d/a, operating on a basis of three 8 h shifts per day. The mine plan requires 416 employees. Labour requirements are based on previous 2013–2014 operational requirements, and a check of the estimated labour requirement will be performed once the mining contractor bid process is finalized. Table 16-9 shows the Santa Luz and mining contractor personnel labour requirements summary for mining operations.

Table 16-9: Estimated Santa Luz and Mining Contractor Labour Force

Labour Force	Operation	Technical Services	Administration	Santa Luz Personnel Subtotal	Mining Contractor	Total
Geology & Mine Planning	-	14	-	14	-	14
Survey	-	10	-	10	-	10
Administration	-	-	4	4	-	4
Operators	48	-	-	48	326	374
Maintenance	10	-	-	10	-	10
Management	2	1	-	3	1	4
Total	**60**	**25**	**4**	**89**	**327**	**416**

16.2.4 Dewatering

The open pits may intercept the water table, and all surface water from direct rainfall and runoff within the pit limits will be absorbed by fractures into the pit (VogBR, 2014). As such, dewatering needs to be considered for the C1 and Antes 3 pits. It is recommended these dewatering quantities be finalized during the pre-production stage of the Project's development.

Water from the nearby Itapicurú River has been pumped into the C1 pit over several years to provide sufficient water for the restart of the process plant. A total of approximately 3.9 Mm^3 of water was stored in the C1 and Antas 3 pits in December 2018 and prior to initiating dewatering and transferring of the water into the TSF in October 2019. As of June 30, 2020, there were 2.59 Mm^3 of water remaining in the C1 and Antas 3 pits. Table 16-10 summarizes the C1 and Antes 3 pit topographies, water levels, approximate water volumes, and estimated time to dewater the pits at a pumping rate of 11,520 m^3/d.

Table 16-10: Estimated Remaining Raw Water Volumes within the Pits

Description	Unit	C1	Antas 3
Current Pit Bottom Elevation	mASL	171.00	234.50
Approximate Water Elevation	mASL	207.96	248.33
Remaining Water Volumes Within the Pits	m^3	2,460,644	129,497
Estimated Days to Dewater Current Volumes at 11,520 m^3/d	d	214	11

Note: As of June 30, 2020.

16.2.5 Mine Infrastructure

It is assumed that Santa Luz Project personnel will use existing office space at the site, including the existing warehouse storage. Most of the mine infrastructure is operational. However, some infrastructure must be relocated in 2023 due to proximity to the Antes 3 North pit (i.e., before mining starts in 2024).

17 RECOVERY METHODS

17.1 Summary

The original processing plant at Santa Luz was built and placed in operation in mid-2013, and was shut down in September 2014 after 14 months of operation. The original plant consisted of a comminution circuit followed by flotation, then CIL of the flotation concentrate with kerosene blanking of the organic carbon. The plant did not operate well, due to process difficulties and mechanical problems.

The new processing facility will incorporate the crushing, crushed ore storage, and SAG mill sections of the original plant. The rest of the plant will be new, except the refinery. The new facility will process a blend of carbonaceous and dacitic ore, using the following processing sequences:

- Two-stage crushing
- Two-stage grinding
- Gravity concentration and intensive cyanidation
- Pre-aeration and kerosene conditioning
- RIL
- Detoxification of leached tailings
- Containment of tailings in a TSF
- Loaded-resin washing
- Acid wash
- Zadra-type elution and electrowinning of gold from resin eluate
- Electrowinning of gold from the gravity concentrate intensive cyanidation
- Refining to generate gold-copper bars.

Plant throughput will be 2.7 Mt/a ore (7,400 t/d ore) grading 1.39 g/t Au (or 1.34 g/t with stockpiles). Projected gold recovery is 84%.

17.2 General

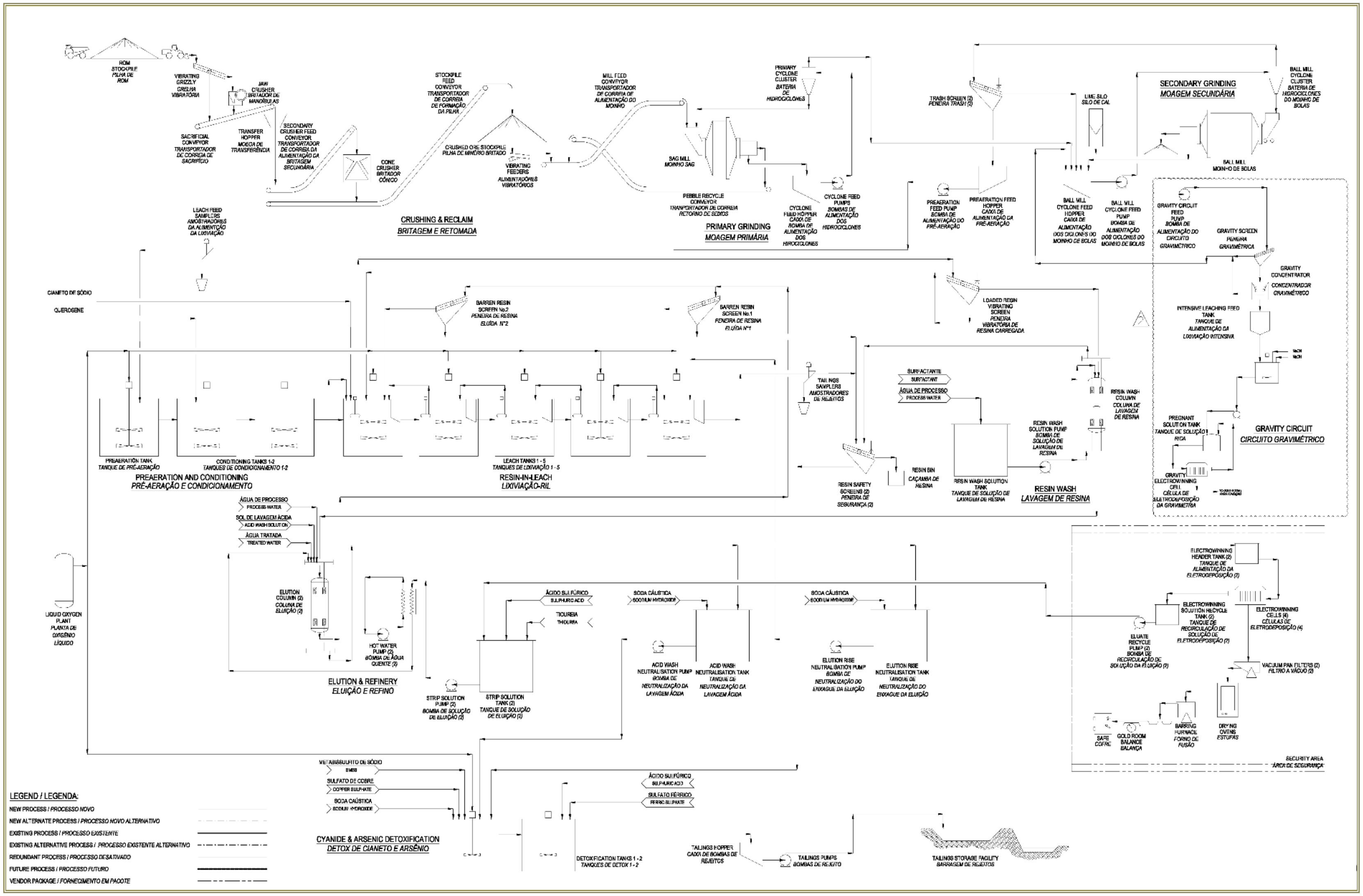
The principal parameters for the processing plant are shown in Table 17-1. A listing of principal equipment is provided in Table 17-2. A simplified flow diagram of the plant is shown in Figure 17-1, and a general arrangement drawing of the plant is shown in Figure 17-2.

Table 17-1: Principal Process Parameters

Parameter	Unit	Value
Throughput Rate		
Annual	t/a	2,700,000
Daily	t/d	7,400
Operating Period	years	9.5
Ore Grade (Average LOM)		
Gold (Including Stockpiles)	g/t	1.34
TOC	%	0.6
Arsenic	g/t	500
Gold Recovery	%	84
Gold Production (Average)	oz/a	95,000
Ore Physical Characteristics		
Work Index	kWh/t	19
Abrasion Index		0.5
Primary Crush Size	80% passing, mm	150
Secondary Crush Size	80% passing, mm	50
Primary Mill Grind Size	80% passing, µm	860
Secondary Mill Grind Size	80% passing, µm	75
Gravity		
Recovery	%	20%
Retention Times		
Conditioning	hours	6
Leaching	hours	20
Detoxification	hours	3
Employees		
Management	number	12
Operation	number	71
Maintenance	number	74
Utilities Consumption		
Power	kWh/t	42
Fresh Water (Make-Up)	m^3/t	0.40
Consumables		
Resin	m^3/t	0.00003
Grinding Balls	kg/t	1.80
Quick Lime	kg/t	1.00
Kerosene	kg/t	2.00
Sodium Cyanide	kg/t	0.75
Sodium Metabisulphite (SMBS)	kg/t	0.75
Thiourea	kg/t	0.25
Operating Cost (LOM, all ores)	US$/t	13.43

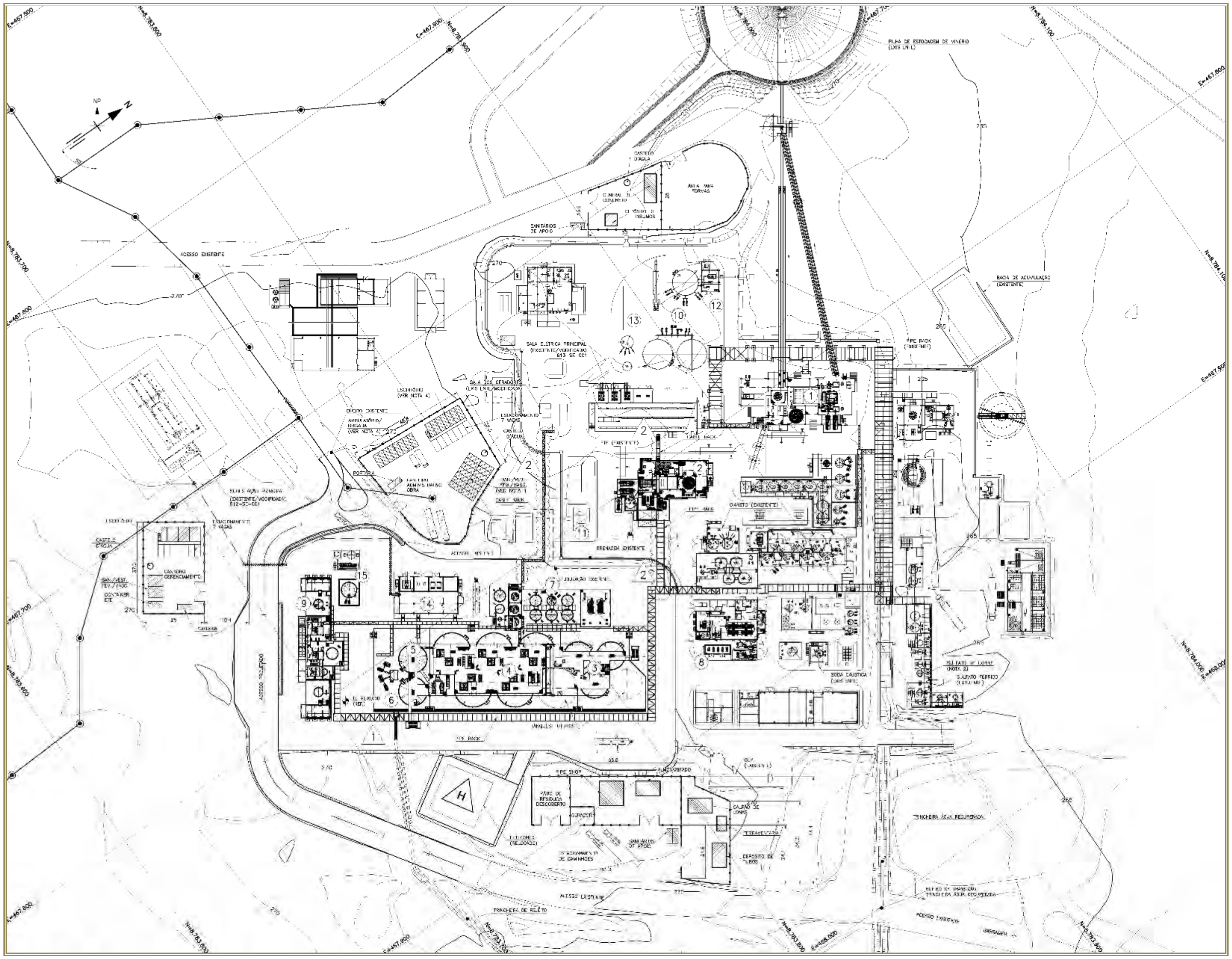
Table 17-2: Principal Process Equipment

Item	Size	(kW ea.)	Quantity	Standby
Crushing System				
Vibrating Feeder/Grizzly	Component of Lokotrack crushing unit	37	1	
Jaw Crusher	Single-toggle, 1,220 mm x 914 mm	185	1	
Cone Crusher	HP 400	300	1	
Stockpile Feed Belt	900 mm wide	185	1	
Stockpile	3,600 t live	-	1	
SAG Mill				
Feeders	Vibrating, 1,066 mm wide x 1,525 mm	4	2	
SAG Mill Feed Belt	600 mm wide	15	1	
Mill (with Trommel Screen)	5.79 m dia. x 10.36 m EGL, fixed speed	6,200	1	
Cyclone Feed Pumps	14 x 12, Warman 300-MCR	525	1	1
Cyclones	gMAX 20	-	12	4
Ball Mill				
Mill (with Trommel Screen)	Overflow, 5.0 m dia. x 8.1 m EGL	3,500	1	
Lime Silo		-	1	
Cyclone Feed Pumps	14 x 12, Warman 300-MCR	525	1	1
Cyclones	gMAX 20	-	12	4
Trash Screens	1.2 m x 2.4 m, vibrating, 420 µm openings	5	3	
Pre-aeration Feed Pumps	12 x 10	220	1	1
Gravity Circuit				
Gravity Feed Pumps	8/6AH		1	1
Scalping Screen	1.2 m x 3.05 m		1	
Gravity Concentrators	1016 mm dia. bowl	45	2	
Intensive Leach Reactor	2 t/d capacity		1	
Leaching System				
Conditioning Tanks	11.3 m dia. x 14.0 m high	260	3	
Leach Tanks	14.25 m dia. x 15.0 m high	185	5	
In-Tank Resin Retention Screens	Cylindrical, 500 µm wedge wire	22	10	1
Safety Screens	1.2 m x 2.4 m, vibrating, 500 µm openings	5	2	
Detoxification Tanks	11.4 m dia. x 11.5 m high	150	2	
Tailings Pumps, Stage 1	12 x 10	185	1	1
Tailings Pumps, Stage 2	12 x 10	300	1	1
Tailings Reclaim Pumps		150	1	1
Resin System				
Loaded Resin Screen	1.2 m wide x 3 m long	11	1	
Loaded Resin Wash Tank	3 m dia. x 3.8 m	3	1	
Elution Columns	3 m dia. x 5.8 m high	-	2	
Barren Resin Screen No. 1	1 m x 1.7 m, vibrating	5	1	
Barren Resin Screen No. 2	2 m x 2 m-dia. 45° arc, DSM-type	5	1	
Elution Solution System				
Hot-Water Heating Unit		-	2	
Heat Exchangers		-	2	
Strip Solution Tanks	4 m dia. x 5 m high	-	2	
Neutralization Tank, Acid Wash	3.5 m dia. x 4.4 m high	-	1	
Neutralization Tank, Elution Rinse	3.5 m dia. x 4.4 m high	-	1	
Refinery				
Electrowinning Header Tanks	1 m dia. x 1.5 m high	-	2	
Electrowinning Cells	5 kA	45	4	
Electrowinning Recycle Tanks	4 m dia. x 5 m high	-	2	
Smelting Furnace	Gas-fired, crucible	-	1	



Source: Ausenco

Figure 17-1: Simplified Process Flowsheet



Source: Ausenco

Figure 17-2: General Arrangement Diagram

A description of the plant and its operation is discussed below.

17.3 Crushing

The crushing section of the plant is part of the original facility. It consists of a jaw crusher followed by a cone crusher, both of which are in open circuit, with the product conveyed to a conical stockpile. Run-of-mine ore is fed by a front-end loader, from stockpiles on a pad adjoining the crushing plant, and into a hopper in front of the jaw crusher. Ore is withdrawn from the hopper by a vibrating feeder-grizzly, with grizzly oversize feeding the jaw crusher and grizzly undersize dropping to a conveyor that transports the grizzly undersize and the jaw crusher product to a transfer hopper, thence to a second conveyor that feeds a cone crusher. Prior to entering the grinding circuit, a third conveyor takes the crushed ore from the secondary crusher to a crushed ore stockpile.

17.4 Grinding

The grinding section will consist of two mills: a SAG mill and a ball mill operated in series. The SAG mill was part of the original facility; the ball mill is new. The SAG mill is unconventional in the following respects:

- The 10.4 m (34 ft) length is greater than the 5.8 m (19 ft) diameter—known as a low-aspect SAG mill
- The feed (minus 50 mm) is finer than that normal
- The product (minus 1 mm) is finer than normal.

The SAG mill, as originally installed, operated as a single-stage SAG mill to produce the final grind size. The new ball mill is a conventional overflow ball mill, and will grind the SAG mill product to the final grind size.

Ore from the stockpile is drawn by two underlying vibrating feeders onto the SAG mill feed conveyor. The SAG mill is equipped with a trommel screen that screens out plus 12 mm pebbles. The pebbles are conveyed back to the SAG mill feed conveyor and are expected to constitute approximately 5% of new feed. Trommel screen undersize will be pumped via new pumps to the new primary mill cyclones; cyclone underflow will be recycled to the SAG mill, and the cyclone overflow will pass to the ball mill discharge pump box. From the ball mill on, all process equipment is new.

The ball mill discharge pumps (one operating, one spare) will pump the combined SAG mill cyclone overflow and the ball mill discharge to the secondary mill cyclones. Cyclone underflow flows to the ball mill feed. Dry lime will be fed from a silo adjoining the ball mill pump box directly into the pump box. Cyclone overflow will flow to a trash screen. Trash screen oversize will be recycled to the ball mill pump box; trash screen undersize will flow to the plant feed sampler and thence to a pump box from which it will be pumped to the RIL circuit. This sampler will provide samples for determination of the plant head grade assays.

17.5 Gravity Circuit

The ball mill discharge pump box receives the material from the ball mill as well as SAG mill cyclone overflow, as previously described. In addition to the ball mill discharge pumps (one operating, one spare), gravity circuit feed pumps are installed in this box (one operating, one spare). The box is built to allow the gravity feed pumps to preferably pump the product fed from the SAG mill circuit.

The pumped slurry feeds the two gravity scalping screens, whose oversize is returned to the ball mill discharge pump box. The screen undersize feeds the gravity concentrators. Concentrate from the gravity concentrators feeds the intensive leach reactor, and the tailings also return to the ball mill discharge pump box, along with intensive leach reactor tailings.

The pregnant solution from the intensive leach reactor is pumped to the intensive leach pregnant solution tank, from where it will be recirculated in a dedicated electrowinning cell, given the different chemistries between this pregnant solution and the elution circuit pregnant solution.

17.6 Conditioning, Leaching, and Detoxification

Leaching will be preceded by two sequential stages of conditioning using three tanks in series. Pre-aeration will occur in the first conditioning tank and kerosene-conditioning in the subsequent two tanks. Oxygen will be added in the pre-aeration tank. It has been found that pre-aeration minimizes surface coating of the resin by a kerosene ore film. In the kerosene conditioning step, kerosene will be added to blank organic carbon in the ore to minimize its propensity for soluble gold adsorption (preg-robbing).

Following conditioning, sodium cyanide solution will be added to the ground slurry in the first of the RIL tanks. The RIL circuit will consist of five stirred tanks in series. Oxygen from a liquid oxygen plant will be added to all but the last RIL tank, to provide oxygen for cyanide dissolution of the gold while minimizing froth formation on the surface of the tanks. The RIL tanks will contain approximately 16 mL of Dow Ambersep XZ-91419 resin per litre of slurry. The resin will be in the form of spherical beads approximately 1 mm in diameter; the ground ore particles will be mostly less than one-tenth the size of the resin at approximately 90% <0.1 mm in diameter. Each tank will be fitted with mechanically swept screens to retain the resin while allowing the ground ore slurry to flow from tank to tank. Each tank will be fitted with pumps to intermittently pump resin-containing slurry counter-current to the normal slurry flow. Eluted resin will be periodically added to the first and last RIL tanks, and loaded resin will be periodically removed from the first RIL tank.

Leached slurry from the last RIL tank will flow to a tailings slurry sampler. This sampler will provide samples for determination of the plant tailings grade assay. Discharge from the tailings sampler will pass to a safety screen to recover any partially abraded resin. The safety screen underflow will flow to a tailings detoxification circuit.

The detoxification circuit will consist of two tanks in series. Cyanide detoxification will occur in the first tank, and arsenic precipitation in the second tank. Oxygen, sodium metabisulphite, copper sulphate, and sodium hydroxide will be added to the cyanide detoxification tank to oxidize the residual cyanide in the leached slurry. Ferric sulphate will be added to the arsenic precipitation tank to precipitate any arsenic in the leached slurry, though testwork indicates that the amount of soluble arsenic will be so low that it is unlikely to be required.

Slurry from the detoxification tanks will flow to the tailings pump box and the detoxified tailings will be pumped to the TSF.

17.7 Elution, Electrowinning, and Refining

Loaded resin in the first RIL tank will be periodically pumped to a screen to separate the resin from the ore slurry. The slurry in the screen underflow will flow back to the first RIL tank. Loaded resin will flow to a resin wash tank to remove the kerosene-ore coating on the resin. The resin will be washed and eluted on a batch basis. The resin wash tank will be fitted with a mixer; wash solution containing surfactant will be added to the base of the resin wash tank and will flow upwards to wash the coating from the resin. The wash solution containing the suspended solids removed from the resin will flow to the last RIL tank.

Washed resin will be periodically transferred from the wash tank to one of two combined acid wash and elution tanks operated in parallel. In the acid wash circuit, a sulphuric acid solution at 60°C will be recirculated in the column. The base metals removed during the acid wash phase will be displaced by a rinse stage with wash water, also at 60°C. After completion of the acid wash and rinse cycle, an elution solution consisting of a mixture of sulphuric acid and thiourea heated to 60°C will be pumped to the top of the elution column; it will flow downwards through the column to remove adsorbed gold and other adsorbed metals from the resin (primarily copper and iron). A hot water-heated heat exchanger will be used to heat the elution solution. Provision is made in the design for the first 0.6 BV of elution solution will be diverted to a copper precipitation tank to improve doré bullion grade; the tank will be installed in the future.

Elution will take place in closed circuit with four electrowinning cells, with two electrowinning header tanks ahead of the cells and two recycle solution tanks following the cells. Following elution, the eluted resin will be washed with caustic solution to neutralize any remaining acid, and will then be recycled to the RIL tanks via two barren resin screens, one over the first RIL tank and one over the last RIL tank. Barren resin screen oversize will gravitate to the RIL tanks below the screens. Underflow from the barren resin screens will pass to the resin safety screen located between the RIL and detoxification tanks and thence to the detoxification tanks.

The loaded eluate solution containing gold and other metals will flow to two electrowinning header tanks in parallel, thence to four electrowinning cells. The cells' cathodes will be stainless steel-wool. Gold and other metals in the eluate will be electrodeposited on the cathodes. Periodically the electrodeposited metals will be washed from the cathodes and the cathode sludge collected, filtered, and dried. The dried cathode sludge will be mixed with oxidizers and flux, and the mixture smelted to produce doré metal and slag. The eluate is expected to contain 10 to 15 times as much copper as gold; accordingly, the metal produced will consist of an alloy of copper and gold.

There will be a separate electrowinning cell for the pregnant solution from the gravity concentrate intensive leach reactor (ILR). The ILR will operate in batches in accordance with production of gravity concentrate.

The copper-gold alloy will be shipped to a custom refinery, where the slag will be processed to recover any coarse metal, and will then be recycled to the plant feed.

17.8 Dacitic-High-Sulphide Ore Processing

Dacitic-high-sulphide ore from the Antas 3 deposit will be processed in the last year of operation. Principal processing parameters relating to the planned dacitic-high-sulphide ore processing facility are provided in Table 17-3, and a listing of principal process equipment is shown in Table 17-4.

The dacitic-high-sulphide ore will be crushed and ground in the same crushing and grinding facilities as those used for regular ore. The ground ore will be floated in the flotation circuit that was constructed for the prior operation, which will be refurbished. Flotation concentrate will be filtered in a new filtering plant and the filter cake stored in a new concentrate storage building with facilities for loading the concentrate into super sacks. The concentrate will be shipped off site, either to copper smelters or to other plants offering toll processing of the concentrate.

Table 17-3: Dacitic-High-Sulphide Ore, Principal Process Parameters

Parameter	Unit	Value
Throughput Rate		
Annual	t/a	2,251,000
Daily (360 days)	t/d	6,250
Ore Grade (Average LOM)		
Gold	g/t	0.97
TOC	%	0.05
Arsenic	%	0.2
Recovery		
Gold	%	84
Arsenic	%	90
Concentrate Grade		
Gold	g/t	80
Arsenic	%	16
Concentrate Production	t/a	30,375
Gold Production	oz/a	58,696
Ore Physical Characteristics	Same as that for RIL	
Flotation Retention Times		
Roughing	minutes	20
Cleaning	minutes	20
Concentrate Regrind Size	80% passing, µm	45
Concentrate Dewatering		
Thickening Area (High Rate)	$(t/h)/m^2$	0.2
Filtration Area (Pressure)	$(t/h)/m^2$	0.2
Employees		
Management	number	12
Operation	number	71
Maintenance	number	74

Parameter	Unit	Value
Utilities Consumption		
Power	kWh/t	42
Fresh Water (Make-Up)	m^3/t	0.40
Consumables		
Resin	m^3/t	0.00003
Grinding Balls	kg/t	1.80
Sulphuric Acid	kg/t	0.5
Copper Sulphate	kg/t	0.1
Collectors	kg/t	0.15
Frother	kg/t	0.05
Flocculent	kg/t	0.01
Operating Cost	US$/t	9.17
Dacitic-High Sulphide Plant		
Refurbishment	US$ M	0.57
New Equipment/Facilities	US$ M	1.91
Engineering & Indirects	US$ M	0.29
15% Contingency	US$ M	0.41
Total Capital Cost	US$ M	3.18
Projected Start-Up	Date	Year 10

Table 17-4: Dacitic-High-Sulphide Ore, Principal Process Equipment

Item	Size	Existing (E) or New (N)	Power (kW ea.)	Oper.	Standby
Primary Comminution System	Same as that for RIL	E		1	
Flotation System					
Roughing	Metso 50 m^3 cells	E	11	6	
Cleaning	Metso 50 m^3 cells	E	11	2	
Regrind Mill					
Mill	3.05 m dia. x 3.66 m long	E	485	1	
Cyclones	250 mm	E		8	2
Cyclone Feed Pumps		E	95	1	1
Concentrate Dewatering					
Thickener	10 m diameter	E		1	
Filter Feed Storage Tank	8 m dia. x 8 m high	N	75	1	
Filter	40 m^2 pressure filter	N	75	1	
Concentrate Storage and Loadout					
Storage Building	400 m^2	N		1	
Super-Sack Loading	Front-end loader & hopper	N		1	

18 PROJECT INFRASTRUCTURE

The listing of the current (June 2020) Santa Luz site infrastructure is provided in Table 18-1and the infrastructure facilities project is shown on the site map provided in Figure 18-1.

