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Preliminary Economic Assessment NI 43-101 Technical Report of the Velardeña Project

Durango State, Mexico

Project No. 117-8133003

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LIST OF ABBREVIATIONS

Abbreviation	Unit or Term
2D	two-dimensional
3D	three-dimensional
Ag	silver
AR	risk analysis
As	Arsenic
ASARCO	American Smelting & Refining Company
Au	gold
BLS	Bureau of Labor Statistics
Broad Oak	Broad Oak Associates
CAM	Chlumsky, Armbrust & Meyer, LLC
CCP	Comprehensive Closure Plan
CFE	Comisión Federal de Electridad
°C	degrees Celsius
cm	centimeter
cm^3	cubic centimeters
CONAGUA	Comisión Nacional del Agua
Cu	copper
CUS	Land Use Change
CUSF	change in forestry land use
ECU	ECU Silver Mining
EIA	Environmental Impact Assessment
ERSA	Ensayes y Representaciones, S.A.
ETJ	Technical Justification Study
g	gram
g/T	grams per tonne
g/cm^3	grams per cubic centimeter
Golden Minerals	Golden Minerals Company
Golden Tag	Golden Tag Resources Ltd.
GxT	grade multiplied by thickness
ha	hectare
ID	identification
IMMSA	Industrial Mineral de Mexico S.A.
INAH	National Institute of Anthropology and History
kg	kilogram
km	kilometer

Abbreviation	Unit or Term
km^2	square kilometers
km/hr	kilometers per hour
KVA	kilovolt amp
K-Ar	potassium argon
LAU	Licencia Única Ambiental
lb	pound
LGEEPA	General Law of Ecological Equilibrium and Environmental Protection
LGPGIR	Ley General para la Prevención y Gestión Integral de los Residuos
LoM	Life of Mine
m	meter
M	million
MIA	Manifestación de Impacto Ambiental
Micon	Micon International Limited
Minera Labri	Minera Labri S.A. de C.V.
Minera William	Minera William S.A. de R.L. de C.V.
Mm	millimeter
mm/yr	millimeters per year
Mya	million years before present
m/s	meters per second
MXN	Mexican Peso
NEAP	Mexican National Environmental Auditing Program
NI 43-101	Canadian Securities Administrators' National Instrument 43-101
NOM-120-SEMARNAT-1997	Mexican Official Standard
NSR	Net Smelting Return
Pb	lead
PEA	Preliminary Economic Assessment
PMLU	Post-Mining Land Use
PPA	Accident Prevention Plan
PPI	producer price indexes
ppb	parts per billion
ppm	parts per million
PROFEPA	Federal Bureau of Environmental Protection
Project	Velardeña
QA/QC	quality assurance/quality control
RoM	Run of Mine
RPA	Roscoe Postle and Associates, Inc.
Sb	Antimony

Abbreviation	Unit or Term
SEDENA	Ministry of Defense
SEMARNAT	Secretaria del Medio Ambiente y Recursos Naturales
SMT	Special Mining Taxes
T	metric ton
TEM	Technical Economic Model
oz	Troy ounces
t/d	Tonnes per day
t/yr	Tonnes per year
US$	United States dollars
V	volt
µm	micrometer
yd^3	cubic yard
yr	year
/	per
Zn	Zinc