Table 18-1: Santa Luz Infrastructure Summary

Item	Type and Size
Access Road	Existing two-lane gravel road, 35 km long from Santa Luz, which is paved in areas adjoining communities to minimize dust.
Employee Transport	Employees will be bussed from Santa Luz.
Process Water System	Existing system for water pumped from local river (Rio Itapicurú) during rainy season will be stored in the leach TSF, which will be converted to a WSF, the Antas 3 pit, and the flotation TSF. Existing wells will supply water for the resin elution operation.
Potable Water System	Existing tank with 10 m^3 volume will be used to store potable water for human consumption. The water will be provided by a contract with EMBASA—Public agency of Bahia State
Power Supply	Existing 138 kV power line, capable of transmitting up to 15 MW, and linked to the grid and Coelba power plant; mine-site substation will be expanded.
Fuel Supply and Storage	Existing steel-frame open shed of ~100 m^2 for 5,000 L diesel tanker trailer. Fuel storage for mine vehicles will be provided by the mining contractor. Storage will be expanded.
Ancillary Systems	
Communication	Existing system linked to national network for voice and data communication.
Security	Existing gatehouse at site entry staffed by contracted security service; existing site fencing with additional fencing in certain areas.
Medical	Existing staffed clinic; ambulance on site; helicopter pad at plant.
Waste	Compostable refuse is composted; non-composting refuse is buried on site; recyclable material is transported off site.
Sewage	Existing compact sewage treatment systems (anaerobic system) will be used to treat all sewage.
Buildings	
Administrative Office	Existing.
Cafeteria	Existing.
Laboratory & Plant Office	Existing.
Workshop	Existing steel building of ~540 m^2 for mechanical and electrical maintenance. Workshop structure will be expanded.
Explosive Magazine	Existing fenced area of ~5,400 m^2 prepared for the installation of steel buildings. Explosive Magazine will be provided by a contractor.
Community Relocation	New village, Nova Esperança, of 97 houses (located 470,620.30E and 878,6022.275 N).

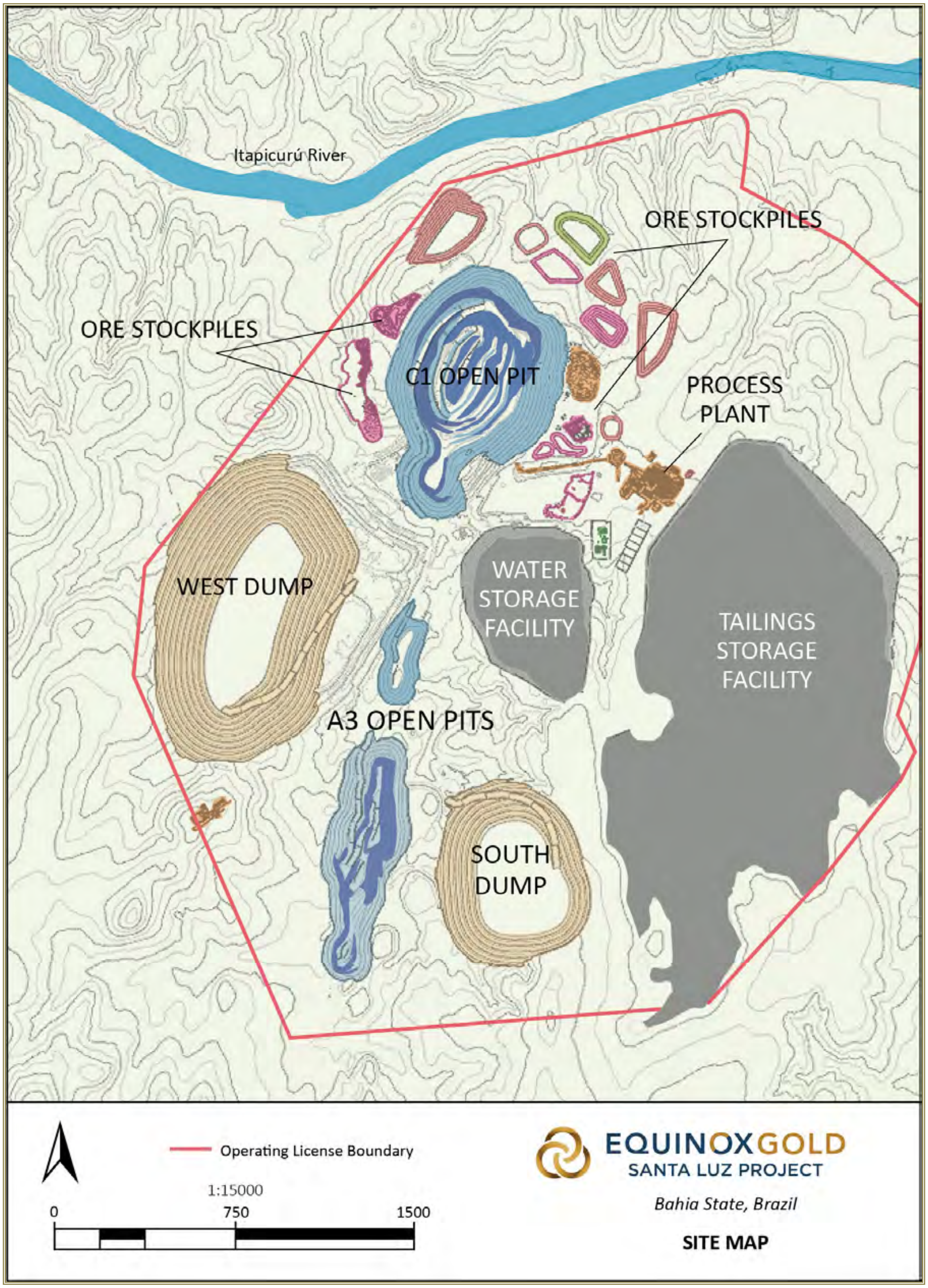


Figure 18-1: Site Infrastructure

All employees and contractors will live in Santa Luz or its vicinity and will be bussed to the site daily. A municipal all-season two-lane gravel road provides access to the mine from the town of Santa Luz.

18.1 Water Supply

A water balance for the Project is shown in Table 18-3. The water is required to provide make-up water for the processing plant and for dust suppression of the mine roads. All raw water for the operation will be pumped from the Rio Itapicurú (which runs west–east on the northern boundary of the property) at a maximum rate of 500 m^3/h, and a maximum withdraw amount of 1.17 Mm^3/a. The amount of water in the Rio Itapicurú is strongly seasonal and can be erratic. Due to environmental obligations, water can be collected only in January, and from March to August. In the other months of the year, there will be no withdraw allowed from the river. Because of the seasonality and erratic fluctuation of water availability in the Rio Itapicurú, it is necessary to build up an inventory of water to last through the dry season.

The available fresh water as of June 30, 2020 is stored in the areas as shown in Table 18-2.

Table 18-2: Santa Luz Fresh Water Storage

Location	Volume (m^3)
Antas 3 Open Pit	129,497
C1 Open Pit	2,460,644
TSF	1,228,342
WSF	81,151
Total	**3,899,634**

There are also fourteen additional deep wells, with intake capacities ranging from 152 m^3/d to 1,471 m^3/d, as approved by the local environmental agency, Instituto do Meio Ambiente e Recursos Hidricos (INEMA). Well water will be treated and used for resin elution, amounting to approximately 27,000 m^3/a. A water balance with the different consumptions is shown in Table 18-3.

Santa Luz currently has a total water storage of approximately 3.9 Mm^3, which allows a safe operation from a water supply perspective, considering the expected annual process water consumption of approximately 1 Mm^3/a. Potable water will be supplied by a contract with Empresa Baiana de Águas e Saneamento S.A. (EMBASA)—Bahia' state water utility—amounting to approximately 30 m^3/wk.

Table 18-3: Santa Luz Water Balance Summary

Input		Output	
Source	**(Mm^3/a)**	**Source**	**(Mm^3/a)**
Water Storage			
Itapicurú River	1.17	Road watering	0.10
Precipitation	0.14	Evaporation	0.37
		Process make-up	0.84
Total	**1.31**	**Total**	**1.31**
Process Plant			
Process Make-Up	0.84	Evaporation	0.06
Moisture in Ore	0.09	Water in tailings slurry	6.24
Wells (Resin Elution Water)	0.03		
TSF Decant Water	5.34		
Total	**6.30**	**Total**	**6.30**
Tailings Storage Facility			
Water in Tailings Slurry	6.24	Entrained in tailings	1.00
Precipitation	0.55	Evaporation	0.45
		TSF decant water	5.34
Total	**6.79**	**Total**	**6.79**

Notes: *Comprises water stored in the Raw Water Pond, Antas 3 Pit, and TSF.

18.2 Power

Power for the Project will be provided by the existing power line connected to the Brazilian grid. The existing substation will be expanded to approximately 16 MW to accommodate the additional power draw of the modified and expanded plant. The system has the capacity to provide 17.8 MW. The Santa Luz Project also has contract No. 5040699/CUSD with Companhia de Eletricidade do Estado da Bahia (COELBA), Bahia's state power utility, which covers demand for 10 MW in non-peak hours and 0.5 MW during peak hours.

18.3 Ancillary Infrastructure

Ancillary infrastructure requirements for the Project are in place, including communication, security, medical, waste, and sewage systems. Most of the Project buildings are already in place, including administrative offices, cafeteria, laboratory, plant office, and a workshop. Areas for explosive magazine and fuel storage are available and will be constructed by contractor.

18.4 Labour Force

The Project will employ approximately 749 people with approximately a 40:60 split of employees and contractors. A listing of projected personnel for the operation, according to the new production schedule, is shown in Table 18-4.

Table 18-4: Santa Luz LOM Projected Personnel

Department	Employees	Contractors	Total
Mining			
Management	3	1	4
Operators	48	316	364
Maintenance	10	-	10
Support (Geologists, Engineers, Surveyors)	28	10	38
Subtotal	**89**	**327**	**416**
Processing			
Management	12	-	12
Operations	53	18	71
Maintenance	64	10	74
Subtotal	**129**	**28**	**157**
Administration			
Management	2	-	2
Administration	27	-	27
Laboratory	19	-	19
Human Resources	5	-	5
Controller	6	-	6
Community Relations	1	-	1
Safety, Health, Environmental Coordination	17	-	17
Financial Planning and Analysis	2	-	2
Food Service	-	16	16
Bus Transportation	-	20	20
Site Security and Maintenance	-	44	44
Others	-	17	17
Subtotal	**79**	**97**	**176**
Total	**297**	**452**	**749**

18.5 Tailings Storage Facility

There are currently two TSFs at the Project, one for flotation tailings and one for leached tailings, from the prior operations. The existing flotation tailings TSF will be used for storing the tailings for the new operation. The new process plant design will eliminate the flotation process and will be based on whole ore leaching, which means that all the tailings generated from the plant will have been in contact with leaching reagents, and will be best stored in a geomembrane-lined tailings impoundment. The current flotation TSF has already been modified, with the installation of a geomembrane to accept the leached tailings, and the prior leach TSF will be expanded and used only for water storage.

The existing TSFs and their locations with respect to the process plant and other facilities are shown in Figure 18-2.

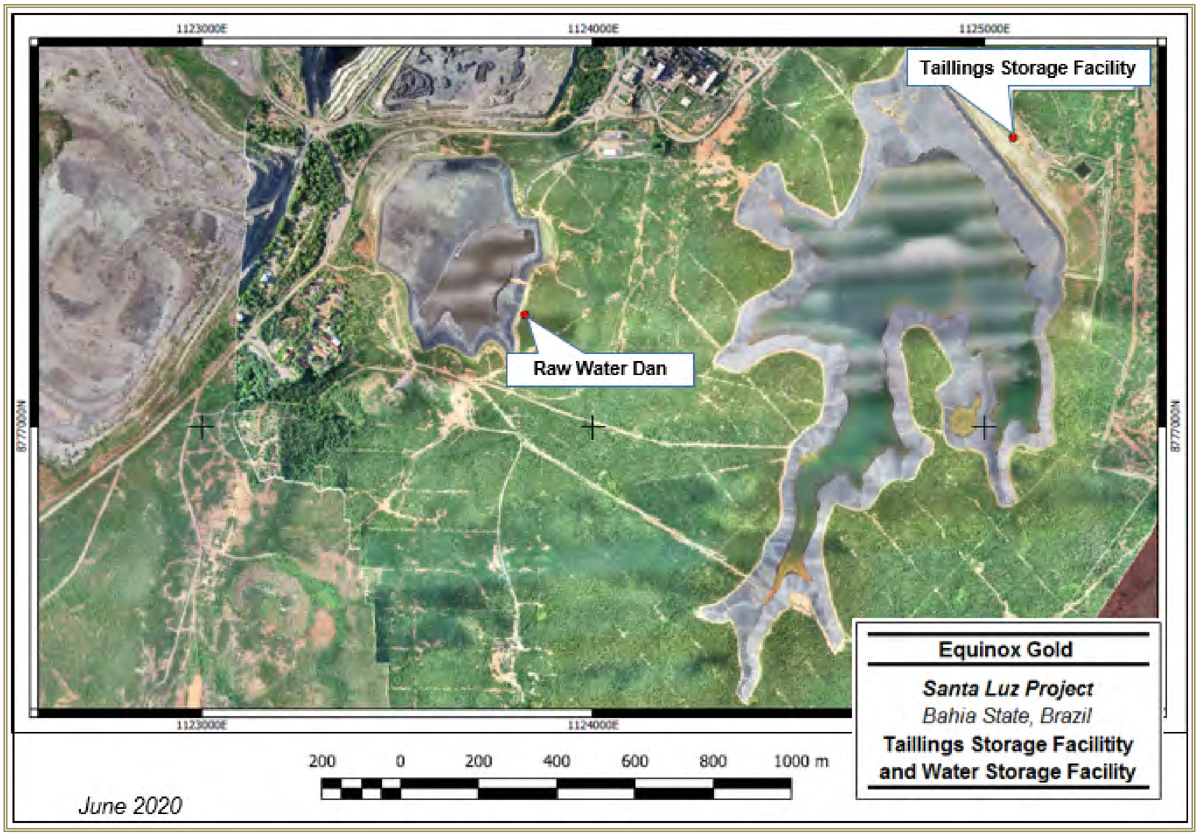


Figure 18-2: Tailings and Water Storage Facilities

The existing flotation TSF is in a broad, shallow valley adjoining the processing plant. A 750 m long and up to 25 m wide dam made of compacted clay and waste rock closes the valley to impound the tailings. The upstream slope of the dam was built of low permeability clayey material sourced from local borrow areas. The downstream zone was constructed of waste rock. An inclined sand-and-gravel chimney drain was placed between the clay and waste rock materials and is linked to a horizontal filter blanket below the base of the dam that leads to the downstream toe.

The elevation of the dam crest is currently 260 mASL. The dam's rockfill zone, which is the largest zone, will be built up to its maximum elevation of 273 mASL during the Project construction phase, using waste rock stripped from the C1 open pit. The clay zone and chimney drain will be periodically raised in stages throughout the LOM, first to a crest elevation of 266 mASL, then to 270 mASL, and finally to 273 mASL. The dam will be raised using the downstream construction method and will include the same pattern of construction as the original dam, with extending the clayey material and a gravel sand chimney drain on the upstream side and waste rock on the downstream side.

The flotation TSF, as previously operated, did not include a geomembrane liner. For the new operation, the TSF was lined with a 1.5 mm thick, low-density polyethylene (LDPE) geomembrane that was installed by 2018. Additional liner will be subsequently installed and extended in the basin, and on the upstream face of the embankment in parallel with the periodic raising of the dam.

The TSF includes an overflow system (spillway) on the right (east) abutment. The base (invert) of the spillway is currently at 259 mASL and is 3 m wide along the base; the spillway is excavated into bedrock up to a depth of 5 m, with 1H:1V side slopes. This portion of the spillway extends for approximately 100 m prior to changing into a concrete-lined spillway measuring 3 m wide by 1 m high that extends approximately 300 m and discharges into a collection pond downstream of the toe of the dam. To allow for storm events, the TSF is designed to maintain a minimum freeboard of 2.5 m between the dam crest and maximum water level in the basin. The capacity of the TSF at the various crest elevations, including 2.5 m of freeboard, is shown in Table 18-5.

Table 18-5: Tailings Storage Facility Parameters

Parameter	Unit	Values				
Planned Construction	year	Existing	Year 2	Year 5	Year 7	Final
Crest Elevation	mASL	260	266	270	273	273
Available Capacity (cumulative)						
Volume[a]	Mm^3	1.3	9.1	13.9	18.7	23.9
Tailings solids[b]	Mt	1.9	12.7	19.5	24.9	33.4
Operating time	years	1.5	4.5	7.0	9.5	-

Notes: [a] Volume of 1.345 Mm^3 of flotation tailings from prior operation is not included.
[b] Based on a tailings dry density of 1.4 t/m^3.

Placement of tailings in the TSF will follow past practice, with the slurry spigotted from tailings distribution pipes located on the perimeter of the TSF basin, and from the crest of the dam. A pump barge in the TSF will recover decant water and pump it to the plant water tank.

18.6 Water Storage Facility (Raw Water Dam)

The existing leach TSF is located adjacent to the flotation TSF and the processing plant. The leach TSF will be converted to a WSF to store the currently impounded water in the C1 and Antas 3 pits. The leach TSF was historically used for tailings deposition during the 2013–2014 period but only minor amounts of tailings were deposited into the facility.

Like the flotation TSF, the leach TSF embankment is constructed of compacted clay and waste rock, measuring approximately 500 m long and up to 16 m high, with the upstream zone built of clayey material and the downstream zone of waste rock. An inclined sand-and-gravel chimney drain was placed between the clay and waste rock materials and is linked to a horizontal filter blanket below the base of the dam that leads to the downstream toe.

A second dam, the saddle dam, is located on the opposite side of the facility from the main dam. It is smaller than the main dam and measures approximately 115 m long and up to 11 m high. It is built in a similar style to the main dam. The saddle dam was built to enclose the basin for tailings storage.

The elevation of the dam crests is currently 260 mASL. The dams will be raised to a crest elevation of 266 mASL to provide additional water storage capacity from the current 0.5 Mm^3 to a total of 2.0 Mm^3. A second expansion is planned to raise the dams to a final crest elevation of 273 mASL to provide a total storage capacity of 3.0 Mm^3. Construction of the dams will use the same downstream construction method as the original dams with extending the clayey material and the sand-and-gravel chimney drain on the upstream side and waste rock on the downstream side.

The leach TSF as previously operated included a 1.5 mm thick high-density polyethylene (HDPE) geomembrane liner installed throughout the basin and along the upstream slopes of the dams. Additional liner will be installed subsequently, and extended in the basin and on the dams in parallel with the dam raisings.

The leach TSF dam has an overflow system (spillway) along the eastern side of the basin and adjacent to the right abutment of the saddle dam. To allow for storm events, the facility is designed to maintain a minimum freeboard of 1.5 m between the dam crest and the maximum water level in the basin. For the future dam raises, the spillway will be rebuilt, and will measure 1.0 m wide by 1.0 m deep to maintain overflow capability from the basin.

18.7 Infrastructure Relocation

The administrative buildings, such as offices and mess hall, must be moved from their current position to allow for the development of the Antas 3—North pit. This change is planned to happen in 2022.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

Gold is the principal commodity produced at Santa Luz and is freely traded, at prices that are widely known, so prospects for sale of any production are virtually assured. Prices are quoted in US dollars per troy ounce unless specified otherwise. A gold price of US$1,350/oz and exchange rate of R$5.00:US$1.00 have been used for the LOM production schedule and capital and operating cost estimates.

Equinox is of the opinion that the R$5.00:US$1.00 exchange rate and gold price of US$1,350/oz for the Mineral Reserves and a gold price of US$1,500/oz and R$5.00:US$1.00 exchange rate for the cash flow are reasonable assumptions.

19.2 Contracts

Santa Luz will use contractors for loading, hauling, drilling and road maintenance, like the contracts that were in place during previous operations. The explosive supply will include blasting services at the mine. There will be supply contracts for electricity, fuel, spare parts, steel balls, and processing reagents following similar supplier structure as in the previous operation.

Santa Luz will prioritize sourcing goods and services through local suppliers, to contribute to the sustainable economic development of local communities.

In Equinox's opinion, contractor costs are within reasonable ranges.

Estimates of major consumables for Santa Luz are listed in Table 19-1.

Table 19-1: Primary Consumables

Description	Consumption Unit	Estimated Monthly Consumption	Unit	Estimated Unit Costs	Monthly Spend[a] (US$ '000s)
Diesel	L/month	1,154,000	US$/L	0.57	660
Kerosene	L/month	375,000	US$/L	0.81	304
Explosives	t/month	271	US$/kg	0.83	225
Resin	m³/month	6.8	US$/m³	17,243	117
Cyanide (33% solution)	t/month	511	US$/t	640	327
Grinding Media	t/month	405	US$/t	662	268
Electric Power	MWh/month	9450	US$/kWh	0.06	521

Note: [a] Median value for the LOM operation considering R$5.0:US$1.00.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Project Permitting

According to the Brazilian Federal Resolution CONAMA No. 237/97, the environmental licensing for a mining project is handled by the state in which the Project resides. The environmental licensing process in Bahia State is under the responsibility of INEMA (the Environmental and Water Resources Institute) and CEPRAM (Conselho Estadual de Meio Ambiente, or Environment State Council). INEMA analyzes and technically approves environmental projects proposed for mining activities as follows:

- Preliminary Licence (Licença Prévia—LP) is required for the preliminary phase of the Project planning or activity, approving its location and conception, attesting to environmental viability, and establishing the basic requirements to be fulfilled in the next phases of the Project implementation.
- Implementation Licence (Licença de Instalação—LI) authorizes the installation of the Project or activity according to the specifications contained in the plans, approved programs, and designs, including environmental constraints and control measures.
- Operation Licence (Licença de Operação—LO) authorizes the Project's operation, after verification of effective fulfillment of the conditions, which appear on the previous two licences, with the environmental constraints and control measures determined for the operation.

The Santa Luz Project maintains operational licences with several conditions that comprise monitoring and mitigation actions to compensate all environmental and social impacts, such as monitoring water quality, noise levels, and particulate matter. In the years since the shutdown of the Project, the Project has maintained compliance with the general conditions established by INEMA, as demonstrated by several environmental reports.

Equinox requested the renewal of its operating licences following the requirements of Brazilian law, where the renewal application must be submitted at least 120 days before the expiration date. This means its permits are valid until the publication of the license renewed.

Since 2018, Equinox obtained a fauna management licence and a new water permit to its operating licence considering the future operational process, which includes constructing the processing plant and the TSF expansion.

A review of the status of current permitting prepared by RPA in 2018 has been performed and the results are presented in Table 20-1.

Table 20-1: Santa Luz Permitting Status

Permit	Process Number	Issue Date	Expiration Date
Operation Licence—Mine, Plant and Tailings Dam	Portaria No. 14.666	22/08/2017	22/08/2020[a]
Operation Licence—Mine (CBPM Area)	Portaria No. 14.688	26/08/2017	26/08/2020[b]
Alteration Licence—Change Dam and Plant	Portaria No. 14.867	20/08/2017	22/08/2020[c]
Water Permit—Pumping 4 Groundwater Wells	Portaria No. 17.450/2018	08/12/2018	08/12/2022[d]
Water Permit—Pumping 6 groundwater Wells	Portaria No. 17.444/2018	07/12/2018	09/08/2022[e]
Freshwater Pumping Permit	Portaria No. 19.971/2020	22/01/2020	22/01/2024[f]
Fauna Management	Portaria No. 18.297/2019	29/04/2019	22/08/2020[g]
Water Permit—Pumping 4 Groundwater Wells	Portaria No. 20.323/2020	31/03/2020	02/12/2023
Renewal of Operation Licence—Portaria No. 14.666	2017.001.000514/INEMA/LIC-00514	28/02/2020	-
Renewal of Operation Licence—Portaria No. 14.688	2017.001.001968/INEMA/LIC-01968	17/03/2020	-
Renewal of Alteration Licence—Change Dam and Plant—Portaria No. 14687	2017.001.002109/INEMA/LIC-02109	07/04/2020	-
Renewal of Fauna Management—Portaria 18.297/2019	2018.001.006989/INEMA/LIC-06989	02/04/2020	-

Notes: [a] Renewal requested on 28/02/2020.
[b] Renewal requested on 17/03/2020
[c] Renewal requested on 07/04/2020
[d] Portaria 17.450/2018 replace previous Portaria No. 6563
[e] Portaria 17.444/2018 replace previous Portaria No. 6269
[f] Portaria 19.971/2020 replace previous Portaria No. 7573 and 7574
[g] Renewal requested on 02/04/2020

20.2 Environmental Impacts and Mitigation Actions

The control actions for mitigation, compensation, and monitoring of environmental impacts are encompassed in the following plans and programs.

- Surface and Underground Water Monitoring Program: refers to the monitoring of Itapicurú River by means of sampling points up- and downstream of the Project, considering the main parameters of CONAMA Resolution 357/2005. Water from points where the pits intercept groundwater and/or water from pumping stations that lower the water table are also sampled.
- Noise Monitoring Program: noise levels generated in the mine and industrial installations are monitored at the closest reception points.
- Hydrogeological Monitoring Program: the final pits will surpass the water table elevation. Although the groundwater is not being used locally, the alterations will be determined by sampling monitoring wells. The same wells will be used to sample groundwater to provide information for the Water Quality Program.
- Compensation actions relative to the Sistema Nacional de Unidades de Conservação (SNUC—National System of Protected Areas) law 9985/2000: this law considers the environmental compensation causing an impact of at least 0.5% of the total investment value determined in the licensing process. In the case of mining enterprises, the percentage of compensation is typically approximately 1% to 2% of investment value.

- Archaeological Rescue Program: considers the rescue of archaeological sites identified in the directly affected area, according to their approved Rescue Plan.
- Waste Management Program: consists of defining the types and quantities of waste generated by the Project and its final destination—reuse, recycling, treatment, and destination following the legal requirements to not generate environmental liabilities.
- Social Communication Program: aims to establish a permanent communication mechanism between the company and local communities, from the initial stages of planning and implementation until the mine is in operation, with the objective of identifying problems or conflicts and to keep the communities informed of the actions taken by the company in relation to social and environmental aspects.
- Environmental Education Program: involves the contribution of the company to the environmental understanding of the communities in its area of influence, and for environmental training of staff, from implementation to operation stages.
- Reclamation Plan: establishes the actions to be implemented from the early stages of Project implementation, aimed at restoring the impacted areas to allow use of the area after mining closure.
- Erosion and Water Courses Sanding Prevention Program: monitors the erosion and sanding processes during implementation, operation, and Project closure. It aims to detect the events as early as possible and to stabilize them before major environmental impacts take place.
- Control of Water and Energy Consumption Program: aims to monitor the usage of water and energy during Project operation, and to contribute to development of actions that will reduce the consumption.

Santa Luz site personnel develop and submit to INEMA several monitoring reports on a regular basis to demonstrate compliance with the relevant conditions of the licences.

There are no specific requirements for wastewater discharges, as the beneficiation (processing) plant operates in a closed circuit. Water supply is provided by fourteen groundwater wells, and by a freshwater withdraw from the Itapicurú River. With the new operational scheme, the current leach TSF will be transformed into a WSF (Raw Water Dam), improving the water storage system. More details on the water supply and balance are presented later in this section.

20.2.1 Wastewater Treatment and Disposal Scheme

The Santa Luz Project wastewater is treated in a closed circuit, with 100% recirculation of industrial water, which means that there are no discharges of process wastewater from operations to surface water. There are two compact treatment sewage (anaerobic system) for treating all domestic wastewater.

20.2.2 Acid Rock Drainage Evaluation

Representative samples were selected from a total of 14 ore panels for gold assays, as well as total sulphur and carbon forms (total C, inorganic C, and carbonate C). Figure 20-1 shows the sulphur grade of each sample assayed and the average grade within each panel.

These data show that the sulphur grade is low for Panels 1 to 4 with average values <0.05% S. There is a noticeable increase in sulphur grade at Panel 5; however, this is due to only one of the nine samples assayed containing more than 0.02% S. The average grade for Panels 6 to 8 is 0.3% S to 0.4% S with some samples containing more than 1% S. The sulphur grade increases to an average of 1% S at Panels 9 and 10, and decreases to 0.5% S in Panels 11 and 12. The grade returns to approximately 1% S in Panels 13 and 14. The sulphur trend is consistent with the transitions from oxide to semi-oxidized and fresh ore.

Preliminary correlations between carbonate carbon and acid neutralizing capacity (ANC) were developed from the 2010 pilot plant samples. Figure 20-2 shows good correlation between carbonate C (C_{carb}) and ANC for samples with <2% C_{carb}, and a reasonable relationship for samples with >2% C_{carb}. Note that there was only poor correlation for the CIL tailings, possibly due to analytical error, and hence the relationships shown should not be used for CIL tailings pending further investigations.

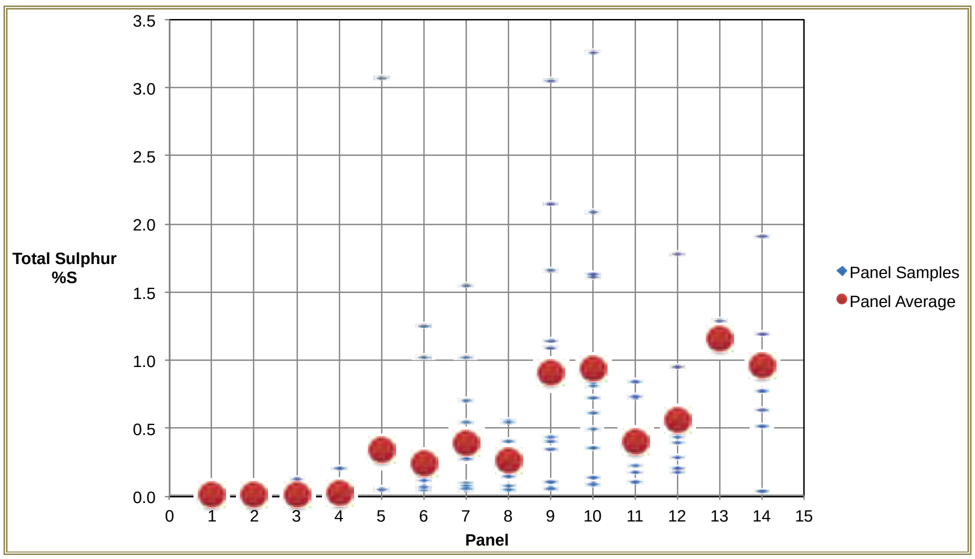


Figure 20-1: Mine Panel Sulphur Grade

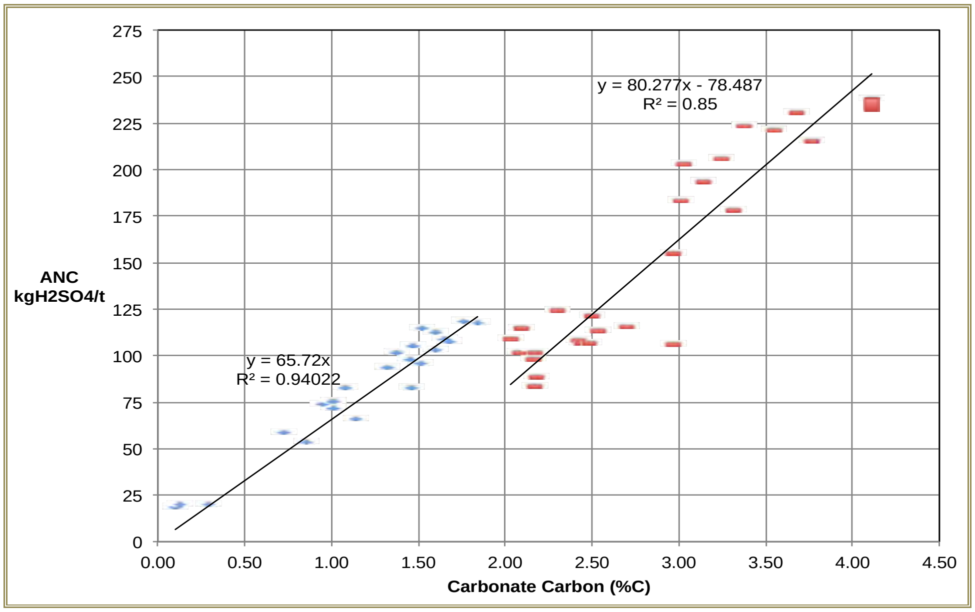


Figure 20-2: Carbonate Carbon and ANC for Ore, Flotation, and Combined Tailings

Figure 20-3 shows the calculated ANC (ANC_{cal}) for the panel ore samples. The individual sample and average values are shown. A minimum ANC_{cal} value of 15 kg H_2SO_4/t was assumed based on actual ANC values for samples with less the 0.2% C_{carb}.

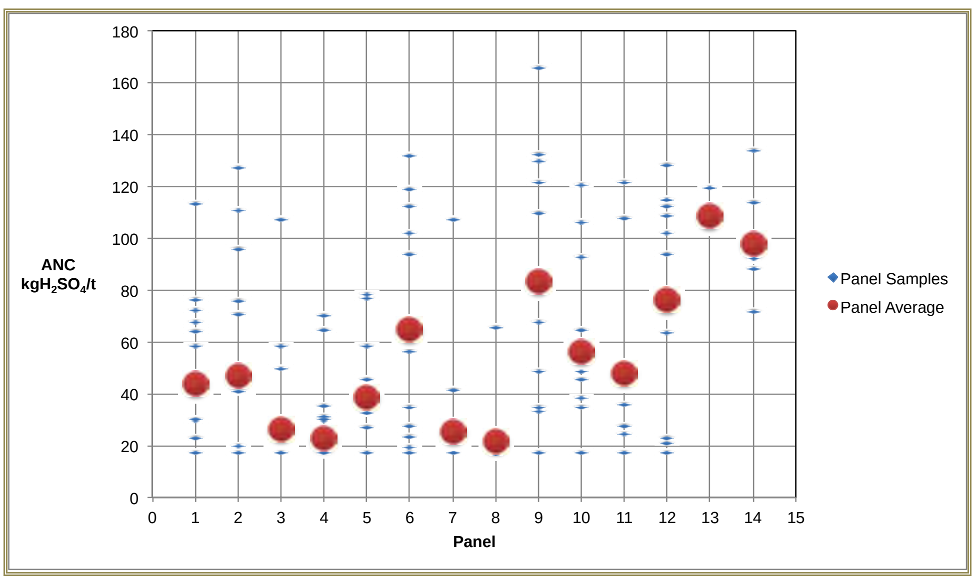


Figure 20-3: Mine Panel Calculated ANC (ANC_{cal})

The results indicate that ANC_{cal} varies widely from typically <20 kg H_2SO_4/t to 130 kg H_2SO_4/t. Overall, the average for Panels 1 to 8 ranged from low to moderate (20 kg H_2SO_4/t to 60 kg H_2SO_4/t), and for Panels 9 to 14 the range from moderate to high values (50 kg H_2SO_4/t to 110 kg H_2SO_4/t). There is a discernable trend of increasing ANC with depth from oxide to fresh ore.

Figure 20-4 shows the ANC/maximum potential acidity (MPA) based on the ANC_{cal} and total sulphur results. The values indicate that all the individual oxide ore and most of the fresh ore samples have ratios >1, with overall average values >2. The ratios for the semi-oxidized ore are lower, and the number of individual samples with ratios less than one is higher than for oxide and fresh ore. However, overall average values indicate a good factor of safety for prevention of acid rock drainage (ARD) from ore.

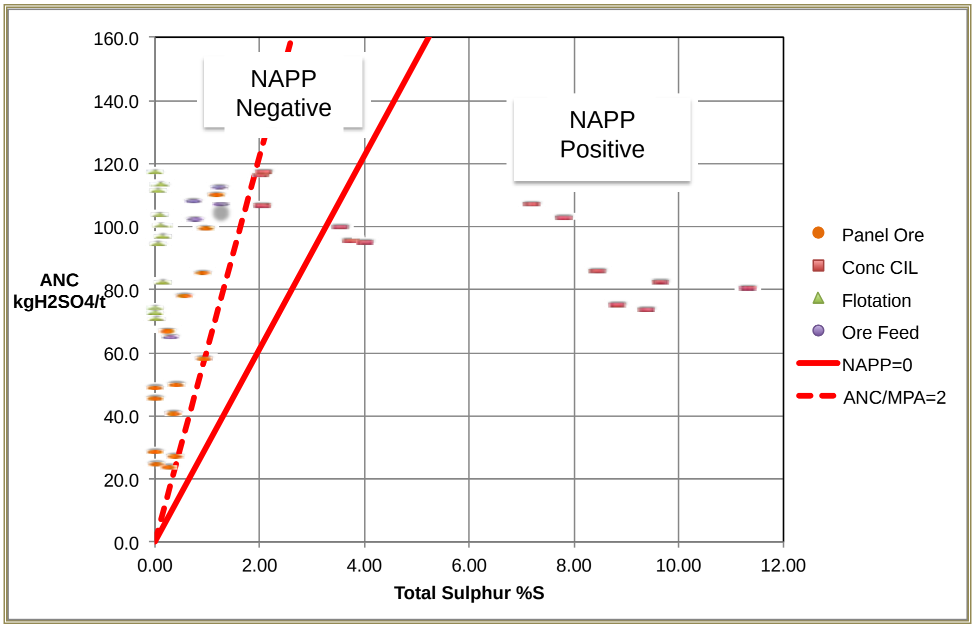


Figure 20-4: Acid Base Account Plot

Figure 20-4 shows that all ore grade samples plot in the net acid production potential (NAPP) negative domain and are likely to be non acid-forming (NAF) with a high factor of safety. The original flotation tailings, produced in 2013 and 2014, are also NAF, with a low sulphur grade (<0.2% S) and high ANC. The ANC/MPA value is >10, confirming a high factor of safety for prevention of ARD from this tailings stream.

The CIL concentrate tailings are high-risk ARD material; however, due to the inherent moderate-to-high ANC (80 kg H_2SO_4/t to 100 kgH_2SO_4/t), these tailings are likely to have a lag period of at least one year before low pH conditions develop. Kinetic non-acid generating testing and column leach testwork on representative samples of CIL tailings are required to confirm the lag period, and element solubility and leaching during the lag.

Based on the ANC and sulphur grades of the ore feed and CIL tailings samples, it is likely these samples were generated from fresh ore material. Hence the oxide and transition flotation tailings are likely to be NAF, and CIL tailings are likely to be potentially acid-forming (PAF). To reduce the risk of potential environmental damage due to the generation of acid drainage by this structure, installing waterproofing of the TSF was concluded in February 2018.

20.3 Requirements for New Environmental Licences and Permits

As part of the expansion of the Santa Luz Project, tree deforestation licences were requested to support the TSF and WSF raises and the Antas 3 pit expansion. A summary of the new licence requirements is presented in Table 20-2.

Table 20-2: Santa Luz Status of New Deforestation Permits

Permit Requested	Process Number	Issue Date	Expiration Date
Deforestation Licence—CIL Tailings Dam Raise	2020.001.001629/INEMA/LIC-01629	09/03/2020	-
Deforestation Licence—Antas 3 Pit Expansion	2018.001.006928/INEMA/LIC-06928	14/11/2018	-
Deforestation Licence—Flotation Tailings Dam Raise	2018.001.002589/INEMA/LIC-02589	08/05/2018	-

In the medium term, additional environmental and social (E&S) studies may be necessary if the mining area exceeds the limits outlined in the current operational licences. In this case, the company will consult INEMA regarding the required E&S studies to obtain the necessary installation licences.

20.4 Mine Closure

Considering the lack of specific mine closure environmental legislation in Brazil, the closure plan for Santa Luz will follow the directions given by DNPM, in terms of what is requested in the Economic Exploitation Plan (Plano de Aproveitamento Econômico or PAE). This also incorporates the recommendations of the Reclamation Plan (RP) presented in the Environmental Impact Assessment—Environmental Impact Report (EIA-RIMA).

The closure plan contains:

- Characterizing the remaining Mineral Reserves
- Plan for removing installations and equipment
- Updating topography to define the recovered mined areas, tailings facilities, waste dumps, and other remaining installations
- Identifying and remediating liabilities
- Environmental monitoring plans relative to the open pits; waste dumps and tailings embankment slope stability and piezometer levels; surface water and groundwater quality; revegetation, and wildlife
- Future uses of the area, according to reclamation plans
- Social communication plans pertaining to the mine closure
- Physical and financial schedule of the closure works.

The closure plan is periodically updated and submitted to INEMA for approval.

The following actions are required for the TSF closure:

- Draining and neutralizing any remaining process water
- Drying surface tailings and establishing a competent tailings beach
- Spreading and compacting a 0.2 m thick soil layer over the tailings beach
- Revegetating with native species to protect the soil surface against erosion.

The following actions are required for waste dump closure:

- Spreading and compacting a 0.2 m thick soil layer over the final plateau
- Revegetating with native species to protect against erosion.

The following actions are required for the process plant and other utility area closure:

- Dismantling and removing all physical structures, sending the unusable and inert materials to the inactive open pit
- Sending to recycling all recyclable materials, such as metals
- Approximately restoring original topography by means of reshaping the altered surfaces and foundations
- Mechanically restoring the terrain with a bulldozer to blend into the surrounding areas
- Revegetating with native species.

The water dam and storage facility will be preserved for future use, including the water pumping system from the Itapicurú River. There are no costs related to this item, with future maintenance and operation costs being passed onto future users.

The open pit and satellite deposit closure costs will be defined only after completing the final mining plan, considering the numerous alternatives available for their closure.

The costs estimated for mine closure are presented in Table 20-3.

Table 20-3: Estimated Closure Costs

Area	Cost (US$ '000s)
Open Pits	1,199
Waste Dumps	1,948
TSF	1,845
Process Plant	1,388
Support Areas	144
Crushing	636
Environmental Monitoring	1,370
Technical Studies	285
Total Estimate	**8,816**

Note: **1.** Assumes an exchange rate of R$5,00:US$1.00.
Total reflects rounding of cost estimates.

20.5 Social or Community Requirements

The areas of direct influence, where changes to the environment will clearly result from the Santa Luz operations, include:

- Santa Luz municipality, part of the Araci municipality
- Campo Grande district, part of the Cansanção municipality
- Nova Esperança settlement
- Alto Bonito settlement
- Nova Vida settlement
- Nordestina municipality, where a garimpeiros village exists.

Yamana previously committed to several community concessions to the original nearby village of Nova Esperança, including village relocation, community compensation, and other environmental considerations, for a total of R$20.6 million. The new village was completed in 2018. Since 2019 and up to June 30, 2020, the Santa Luz Project spent an additional US$0.25 million in community concessions.

Yamana implemented a series of programs, such as *Open Doors*, partnership seminars, environmental education programs, and lectures in the schools and communities in the vicinity of the Project area, which have been continued to date by Equinox. Equinox has not identified any significant issues with the local communities at Santa Luz.

An agreement was established between the Brazilian Institute of Settlement and Land Reform (Instituto Nacional de Colonização e Reforma Agraria or INCRA), the Santa Luz Project, and the Nova Esperança Association for the relocation of the *Agrovila* (Nova Esperança Village) to a new area further away from the Project site, which included constructing houses, along with schools, a community centre, and a health centre. Individual contracts were signed for relocating families from the area, including compensation measures, and construction of new 60 m^2 houses.

20.6 Archaeology

The archaeological prospecting works executed at surface and in underground openings in the Project area have identified one site (coordinates 24 L 468131/8785813) and one occurrence (coordinates 24L 466431/8782046). This archaeological prospecting program was previously evaluated and approved by the Institute of National Historical and Artistic Heritage (*Instituto do Patrimônio Histórico e Artístico Nacional* or IPHAN-MinC)), with a permit being issued through Resolution No. 121 on April 11, 2007 and published in the Official Gazette on April 12, 2007.

The Itapicurú archaeological site is approximately 100 m from the margin of the Itapicurú River, in the vicinity of Mansinha. This site is <100 m wide, and the pedological cover is constituted mainly of fine sandy sediments. The archaeological remains consist of pebble-sized quartz chips. The chips originated through the fracturing or grinding of bigger stones, and occur as small groupings dispersed across a 150 m^2 area. In two of the test excavations opened in the site area, rock chip remains were observed at 10 cm depth.

21 CAPITAL AND OPERATING COSTS

All Project capital costs, operating costs, and off-site costs are summarized in Table 21-1.

Table 21-1: Estimated Capital and Operating Costs Summary

Area	Total Cost (US$ M)	Tonnes ('000s)	Operating Cost (US$/t)
Capital Costs			
Initial Capital			
Initial Mine Stripping and Development	16.4	8.1	2.04
Initial Plant and Infrastructure	73.6	-	-
Working Capital	13.0	-	-
Reclamation	0.1	-	-
Initial Capital Subtotal	**103.1**		
Sustaining Capital			
Tailings	10.2	-	-
Other	10.8	-	-
Sustaining Capital Subtotal	**21.0**	-	-
Mine Stripping Deferred Capital	60.6	-	-
Working Capital Recovery	-5.6	-	-
Reclamation Cost	8.8	-	-
Salvage Value	-15.0	-	-
Total Costs	**172.9**	**-**	**-**
Operating Costs			
Mining			
Ore + Waste		129,266	
Total Mining Cost[a]	**295.2**	**24,938**	**11.84**
Ore Re-Handle (ROM Pad to Crusher)	11.2	24,938	0.45
Ore Re-Handle (Stockpiles to ROM Pad)[a]	16.9	10,210	1.50
Ore Grade Control	4.4	22,747	0.19
Processing	-	-	-
Dacitic Ore	-	10,285	13.67
Carbonaceous Ore	-	12,953	13.81
Dacitic + Carbonaceous Ore	-	23,238	13.74
Dacitic-High-Sulphide Ore	-	1,700	9.17
All Ore	**334.9**	**24,938**	**13.43**
G&A	68.6	24,938	2.75
Total Operating Cost[b]	**698.7**	**24,938**	**28.02**
Off-site Costs			
Transport & Refining	9.0	-	-
Sulphide Concentrate	7.8	-	-

Note: [a] Mining cost excluding Initial Mine Stripping & Development [b] Ore re-handle excluding pre-stripping period

21.1 Capital Costs

The total remaining initial capital costs estimated for the pre-production period (Year -1 and part of Year 1) is US$103.1 million. The total initial and sustaining capital cost is estimated to be US$172.9 million, including the capitalized stripping of the pit (US$60.6 million) and the initial working capital (US$13.0 million) but excluding the working capital recovery (US$5.6 million), reclamation costs (US$8.8 million) or salvage value (US$15.0 million). There are three main components to this cost:

- Capitalized mine stripping
- Plant and infrastructure
- Sustaining capital.

Mine stripping and development costs are estimated to be US$60.6 million. Details of these costs are provided in Table 21-2.

Table 21-2: Estimated Capital Mine Stripping Cost Schedule

Year	Excess Waste Stripped Over LOM Average (t '000s)	Capitalized Stripping Costs (US$ '000s)
1	10,919	22,827
2	8,401	17,563
3	6,027	12,599
4	3,641	7,612
Total	28,988	60,600

A breakdown of the plant and Infrastructure cost is provided in Table 21-3.

Table 21-3: Plant and Infrastructure Total Capital Cost Estimate

Area	Pre-Production (Year -1 to Year 1) (US$ M)
Ore Processing Plant	
Process Plant Alterations	37.5
Tailings and Water Storage Facilities Expansions	7.5
EPCM and Others	4.8
Owner's Costs, Including Pre-Operation Staff Hiring	10.2
Contingency	9.5
Subtotal	69.5
Pre-stripping	
Contractor Mobilization and Mining Infrastructure	4.0
Pre-Stripping Activities	16.4
Subtotal	20.4
Working Capital and First Fills	
Inventories, Other Working capital	5.6
Plant First Fills, Consumables	7.4
Total	**103.1**

Notes: Exchange Rate R$5.0:US$1.00
Projected Accuracy: -5/+20%
Numbers may not sum due to rounding.

For the initial capital costs, July to December (H2 2020) forecast is US$25.10 million. Current status of 2020 expenditures are indicated below:

- Plant alterations: US$25.52 million spent in H1 2020 for new equipment, with an additional US$16.92 million scheduled in H2 2020
- Owner's US$4.63 million spent in H1 2020; the remaining US$3.16 million to be spent in H2 2020
- TSF and WSF: no expenditure in H1 2020 for the installation of the geomembrane liner; US$3.13 million to be spent in H2 2020
- Pilot plant: US$0.71 million spent in H1 2020; the remaining US$0.05 million to be spent in H2 2020
- Community: US$8.13 million spent in H1 2020; the remaining US$0.01 million to be spent in H2 2020
- Care and Maintenance: US$11.1 million spent in H1 2020 to maintain the Santa Luz Project site; the remaining US$1.84 million to be spent in H2 2020.

For the sustaining capital costs, these are estimated to be US$21.0 million (excluded capitalized stripping). A breakdown of these costs is provided in Table 21-4. The principal components of these costs are as follows:

- Three embankment raises and basin expansions for the TSF
- Dacitic-high-sulphide plant capital consisting of a facility to process dacitic-high-sulphide ore in Years 8 and 9
- Administrative buildings relocation prior the start of Antas 3 production
- Other; constituting general costs incurred in operation such as mobile equipment replacements.

Table 21-4: Sustaining Capital Cost Estimate

Area	Cost (US$ M)
Tailings Storage Facility	
Year 2 Expansion	3.4
Year 5 Expansion	3.4
Year 7 Expansion	3.4
Subtotal	10.2
Dacitic-High-Sulphide Plant Modifications	
Machinery and Equipment	6.8
Hardware, Software, and Automation	0.7
Furniture and Fixture	0.2
Subtotal	7.7
Infrastructure	2.8
Other	0.4
Total	**21.0**

Notes: Projected Accuracy: -5/+30%
Exchange Rate: R$5.0:US$1.00

A summary of the Project's capital costs, including the deferred mine stripping, plant and infrastructure, and sustaining capital cost, are presented in Table 21-5.

Table 21-5: Santa Luz Summary of Project Capital Costs

Capital Category	Pre-Production (US$ '000s)	Year 1 (US$ '000s)	Year 2 (US$ '000s)	Years 3 to 10 (US$ '000s)	Total (US$ '000s)
Plant Alterations	37.5	-		-	37.5
TSF and WSF Raises	7.5	-		-	7.5
EPCM and others	4.8	-		-	4.8
Owner's Costs	10.2	-		-	10.2
Pre-Stripping	20.5	-		-	20.5
Contingency	9.5	-		-	9.5
Working Capital	13.0	-	-	(5.6)	7.4
Salvage	-	-	-	(15.0)	(15.0)
Deferred-Stripping Capital Cost	-	22.8	17.6	20.2	60.6
Sustaining Capital Cost	-	6.9	1.7	12.5	21.0
Reclamation Cost	0.1	0.1	0.0	8.6	8.8
Total Capital Cost	**103.1**	**29.8**	**19.3**	**20.7**	**172.9**

Note: **1.** LOM exchange rate R$5.00:US$1.00.

21.2 Operating Costs

The average operating costs are estimated at US$28.02/t of ore processed. There are three components to this cost, as follows:

- Mining
- Processing
- General and administrative (G&A) and Other.

21.2.1 Mining

Mining costs per ore processed is US$11.84/t, as shown in Table 21-1. LOM base mining costs are estimated to be US$2.41/t of ore and waste. Details of these costs are provided in Table 21-6. Unit costs were defined based on the previous operation of Santa Luz and the costs from Equinox's Mineração Riacho dos Machados operation, which uses similar-sized mining equipment and a similar production rate.

Table 21-6: Mining LOM Operating Costs

Category	LOM (US$/t mined)
Drilling	0.17
Blasting	0.21
Loading	0.40
Hauling	1.08
Road and Dump Maintenance	0.09
Stockpile Maintenance	0.06
Mine G&A	0.13
Subtotal	2.15
Rehandle Tonnes (Stockpiles)	0.14
Rehandle Tonnes (ROM Pad to Crusher)	0.09
Grade Control	0.04
Total	**2.41**

Notes: Exchange rate: R$5.0:US$1.00

21.2.2 Process

Process costs are estimated to average US$13.43/t of ore processed for the carbonaceous and dacite ores as well as the high sulphide dacite ore. The following three types of ore will be processed:

- Dacitic ore
- Carbonaceous ore
- Dacitic-high-sulphide ore from Antas 3 and stockpiles 2, 3 and 4 (Table 15-3).

The process plant will have the flexibility to process dacitic ore and carbonaceous ore separately if warranted. However, testwork has shown that a blend of these two ores can be processed at the same cost and achieve the same recovery as would occur with separate campaign processing; therefore, the plan is to process a blend of the two ores. A breakdown of the estimated operating cost for processing the ore blend is provided in Table 21-7. In common with most gold ore processing plants, and as indicated in the table, the three largest cost components are power, grinding media, and cyanide. Other significant costs are labour, maintenance materials, and kerosene.

Table 21-7: Process LOM Operating Costs (Carbonaceous + Dacite Ores)

Commodity	Consumption		Commodity		Operating Cost (US$/t)
	Rate	Rate Basis	Cost (US$)	Cost Basis	
Labour	129	employees	17,069	employee/year	0.82
Power	42	kWh/t	0.04	kWh	1.68
Maintenance Materials		fixed cost/t	1.18	tonne	1.18
Miscellaneous		fixed cost/t	0.46	tonne	0.46
Crusher Liners					
Jaw Crusher	6	sets/a	5,616	per set	0.02
Cone Crusher	6	sets/a	8,320	per set	0.01
Mill Liners					
SAG Mill Liners	2	sets/a	832,000	per set	0.59
Ball Mill Liners	1	sets/a	624,000	per set	0.23
Consumables					
Grinding Balls	1.81	kg/t	672	tonne	1.21
Quicklime	1.40	kg/t	55	tonne	0.05
Oxygen	4.56	kg/t	71	tonne	0.12
Kerosene	2.00	kg/t	989	tonne	1.98
Cyanide	1.00	kg/t	640	tonne	1.94
Resin	0.00003	m^3/t	17,243	m^3	0.60
Sodium Metabisulphite (SMBS)		kg/t	487	tonne	0.36
Sulphuric Acid	1.80	kg/t	462	tonne	0.31
Copper Sulphate	0.19	kg/t	2,157	tonne	0.15
Thiourea	0.79	kg/t	2,000	tonne	0.58
Caustic	0.80	kg/t	238.57	tonne	0.19
Ferric Sulphate	0.66	kg/t	328	tonne	0.08
Other Costs					0.86
Total					**13.43**

Notes: R$5.0:US$1.00 exchange rate
Projected Accuracy: -5/+25%

Table 21-8 provides a breakdown of the estimated operating cost for processing dacitic-high-sulphide ore, which will be processed in the last year of operation. Except for the reagents, the estimated operating costs for the dacitic-high-sulphide ore are the same as those for the blend of dacitic and carbonaceous ore. The cost for reagents is significantly lower for the dacitic-high-sulphide ore than that of the blend of dacitic and carbonaceous ore, since no reagents are required for leaching and detoxification; the reagents will consist of flotation chemicals and flocculent.

Table 21-8: Process LOM Operating Costs (High-Sulphide Dacite Ore)

Commodity	Consumption		Commodity		Operating Cost (US$/t)
	Rate	Rate Basis	Cost (US$)	Cost Basis	
Labour	129	Employees	22,500	employee/year	1.18
Power	42	kWh/t	0.055	kWh	2.30
Maintenance Materials		fixed cost/t	1.50	tonne	1.50
Miscellaneous		fixed cost/t	0.50	tonne	0.50
Crusher Liners					
Jaw Crusher	6	sets/a	8,500	per set	0.02
Cone Crusher	6	sets/a	13,000	per set	0.03
Mill Liners					
SAG Mill Liners	2	sets/a	800,000	per set	0.59
Ball Mill Liners	1	sets/a	600,000	per set	0.22
Grinding Balls	1.75	kg/t	1,000	tonne	1.75
Reagents					
Sulphuric Acid	0.50	kg/t	650	tonne	0.33
Copper Sulphate	0.10	kg/t	3,000	tonne	0.30
Collectors	0.15	kg/t	2,000	tonne	0.30
Frother	0.05	kg/t	500	tonne	0.03
Flocculent	0.01	kg/t	5,000	tonne	0.05
Other	0.50	kg/t	1,000	tonne	0.50
Subtotal					**1.50**
Total Before Currency Adj.					**9.59**
Currency Adj. to 3.70					(0.42)
Total					**9.17**

Notes: Estimate Date: Q2 2017 adjusted to 3.70 exchange rate
Projected Accuracy: -5/+25%

21.2.3 General and Administrative and Other

G&A costs are estimated to be US$2.75/t of ore processed, as shown in Table 21-9.

Table 21-9: G&A and Other LOM Operating Costs

Item	LOM Total Cost (est.) (US$ M)	Operating Cost (US$/t Processed)
G&A (US$7.42 M/a)	68.58	2.75

Notes: Exchange Rate: R$5.0: US$1.00
Projected Accuracy: -5/+30%

21.2.4 Operating Costs Summary

Table 21-10 shows the LOM operating costs summary for mining, processing, and G&A.

Table 21-10: Santa Luz Summary of Project LOM Operating Costs

Total Operating Costs	LOM Total (US$ '000s)	Unit Costs (US$/t Processed)
Mining Cost	262,724	10.54
Grade Control	4,357	0.17
Ore Re-Handle (ROM Pad to Crusher)	11,222	0.45
Ore Re-Handle (Stockpiles)	16,921	0.68
Processing	334,875	13.43
Fixed G&A	68,579	2.75
Total Operating Costs	**698,678**	**28.02**

21.2.5 Off-Site Costs (for High-Sulphide Dacite Concentrate)

Off-site costs, constituting the freight, smelting, and refining (FSR) costs for processing the concentrate generated from the dacitic-high-sulphide ore, are provided in Table 21-11. While the presumption is that the concentrate will be processed by a copper smelter, testwork will be required on other processing methods that will likely be more economic.