ABBREVIATIONS OF THE PERIODIC TABLE

actinium = Ac
aluminum = Al
americium = Am
antimony = Sb
argon = Ar
arsenic = As
astatine = At
barium = Ba
berkelium = Bk
beryllium = Be
bismuth = Bi
bohrium = Bh
boron = B
bromine = Br
cadmium = Cd
calcium = Ca
californium = Cf
carbon = C
cerium = Ce
cesium = Cs
chlorine = Cl
chromium = Cr
cobalt = Co
copper = Cu
curium = Cm
dubnium = Db
dysprosium = Dy
einsteinium = Es
erbium = Er
europium = Eu
fermium = Fm
fluorine = F
francium = Fr
gadolinium = Gd
gallium = Ga
germanium = Ge
gold = Au
hafnium = Hf
hassium = Hs
helium = He
holmium = Ho
hydrogen = H
indium = In
iodine = I
iridium = Ir
iron = Fe
krypton = Kr
lanthanum = La
lawrencium = Lr
lead = Pb
lithium = Li
lutetium = Lu
magnesium = Mg
manganese = Mn
meitnerium = Mt
mendelevium = Md
mercury = Hg
molybdenum = Mo
neodymium = Nd
neon = Ne
neptunium = Np
nickel = Ni
niobium = Nb
nitrogen = N
nobelium = No
osmium = Os
oxygen = O
palladium = Pd
phosphorus = P
platinum = Pt
plutonium = Pu
polonium = Po
potassium = K
praseodymium = Pr
promethium = Pm
protactinium = Pa
radium = Ra
radon = Rn
rhodium = Rh
rubidium = Rb
ruthenium = Ru
rutherfordium = Rf
rhenium = Re
samarium = Sm
scandium = Sc
selenium = Se
silicon = Si
silver = Ag
sodium = Na
strontium = Sr
sulphur = S
technetium = Tc
tantalum = Ta
tellurium = Te
terbium = Tb
thallium = Tl
thorium = Th
thulium = Tm
tin = Sn
titanium = Ti
tungsten = W
uranium = U
vanadium = V
xenon = Xe
ytterbium = Yb
yttrium = Y
zinc = Zn
zirconium = Zr

UNITS OF MEASURE

All dollars are presented in U.S. dollars unless otherwise noted. Common units of measure and conversion factors used in this report include:

English to metric conversion factors:

- 1 oz (troy) = 31.1035 g
- 1 short ton = 0.907 tonne (T)
- 1 troy ounce/short ton = 34.286 grams per tonne (g/t);
- 1 foot = 30.48 centimeters (cm) = 0.3048 meters (m);
- 1 mile = 1.61 kilometer (km); and
- 1 acre = 0.405 hectare (ha).

1 SUMMARY

This report has been prepared as a Preliminary Economic Assessment (PEA) Technical Report (TR) for Golden Minerals Company (Golden Minerals) for the Velardeña Project in Durango, México; the Project is held by Minera William S.A. de R.L. de C.V. (Minera William) a wholly owned subsidiary of Golden Minerals.

This TR is prepared as an update of a previous PEA TR prepared by Tetra Tech (Tt) for Golden Minerals dated December 31, 2014 with effective date of February 20, 2015.

This PEA incorporates additional information developed by Golden Minerals since the February 2015 report. This includes: the review and update of the project database to refine samples that included dilution and those that did not. This TR investigates the potential mineability of Measured, Indicated and Inferred Sulfide Mineral Resources for the primary veins.

Following the success of two series of metallurgical tests to evaluate the amenability of Velardeña gold and silver bearing concentrate to bio-oxidation, this updated PEA includes a circuit to oxidize the Fe concentrate from Plant #1 for recovery of the contained gold and silver to doré on site.

1.1 Location, Property Description & Ownership

The Project is located in the Velardeña mining district, within the municipality of Cuencamé, in the northeastern portion of the State of Durango, Mexico. The property is situated approximately 75 km south-southwest of the city of Torreón in the State of Coahuila, and 170 km northeast of Durango City, Capital of the State of Durango. The location of the Project is shown in **Figure 1-1**.