Table 21-11: Off-Site Costs for High-Sulphide Dacitic Concentrate

Parameter	Unit	Value
Basis		
Concentrate Quantity		
Dry	dry tonnes	24,000
Wet (8% Moisture)	wet tonnes	26,000
Concentrate Grade		
Gold	g/t	80
Arsenic	g/t	166,000
Concentrate Production	t	24,000
Gold Production	oz	63,000
Payable Gold (93.4%)	oz	59,000
Cost		
Transport (US$90/wet tonne)	US$	2,340,000
Treatment (US$90/dry tonne)	US$	2,160,000
Penalty (US$1.50/1,000 g/t As)	US$	5,976,000
Refining (US$5.00/payable ounce)	US$	295,000
Total Before Currency Adj.	**US$**	**10,771,000**
Currency Adjusted to 3.70		(396,000)
Total		**10,375,000**

Notes: Projected Accuracy: -5/+50%

21.2.6 Power

Total power consumption is derived from the average power draw of all duty drives on the mechanical equipment list. Factors for expected run time and loading are applied to the installed power to determine the average power drawn. Power unit costs, as discussed in Data Sources, are as per invoice rates from the Bahia State electrical company, Companhia de Eletricidade do Estado da Bahia (COELBA). Power costs are constant for processing both dacitic and carbonaceous ore types. Average power consumption per tonne of ore to be milled is 41 kWh/t. Annual power consumption is estimated to be 110,600 MWh.

21.2.7 Maintenance

Maintenance costs include the costs to conduct routine maintenance activities required to maintain the plant, including the consumables that are used during these activities. Maintenance consumables, for example on-site piping, valve and platework replacement/refurbishment, lubricants, and pump liners, are included in the maintenance costs. Speciality wear items, such as grinding mill liners and crusher wear components, are separately accounted for in the Reagents and Consumables section.

21.2.8 Reagents and Consumables

Equinox and Ausenco have provided unit reagent costs (with the exception of antiscalant and water treatment reagents), based either on purchase prices during the previous operations phase of the process plant or current reagent prices obtained from Equinox's other operating plants in Brazil. Antiscalant and water-treatment reagent costs have been benchmarked by Ausenco.

Santa Luz Project personnel and Ausenco provided unit costs for consumables (except for ball mill liners, ball mill grinding media, and vibrating screen panels), based on purchase prices during the previous operations phase of the process plant (and benchmarked against current operations). The unit costs for ball mill liners, ball mill grinding media, and vibrating screen panels were more recently sourced from vendor quotations or benchmarked against similar operating plants.

Reagent and consumables costs should not vary by ore type because processing will occur on a blend of dacitic and carbonaceous ores.

21.3 Project Schedule

The overall Project implementation milestones, as of June 30, 2020, are shown in Table 21-12, based on the final investment decision made on May 15, 2020. The planned completion date for the construction project is year-end 2021.

Table 21-12: Project Construction Milestones

Milestone	Date
Restart Authorization	May 15, 2020
Construction Start	August 3, 2020
Cold Test Equipment Start-up	June 1, 2021
Plant Commissioning Start-up	September 1, 2021
Operational Ramp-up Start	October 1, 2021
Construction Completion	December 31, 2021
Commercial Operation	January 1, 2022

22 ECONOMIC ANALYSIS

The summary results for the Santa Luz Project is summarized in Table 22-1 and is based on the LOM production schedule and capital and operating cost estimates. A more detailed cash flow summary is presented in Table 22-2.

Table 22-1: Santa Luz Cash Flow Summary (US$1,500/oz)

	Unit	LOM Total
Total Ore Mined	kt	22,747
Total Waste Mined	kt	106,519
Total Material Moved	kt	129,266
Strip Ratio	w:o	4.7
Au Grade	g/t	1.39
Contained Gold	oz	1,014,263
Stockpiled Ore Processed	kt	2,191
Au Grade	g/t	0.86
Contained Gold	oz	60,654
Total Ore Processed	kt	24,938
Processed Au Grade	g/t	1.34
Contained Gold	oz	1,074,917
Recovery	%	84
Recovered Gold	oz	902,549
Mine Life	years	9.5
Initial Capital	US$ M	103.1
Sustaining Capital	US$ M	21.0
Average Annual Production (LOM)	oz	95,000
Average Annual Production (2022–2026)	oz	110,500
Average Annual Production (2022–2029)	oz	104,500
Average Annual EBITDA (LOM)	US$ M	68.7
Average Annual Net Cash Flow (LOM, after tax)	US$M	56.9
Net Cumulative Cash Flow (LOM, after tax)	US$ M	436.0
NPV 5% (after tax)	US$ M	305.1
IRR (after tax)	%	57.6
Payback Period	years	1.6
Cash Costs (LOM, including royalties)	US$/oz	776
AISC[1]	US$/oz	877

Note: 1. All-in sustaining costs (AISC) includes mine cash costs per oz sold, royalties, sustaining capital costs, and operational waste stripping costs.

A summary of the key criteria is provided below.

22.1 Economic Criteria

22.1.1 Revenue

- Approximately 7,400 t/d of ore processed (approximately 2.7 Mt/a).
- Processing gold recoveries of 84% were used in the cash flow for a blended feed of high carbonaceous material, low carbonaceous material, and dacitic ore. Gold recovery for dacites with high-sulphides is also projected to be 84%.
- Metal prices for cash flow: US$1,500/oz Au.
- Salvage value of US$15 million was applied to equipment or infrastructure at the end of the LOM.
- 9.5-year project life during production.
- Yearly revenues were calculated by subtracting the applicable refining charges and transportation costs (US$10/oz) from the payable metal value generated by carbonaceous and dacitic ore, and US$177/oz from dacites with high-sulphide ore.
- Revenue is recognized at the time of production.
- The production schedule includes only Proven and Probable Mineral Reserves.

22.1.2 Costs

- There are 6.9 Mt mined as pre-strip, and stockpile relocation of 1.2 Mt prior to the start of commercial production.
- Unit operating costs for mining, processing, re-handle, grade control, and G&A were applied to determine the overall yearly operating cost.
- Closure costs for the Project have been estimated at US$8.8 million and these costs are included in the cash flow.
- Initial capital cost totals US$103.1 million.
- Local currency denominated capital and operating costs are based on a nominal exchange rate of R$5.00:US$1.00.
- Project LOM AISC is US$877/oz.
- Royalties: An existing royalty agreement with the Brazilian federal government for 1.5% gross revenue; another agreement with COSIBRA for 1% gross revenue was included in the cash flow and pit optimization analysis. An additional 2% royalty was included for the CBPM area of the C1 deposit, which represents 397,810 oz in the production schedule.

22.1.3 Taxation

For the calculation of income taxes, it has been assumed that the Superintendência de Desenvolvimento do Nordeste (SUDENE) mining tax incentive would be approved for the duration of the LOM, which results in an income tax rate of 15.25%. An average rate of 9.25% was assumed for operating and capital costs subject to Brazilian federal value-added-taxes (PIS and Cofins) and 12% was assumed for items subject to state value-added-taxes (Imposto Sobre Operações Relativas à Circulação de Mercadorias e Serviços de Transporte Interestadual de Intermunicipal e de Comunicações or ICMS).

22.2 Cash Flow Analysis

Table 22-2 shows the LOM cash flow for Santa Luz Project.

Table 22-2: Santa Luz LOM Cash Flow

	Unit	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032-2038
Open Pit															
Ore Mined	kt	22,747	0	570	1,771	2,140	2,521	3,230	4,454	3,546	3,697	819	0	0	0
Waste Mined	kt	106,519	0	6,304	19,213	18,422	17,831	18,765	13,692	8,610	3,427	254	0	0	0
Total Mined	**kt**	**129,266**	**0**	**6,874**	**20,984**	**20,562**	**20,351**	**21,994**	**18,146**	**12,156**	**7,124**	**1,073**	**0**	**0**	**0**
Capitalized Waste	kt	31,152	0	2,164	10,919	8,401	6,027	3,641	0	0	0	0	0	0	0
Total Mined (Exc. Capitalization)	**kt**	**98,113**	**0**	**4,710**	**10,064**	**12,161**	**14,325**	**18,353**	**18,146**	**12,156**	**7,124**	**1,073**	**0**	**0**	**0**
Strip Ratio	w:o	4.7	0	11.1	10.8	8.6	7.1	5.8	3.1	2.4	0.9	0.3	0.0	0.0	0
Grade	g/t	1.39	**0**	1.45	1.73	1.75	1.40	1.36	1.28	1.06	1.39	1.66	0.00	0.00	0
Contained Gold	oz	1,014,263	0	26,602	98,505	120,554	113,600	141,722	183,360	120,440	165,757	43,723	0	0	0
Processed															
Dacite-Carb Processed	kt	23,238	0	0	2,475	2,700	2,700	2,700	2,700	2,700	2,700	2,025	1,675	863	0
Grade	g/t	1.37	0	0	1.70	1.58	1.37	1.49	1.57	1.15	1.60	1.07	0.68	0.69	0
Contained Gold	oz	1,022,445	0	0	135,531	137,061	119,154	129,196	136,646	99,965	139,313	69,832	36,665	19,083	0
Recovery Rate	%	84	0	0	84	84	84	84	84	84	84	84	84	82	0
Dacite-Carb Recovered Gold	oz	858,472	0	0	113,846	115,131	100,090	108,525	114,782	83,970	117,023	58,659	30,798	15,648	0
Dac-High Sulphide Ore Processed	kt	1,700	0	0	0	0	0	0	0	0	0	675	1,025	0	0
Grade	g/t	0.96	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.30	0.73	0.00	0
Contained Gold	oz	52,473	0	0	0	0	0	0	0	0	0	28,305	24,168	0	0
Recovery Rate	%	84	0	0	0	0	0	0	0	0	0	84	84	0	0
Dac-High Sulphide Recovered Gold	oz	44,077	0	0	0	0	0	0	0	0	0	23,776	20,301	0	0
Processed Tonnes	kt	24,938	0	0	2,475	2,700	2,700	2,700	2,700	2,700	2,700	2,700	2,700	863	0
Grade	g/t	1.34	0.00	0.00	1.70	1.58	1.37	1.49	1.57	1.15	1.60	1.13	0.70	0.69	0
Contained Gold	kt	1,074,917	0	0	135,531	137,061	119,154	129,196	136,646	99,965	139,313	98,137	60,833	19,083	0
Recovery Rate	%	84	0	0	84	84	84	84	84	84	84	84	84	82	0
Recovered Gold	oz	902,549	0	0	113,846	115,131	100,090	108,525	114,782	83,970	117,023	82,435	51,099	15,648	0
Gold Produced	oz	902,549	0	0	113,846	115,131	100,090	108,525	114,782	83,970	117,023	82,435	51,099	15,648	0
Ounces Subject to CBPM Royalty	oz	397,810	0	0	93,938	73,823	37,056	64,635	76,484	3,416	48,146	313	0	0	0

	Unit	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032-2038
Cash Flow Summary (US$M)															
Revenue		1,353.8	0.0	0.0	170.8	172.7	150.1	162.8	172.2	126.0	175.5	123.7	76.6	23.5	0.0
Refining transportation		(9.0)	0.0	0.0	-1.1	-1.2	-1.0	-1.1	-1.1	-0.8	-1.2	-0.8	-0.5	-0.2	0.0
High Sulphide Gold Concentrate Transport & Refining		(7.8)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-4.2	-3.6	0.0	0.0
CBPM Royalty (for east half of C1 pit)		(11.9)	0.0	0.0	-2.8	-2.2	-1.1	-1.9	-2.3	-0.1	-1.4	0.0	0.0	0.0	0.0
COSIBRA Royalties		(13.5)	0.0	0.0	-1.7	-1.7	-1.5	-1.6	-1.7	-1.3	-1.8	-1.2	-0.8	-0.2	0.0
Gov't Royalties (CFEM)		(20.3)	0.0	0.0	-2.6	-2.6	-2.3	-2.4	-2.6	-1.9	-2.6	-1.9	-1.1	-0.4	0.0
Net revenue		1,291.2	0.0	0.0	162.5	165.0	144.3	155.7	164.4	121.9	168.5	115.5	70.6	22.7	0.0
OP Mining		(202.1)	0.0	0.0	-21.6	-26.1	-30.7	-39.4	-39.3	-26.5	-16.0	-2.5	0.0	0.0	0.0
Processing		(334.9)	0.0	0.0	-34.0	-37.1	-37.1	-37.1	-37.1	-37.1	-37.1	-34.0	-32.4	-11.9	0.0
Site G&A		(68.6)	0.0	0.0	-6.8	-7.4	-7.4	-7.4	-7.4	-7.4	-7.4	-7.4	-7.4	-2.4	0.0
Rehandle		(28.1)	0.0	0.0	-2.9	-2.9	-2.5	-2.3	-1.8	-2.6	-1.7	-4.5	-5.3	-1.7	0.0
Grade Control		(4.4)	0.0	0.0	-0.3	-0.4	-0.5	-0.6	-0.9	-0.7	-0.7	-0.2	0.0	0.0	0.0
Total Operating Costs		(638.1)	0.0	0.0	-65.7	-74.0	-78.2	-86.8	-86.5	-74.3	-62.9	-48.6	-45.1	-16.0	0.0
Operating Margin		653.1	0.0	0.0	96.9	91.0	66.0	68.9	77.9	47.6	105.6	66.9	25.5	6.8	0.0
Sustaining Capital		(81.6)	0.0	0.0	-29.7	-19.2	-18.4	-8.4	-4.0	-1.6	-0.3	0.0	0.0	0.0	0.0
Non-Sustaining CAPEX		(103.1)	-1.7	-97.4	-3.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Working Capital Recovery		5.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5.6	0.0
Reclamation Costs		(8.8)	0.0	-0.1	-0.1	0.0	0.0	-0.1	0.0	0.0	-0.3	-0.9	-0.9	-1.5	-5.0
ICMS		(14.1)	0.0	0.0	-1.4	-1.6	-1.6	-1.6	-1.6	-1.6	-1.6	-1.4	-1.4	-0.5	0.0
PIS/COFINS		(2.1)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-0.3	0.0	0.0	-1.0	-0.7	0.0
Taxes CIT		(28.1)	0.0	0.0	-3.2	-5.0	-1.9	-1.1	-2.4	-0.7	-6.9	-5.7	-1.1	0.0	0.0
Salvage		15.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	15.0	0.0
Net Cash Flow		436.0	(1.7)	(97.5)	58.5	65.3	44.2	57.8	69.9	43.3	96.5	58.9	21.1	24.6	(5.0)
Cash cost/oz		776	-	-	649	709	840	865	821	933	598	688	1,000	1,068	-
AISC/oz		877	-	-	911	877	1,024	944	856	953	603	699	1,018	1,164	-
NPV 5%	%	305.1													
IRR	%	57.6													
Payback Period	year	1.6													

22.3 All-In Sustaining Cost

The LOM plan shows production averaging 110,500 oz of gold per year for the first five years (2022–2026) with an average AISC of $922/oz. Production for the first eight years (2022–2029), when the mine is processing predominantly fresh ore, averages 104,500 oz of gold per year with an average AISC of $858/oz, followed by 1.5 years processing residual stockpile feed for a LOM production average of 95,000 oz of gold per year and a LOM average AISC of $877/oz. The Project's AISC includes capitalized stripping and reclamation costs. Annual gold production and AISC are shown on Figure 20-1.

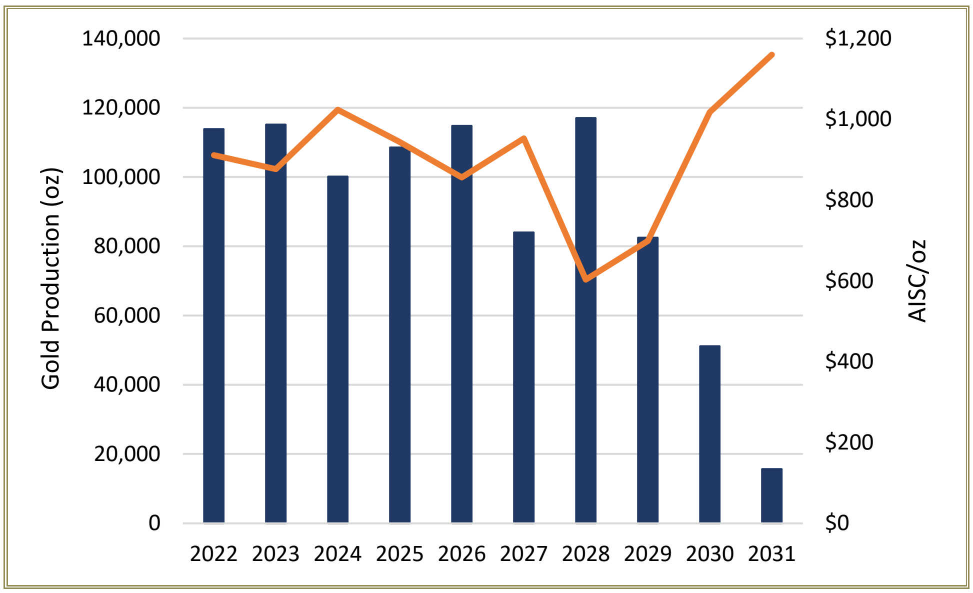


Figure 22-1: Annual Gold Production and All-In Sustaining Cost

22.4 Sensitivity Analysis

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities for the following:

- Gold price
- Operating costs
- Capital costs
- Foreign exchange.

The net present value (NPV) at 5% discount rate sensitivities are shown in Figure 22-2, Figure 22-3, and Table 22-3.

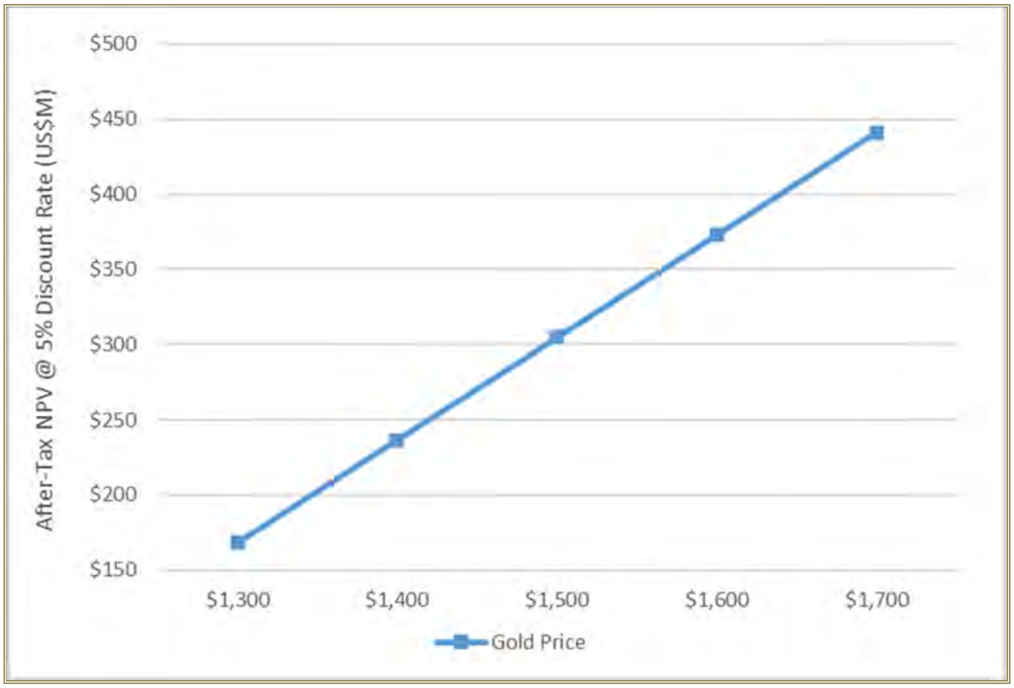


Figure 22-2: Gold Price Sensitivity Analysis to After-Tax NPV

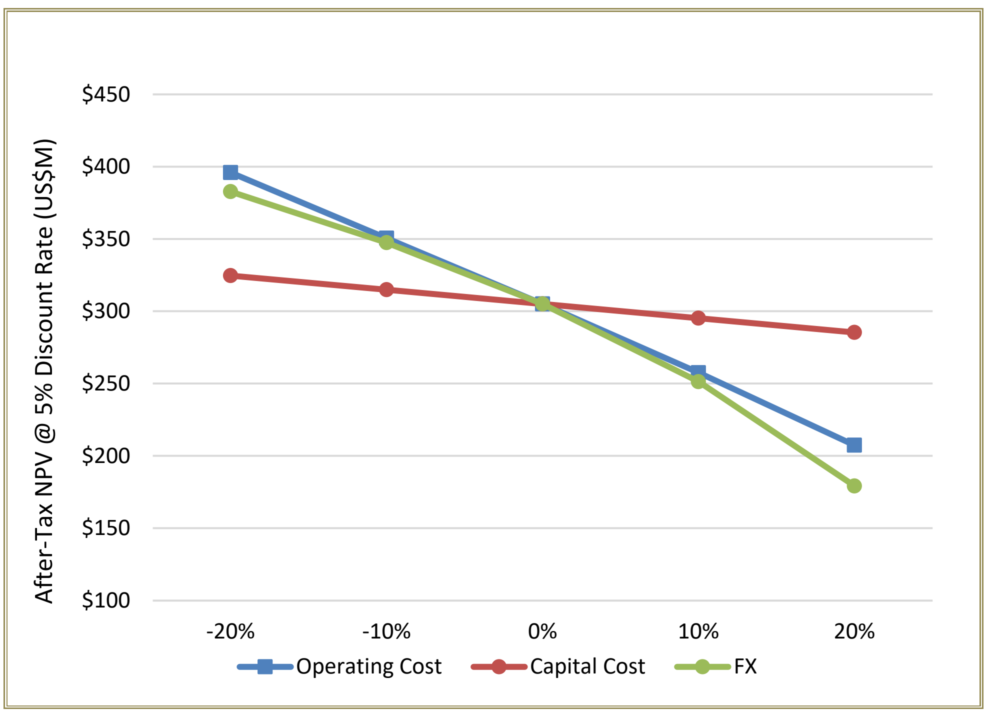


Figure 22-3: Operating Cost, Capital Cost, and Foreign Exchange Sensitivity Analysis to After-Tax NPV

Table 22-3: Sensitivity Analyses

Factor	After-Tax NPV at 5% (US$ M)	IRR (%)
Gold Price		
US$1,300	186	38
US$1,400	247	48
US$1,500	305	58
US$1,600	362	67
US$1,700	419	76
US$1,800	475	85
Operating Cost		
-20%	396	73
-10%	351	65
0%	305	58
10%	258	49
20%	207	41
Capital Cost		
-20%	325	73
-10%	315	65
0%	305	58
10%	295	52
20%	285	47
Foreign Exchange		
-20%	383	79
-10%	347	69
0%	305	58
10%	251	46
20%	179	32

23 ADJACENT PROPERTIES

There are no adjacent properties relevant to the Santa Luz Project.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 C1 Underground

24.1.1 Summary

SLR and Equinox have completed a PEA-level study of the potential to exploit the Mineral Resources below the open pit using underground mining methods (RPA, 2018).

SLR considers this section to meet the requirements of a PEA as defined in Canadian NI 43-101 regulations. The PEA relates to the economic potential of the Santa Luz C1 Underground (C1 Underground) Mineral Resources and is not part of, and should be distinguished from, the planned open pit mining of the Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The economic analysis contained in this section is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. Additional drilling and technical studies will be required to convert the C1 Underground Mineral Resources to Mineral Reserves. There is no certainty that the results contemplated in the PEA will be realized.

The C1 Underground resources are a proximal extension of the Mineral Resources exploited by the C1 open pit.

Host rocks to the underground resources include carbonaceous metasedimentary rocks, dioritic and dacitic intrusive rocks, and metavolcanic rocks. The majority of the C1 underground resource is classified as carbonaceous breccia. The mineralization style is quartz-carbonate-sulphide veins and breccia fillings hosted in a major, district-scale shear zone, typical of orogenic gold deposits.

The shear zone is north to northeast trending and dips at 30° to 40° to the west. The shear zone and mineralization range in thickness from several metres to over twenty metres.

The C1 Underground Mineral Resources considered in this study exist in four separate mining zones (A, B, C, and F). The largest is the B-Zone.

The design anticipates a nominal 2,500 t/d underground long hole mining operation using cemented paste backfill to allow for maximum extraction of the deposit. Over the 9.5-year LOM, a total of 7.1 Mt of mill feed would potentially be extracted at a grade of 2.65 g/t Au.

Primary and secondary long hole stoping using paste backfill is considered the most practical and economic method for extracting the C1 Underground Mineral Resources. The underground mineralization is a down-dip continuation of the C1 deposit scheduled to be extracted by the open pit. These Mineral Resources are a portion of the C1 Santa Luz Mineral Resources that are described in Section 14.

The projected preliminary development access and mining method design for the C1 Underground is based on current practices at Equinox Gold's Fazenda Brasileiro mine mining operation located 115 km by road northeast of the Santa Luz property. SLR has used the same development heading profiles, stope drilling, and blasting patterns and mobile equipment fleet for the C1 Underground as are currently in use at the Fazenda Brasileiro mine. Unit productivities (except for development) and unit costs for all component development and stoping activities (except for backfilling) proposed for the C1 Underground are based on the actual Fazenda Brasileiro mine 2016 and 2017 results (Brio, 2016, 2017).

If Equinox elects to develop the C1 Underground, development of the main decline will take two years (identified as Year -2 and Year -1 in the LOM). Production would begin ramp-up in LOM Year 1, to be completed by Year 10.

Total underground mine production would reach full capacity of 2,500 t/d by mid-Year 2 and continue at this rate until early Year 7. Over the expected 9.5-year LOM, the C1 Underground is forecast to contribute a total production of 511,000 oz Au.

LOM C1 Underground development will include excavation of slightly more than 24 km of decline and lateral development and 630 m of bored ventilation raises as summarized in Table 24-1. Waste development will generate 778,000 tonnes of waste rock that will be hauled to surface for disposal on one of the open pit waste dumps.

Table 24-1: C1 Underground Development

Description	Dimension	Quantity (m)
Decline	5 m wide by 5.5 m high	6,687
Stope Access Drifts	5 m wide by 5 m high	4,175
Stope Sill Drifts	5 m wide by 5 m high	13,155
Intake Ventilation Raise	3 m diameter	300
Exhaust Ventilation Raise	4 m diameter	330

SLR prepared a stand-alone LOM plan with production from the underground Mineral Resources. SLR is of the opinion that the key elements remaining to be confirmed for the underground Mineral Resources are the spatial location, thickness, and geological continuity of the mineralization, as well as the geotechnical properties that impact access development and stope design.

The mill feed from the C1 Underground would potentially be blended with open pit ore in the proposed 7,400 t/d process plant and no modifications to the process plant are included in this analysis.

A large proportion of the tailings generated from processing C1 Underground mill feed will be returned underground as paste backfill for the mined-out stopes. Paste fill production is estimated at 5.1 Mt. The remaining tailings (2.0 Mt) will be placed in the existing TSF.

The estimated pre-production capital cost for the C1 Underground is US$74.1 million and the total Project capital is US$98.3 million. The estimated operating cost is US$50.28/t. The key Project parameters, based on a foreign exchange rate of R$5.00:US$1.00, are shown in Table 24-2.

Table 24-2: C1 Underground Key Project Metrics

Description	Unit	Value
Tonnes Mined and Processed	Mt	7.132
LOM (Including Production Ramp-Up)	years	9.5
Mill Throughput (Full Production)	t/d	2,500
Mill Throughput (Annual)	Mt/a	0.75
Average Grade Gold (Diluted)	g/t	2.65
Average Operating Cost	US$/t	50.28
Pre-production Capital Cost	US$ M	74.1
Sustaining Capital Cost	US$ M	23.2
Closure Allowance	US$ M	1.0

At a gold price of US$1,500/oz and mill recovery averaging 84%, net smelter return (NSR) revenues are forecast to equal US$104.86/t milled.

The estimation of potentially mineable underground stoping blocks was initially prepared by RPA in 2018 using a gold price of US$1,300/oz Au and a R$:US$ exchange rate of 3.5:1, which resulted in an estimated break-even cut-off grade of 1.6 g/t Au. The current review (2020) assumes a higher gold price of US$1,500/oz and a more favourable R$:US$ exchange rate of 5.0:1. These new parameters result in higher revenues and lower operating costs than those anticipated in the original estimation. SLR therefore considers the Mineral Resources included in the underground mine plan to be conservative.

24.2 C1 Underground Geology and Mineral Resources

The C1 Underground Mineral Resources are a proximal down-dip extension of the Mineral Resources exploited by the C1 open pit of Equinox Gold's Santa Luz Project. Figure 24-1 shows the location of the C1 Underground Mineral Resources.

Details of the Santa Luz regional and local geology are described in Section 7.

Host rocks to the underground Mineral Resource include carbonaceous metasedimentary rocks, dioritic and dacitic intrusive rocks, and metavolcanic rocks. The majority portion of the underground resource is classified as carbonaceous breccia. The mineralization style is quartz-carbonate-sulphide veins and breccia fillings hosted in a major, district-scale shear zone, typical of orogenic gold deposits.

The shear zone is north to northeast trending and dips moderately to the west. In the underground resource area, the shear zone is overlain by a dioritic intrusive mass and underlain by andesitic metavolcanics rocks.

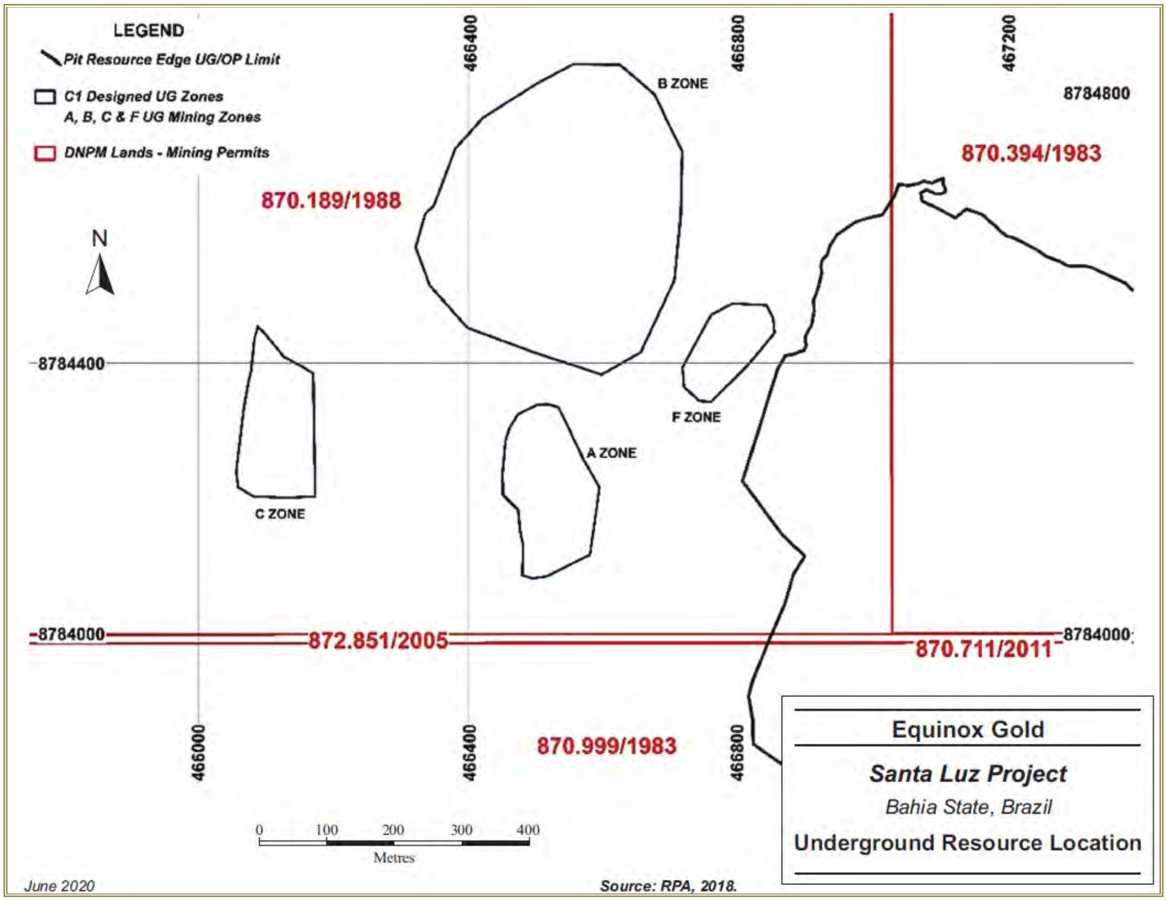


Figure 24-1: C1 Underground Mineral Resource Location

The shear zone and mineralization range in thickness from several metres to over twenty metres. The planned stopes for the C1 Underground are generally located in areas where the shear zone is thickest and contains the highest gold grades, as can be seen in the contoured grade x thickness (GT) values calculated from drill hole intercepts across the mineralized shear zone. Figure 24-2 shows GT contours and the location of the planned stopes. Cross sections A-A’ and B-B’ illustrate the geology of the C1 reserve pit and planned stopes (Figure 24-3).

Equinox believes that there is the potential for significant additional Mineral Resources to be delineated in down-dip and lateral extensions of the C1 Underground Mineral Resource. Additional definition drilling is planned to reduce the space between drill holes to achieve Measured or Indicated Mineral Resources and enable a full underground mine design.

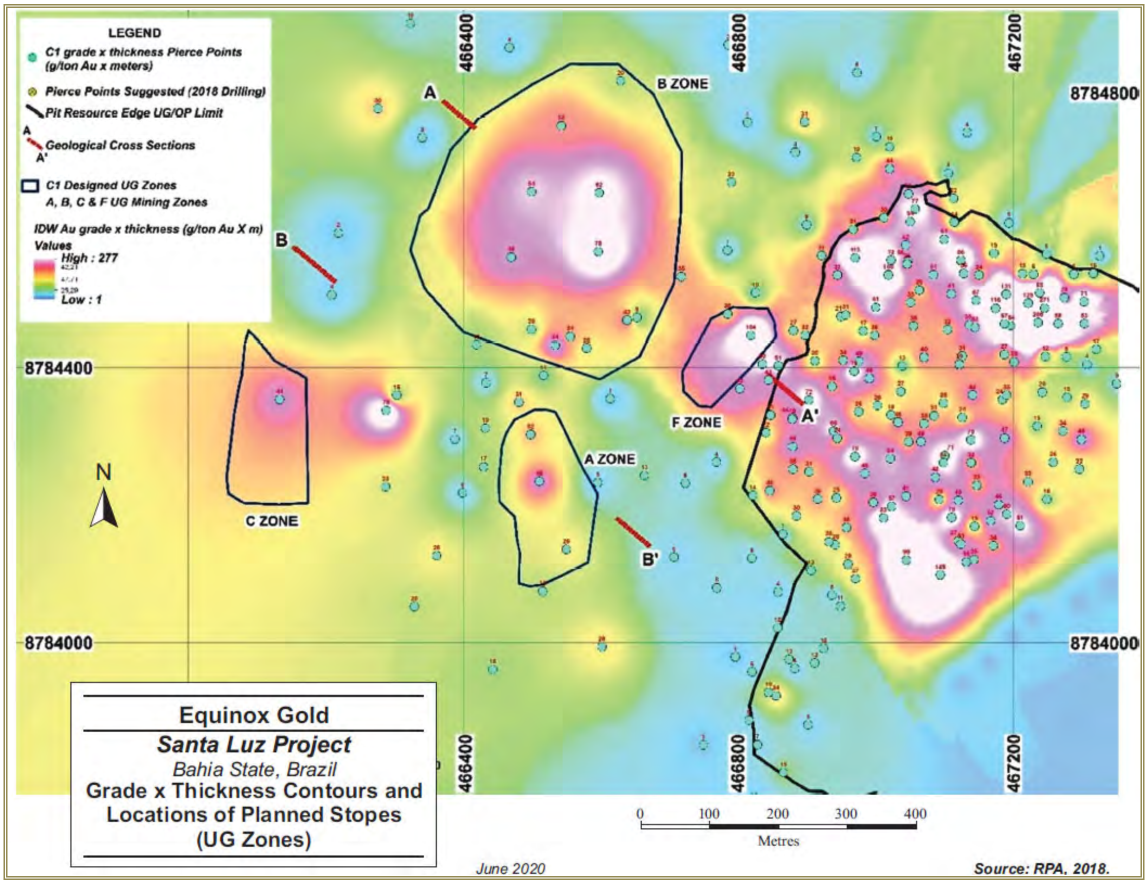


Figure 24-2: C1 Underground Grade x Thickness Contours and Locations of Planned Stopes (Underground Zones)

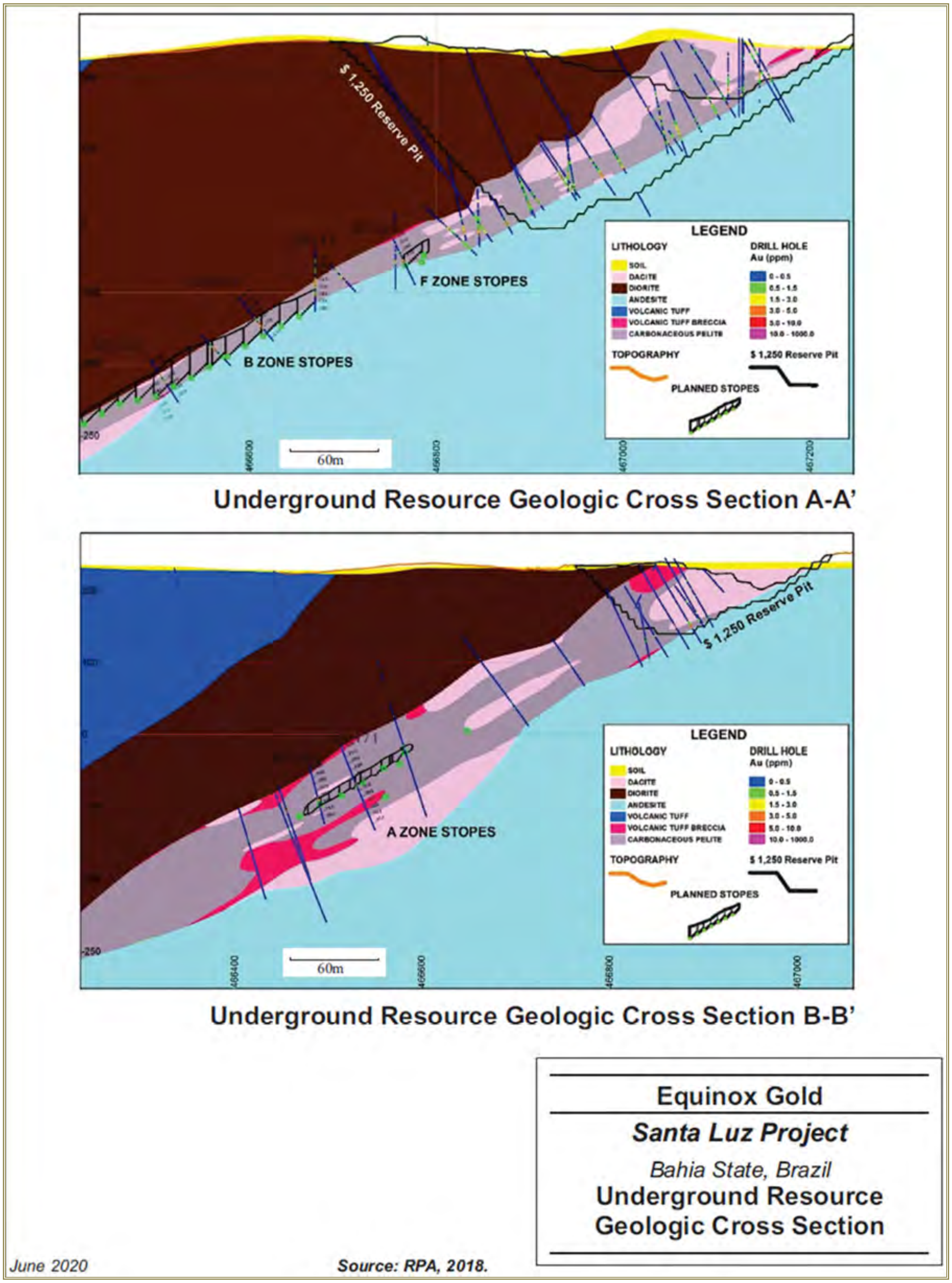


Figure 24-3: C1 Underground Resource Geologic Cross Section A-A' and Cross Section B-B'

24.2.1 Mineral Resource Estimate

The Mineral Resource estimate and the underlying work are described in Section 14, and the C1 Underground Mineral Resources are a portion of the total estimate. The C1 Underground Mineral Resources used in the PEA are summarized in Table 24-3.

Table 24-3: C1 Underground Mineral Resource as of October 22, 2018

Category	Tonnes ('000s)	Grade (g/t Au)	Contained Gold (oz)
Measured	121	1.94	7,561
Indicated	5,913	2.55	484,066
Measured & Indicated	**6,034**	**2.53**	**491,627**
Inferred	6,560	2.19	461,367

Notes: CIM Definition Standards (2014) were followed for Mineral Resources.
Underground Mineral Resources are reported at a cut-off grade of 1.5 g/t Au.
Bulk density of 2.70 t/m^3 used.
No minimum thickness was used in the resource estimation.
Mineral Resources are estimated using a gold price of US$1,500/oz.
Totals may not add due to rounding.

24.3 Identification of Potential Mining Zones

Mineral Reserves have not yet been estimated for the C1 Underground. All production from the C1 Underground Project is based upon expected conversions of Mineral Resources to Mineral Reserves; however, the planned production quantities are based on Mineral Resources only.

The definition of potentially economic underground mining zones was initiated with grade contouring of the portion of the block model located below the planned open pit. This block model was developed for the Open Pit Feasibility Study and is based on 5 m x 5 m x 5 m blocks. As an initial target, grade shells of material exceeding 2.0 g/t Au were generated. From this initial work, four separate zones of >2.0 g/t Au mineralization with good continuity and sufficient thickness (minimum 5 m) were defined. These were identified as the A, B, C, and F-Zones. The four zones were then split along their entire strike length into 25 m wide stoping corridors. Each stoping corridor was then subdivided into 25 m long individual stopes.

These individual stope shapes and tonnage/grades became the basis for stope design and scheduling. Dilution and extraction were estimated to be 10% (at a grade of 1.0 g/t Au) and 95% respectively, resulting in recovery of 97% of the in-situ tonnage and 98% of the contained gold. A more detailed analysis of the dilution from the wall rocks and the backfill along with a review of the mining extraction is recommended as part of any future studies.

In any subsequent phase, evaluation of the current block model will be required to determine the modifications needed (e.g., smaller block size) to better define the deposit for underground mine design purposes.

24.4 Geotechnical Considerations

The most comprehensive geotechnical characterization for the underground mining is presented in Itasca (2006). In designing the long hole mining method for C1 Underground, SLR incorporated the recommendations included in the Itasca report with respect to allowable spans and expected ground support. As a starting point, SLR has assumed that, for all decline and stope development, standard ground support (bolts and mesh) will be sufficient for long-term stability and that shotcrete will, in general, not be required. SLR has also assumed that, with cable bolting, the open stope backs and walls will remain stable through the blasting, extraction, and backfilling phases. Primary and secondary stopes will be 25 m wide, and stopes will be mined 25 m on strike, then backfilled with cemented paste fill.

24.4.1 Geotechnical Drilling

Further drilling campaigns are planned. It is recommended that some specifically oriented geotechnical holes be drilled to identify the joint sets in the hanging wall rocks. These holes should be inclined at approximately 30° to 40° from vertical and drilled with triple tube core barrels using oriented coring techniques. The holes should be drilled dipping north to northwest to ensure intersection of the identified steeply dipping joint set that strikes east–west as well as the flatter southeast dipping set.

24.5 Underground Development

24.5.1 Primary Development

The underground mine will be accessed from a portal to be established in the southwest corner of the open pit. The decline will be driven 5.0 m wide by 5.5 m high, incorporating a roof line back-excavated along the angle of the rock foliation. This heading size is the same as currently used by Equinox for declines at the Fazenda Brasileiro mine.

The main decline will be driven at a maximum gradient of -12%, and will be divided into a number of segments that will be used to connect the surface to the four major zones and the stope access sub-levels within each zone. An isometric view of the zones and primary development is shown in Figure 24-4.

Over the course of the LOM, a total of 6,342 m of decline will be driven to access and extract the four major underground zones extending to a vertical depth of 592 m from the surface portal. For scheduling and costing purposes, the decline measured distances have been increased by 5% to account for miscellaneous additional development (re-muck stations, sumps, storages, ventilation raise accesses) currently not included in the design layout.

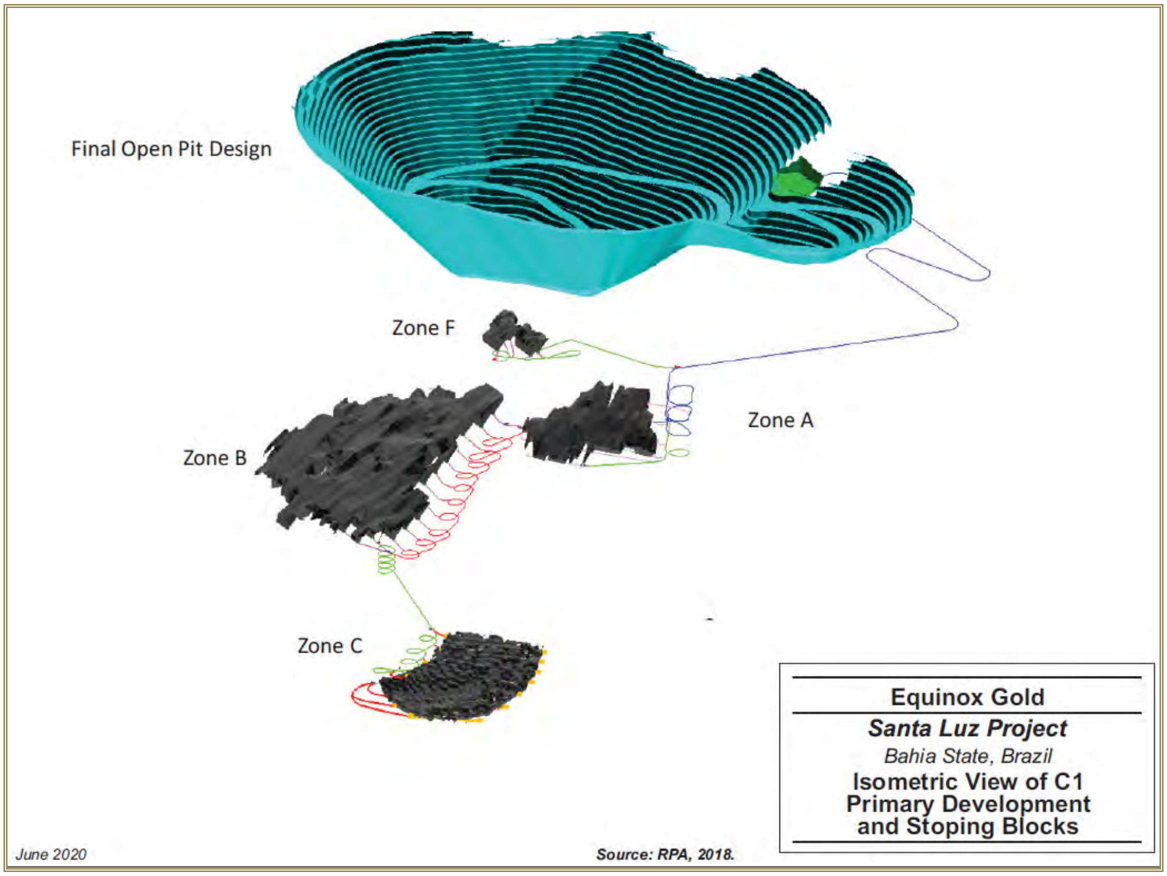


Figure 24-4: Isometric View of C1 Underground Primary Development and Stoping Blocks

The typical advance rate for headings of this size (5.0 m x 5.5 m) with bolt/mesh ground support is one round (3.3 m long) every 10 to 12 hours. For this study, SLR has assumed a development cycle time of 12 hours. Based on SLR's experience and results from other similar operations, this development cycle time is reasonable and achievable. Effective underground work time per day at the Fazenda Brasileiro mine is 18 hours (6 hours per shift, 3 shifts per day). On this basis, the schedule has been prepared using advance rates of 150 m/month. The primary declines are driven on a single face basis. Current single face development productivities (75 m/month) at the Fazenda Brasileiro mine are approximately 50% of the advance rate assumed in the scoping study schedule. Detailed review of the development rate is recommended in the next level of study.

For costing purposes, SLR has assumed that a contractor will be used to excavate the decline from the surface portal to the bottom of the B-Zone. Santa Luz underground crews will drive the other decline segments (F-Zone, A-Zone bottom, and C-Zone). To reduce the initial capital costs and improve the economics, the capital development schedule should be reviewed in more detail to determine if some of the development can be delayed without negative impacts to the stope preparation schedule.

SLR considered a shaft access option for the deposit given that the lowest elevation of the C-Zone was at a depth of 600 m below surface. This depth is often the elevation where the economics of decline access versus shaft access begin to shift in favour of the shaft option. The shaft option was not chosen at this stage; however, given the shallow dipping flat nature of the deposit (long access drives from the shaft) and 75% of the deposit tonnage is located above the 400 m elevation, a shaft option may be reconsidered.

24.5.2 Stope Access Development

Stope access development has been designed to enable direct mobile equipment access to the individual primary and secondary stopes within each zone, and provide for separate intake and exhaust ventilation circuits. The stope drifts will be driven 5 m wide x 5 m high.

When the decline is adjacent to the mineralized zones, each loop of the main decline has been designed to intersect the elevation of the stope lower sill drifts, which are spaced every 25 m horizontally. From the decline, a 50 m long access drift is driven perpendicular to the decline, extending to the end of the zone. This access drift will then be extended along the entire strike of the deposit, and becomes the lower sill for long hole drilling, blasting, and loading. Approximately 25 m from the decline along the access drift, a drift is driven perpendicular to the access drift for a distance of approximately 50 m, then turned 90° and driven an additional 25 m to intersect the zone at the upper sill drift elevation (for the adjacent down-dip primary/secondary stope). This access drift will then be extended along the entire strike of the deposit, and becomes the upper sill for long hole drilling, blasting, and backfilling of the void. Approximately 20 m along the upper sill access drift, a 5 m high by 5 m wide ventilation cross-cut is driven perpendicular to the access drift for a distance of approximately 6 m. A parallel 5 m wide by 5 m high drift is then excavated to access the exhaust raise from the sub-level above and eventually to the raise that will be excavated to the sub-level below. The stope access development is illustrated in Figure 24-5.

Over the course of the LOM, a total of 17,330 m of stope access and stope sill drifts will be driven to access and extract the four major underground zones. Typical advance rates for headings of this size (5 m x 5 m) with bolt/mesh ground support on a multi-faced basis is one round (3.2 m length each) every eight to nine hours. For this study, SLR has assumed a development cycle time of 9 h. Based on SLR's experience and results from other similar operations, this development cycle time is reasonable and achievable. On the basis of an 18 h effective work day as described in the primary development section above, the schedule has been prepared using advance rates of two rounds per day or 200 m per month. The stope accesses and stope sill drifts are driven on a multi-face basis. The current Fazenda Brasileiro mine multi-face development productivities (125 m per month) are approximately 60% of the advance rate assumed in the scoping study schedule. Detailed review of the development rate is recommended in the next level of study.

For costing purposes, SLR has assumed that Santa Luz crews will develop all stope access and stope sill drifts. Two separate crews will be required to meet the development schedule quantities.

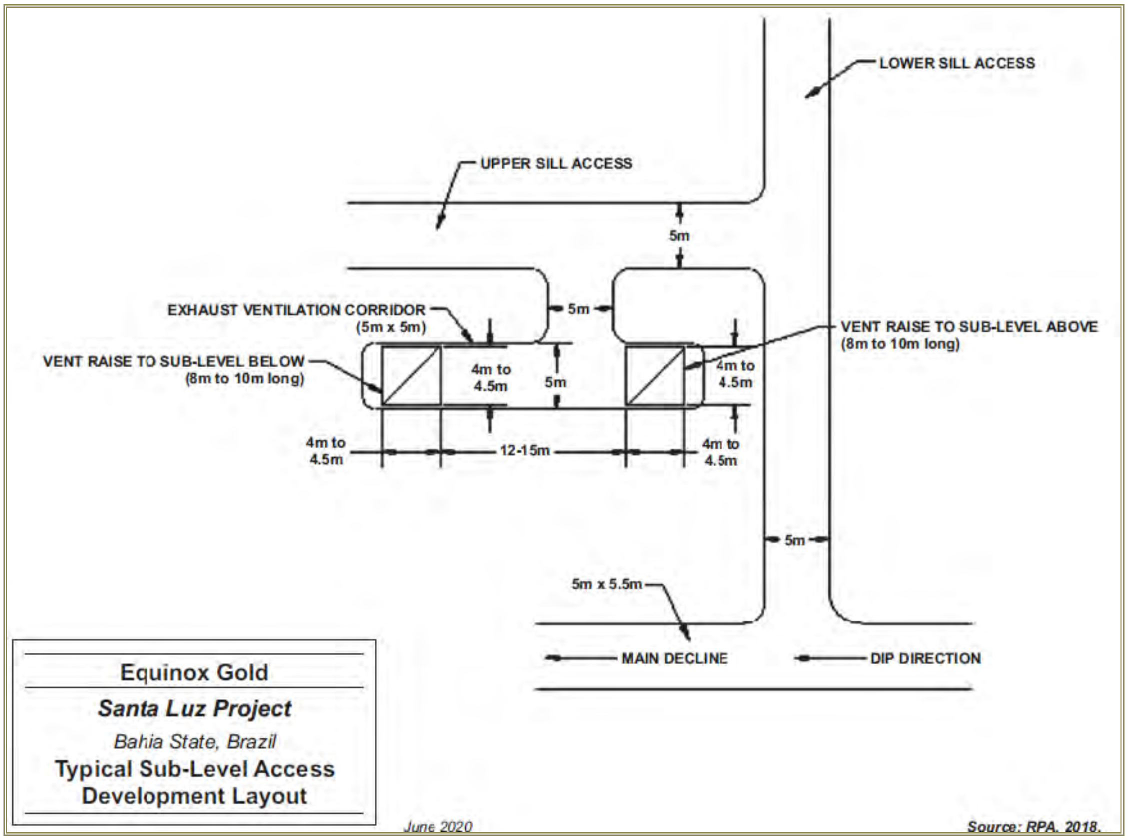


Figure 24-5: C1 Underground Typical Sub-Level Access Development Layout

24.5.3 Stope Design

Mechanized long hole stoping design and productivities are based directly on the Fazenda Brasileiro mine actual designs and performances. It is SLR's assumption that the Fazenda Brasileiro unit stoping operations (drilling, blasting, and loading) design parameters are directly applicable to the C1 Underground and provide the basis for stope layouts and scheduling. The actual Fazenda Brasileiro mine performances already account for equipment use and availability, shift effective hours, typical delays, and labour capabilities. The only major difference between the two operations is that due to the much steeper deposit geometry combined with good ground conditions at the Fazenda Brasileiro mine, backfilling of stopes is not required. With the much shallower dip and extensive expanse of the zones at the C1 Underground, this additional unit operation is needed and must be included in the stope life-cycle calculation.

Long Hole Drilling and Blasting

Long hole drilling and blasting patterns assumed for C1 Underground are identical to those currently in use at the Fazenda Brasileiro mine. This includes drilling rings of 2.5 inches (64 mm) diameter drill holes on a pattern of 1.6 m between rows and 1.8 m between the toes of holes. This pattern results in 2.8 tonnes of ore per metre drilled. The Simba H1254 long hole drills used at the Fazenda Brasileiro mine average 250 m/d/rig thereby preparing 700 t/d/rig of stoping ore. This same productivity is assumed for the C1 Underground.