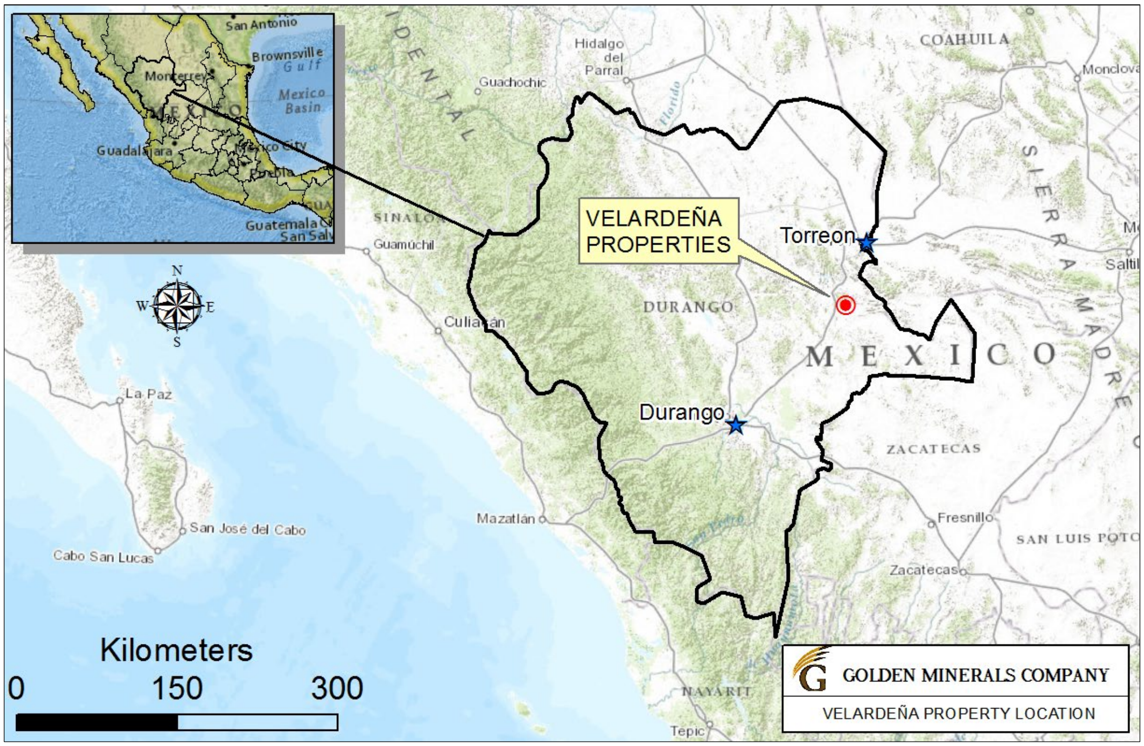


Figure 1-1: General Location Map

1.2 Geology & Mineralization

The Project is located within the eastern limit of the Sierra Madre Occidental, near its boundary with the Mesa Central. The deposits of the Sierra de Santa María (where Peñoles' Santa María mine is located), and Sierra San Lorenzo (where Golden Minerals' Velardeña Project is located), like many other polymetallic, hydrothermal deposits in northern Mexico, are situated along the 1,200-km Mexican Silver Belt held by the regional geologic boundary, which separates thick Tertiary volcanic sequences with Mesozoic and Paleozoic carbonaceous rocks to the east.

The regional geology is characterized by a thick sequence of limestone and minor, calcareous clastic sediments of Cretaceous age, intruded by Tertiary plutons of mostly felsic to intermediate composition. During the Laramide geologic event, sediments were subject to an initial stage of compression which resulted in formation of large amplitude, upright to overturned folds generating the distinctive strike ridges of limestone, which dominate local topography. Fold axes trend northerly in the northern part of the region but are warped or deflected to west northwest azimuths in the south. The northeast trending hinge line or deflection which controls this fundamental change in strike passes through the Velardeña district.

1.3 Exploration, Sampling & QA/QC

The Project has been extensively explored from the surface using geologic mapping, vein mapping and vein sampling. Underground exploration consisted of geologic mine level mapping, vein level mapping, vein sampling, drilling, drifting and stope development. Mining and metallurgical testing and small-scale production have been carried out throughout the long historical development along some of the numerous mineralized structures.

Sample preparation, analyses and security procedures followed by Minera William staff meet industry best practice standards and are sufficient to support the estimation of Resources. The quality control sampling results are typical of an operation given the amount of throughput and data handling. Previous quality control procedures and results have been reviewed by previous authors, and those reviews resulted in improved protocols and performance, but previous authors ultimately concluded the data was sufficient to support estimation of Resources.

Current drill hole analysis is completed by ALS Chemex in Vancouver, Canada and mine channel and mill samples are tested at the onsite laboratory facility constructed in 2013. ALS Chemex in Vancouver is independent of the issuer and is ISO 17025 accredited, and the accreditation of ALS Vancouver encompasses preparation processes completed at ALS Chihuahua. The onsite laboratory is not independent of the issuer and is not accredited; however, it is well qualified and maintains regular checks with duplicate samples with the Certified and Accredited laboratory SGS-Durango-Vancouver. During Tt's December 2019 site visit, QP's inspected the onsite laboratory and found the facility and procedures appropriate and following the procedures to be of excellent standard. Velardeña's laboratory also performs metallurgical test investigations.

1.4 Mineral Processing & Metallurgical Testing

There are two processing plants at the Project. Plant #1 treats sulfide material by conventional crush, grind, and differential flotation technologies to produce Pb, Zn and pyrite concentrates. Process Plant #2 has two production circuits for separately processing oxide and sulfide Au-Ag material to produce Au-Ag doré by cyanide leach/Merrill-Crowe bulk and Au-Ag sulfide concentrate by flotation, respectively. Production at both plants was suspended in June 2013. Golden Minerals subsequently restarted production at Plant #1 in November 2014 and ultimately suspended production in November 2015. Plant #2 has been leased to Hecla since July 2015 for processiong by cyanidation only, 400 tpd Hecla's mineralized material; however, Plant #2 remains on care and maintenance.

Because of the historical production for Plant #1, the liberation characteristics of the material and their response to differential flotation are within typical design criteria and known by the operations personnel. There are no geological, lithological or mineralogical changes in the process plant feed anticipated for the envisaged future production as compared to previous operations. Existing legacy operational data supports the existing process flow sheet for potential future production at Plant #1. Further, the use of existing and refurbished equipment within the pre-existing facilities is Golden Minerals' preferred method of future treatment.

In 2007 the former owner of Velardeña explored the potential to increase gold recovery from Plant #1 and improve project economics by installing a bio-oxidation circuit to treat pyrite (Fe) concentrates on site and recover gold and silver to doré by sending samples to SGS in South Africa for testwork. Treatment on site would allow operating the flotation circuit to pull more mass to the pyrite circuit containing gold and silver previously lost to tailings. The testwork indicated Velardeña pyrite concentrate could be successfully oxidized with the BIOX® process prior to cyanidation. Golden Minerals sent additional samples for testing at Outotec facilities in 2019, which confirmed the initial findings. Accordingly, this updated PEA incorporates a bio-oxidation circuit.

1.5 Mineral Resource Estimation

Resources have been estimated independently for 60 vein surfaces representing main veins, fault offsets and splits of 39 known veins. Additional work has been completed on the project database since the 2015 update. Intervals were evaluated and recoded by vein as necessary, which lead to the update of the principal veins, including wireframe models, at site for this TR update. These veins are: CC, C1, A4, F1, G1, San Mateo, Roca Negra, Hiletas, Terneras, Chicago, and Escondida. Parameters were updated for the Secondary veins, but no additional 3D modeling was completed for these veins at the time of this report. Attributes have been estimated using inverse distance to a power of 2.5.

Estimated Mineral Resources for the Velardeña project are shown in **Table 1-1** below, as well as the mineral type portions for each Resource class. Resources were calculated as diluted to a minimum of 0.7 meters and are reported at a $125 NSR cutoff.

Table 1-1: Velardeña Project Resources

Classification	Mineral Type	NSR Cutoff	Tonnes	Grade Ag g/t	Grade Au g/t	Grade Pb%	Grade Zn%	Ag oz	Au oz	Pb lb	Zn lb
Measured	Oxide	125	135,000	260	5.55	1.72	1.54	1,130,000	24,000	5,120,000	4,570,000
Indicated	Oxide	125	301,000	250	4.89	1.70	1.47	2,420,000	47,000	11,300,000	9,750,000
Measured + Indicated	Oxide	125	436,000	253	5.10	1.71	1.49	3,550,000	71,000	16,430,000	14,310,000
Inferred	Oxide	125	372,000	399	4.82	2.52	1.46	4,770,000	58,000	20,680,000	11,950,000
Measured	Sulfide	125	269,000	346	5.38	1.53	1.88	3,000,000	47,000	9,100,000	11,140,000
Indicated	Sulfide	125	645,000	327	4.62	1.43	1.94	6,790,000	96,000	20,300,000	27,530,000
Measured + Indicated	Sulfide	125	915,000	333	4.84	1.46	1.92	9,790,000	142,000	29,410,000	38,670,000
Inferred	Sulfide	125	1,393,000	342	4.70	1.51	1.97	15,320,000	211,000	46,380,000	60,400,000
Measured	All	125	404,000	317	5.43	1.60	1.76	4,120,000	71,000	14,220,000	15,710,000
Indicated	All	125	946,000	303	4.71	1.52	1.79	9,220,000	143,000	31,610,000	37,280,000
Measured + Indicated	All	125	1,351,000	307	4.92	1.54	1.78	13,340,000	214,000	45,830,000	52,990,000
Inferred	All	125	1,765,000	354	4.73	1.72	1.86	20,080,000	268,000	67,060,000	72,350,000