Long hole drilling/blasting are separated between the lower sill drift and the upper sill drift of each stope. For the thicker stopes (>10 m) prevalent in the B-Zone, the lower sill drift is located at or slightly below the footwall contact along the down-dip stope boundary. All drill holes from this sill are drilled vertically and to the up-dip side of the sill drift. Since the up-hole slot raises needed to initiate blasting of each 25 m long stope must be successfully completed in one attempt, the raises are limited to 15 m long, which is, from SLR's experience, the maximum recommended length for blind up-hole long hole slot raises. The upper sill drift is located higher up in the zone, below the hanging wall contact, along the up-dip stope boundary. All drill holes from this sill are drilled in a semicircular fan pattern towards the down-dip side of the upper sill drift. The downhole slot raises in this case can be any length, since they can be blasted in several lifts. Layout of the >10 m thick secondary stopes is the same as the primary stopes, except that the upper and lower sill drifts have been inset 4 m from the stope boundaries. This is to ensure that the sill drifts are not driven against primary stope backfill. A schematic of the sill positioning and long hole drilling patterns for the >10 m thick primary stopes is illustrated in Figure 24-6. A schematic of the sill positioning and long hole drilling patterns for the >10 m thick secondary stopes is shown in Figure 24-7.

For the thinner stopes (<10 m) prevalent in the A-, C-, and F-zones, the lower sill drift is located at or slightly below the footwall contact along the down-dip stope boundary. All drill holes from this sill are drilled vertically and to the up-dip side extending to the centreline of the 25 m wide stope. The upper sill drift is higher in the zone and is also at or slightly below the footwall contact along the up-dip stope boundary. All drill holes are drilled vertically and to the down-dip side of the upper sill drift extending to the centreline of the stope. A schematic of the sill positioning and long hole drilling patterns for the thinner stopes is illustrated in Figure 24-8. A schematic of the sill positioning and long hole drilling patterns for the thinner secondary stopes is shown in Figure 24-9.

It is anticipated that the stope extraction sequence will involve opening a 1.6 m wide slot along the entire end of the stope. The remaining mineralization would be blasted in two large blasts with the blasting of the lower sill drift rings (undercut) slightly leading the blasting of the upper sill drift rings (overcut). The powder factor at the Fazenda Brasileiro mine is 1.0 kg of explosives per tonne, and the same is assumed for the C1 Underground.

SLR cautions that some holes in the proposed drill rings for the thicker stopes (>10 m) are up to 25 m long. High drilling accuracy will be needed to ensure complete extraction of the ore and to minimize the risk of benching. Other drill patterns should be evaluated in the future. For the thinner stopes (<10 m), the long hole drilling lengths are lower (15 m or less) so there is less risk of benching.

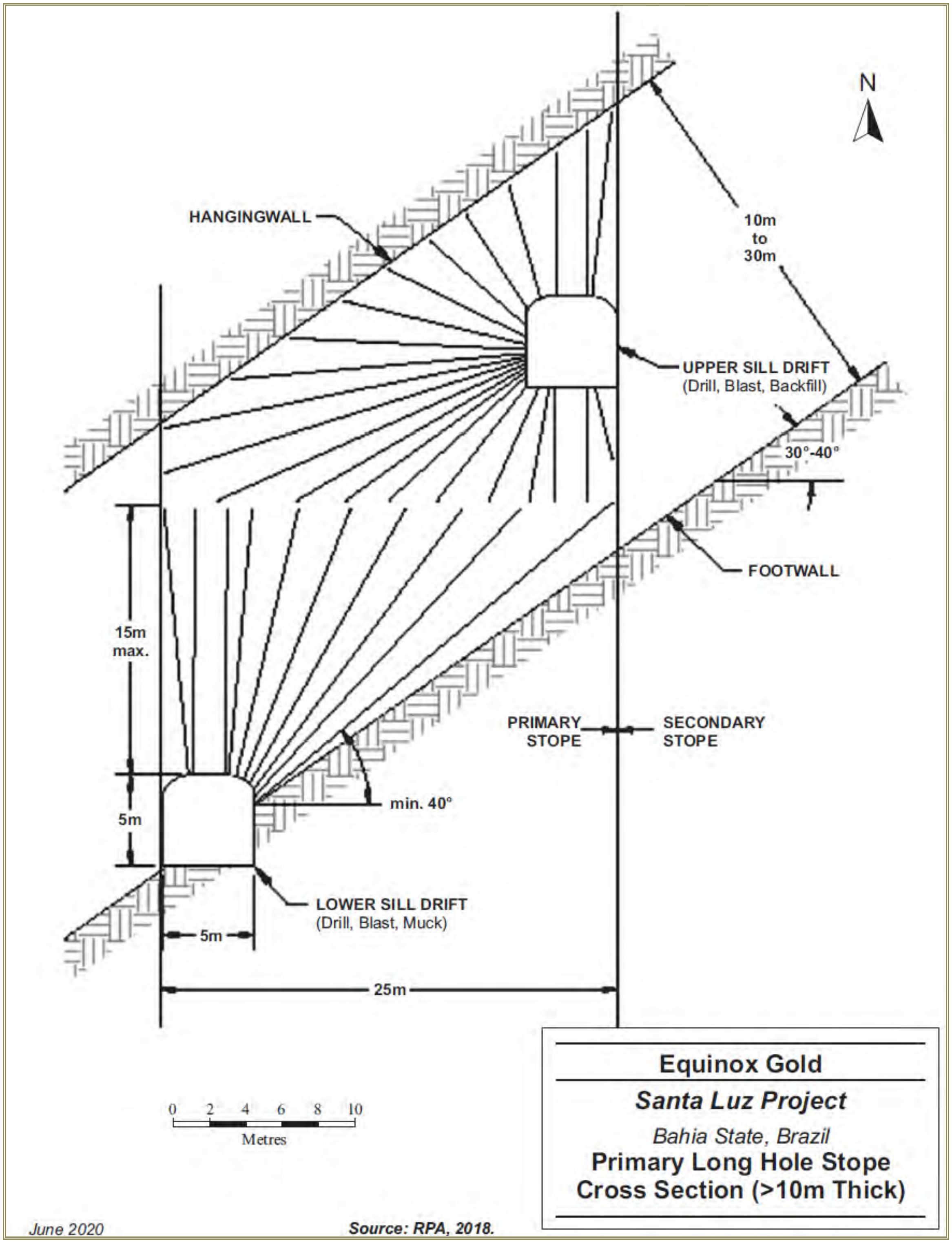


Figure 24-6: Primary Long Hole Stope Cross Section (>10 m Thick)

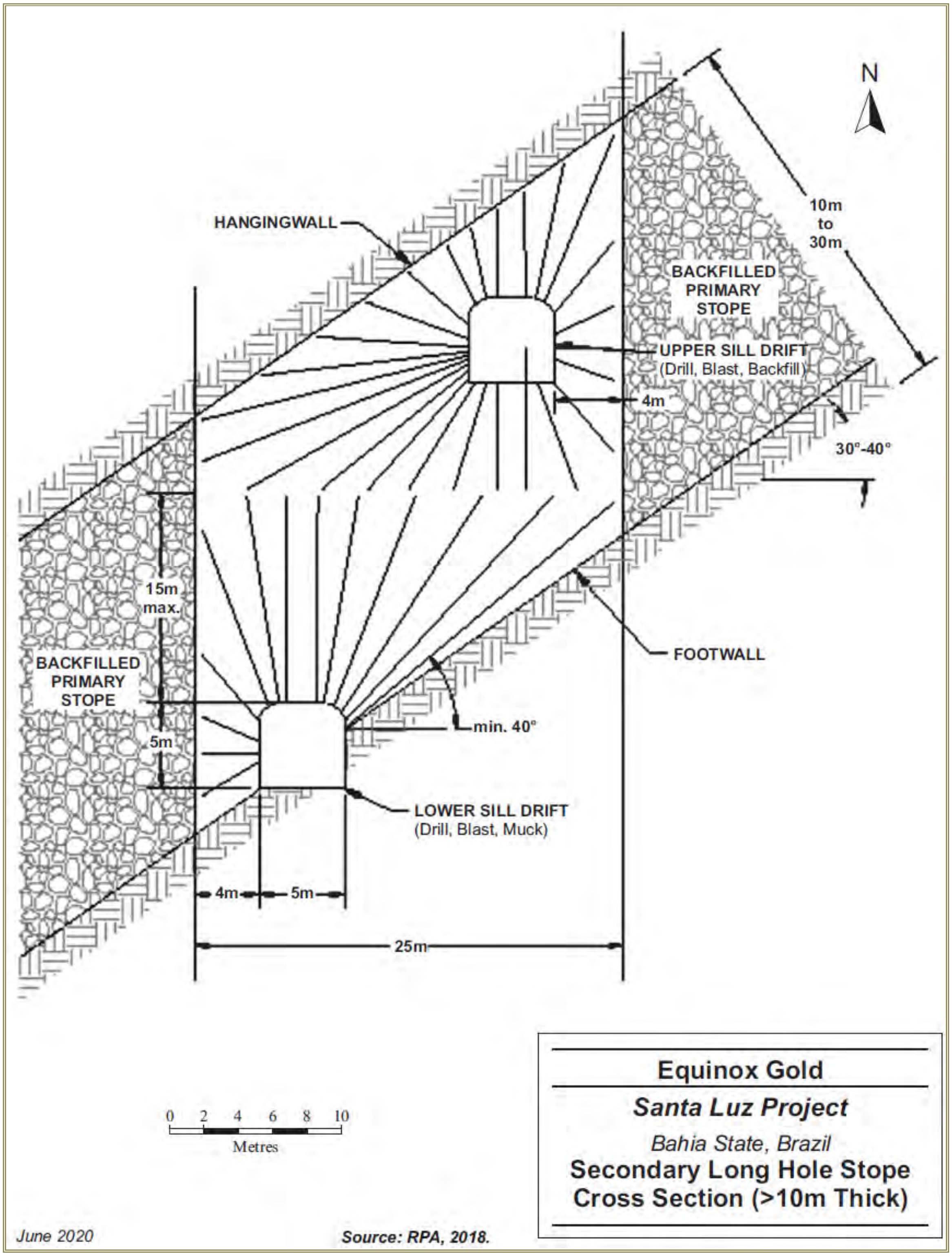


Figure 24-7: Secondary Long Hole Stope Cross Section (>10 m Thick)

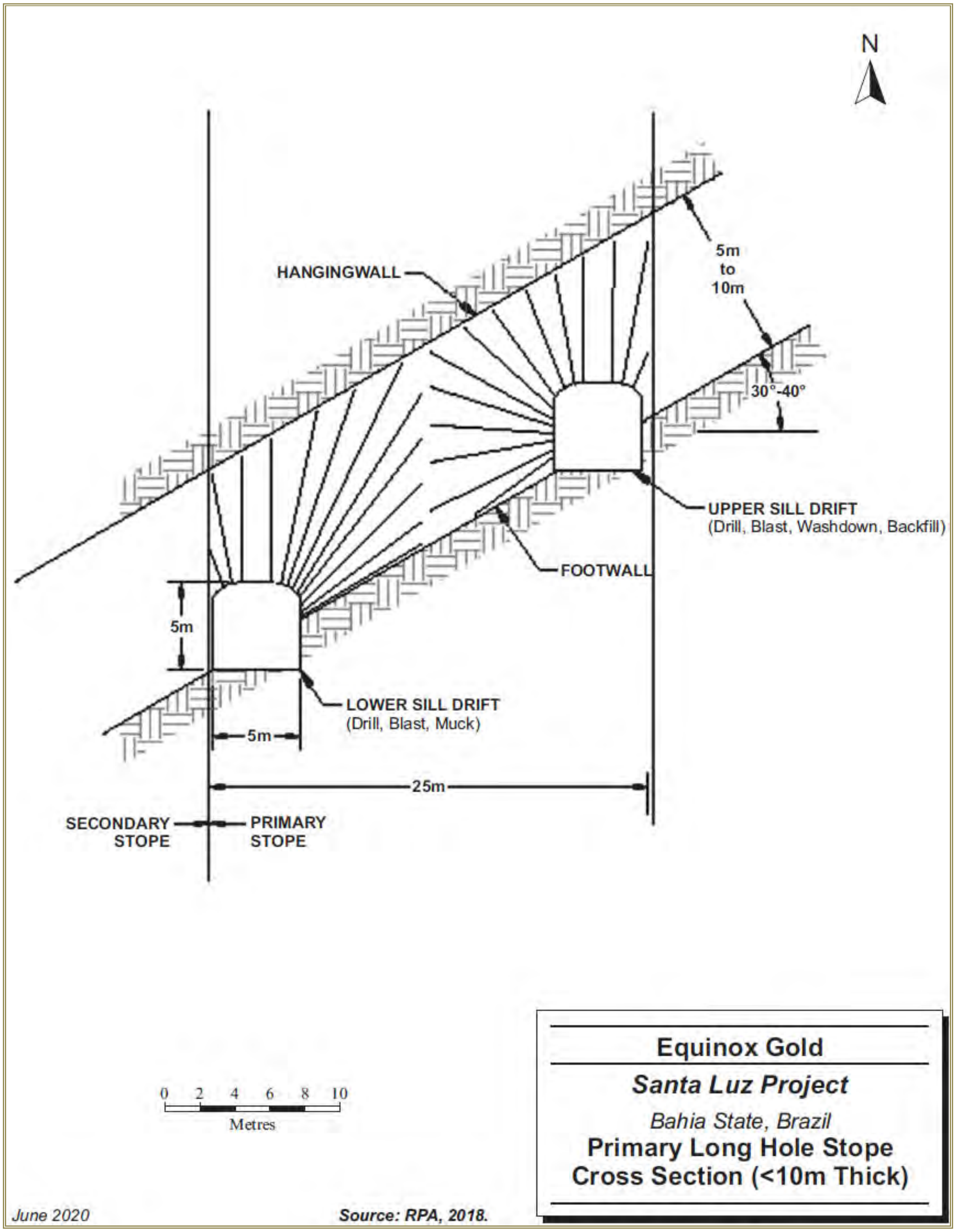


Figure 24-8: Primary Long Hole Stope Cross Section (<10 m Thick)

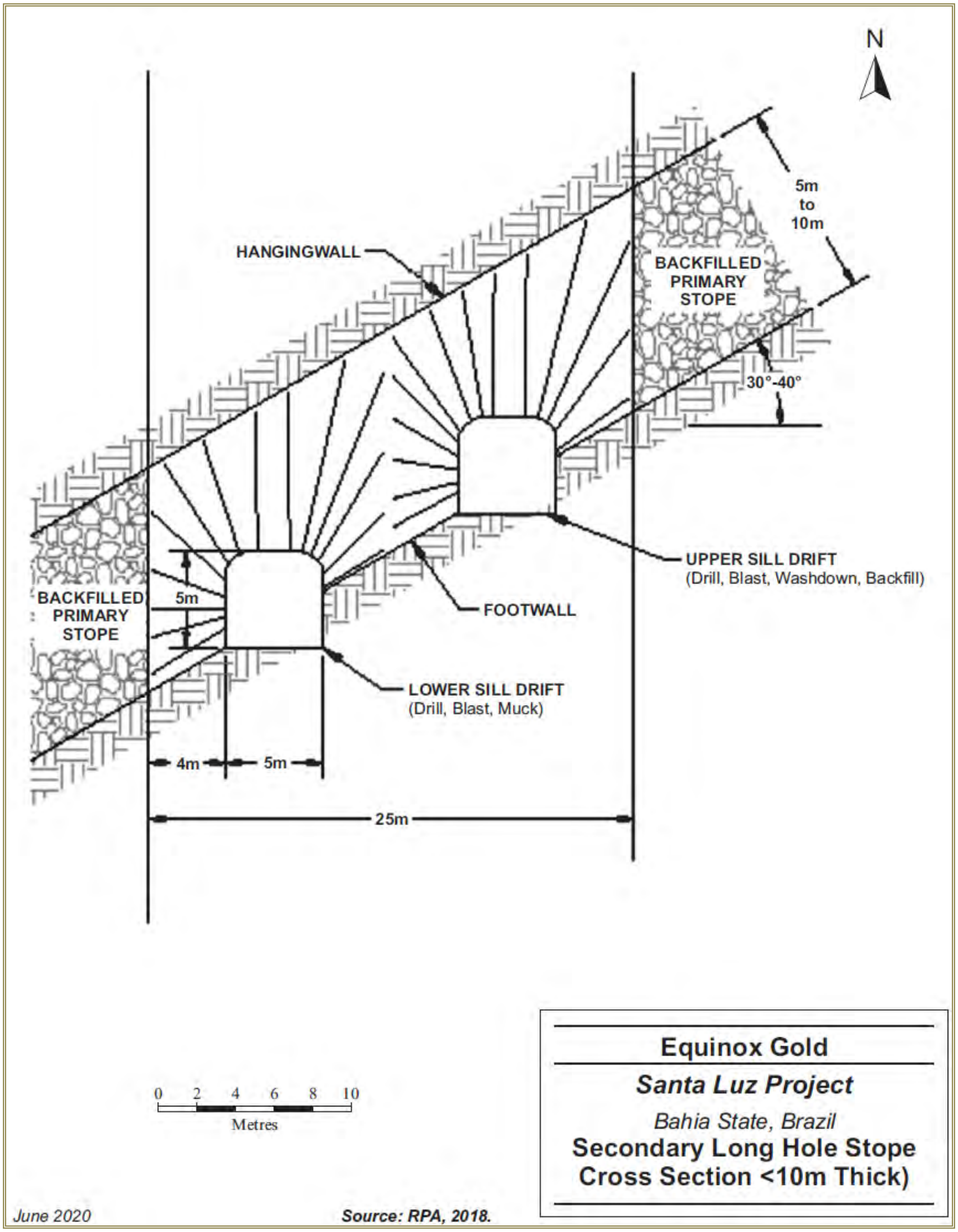


Figure 24-9: Secondary Long Hole Stope Cross Section (<10 m Thick)

Loading and Haulage

Stope loading rates are based on actual Fazenda Brasileiro results using Caterpillar R1700G 7.5 yd^3 load-haul-dump units (LHD) with remote control capabilities. It is SLR's assumption that half the loading will be undertaken under manual control (71 t/h) when the stope brow is filled; half the loading will occur when the stope brow is open, requiring use of the unit's remote-control capabilities (50 t/h). Stope loading is estimated to average 1,080 t/d/LHD unit based on 18 h/d of effective underground shift time at an average of 60 t/h. The maximum LHD haulage distance is 300 m but will average in the 100 m to 150 m range. This is well within the unit's efficient operating range. Trucks will be loaded in the access drift near the main decline; at this location, access drift height will be increased to allow for direct loading.

Truck haulage capacities are also based on the Fazenda Brasileiro mine actual results. Using Scania P420 underground trucks, the Fazenda Brasileiro mine averages 450 t/d/unit (25 tph) at an average haul distance of 4,000 m. As shown in Table 24-4, the LOM average haul distance is slightly lower, at 3,200 m, with most of the mill feed obtained from the B-Zone.

Table 24-4: C1 Underground Truck Haulage Distances

Zone	Average Haul Distance (m)	Tonnes Hauled (t '000s)	% Total Tonnes Hauled
A	2,400	650	9.1
B	3,100	5,348	75.0
C	4,600	864	12.1
F	2,100	269	3.8
Total	**3,200 (Weighted Average)**	**7,131**	**100**

Backfilling

The paste backfill plant is normally designed to prepare a 250 mm (10" slump) backfill using milled tailings. The paste backfill plant is developed around the dewatering of tailings, then adding cement before the mixture is pumped to the stopes. The paste backfill system can be summarized as follows:

- Thickened tailings are pumped from the tailings thickener underflow to an agitated filter feed storage tank.
- From the filter feed-storage tank, the thickened tailings can be split, with a large portion directed to a vacuum disc filters, and a small portion sent directly to a conditioning mixer for slump adjustment.
- Filter cake from the vacuum disc filters is conveyed to a conditioning mixer, while the filtrate is directed to a seal water sump and back to the thickener feed box. The filter drain and clean-up of the paste backfill plant area are collected in a separate sump, and also directed back into the thickener feed box.
- The continuous mixer discharge and normal Portland cement are weighed and fed into a batch mixer to which process water is added so that the mixer discharge is of a consistent paste slump.

- Normal Portland cement will be delivered to site, more than likely by truck, and discharged into one large-capacity silo designed to offer capacity for several days of continuous operation at the highest binder consumption-rate.
- The discharge of paste from the batch mixer will be pumped underground via a surface pipeline connecting to multiple holes drilled to intersect the main decline and each of the major mining zones. Pipelines will direct the backfill from the underground collection points to the stope backfill drifts.
- The bottom stope sill accesses will be barricaded using mesh and waste rock. Paste will be directed to the upper sill of each stope and free poured until the floor elevation of the upper sill is reached. Backfill will then be pumped through drill holes/breather holes to reach the highest points of the stope void.
- The backfill plant will be designed to prepare 1,875 t/d of backfill which, due to the different densities, is equivalent to the quantity of tailings generated from the processing of 2,500 t/d of ore. This will allow for backfill to be placed at a continuous rate of 80 t/h.

Stope Life Cycle

Stope cycles have been developed for every defined underground long hole stope. The unit operations include, in sequence, long hole drilling, stope blasting and loading, backfilling, and backfill curing. The time required for these four major activities is totaled, and a factor of 10% is added to account for delays. This calculation defines the stope cycle for each specific underground long hole stope. The process is as follows:

1. From the stope designs, sill development tonnage is removed, leaving available long hole stoping tonnage. This tonnage is divided by 2.8 to calculate the metres of long hole drilling required. The metres of long hole drilling are then divided by 250 to calculate the total number of long hole drilling days.
2. The long hole stoping tonnage is divided by 1,080 to calculate the total number of blasting and loading days.
3. The long hole stoping tonnage is divided by 1,875 to calculate the total number of paste filling days.
4. A final cure time of seven days is added to the stope cycle times.
5. Since long hole drilling can be re-initiated prior to the end of backfill curing, the total long hole drilling days are reduced by the final cure time.
6. A 10% allowance is added to the stope cycle total to account for miscellaneous delays.

Average stope production is estimated at approximately 9,000 t/month and cycle times for the smaller A-, C-, and F-Zone stopes average one to two months (9,000 to 18,000 t/stope). Cycle times in the larger B-Zone stopes range between two and six months (18,000 to 54,000 t/stope). To maintain the desired production rate, a total of seven operating stopes are required. At any one time these stopes will be either in the drilling–blasting/loading–backfilling, or curing phases. Should unexpected difficulties occur in one of the seven active stopes, other workplaces will be available, as sill development is completed well ahead of mining.

24.5.4 Ventilation

SRK developed raise and shaft sizing using the VentSim Visual software package based on drift dimensions and mine projection maps provided by RPA. Models were constructed to represent four ventilation stages in the LOM. These models were used to determine fan duties and provide recommendations based on the results.

Airflow requirements in the mine are based on the vehicle motor power (kW) use, and recommended factor of 0.06 $m^3/s/kW$ of engine power, to ensure safe working conditions in areas with diesel equipment and to meet or exceed Brazilian regulations.

Primary ventilation circuits are achieved with a single surface-fan installation operating at approximately 1120 kW (assuming 75% efficiency). The fan installation will need to apply a pressure of 3.8 kPa to the system and have a total flow of 235 m^3/s. This accounts for maximum-demand ventilation needs. The main fan installation is assumed to be placed at the collar of the return shaft on surface.

24.5.5 Life-of-Mine Development and Production Schedule

The mine scheduling strategy involves developing the main decline with a contractor from surface to the B-Zone, the largest of the four mining zones, followed by excavation of the stope access ramps. Single face development is scheduled at 150 m of advance per month. As soon as faces are available, a Santa Luz development crew will begin excavating the stope accesses and primary stope sills in sequence from the top of the zone down dip until the bottom is reached. Multiple face development is scheduled at 200 m of advance per month. Mining of the primary stopes is initiated as soon as development is completed.

Secondary stope sills are then excavated in sequence from bottom up to mirror the secondary stope mining sequence. While development and mining of the B-Zone is occurring, a second Santa Luz development crew is preparing the F-, A-, and C-zones in sequence as supplements to the B-Zone production. At full production, seven stopes will be in production: five from the B-Zone and two from the F-, A-, or C-zones. A summary project schedule is shown in Figure 24-10.

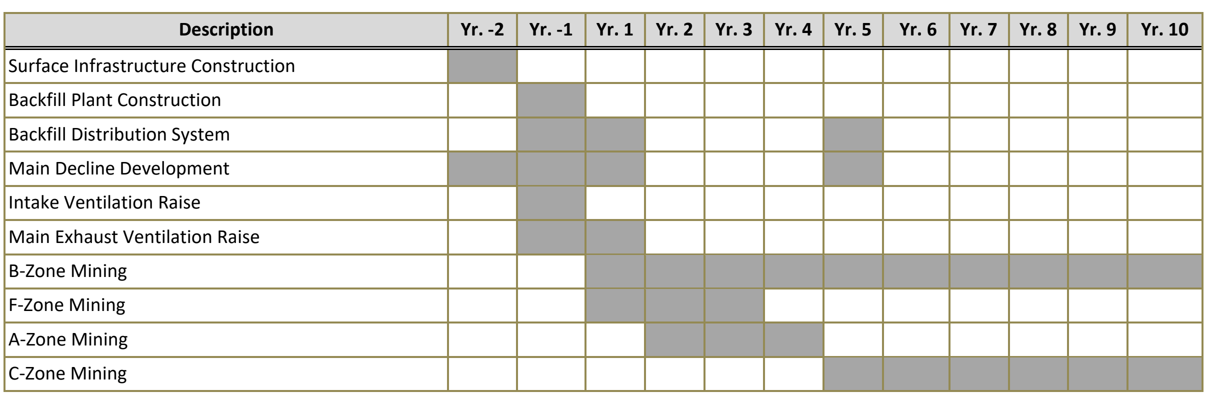


Figure 24-10: C1 Underground Summary LOM Schedule

The general stoping sequence is illustrated in Figure 24-11 and Figure 24-12.

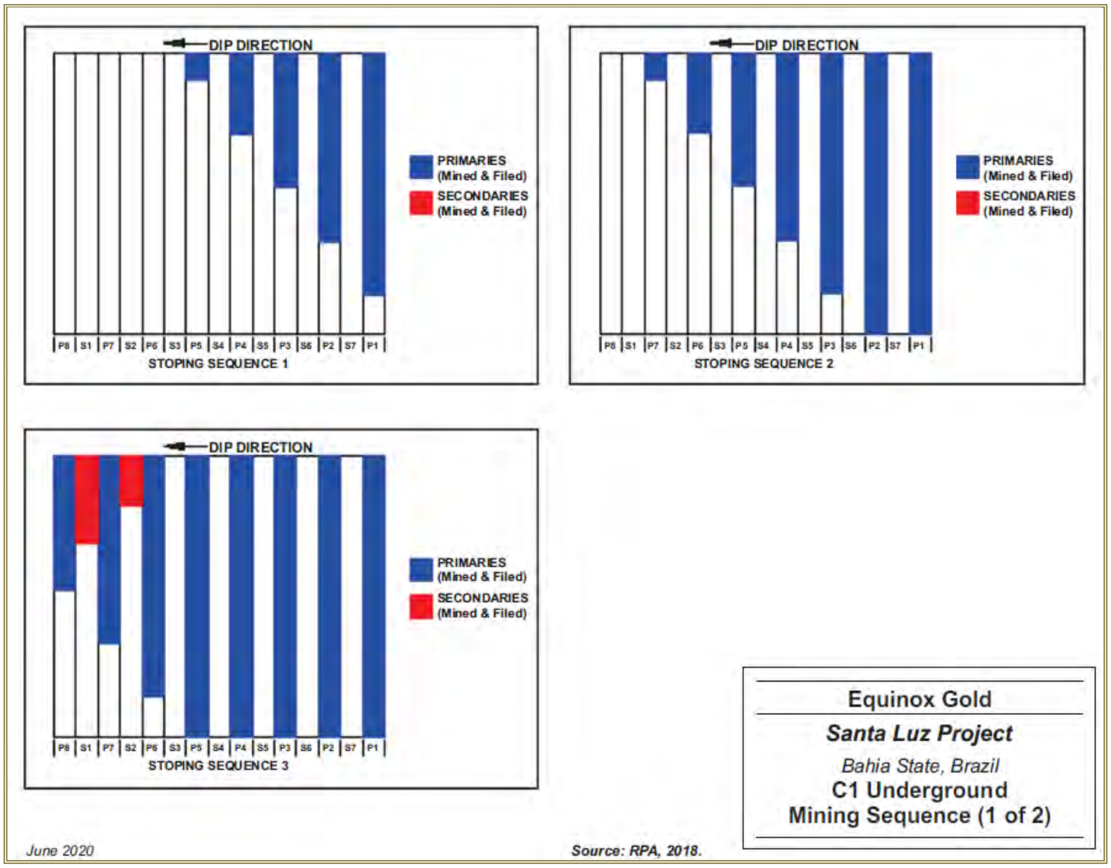


Figure 24-11: C1 Underground Mining Sequence (1 of 2)

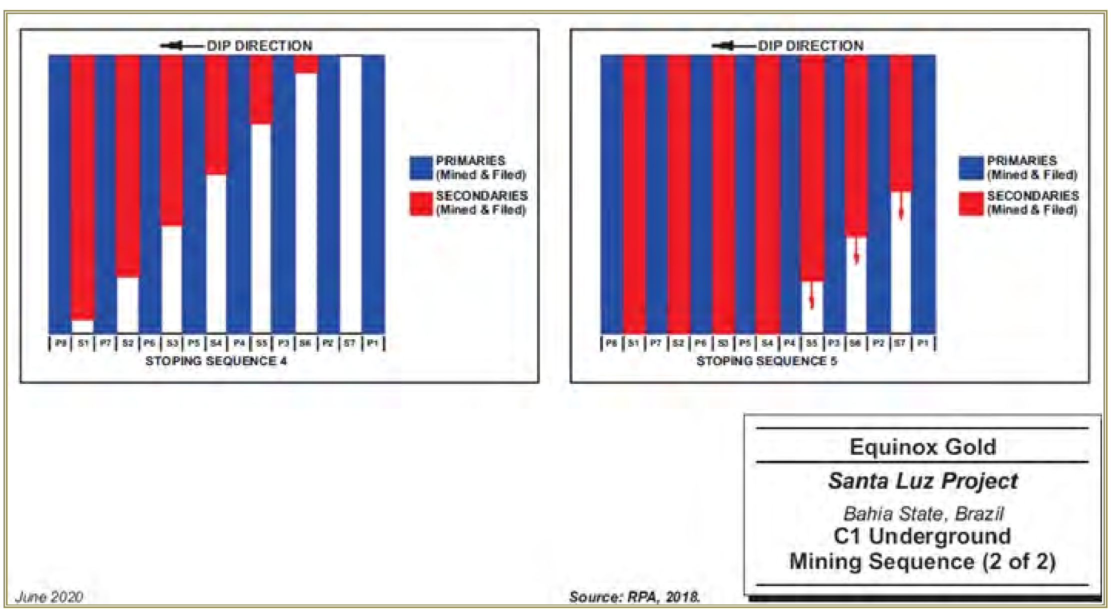


Figure 24-12: C1 Underground Mining Sequence (2 of 2)

The LOM schedule was prepared on a monthly and crew-by-crew basis, and encompasses the period from the expected start of construction in Year −2, the start of stope mining two years later in Year 1, and spans until stoping is completed, 9.5 years later in Year 10. The basis for this schedule is the total potentially mineable underground Measured, Indicated, and Inferred Resources of 7.1 Mt grading 2.65 g/t Au.

An annual summary of the schedule metrics is presented in Table 24-5.

Table 24-5: Annual C1 Underground Mine Production Schedule

	Unit	Total	Pre-Production		Operations									
			Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Primary Decline Development (waste) Contractor)	m	3,857	1,275	1,975	607	0	0	0	0	0	0	0	0	0
Primary Decline Development (waste)	m	2,830	0	726	786	400	0	150	768	0	0	0	0	0
Stope Access Development (waste)	m	4,175	0	299	976	750	325	375	650	750	50	0	0	0
Sill Development (ore)	m	13,155	0	250	2,147	3,273	1,335	2,025	2,275	1,650	200	0	0	0
Total Development	**m**	**24,017**	**1,275**	**3,250**	**4,516**	**4,423**	**1,660**	**2,550**	**3,693**	**2,400**	**250**	**0**	**0**	**0**
Ore Produced – Zone A	tonnes	650,448	0	0	2,498	287,952	189,300	170,698	0	0	0	0	0	0
Ore Produced – Zone A	grade	2.81	-	-	2.45	2.87	2.83	2.68	-	-	-	-	-	-
Ore Produced – Zone A	Au oz	58,741	0	0	197	26,616	17,221	14,707	0	0	0	0	0	0
Ore Produced – Zone B	tonnes	5,348,108	-	13,500	327,523	544,011	644,086	728,079	758,948	690,401	689,677	632,496	261,636	68,751
Ore Produced – Zone B	grade	2.59	-	3.09	2.36	2.30	2.72	2.73	2.71	2.48	2.56	2.72	2.38	2.58
Ore Produced – Zone B	Au oz	444,611	0	1,342	24,819	40,263	55,386	64,000	66,086	55,082	56,685	55,236	20,001	5,712
Ore Produced – Zone C	tonnes	864,411	0	0	0	0	0	0	115,849	215,926	120,893	137,400	213,696	60,647
Ore Produced – Zone C	grade	3,04	-	-	-	-	-	-	2.77	2.93	3.23	2.99	3.25	2.93
Ore Produced – Zone C	Au oz	84,483	0	0	0	0	0	0	10,331	20,363	12,540	13,206	22,326	5,717
Ore Produced – Zone F	tonnes	268,540	0	3,375	134,453	104,272	26,440	0	0	0	0	0	0	0
Ore Produced – Zone F	grade	2.35	-	2.11	2.48	2.35	1.78	-	-	-	-	-	-	-
Ore Produced – Zone F	Au oz	20,313	0	229	10,700	7,874	1,511	0	0	0	0	0	0	0
Total Ore Produced	**tonnes**	**7,131,507**	**0**	**16,875**	**464,474**	**936,235**	**848,827**	**898,777**	**874,797**	**906,327**	**810,570**	**769,895**	**475,332**	**129,397**
Daily Ore Production	t/d	1,981	0	47	1,290	2,601	2,358	2,497	2,430	2,518	2,252	2,139	1,320	359
Ore Grade	g/t Au	2.652		2.895	2.392	2.483	2.716	2.724	2.717	2.589	2.656	2.765	2.770	2.747
Total Ounces Gold Contained	**oz**	**608,148**	**0**	**1,571**	**35,715**	**74,753**	**74,118**	**78,708**	**76,416**	**75,445**	**69,225**	**68,442**	**42,327**	**11,429**
Waste Produced from Development	tonnes	778,322	38,981	141,075	258,937	120,980	13,500	25,313	110,349	55,688	13,500	0	0	0
Total Haulage – Waste + Ore	tonnes	7,909,829	38,981	157,950	723,411	1,057,216	862,327	924,089	985,146	962,014	824,070	769,895	475,332	129,397
Daily Haulage – Waste + Ore	t/d	1,897	108	439	2,009	2,937	2,395	2,567	2,737	2,672	2,289	2,139	1,320	359

24.6 Infrastructure

The general surface layout with a projection of the underground development is shown in Figure 24-13.

24.6.1 Mine Offices, Mine Dry, Shops, and Warehouse

SLR has allocated a total of US$1.7 million for constructing a surface building to house mine offices, a dry/change house, warehouse, and mobile equipment maintenance shop. These will be of simple construction, and open-air where possible. The exact location for the facilities remains to be determined (near portal or near existing infrastructure), and SLR notes that a specific evaluation needs to be undertaken in the next stage of study that included the possibility of sharing or expanding existing facilities instead of constructing separate facilities for the underground operation.

SLR has allocated an additional total of US$0.4 million for construction of an access road to the decline portal, and excavation and construction of the portal itself.

24.6.2 Electrical Power

Electrical power feed from the national grid as well as transformation/distribution infrastructure is available on site. SLR assumes that the current main electrical system is of sufficient capacity to handle the existing open pit and mill facilities as well as the future underground requirements. Power will be fed to the main decline via a substation to be constructed near the portal. Electrical feed to the surface ventilation fans is assumed to be from the existing surface distribution network.

SLR has allocated a total of US$1.0 million for construction of an electrical sub-station near the mine portal to provide electrical power to the underground operations. The cost of electrical supply to the two surface ventilation fans is also included in the allowance. An additional US$0.5 million has been included for underground sub-stations and distribution panels to be installed during pre-production.

Electrical loads have been estimated from the underground development and stoping unit activities and the underground equipment fleet. Electrical demand averages approximately 15.1 MkW/a (41600 kWh/d) peaking at 23.0 M kW/a (63300 kWh/d).

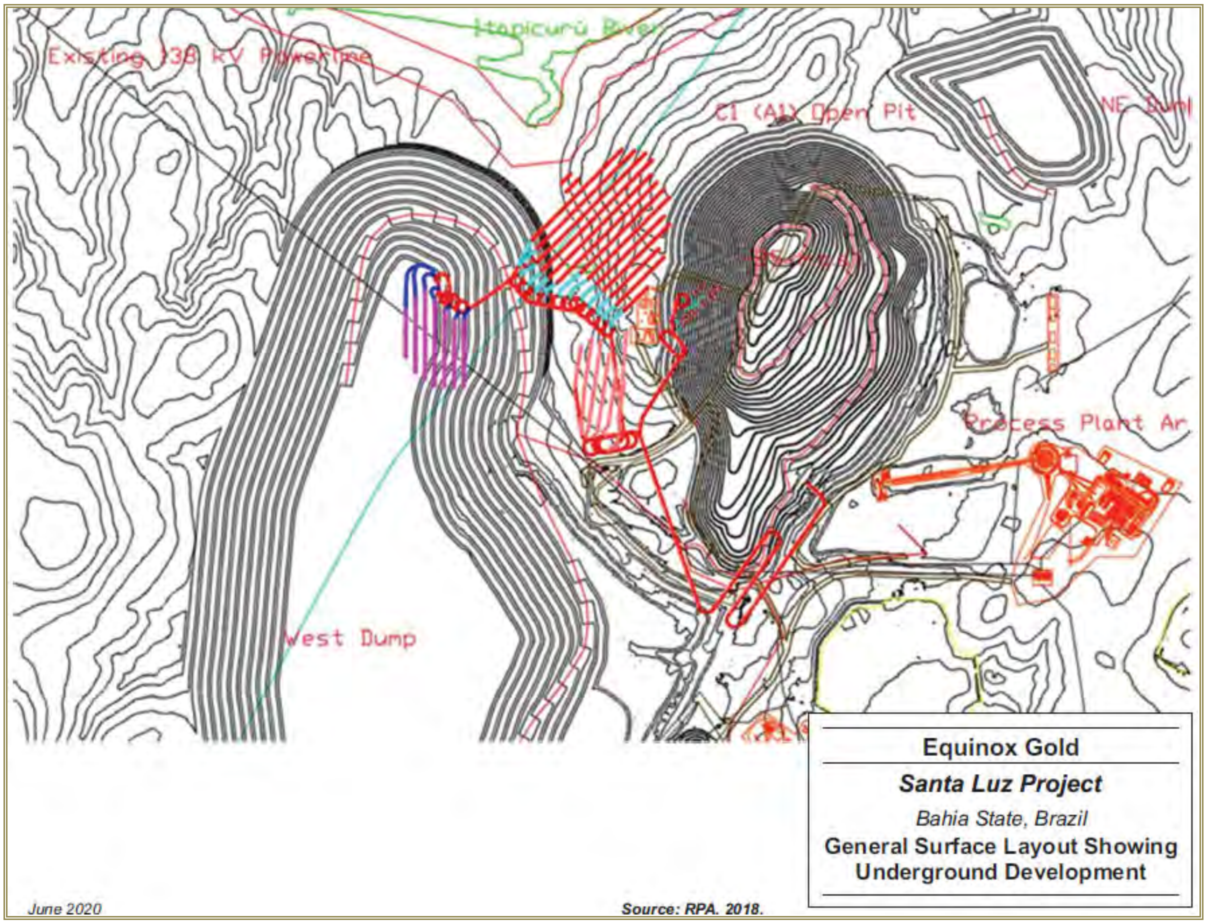


Figure 24-13: General Surface Layout Showing the C1 Underground Development

24.6.3 Pumping and Mine Water Supply

As is generally observed in this area of Brazil, the hydraulic conductivity of the bedrock will reduce with depth. In addition, as less rock is disturbed during underground operations, the drawdown impact tends to be elongated and narrow. Therefore, an increase in groundwater inflow is not expected.

The primary pumps would be mono-type pumps that employ screw action to draw fluid through an inlet port. Cavity displacement action then pushes the fluid forward until it exits through an outlet pipe. The secondary pumps would be electrically powered submersible pumps situated in the development headings. It has been assumed that simple sumps would be established throughout the mine to collect groundwater and mine service-water used for drill hole flushing during drilling, dust control, washing, and other general underground needs. The water would be pumped to surface to a two-compartment settling sump. From that point, decanted water would be returned underground for service water. Any excess water would be directed to the mill or tailings pond.

SLR has allocated a total of US$0.7 million for constructing the surface settling pond and purchasing the primary and secondary pumps capable of dewatering the maximum expected mine inflow of approximately 500 USgpm. SLR assumes that there will not be any until stoping is completed water inflows into the mine but notes that a detailed water balance for the underground mine needs to be undertaken in the next stage of study.

24.6.4 Refuge Stations/Escapeways

Once the B-Zone main exhaust raise is in operation, the A-Zone raise will be converted from exhaust to intake. Ladders, previously installed in the raise after boring, will serve as a second egress from the mine. Given the length of the raise (300 m), the installation of a suitable escape hoist should also be considered. Permanent and portable refuge stations will be installed at appropriate locations near the deeper B-Zone and C-Zone.

SLR has allocated a total of US$0.3 million for purchasing and installing underground refuge chambers.

24.7 Capital Costs

LOM capital costs are estimated at US$98.3 million, an average of US$13.78/t milled as detailed in Table 24-6.

Table 24-6: C1 Underground Life-of-Mine Estimated Capital Costs

Description	Total Cost (US$ '000s)	Unit Cost (US$/t)
Capital Development	12,381	1.74
Paste Fill Plant and Surface Distribution	13,250	1.86
Mobile Equipment	14,252	2.00
Surface and UG Infrastructure	10,000	1.40
Subtotal	49,883	7.00
EPCM/Owner's/Indirect (10%)	4,988	0.70
Subtotal	54,872	7.70
Contingency (35%)	19,205	2.69
Subtotal	74,077	10.39
Sustaining—Primary Development	4,358	0.61
Sustaining—Equipment	10,867	1.52
Sustaining—Other	8,000	1.12
Closure & Reclamation	1,000	0.14
Total	**98,302**	**13.78**

24.7.1 Capital Development

This category includes the anticipated costs (labour, material, equipment) of excavating all of the main, mining zone access declines and the main ventilation raises from surface. This category also includes the anticipated costs (labour, material, equipment) of excavating the F-Zone and A-Zone declines and all stope access and stope sill development completed during the pre-production phase. Development of the main decline and B-Zone decline are scheduled to be excavated by contractor at an all-inclusive unit rate of US$2,500/m. This rate reflects actual Equinox development contractor unit costs equivalent to 1.8 times in-house unit development costs. The pre-production stope development costs are scheduled to be excavated by in-house Santa Luz crews at a unit rate of

US$1,350/m, which is based on 2016 actual Fazenda Brasileiro operating development costs of US$1,342/m. Stope development headings at Fazenda Brasileiro are of similar dimensions and use equivalent mobile equipment units as anticipated for the C1 Underground. Total capital development costs were calculated by applying the unit development costs (contractor or in-house) to the scheduled capital development metres excavated.

24.7.2 Paste Fill

Paste fill plant and distribution system construction costs were estimated for the C1 Underground at US$13.25 million, as detailed in Table 24-7. This estimate was based on a previous RPA (2012) study that anticipated a similar sized paste backfill plant operating at a similar mine depth and distribution distances, as well as equivalent surface pumping distances. The previous study's capital cost estimate was modified to reflect reduced building enclosure construction costs, as relatively dry, temperate weather conditions allow for open-air processing facilities in this area of Brazil.

Table 24-7: C1 Underground Paste Fill Plant Estimated Capital Costs

Description	Cost (US$)
Building and Foundations	5,500
Process Mechanical	3,000
Paste Distribution	1,500
Construction Indirects	3,250
Total Capital Costs	**13,250**

24.7.3 Mobile Equipment

LOM mobile equipment costs are estimated to total US$21.1 million as detailed in Table 24-8. Of this total, pre-production purchases are forecast at US$14.3 million, and sustaining capital purchases are forecast to total US$6.9 million.

Table 24-8: C1 Underground Mobile Equipment Estimated Capital Costs

Description	Unit	Unit Cost (US$ '000s)	Total Cost (US$ '000s)
Development			
Axera DD320 Jumbo	2	830	1,660
Caterpillar R1700G LHD	2	700	1,400
Getman S3120 Scaler	1	687	687
Roofbolter	1	700	700
Getman A64 ANFO Carrier	1	459	459
Production			
Sandvik DL421 Long hole	4	913	3,652
Caterpillar R1700G LHD (RC)	3	700	2,100
Cablebolter	1	700	700
Getman A64 Emulsion Carrier	1	459	459

Description	Unit	Unit Cost (US$ '000s)	Total Cost (US$ '000s)
Scania P420 Haulage Truck	7	541	3,789
Spares			
Caterpillar R1700G LHD	1	700	700
Caterpillar R1700G LHD (RC)	1	700	700
Scania P420 Haulage Truck	2	541	1,083
Service			
Materials Handler with Boom	1	97	97
Boom Truck	1	311	311
Maintenance Utility Truck (Lube)	1	365	365
Maintenance Utility Truck (Fuel)	1	365	365
Paste Fill Utility Vehicle	1	311	311
Electrical Utility Vehicle	1	183	183
Supervision and General Utility Vehicle	3	22	66
Engineering/Geology Utility Vehicle	2	22	44
Surveyor Utility Vehicle	1	168	168
Personnel Transport Vehicle	3	282	845
Grader	1	277	277
Total	**43**		**21,119**

24.7.4 Surface Infrastructure

An allowance of US$10.0 million has been included to cover surface infrastructure construction. This includes constructing a mine office, dry, warehouse, and equipment maintenance shop complex; constructing an access road to the decline portal; excavating and constructing the portal itself; supplying electrical power, compressed air and water to the decline portal; and constructing a settling pond for mine water discharge. Purchasing and installing a ventilation system (fans and motors, installation, ducting, instrumentation), underground refuge stations, and the mine water pumping system are also included. The estimate is shown in more detail in Table 24-9.

Table 24-9: C1 Underground Surface and Underground Infrastructure Estimated Capital Costs

Description	Total (US$ M)
Mine Offices, Mine Dry, Shops, and Warehouse	1.7
Access Road and Portal	0.4
Electrical Distribution	1.5
Pumping and Water Supply	0.7
Primary and Secondary Ventilation System	4.9
Underground Refuge Stations	0.3
Other	0.5
Total	**10.0**

There is no provision of any capital for the process plant.

24.7.5 EPCM/Owner's/Indirect

An allowance equal to 10% of pre-production direct capital costs has been included to cover typical Owner Project costs including engineering, infrastructure construction and mine development contracts management and supervision; supplying electrical power, compressed air, and water to the underground; and handling pumped water from underground. This item is estimated to total US$5.0 million during the two years of pre-production activity.

24.7.6 Contingency

An allowance equal to 35% of pre-production direct and Owner's capital costs has been included to cover Project contingencies. This percentage is typical for projects at this stage of study where detailed engineering of the Project components is not extensive and the risk of omissions is high. This item is estimated to total US$19.2 million during the two years of pre-production activity.

24.7.7 Sustaining Capital

Sustaining capital, estimated to total US$23.2 million without closure and reclamation costs, has been divided into three categories as shown in Table 24-10. Development comprises completion of the A- and B-Zone declines after commercial production has been declared and, later, excavation of the C-Zone decline (US$4.4 million). Mobile equipment includes the purchase of additional mine production and spare units after commercial production has been declared (US$6.9 million) and an additional allowance of US$4.0 million to cover equipment major rebuilds midway through the mine's operating life. The Other category comprises a US$1.0 million allowance per year for miscellaneous expenditures such as the replacement of pumps, fans, drills, and refuge stations.

Table 24-10: C1 Underground Life-of-Mine Estimated Sustaining Capital Costs

Description	Total Cost (US$ '000s)
Development	4,358
Mobile Equipment	10,867
Other	8,000
Total	**23,225**

24.7.8 Closure and Reclamation

An allowance of US$1.0 million has been included to cover the cost of closure and reclamation activities specific to the underground operations. These are not expected to be extensive, as most of the major operations closure costs (open pit waste dumps, processing plant, tailings storage) are already included in the Open Pit Feasibility Study estimate and these areas will not be impacted by underground mine activities. Closure activities specific to the underground operation will be limited to removing equipment and surface infrastructure and sealing of the mine openings (portal and ventilation raises).

24.8 Operating Costs

LOM operating costs are estimated to total US$358.6 million, an average of US$50.28/t milled as detailed in Table 24-11.

Table 24-11: C1 Underground Life-of-Mine Estimated Operating Costs

Description	Total Cost (US$ '000s)	Unit Cost (US$/t)
Operating Development	22,654	3.18
Mining	142,630	20.00
Paste Backfill	64,184	9.00
Processing	108,684	15.24
G&A	23,534	3.30
Excess Power Credit	(3,123)	(0.46)
Total	**358,563**	**50.28**

24.8.1 Operating Development

This category includes the anticipated costs (labour, material, equipment) of excavating all drifts originating from the stope ramps providing access to both the upper and lower stope sill drifts (in waste), as well as the upper and lower strike sill drifts (in ore). Unit development costs are estimated at US$1,350/m, based on the 2016 actual Fazenda Brasileiro mine operating development costs of US$1,342/m. Stope development headings at the Fazenda Brasileiro mine are of similar dimensions and use equivalent mobile equipment units as anticipated for the C1 Underground. Annual total operating development costs were calculated by applying the unit development cost to the annual scheduled development metres excavated.

24.8.2 Stoping and Mine Services

This category includes the costs (labour, material, equipment) of long hole stope mining (drilling blasting, loading, haulage) as well as all underground mine services not directly related to mine development. Unit mining costs are estimated at US$20/t, which reflects 2016 actual Fazenda Brasileiro mining costs of US$19.84/t. Stope mining drill and blast patterns at the Fazenda Brasileiro mine, as well as LHD and truck haulage distances, are similar and use equivalent mobile equipment units as anticipated for the C1 Underground. Annual total underground mining costs were calculated by applying the unit mining cost to the annual scheduled stope mining tonnage excavated.

24.8.3 Backfill

As the Fazenda Brasileiro mine does not use backfill in its operation, there are no direct comparative unit stope backfilling costs available for C1 Underground. Unit operating costs for preparing, distributing, and placing paste backfill is estimated at US$5.80/t milled. This figure does not include electrical power, which, based on estimates prepared by SLR for other similar projects, would add approximately US$1.10/t to the unit cost ($6.90/t total). SLR estimates total unit stope backfilling costs at US$9.00/t, which includes a 30% contingency since no direct comparison is available. This unit cost compares favourably with a previous RPA study that anticipated a similar-sized paste backfill plant operating at a similar mine depth and distribution distances, as well as equivalent surface pumping distances. The previous study's operating cost estimate was modified to reflect the lower Brazilian labour force and power costs, and assumes an average 4% Portland cement binder cost.

24.8.4 Processing

Unit processing costs assumed for the C1 Underground (US$15.24/t) are like those estimated in the Open Pit Feasibility Study Technical Report (Brio, 2018) prepared by RPA to support the proposed restart of the open pit mine.

24.8.5 General and Administration

Unit G&A costs assumed for the C1 Underground (US$3.30/t) are also like those estimated in the Open Pit Feasibility Study Technical Report (Brio, 2018) prepared by RPA to support the proposed restart of the open pit mine.

24.9 Economic Analysis

24.9.1 Base Case

SLR prepared an economic model and LOM cash flow to confirm the economics of the C1 Underground LOM base case (the Base Case) as illustrated in Table 24-12. The Project economics have been evaluated using the discounted cash flow method, by considering annual processed tonnages and grades. The associated process recoveries, metal prices, operating costs, royalties, and capital expenditures (both initial and sustaining) were also considered. All costs are based on third quarter of 2017 estimates presented in US dollars.

Table 24-12: C1 Underground Cash Flow (Stand-Alone Underground)

	Unit	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mining														
Underground														
Operating Days	d	3,510			270	360	360	360	360	360	360	360	360	360
Tonnes Milled per Day	t/d	2,027			1,720	2,601	2,358	2,497	2,430	2,518	2,252	2,139	1,320	359
Development Ore	'000 t	888		17	145	221	90	137	154	111	14			
Stoping Ore	'000 t	6,244			320	715	759	762	721	795	797	770	475	129
Production	'000 t	7,132		17	464	936	849	899	875	906	811	770	475	129
Au	g/t	-												
Ag	g/t	-												
Waste	'000 t	778	39	141	259	121	14	25	110	56	14	-	-	-
Total Moved	**'000 t**	**7,910**	**39**	**158**	**723**	**1,057**	**862**	**924**	**985**	**962**	**824**	**770**	**475**	**129**
Stripping Ratio	w:o	0.11			0.56	0.13	0.02	0.03	0.13	0.06	0.02	-	-	-
Processing														
Mill Feed	'000 t	7,132			481	936	849	899	875	906	811	770	475	129
Au	g/t	2.65			2.41	2.48	2.72	2.72	2.72	2.59	2.66	2.77	2.77	2.75
Ag	g/t	-												
Contained Au	oz	608,156			37,297	74,741	74,122	78,715	76,418	75,442	69,218	68,442	42,333	11,428
Contained Ag	oz	-			-	-	-	-	-	-	-	-	-	-
Recovery														
Au	%	84			84	84	84	84	84	84	84	84	84	84
Ag	%	0												
Total Recovered														
Au	oz	510,851			31,330	62,783	62,262	66,120	64,191	63,371	58,143	57,492	35,559	9,600
Revenue														
Metal Prices														
Au	US$/oz Au	1,500			1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500
Exchange Rate	US$/ US$	1.00			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Au Payable Percentage	%	99.9			99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9
Au Gross Revenue	US$ '000	765,702			46,959	94,103	93,324	99,106	96,214	94,986	87,149	86,173	53,299	14,389
Total Gross Revenue	**US$ '000**	**765,702**			**46,959**	**94,103**	**93,324**	**99,106**	**96,214**	**94,986**	**87,149**	**86,173**	**53,299**	**14,389**
Mining Royalty Tax	US$ '000	15,314			939	1,882	1,866	1,982	1,924	1,900	1,743	1,723	1,066	288

	Unit	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Total Charges	**US$ '000**	**17,868**			**1,096**	**2,196**	**2,178**	**2,313**	**2,245**	**2,217**	**2,034**	**2,011**	**1,244**	**336**
Net Revenue	US$ '000	747,833			45,863	91,907	91,146	96,794	93,969	92,769	85,115	84,162	52,055	14,053
Unit NSR	US$/t milled	104.86			95.28	98.17	107.38	107.69	107.42	102.36	105.01	109.32	109.51	108.60
Operating Cost														
Processing	US$/t milled	15.24			15.24	15.24	15.24	15.24	15.24	15.24	15.24	15.24	15.24	15.24
G&A	US$/t milled	2.86			3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30
Total Unit Operating Cost	**US$/t milled**	**50.28**			**55.28**	**53.34**	**50.18**	**51.14**	**52.05**	**51.11**	**47.96**	**47.54**	**47.54**	**47.54**
Processing	US$ '000	108,684			7,336	14,268	12,936	13,697	13,332	13,812	12,353	11,733	7,244	1,972
G&A	US$ '000	20,411	(3,123)		1,588	3,090	2,801	2,966	2,887	2,991	2,675	2,541	1,569	427
Total Operating Cost	**US$ '000**	**358,563**	**(3,123)**	**489**	**26,610**	**49,940**	**42,594**	**45,968**	**45,537**	**46,327**	**38,872**	**36,601**	**22,597**	**6,152**
ICMS/PIS/COFINS paid	US$ '000	14,200	1,000	1,000	846	1,189	1,189	1,189	1,189	1,189	1,189	1,189	1,189	846
Operating Cashflow	US$ '000	376,070	2,123	(1,489)	18,408	40,779	47,363	49,637	47,244	45,254	45,054	46,372	28,269	7,056
Capital Cost														
Sustaining - Development	US$ '000	4,358			2,579	540		203	1,037					
Sustaining - Equipment	US$ '000	10,867			5,254	913	700		2,000	2,000				
Sustaining - Other	US$ '000	8,000				1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	
Working Capital	US$ '000	-												
Reclamation and closure	US$ '000	1,000												1,000
Total Capital Cost	**US$ '000**	**98,302**	**11,030**	**63,047**	**7,833**	**2,453**	**1,700**	**1,203**	**4,037**	**3,000**	**1,000**	**1,000**	**1,000**	**1,000**
Cash Flow														
Net Pre-Tax Cashflow	US$ '000	277,768	(8,907)	(64,536)	10,575	38,326	45,663	48,435	43,207	42,254	44,054	45,372	27,269	6,056
Cumulative Pre-Tax Cashflow	US$ '000		(8,907)	(73,443)	(62,868)	(24,542)	21,121	69,555	112,762	155,016	199,071	244,443	271,713	277,768
All-In Sustaining Cost	US$/oz	782	-	-	1,134	869	746	748	807	813	721	689	699	676
All-In Cost	US$/oz	927												
Project Economics														
Pre-Tax NPV at 5% Discounting	US$ '000	189,407												
Pre-Tax NPV at 10% Discounting	US$ '000	129,645												
Pre-Tax NPV at 15% Discounting	US$ '000	88,184												

The economic analysis contained in this section is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

Table 24-13 shows some of the key parameters and assumptions for the pre-tax cash flow.

Table 24-13: C1 Underground Pre-Tax Cash Flow (Stand Alone Case)

Description	Unit	Value
Revenue		
Nominal Average Production	t/d	2,500
LOM Head Grade	g/t Au	2.65
Average Mill Recovery	%	84
Metal Price	US$/oz Au	1,500
Metal Payable	%	99.925
NSR	US$/t milled	104.86
Costs		
Pre-production Period	years	2
Mine Life	years	9.5
Pre-production Capital	US$ M	74.1
Sustaining Capital	US$ M	23.2
Average Operating Cost (LOM)	US$/t	50.28
NSR Royalty	%	1
Mining Tax (Brazilian Government)	%	1
All-in Sustaining Cost	US$/oz Au	782
Cash Flow Results		
Undiscounted Cash Flow	US$ M	278
Pre-tax NPV at 5% (based on mid-period discounting)	US$ M	189
After-tax NPV at 5% (based on mid-period discounting)	US$ M	178
IRR (based on mid-period discounting)	%	39

24.9.2 Sensitivities

Project risks can be identified in both economic and non-economic terms. Key economic risks to the C1 Underground Project stand-alone case were examined through analysis of cash flow sensitivities:

- Head grade
- Mill recovery
- Gold price
- Operating costs
- Pre-production capital costs.

NPV at a 5% discount over the Base Case has been calculated for −20% to +20% variations in gold price and head grade, −15% to +35% in capital and operating costs and −5% to +5% in mill recovery. The sensitivities are shown in Table 24-14 and Figure 24-14. The Project return is most sensitive to the product of changes in the head grade, recovery, and gold price followed by changes in the operating and capital costs.

Table 24-14: C1 Underground Pre-Tax Sensitivity Analysis (Stand Alone Case)

Description	Unit	NPV at 5% (US$ M)
Head Grade	g/t	
0.80	2.12	77
0.90	2.39	133
1.00	2.65	189
1.10	2.92	245
1.20	3.18	301
Recovery	%	
0.95	80	161
0.98	82	175
1.00	84	189
1.03	86	203
1.05	88	217
Gold Price	US$	
0.80	1,200	77
0.90	1,350	133
1.00	1,500	189
1.10	1,650	246
1.20	1,800	302
Operating Costs	US$ M	
0.85	305	230
0.93	332	210
1.00	359	189
1.18	421	142
1.35	484	95
Capital Costs	US$ M	
0.85	84	203
0.93	91	196
1.00	98	189
1.18	116	174
1.35	133	158

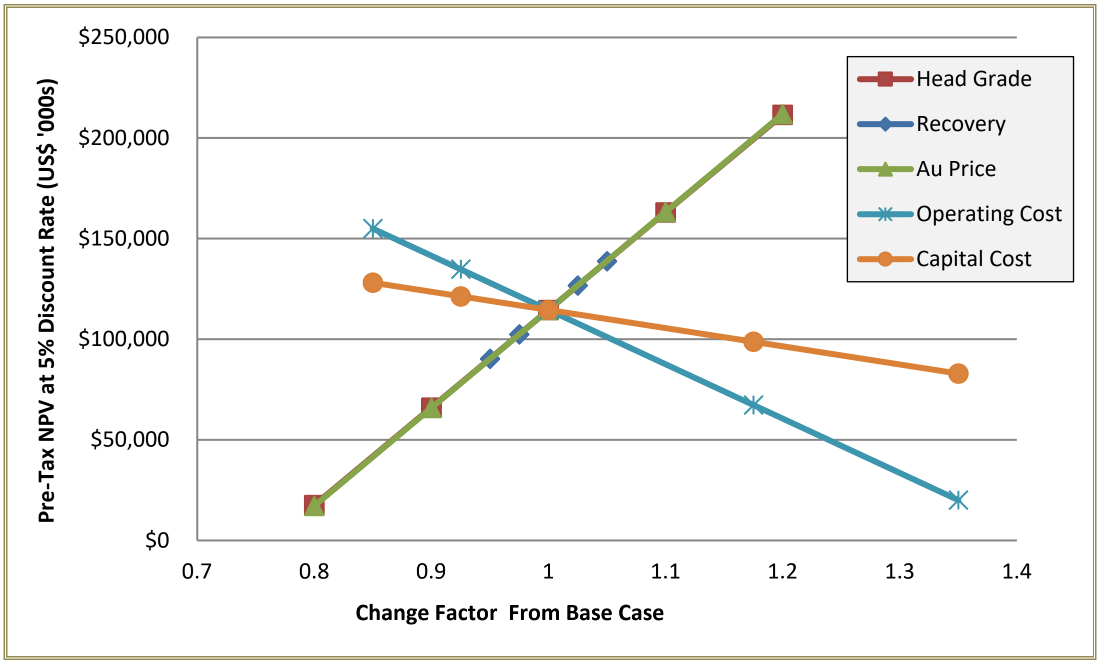


Figure 24-14: C1 Underground Sensitivity Analysis NPV Base Case

25 INTERPRETATION AND CONCLUSIONS

Based on RPA's previous site visit, discussions with Santa Luz Project-related personnel, and the available information, SLR offers the following conclusions for the Santa Luz Project.

25.1 Geology and Mineral Resources

- The regional and property geology is well understood by Equinox and the style of mineralization is consistent with an orogenic gold-type deposit.
- Most of the concessions at Santa Luz are at an early exploration stage, with limited exploration activity other than regional mapping, regional geochemistry surveys, and airborne surveys, which were completed by the previous owners. Many of these concessions remain prospective for gold.
- Drilling and logging methods and the sample preparation, analysis, and security procedures at Santa Luz are adequate for use in the Mineral Resource estimate.
- Based on the data validation and the results of the standard, blank, and duplicate analyses, SLR is of the opinion that the assay and bulk density databases are sufficient quality for Mineral Resource estimation for the Santa Luz deposits.
- The resource database is sufficiently reliable for grade modelling and Mineral Resource estimation.
- The variograms and correlograms are appropriate for the style and nature of the gold mineralization at Santa Luz, and are suitable for grade interpolation in the block models.
- SLR has reviewed the Mineral Reserves and Mineral Resources dated June 30, 2020, and is of the opinion that the parameters, assumptions, and methodology used for Mineral Resource estimation are appropriate for the style of mineralization. The classification of Measured, Indicated, and Inferred Resources conform to CIM Definition Standards (2014). Validation work indicates that the block models are reasonable and acceptable; however, they should be reviewed on an ongoing basis.

25.2 Mining and Mineral Reserves

Conventional open pit mining methods (drilling, blasting, loading, and hauling) will be employed to extract the ore and waste.

- Yamana mined the C1 and Antas 3 pits in 2013–2014. Both pits have stable highwalls, and there is currently water in the bottom of both. The water will be pumped to a WSF prior to resuming mining of the pits.
- Mining will start at C1, with production from a single pit, followed by mining two open pits at Antas 3. Pit benches will be 10 m high, mined as a double flitch, with a safety berm every 10 m.
- Mining will be conducted by contractors with oversight by Santa Luz Project personnel.

- The Open Pit Feasibility Study is based on Mineral Reserves with reasonable prospects for economic extraction by open pit mining and whole ore leaching.
  - Approximately 24.9 Mt at an average grade of 1.34 g/t Au, factored for dilution and extraction, are potentially mineable by open pit methods. This includes approximately 21.6 Mt of Proven Mineral Reserves grading 1.39 g/t Au and 1.17 Mt of Probable Mineral Reserves grading 1.28 g/t Au from the C1 and Antas 3 pits and a low-grade stockpile of Probable Mineral Reserves of approximately 2.19 Mt grading 0.86 g/t Au.
  - The LOM stripping ratio is 4.7:1 waste to ore, or 4.3:1 including stockpiles, with pit designs including benches and roads.
  - At an average processing production rate of approximately 7,400 t/d or 2.7 Mt/a, the Project life is approximately 9.5 years (a partial year for ramp-up of production, eight years of mining plus an additional 1.5 years of processing stockpiles).
- Topographical relief, climate, haul distances, and political jurisdiction present no issues for the Project.

25.3 Metallurgical Testwork and Mineral Processing

- Gold mineralization in the Santa Luz deposits occurs in two main rock types, the carbonaceous schist and dacite, both of which have been variably brecciated along a moderately dipping structural zone. Between the two lithologies the occurrence of gold and amenability to processing varies significantly.
- Since late 2014 there has been a series of laboratory metallurgical testing and pilot plant programs (in 2014, 2015, 2016, 2017, and 2019) to evaluate the existing process facilities; to determine the causes of the low gold recoveries; to develop a revised flowsheet to successfully process the carbonaceous material; and to optimize whole ore RIL processing to increase the gold grade (and reduce the copper grade) of the loaded resin and to optimize gold recovery from the resin.

25.4 Process Plant

- The new ore-processing facility will incorporate the crushing, crushed-ore storage, and SAG mill of the original plant. The rest of the plant, except for the refinery, will be new.
- The new plant will consist of the following sequence of processes:
  - Two-stage crushing
  - Two-stage grinding
  - Gravity concentration
  - Pre-aeration and kerosene conditioning
  - RIL leaching
  - Detoxifying leached tailings
  - Containing tailings in a TSF

- Loaded-resin washing and elution
- Electrowinning of gold from the eluate
- Refining to generate copper-gold bars.

- Plant throughput will be 2.7 Mt/a ore (nominal 7,400 t/d ore) grading approximately 1.34 g/t Au including ore from the existing stockpiles. Projected overall gold recovery is 84%.
- In the final year of operation, high-arsenopyrite, high-sulphide ore from a portion of the Antas 3 deposit will be processed. This ore will be processed by flotation in the existing flotation circuit to recover a gold-arsenic flotation concentrate that will be shipped off site for custom processing.

25.5 Infrastructure

- Existing site services and infrastructure will be reused as part of the new processing facility. This includes water storage and distribution, maintenance workshop, laboratory, site administration buildings, warehousing, and messing facilities.
- Practically all the infrastructure requirements for the Santa Luz Project are already in place because of the prior operations. The mine is only 35 km from the established town of Santa Luz, and is easily accessible from the port of Salvador, minimizing infrastructure requirements; consequently, the Santa Luz Project is not remote or difficult to access.
- The Project is in a semi-arid area, and water must be carefully managed. Raw water is obtained from an adjacent river and will be stored in WSF, an existing small open pit, and the TSF.
- Power is provided by an existing 138 kV connection to the national power grid.
- All the service systems and buildings required for the operation are already in place.
- The operation will employ approximately 749 personnel, approximately 60% of whom will be contractors. Most of the workers live near the mine site (within 75 km).

25.6 Environmental, Permitting, and Social Considerations

- The environmental impacts of the Project have been assessed and appropriate mitigation measures have been put in place.
- The Project has all relevant permits in place. There are no identified environmental liabilities associated with the property.

25.7 Capital and Operating Costs

- The initial mine stripping, plant, and infrastructure capital cost estimate is US$103.1 million including contingencies. Total LOM sustaining capital is estimated at US$21.0 million, and the capitalized stripping is US$60.6 million.
- Reclamation and closure costs have been estimated to be US$8.8 million at an exchange rate of R$5.00:US$1.00.

- Average processing operating costs are estimated at US$13.74/t for the combined dacite-carbonaceous blended ore and US$9.17/t for dacite with high sulphides, for an average LOM processing cost of US$13.43/t.
- The estimated LOM costs for the Project are as follows:
  - Total Capital Cost......................................US$172.9 million (initial and sustaining capital costs)
  - Total Operating Cost ....US$28.02/t of ore processed
  - Off-Site Cost ....US$9.0 million (RIL)
  - Off-Site Cost (High-Sulphide Dacite) ....US$7.8 million
- The capital costs are relatively low for the size of the Project, largely because much of the plant and infrastructure are already in place from the prior operation, and Equinox plans to use a mining contractor.
- The operating costs match the costs of similar operations of similar size.
- The off-site costs comprise the FSR costs for dacitic-sulphide ore concentrates that will be processed in the final year of operation.

25.8 Economics

25.8.1 Project

- 24.9 Mt at 1.34 g/t Au (including stockpiles) head grade producing an average of 95,000 oz/a of gold over a 9.5-year LOM for a total of 903,000 oz, with an AISC of US$877/oz, including capitalized stripping and reclamation cost.
- Positive after-tax NPV at a 5% discount rate of US$305.1 million with an IRR of 57.6% using a US$1,500/oz Au price. Payback period is 1.6 years from the re-start of the Project.
- Both capital and operating costs are based on a LOM exchange rate of R$5.00:US$1.00. Both capital and operating costs are sensitive to changes in the exchange rate.
- The Project return is most sensitive to changes in the gold price, followed by changes in operating costs, foreign exchange, and capital costs.

25.8.2 Risks/Opportunities

- There are no significant risks and uncertainties that could reasonably be expected to affect the reliability or confidence in the Mineral Resource or Mineral Reserve estimates on the projected economic outcomes.

25.9 C1 Underground PEA

- PEA-level designs and analyses indicate that a significant portion of the C1 Underground Mineral Resource at Santa Luz has the potential for economic extraction and warrants more detailed and specific analysis.
- PEA results are summarized in Section 24.

26 RECOMMENDATIONS

Equinox, Ausenco, and SLR offer the following recommendations.

26.1 Geology and Mineral Resources

- Limit the number of CRMs in any drill program to four: one approximating the cut-off grade, two proximal to the average grade, and one representative of high-grade material at the Project site.
- Field duplicate analyses are consistent with the natural variability often seen in orogenic gold deposits; however, Equinox should continue to monitor results of high-grade duplicate samples and consider submitting larger half-core samples in place of quarter-core samples.
- Use a minimum thickness of 5 m for future resource estimation to remove narrow intersections that may not be economic when diluted out. The high grades from narrow intersections will influence block grades in the interpolation process.
- Complete an in-fill drilling program to acquire information on TOC, sulphur, and preg-robbing index (PRI), critical for understanding the mineralization and mineral processing of the C1 and Antas 3 deposits. Further gold analyses from this program will provide higher confidence in gold continuity.

26.2 Mining and Mineral Reserves

- Continue to examine the optimal waste dump locations.
- Begin to evaluate the outlying Mineral Resource areas of Mansinha and Mari.
- Complete a hydrogeology study to determine dewatering parameters.
- Continue to study the Project limits based on waste dump design, and proposed optimizing truck cycle times for the LOM plan.

26.3 Metallurgical Testwork and Mineral Processing

- Confirm that carbonaceous/dacite ore blend(s) processed in the Santa Luz pilot plant over Stages 11–17 can be achieved over the LOM plan (apart from final Antas 3 high sulphide phase).
- Add a gravity treatment stage to the Santa Luz pilot plant to clarify the effect of gravity pre-treatment on whole ore RIL processing and confirm that the 20-h five-tank RIL circuit offers the most cost-effective option.
- Conduct further pilot plant testwork to firm up gold recovery mass balance to confirm that mass recovered from resin lines up with extraction determined from head/tailings assays on the ore.
- Conduct further testwork on gold recovery from resin to:
    - Confirm materials of construction for the elution heat exchanger
    - Clarify effect of multiple cycle eluate reuse on gold electrowinning performance and the most appropriate treatment of bleed solutions from elution to maximize gold recovery.

26.4 Infrastructure

- Consideration should be given to establishing the overburden dewatering parameters needed for designing surface diversions and drainage systems.

26.5 Environmental, Permitting, and Social Considerations

- Maintain renewal of the operating licences. As required by Brazilian law, the renewal application must be submitted at least 120 days before the expiration date. The Santa Luz Project has two operating licences that will expire on August 22 and 28, 2020 and their renewal applications have been submitted, on February, 28 and March 17, 2020, respectively.

26.6 C1 Underground PEA

- Complete further drilling along strike and at depth to expand and better define the current underground Mineral Resources and upgrade the Inferred Mineral Resources to Indicated Mineral Resources.
- Review the current block model to determine and implement necessary modifications that would better define the deposit for underground mining method design purposes.
- Carry out drilling of specifically oriented geotechnical holes to identify the joint sets in the hanging wall rocks. These holes should be inclined at approximately 30° to 40° from vertical and drilled with triple-tube core barrels using oriented coring techniques.
- Review the stope designs based upon the updated geotechnical analysis including:
  - An assessment of the potential ore losses (mining extraction) in the shallow dipping zones
  - A detailed evaluation of the dilution from the wall rock and backfill
  - A review of the overall stability of the hanging wall as the extraction climbs, to determine whether pillars are required.
- Consider alternative stoping methods.
- Undertake a follow-up hydrogeological evaluation of the planned mining area.
- Undertake laboratory testing of the mill tailings for use as underground paste fill. Advance the design of a backfill plant and distribution system.
- Determine which existing surface infrastructure installations (including electrical power supply and distribution) can be used or expanded to meet the needs of the underground operation.
- Advance the design of the surface infrastructure.
- Advance the mine design and scheduling including a review of the development rates.
- Advance the ventilation system design including consideration of the heat loads within the mine.
- Review the capital and operating cost and increase the level of detail in these estimates.

- Undertake further metallurgical testing using drill core from the underground resource area to compare with previous work completed for the open pit to confirm that metallurgical performance will be the same.
- Complete the environmental permitting process with the appropriate involvement of stakeholders.

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28 DATE AND SIGNATURE PAGE

This report titled *NI 43-101 Technical Report on the Santa Luz Project, Bahia State, Brazil* dated November 30, 2020 with an effective date of June 30, 2020, was prepared and signed by the following authors:

November 30, 2020	*Original Signed and Sealed*
Dated at Santa Luz, BA, Brazil	Hugo R. A. Filho, MAusIMM(CP) Mining Engineer
November 30, 2020	*Original Signed and Sealed*
Dated at Bull Creek, WA, USA	Stephen La Brooy, FAusIMM Metallurgist
November 30, 2020	*Original Signed and Sealed*
Dated at Burnaby, BC, Canada	Tommaso R. Raponi, P.Eng. Principal Metallurgist
November 30, 2020	*Original Signed and Sealed*
Dated at Lakewood, CO, USA	Robert L. Michaud, P.Eng. Associate Principal Mining Engineer
November 30, 2020	*Original Signed and Sealed*
Dated at Lakewood, CO, USA	Mark B. Mathisen, C.P.G. Principal Geologist

29 CERTIFICATE OF QUALIFIED PERSONS

29.1 Hugo Ribeiro de Andrade Filho

I, Hugo Ribeiro de Andrade Filho, MAusIMM(CP), residing at Rua 1067, Quadra 142, Lotes 4 a 7, Apt. 1702, Setor Pedro Ludovico, Goiânia, Goiás, Brazil, do hereby certify that:

1. I am employed by Equinox Gold Corp., as Exploration, Geology and Mining Planning Manager, and have been since March 2020.
2. This certificate applies to the technical report titled *NI 43-101 Technical Report on the Santa Luz Gold Project, Bahia State, Brazil* (the "Technical Report") having an effective date of June 30, 2020.
3. I graduated with a Bachelor of Science degree in Mining Engineering in 1997 from the Federal University of Bahia State.
4. I am a Member in good standing of the Australasian Institute of Mining and Metallurgy, MAusIMM(CP), accredited as a Chartered Professional – Membership No. 323096.
5. I have worked as a mine planning engineer, geology and mine planning coordinator, geology, and mine planning manager, and in technical services roles since graduation. I have over 20 years of experience in mine design, planning, evaluation, mining regulatory review and reporting, Mineral Reserve estimation, and development of precious and base metal projects. The majority of my experience is on shear-zone hosted gold deposits in Archean greenstone belts and on copper deposits in Archean ultramafic sills. A summary of my professional career is as follows:
   - Equinox Gold Corp., Exploration, Geology and Mine Planning Manager (2020 to present)
   - Leagold Mining Corporation, Exploration, Geology and Mine Planning Manager (2018-2020)
   - Brio Gold Inc., Exploration, Geology and Mine Planning Manager (2013-2018)
   - Bahia Mineração, Geology and Mine Planning Manager (2012-2013)
   - Mineração Caraíba, Geology and Mining Planning Coordinator (2006-2012)
   - Mineração Fazenda Brasileiro, Geology and Mining Planning Coordinator (2005-2006)
   - Mineração Caraiba, Mining Planning Engineer (1999-2005)
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
7. I am responsible for Sections 2, 3, 15, 16, 19, 20, 22, and 23 and related disclosure in Sections 1.1, 1.2, 1.2.8, 1.2.9, 1.2.13 to 1.2.15 (inclusive), 1.3, 1.5.2, 1.5.6 to 1.5.8 (inclusive), 1.6.2, 1.6.4, 1.6.5, 21.1, 21.2.1, 21.2.3, 21.2.4, 21.3, 25.2, 25.6 to 25.8 (inclusive), 26.2, 26.5, and 27 of this Technical Report.
8. The date of my most recent visit was June 22 to 26, 2020.
9. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have had involvement with the property that is the subject of the Technical Report.
11. I am not independent of the issuer applying the test in Section 1.5 of NI 43-101.
12. I have read NI 43-101, NI 43-101CP, and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective date: June 30, 2020

Signing date: November 30, 2020

Original Signed and Sealed

Hugo Ribeiro de Andrade Filho, MAusIMM (CP)

29.2 Stephen La Brooy

I, Stephen La Brooy, FAusIMM, residing at 7 Forster Court, Bull Creek, Western Australia, Australia, do hereby certify that:

1. I am employed as a Principal Process Consultant by Ausenco Services Pty Ltd., 144 Montague Road, South Brisbane, Queensland 4101, Australia.
2. This certificate applies to the technical report titled *Technical Report on the Santa Luz Gold Project, Bahia State, Brazil* (the "Technical Report") having an effective date of June 30, 2020.
3. I am a graduate of the University of Cambridge and received a PhD degree in Chemical Metallurgy in 1981 and a Bachelor of Arts (Natural Sciences – Metallurgy) in 1973.
4. I am a Fellow in good standing of the Australasian Institute of Mining and Metallurgy (FAusIMM) - Membership No 105358.
5. I have practiced process engineering for over 30 years. I have worked for engineering companies for 20 years and for Ausenco for 13 years.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
7. I am responsible for Section 13, as well as Sections 1.2.6, 1.5.3, 1.6.3, 25.3, 26.3, and 27 of this Technical Report.
8. I have visited the Santa Luz property from February 19-20, 2016.
9. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have not had involvement with the property that is the subject of the Technical Report.
11. I am independent of the issuer applying the test in Section 1.5 of NI 43-101.
12. I have read NI 43-101, NI 43-101CP, and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective date: June 30, 2020

Signing date: November 30, 2020

Original Signed and Sealed

Stephen La Brooy, FAusIMM

29.3 Tommaso R. Raponi

I, Tommaso Roberto Raponi, P.Eng., residing at 15–223 Rebecca Street, Oakville, Ontario, Canada, do hereby certify that:

1. I am employed as a Principal Metallurgist with Ausenco Engineering Canada Inc. with a business address at 11 King Street West, Suite 1550, Toronto, Ontario, Canada M5H 4C7.
2. This certificate applies to the technical report titled *Technical Report on Santa Luz Gold Project, Bahia State, Brazil* (the "Technical Report") having an effective date of June 30, 2020.
3. I am a graduate of University of Toronto with a Bachelor of Applied Science in Geological Engineering, 1984.
4. I am registered as a Professional Engineer in the Province of Ontario (Reg. No. 90225970) and the Association of Professional Engineers and Geoscientists of British Columbia (Reg. No. 23536).
5. I have worked as an independent consultant since 2016. My relevant experience is over 36 years of experience in the development, design, operation, and commissioning of mineral processing plants, focusing on gold projects. A summary of the more recent portion of my professional career is as follows:
    - TR Raponi Consulting Ltd, Independent Consultant (2016-2020)
    - Centerra Gold Inc., Director of Metallurgy (2005-2016)
    - SNC-Lavalin Inc., Senior Metallurgist (1995-2005)
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
7. I am responsible for Sections 17 and 18, and related disclosure in Sections 1.2.10 to 1.2.12 (inclusive), 1.2.15, 1.5.4, 1.5.5, 1.5.7, 1.6.4, 21.2.2, 21.2.4 to 21.2.7 (inclusive) 25.4, 25.5, 25.7, 26.4, and 27 of this Technical Report.
8. I have not personally inspected the property.
9. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I am independent of the issuer applying the test in Section 1.5 of NI 43-101.
12. I have read NI 43-101, NI 43-101CP, and Form 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective date: June 30, 2020

Signing date: November 30, 2020

Original Signed and Sealed

Tommaso Roberto Raponi, P.Eng.

29.4 Robert L. Michaud

I, Robert L. Michaud, P.Eng., as an author of this report titled *NI 43-101 Technical Report on the Santa Luz Project, Bahia State, Brazil* prepared for Equinox Gold Corp. dated November 30, 2020 with an effective date of June 20, 2020, do hereby certify that:

1. I am Associate Principal Mining Engineer with RPA (USA) Ltd. of Suite 505, 143 Union Boulevard, Lakewood, Colorado, USA 80228.
2. I am a graduate of Queen's University in 1976 with a B.Sc. Degree in Mining Engineering. I am a graduate of Queen's University in 1977 with a M.Sc. Degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Provinces of Ontario (Reg. #31570013) and Quebec (Reg. #37287). I have worked as a mining engineer for a total of 39 years since my graduation. My relevant experience for the purpose of this Technical Report is:
    - Review and report as a consultant on numerous mining operations and projects for due diligence and regulatory requirements
    - Operations management of several underground mines
    - Project management of the construction and start-up of several underground mines
    - Management of numerous mine designs and technical studies.
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 did not conduct a site visit to the Project.
6. I am responsible for Section 24 and related disclosure in Sections 1.4, 1.5.9, 1.6.6, 25.9, 26.6, and 27 of this Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have previously prepared a NI 43-101 Technical Report dated January 19, 2018 on the property that is the subject of this Technical Report.
9. I have read NI 43-101, and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective date: June 30, 2020

Signing date: November 30, 2020

Original Signed and Sealed

Robert L. Michaud, P.Eng.

29.5 Mark B. Mathisen

I, Mark B. Mathisen, C.P.G., as an author of this report titled *NI 43-101 Technical Report on the Santa Luz Project, Bahia State, Brazil* prepared for Equinox Gold Corp. dated November 30, 2020 with an effective date of June 20, 2020, do hereby certify that:

1. I am Principal Geologist with RPA (USA) Ltd. of Suite 505, 143 Union Boulevard, Lakewood, Co., USA 80228.
2. I am a graduate of Colorado School of Mines in 1984 with a B.Sc. degree in Geophysical Engineering.
3. I am a Registered Professional Geologist in the State of Wyoming (No. PG-2821) and a Certified Professional Geologist with the American Institute of Professional Geologists (No. CPG-11648). I have worked as a geologist for a total of 20 years since my graduation. My relevant experience for the purpose of this Technical Report is:
    - Mineral Resource estimation and preparation of NI 43-101 Technical Reports.
    - Director, Project Resources, with Denison Mines Corp., responsible for resource evaluation and reporting for uranium projects in the USA, Canada, Africa, and Mongolia.
    - Project Geologist with Energy Fuels Nuclear, Inc., responsible for planning and direction of field activities and project development for an in-situ leach uranium project in the USA. Cost analysis software development.
    - Design and direction of geophysical programs for US and international base-metal and gold exploration joint venture programs.
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 visited the Santa Luz Project on May 2 to 7, 2016.
6. I am responsible for Sections 4 to 12 and 14, and related disclosure in Sections 1.2.1, to 1.2.5 (inclusive), 1.2.7, 1.5.1, 1.6.1, 25.1, 26.1, and 27 of this Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have prepared previous Technical Reports on the property that is the subject of this Technical Report in 2016, 2017, and 2018.
9. I have read NI 43-101, and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective date: June 30, 2020

Signing date: November 30, 2020

Original Signed and Sealed

Mark B. Mathisen, C.P.G.