Notes:

(1) Reported Measured and Indicated Resources are equivalent to mineralized material under SEC Industry Guide 7

(2) Inferred Resource is not a recognized category under SEC Industry Guide 7

(3) Resources are reported as diluted Tonnes and grade to 0.7 m fixed width

(4) Metal prices for NSR cutoff are three- year trailing average as of December 2019: US$16.30/troy ounce Ag, US$1,305/troy ounce Au, US$0.99/lb Pb, and US$1.27/lb Zn

(5) Columns may not total due to rounding

Block attributes were estimated in three passes from small to large. Estimation was completed using anisotropic inverse distance weighting for each block in the model. **Table 1-2** details the search ellipse sizes, orientations along with sample selection criteria, and classification. Resource classification was assessed by pass (maximum search), number of samples and the nearest composite and average distance. Measured or Indicated classification was only permitted in pass one, 75 m maximum search, and was primarily, but not exclusively, defined within blocks haloing the existing drifts and stopes.

Table 1-2: Pass Parameters and Classification

Pass	Method	Max Search	Ratio 1st:2nd:3rd	Sectors	Max Per Sector	Comp Min	Comp Max	Classification
First	IDW 2.5	75	See vein parameter table	4	2	1	8	Inferred, Indicated if; comps >=3 and nearest comp <= 50m, Measured if; comps >=4 and nearest comp <= 16m and average comp distance <= 25
Second	IDW 2.5	150	1:0.25:0.5	1	2	1	2	Not classified, Inferred if; nearest comp <= 125m
Third	IDW 2.5	200	1:0.5:0.5	1	2	1	2	Not classified

1.6 Mining Methods

The Project is currently in care and maintenance. It is planned to be continued as an exclusively underground operation. The current conceptual mine plan includes only the sulfide material in the primary veins.

A site visit was conducted by Tt personnel on December 10, 2019. Cut and fill stoping was observed to be the primary method of extraction, as well as resuing. These methods are suitable to the steeply dipping veins found at the Project. Test mining has been conducted while the operations at the mine have been suspended to prove mining methods and selective mining widths, and to control dilution during mining. These tests were successful at a minimum selective mining width of 0.7 meters.

Conceptually planned stopes for mining are based on Measured, Indicated and Inferred Resources which total 1.138 million tonnes for mining over ten years, from stopes and stope development. **Table 1-3** details the tonnes and grade of the preliminary mine plan.

Table 1-3: Preliminary Mine plan numbers

Category	Total/Avg
Tonnes	1,137,949
NSR	295
Ag (gpt)	337
Ag oz	12,325,300
Au (gpt)	5.15
Au (oz)	188,250
Pb (%)	1.32

Category	Total/Avg
Pb (lbs)	33,096,126
Zn (%)	1.63
Zn(lbs)	40,886,729

1.7 Recovery Methods

There are two existing process plants, Plant #1 and Plant #2, at the Project. Plant #1 is designed to treat sulfide material to produce Pb, Zn and pyrite concentrates, and is located near the village of Velardeña, approximately eight kilometers from the mining operations.

Plant #1 has an operating capacity of 350 tpd with effective capacity of 300 tpd at a 95% availability, equal to 100,000-120,000 tonnes per year (t/yr) on a 350-day schedule. Plant #2 is a process plant with 550 tpd capacity for treating Au-Ag material, with the capacity to separate sulfide material from oxide material and lead and produce bulk Au-Ag flotation concentrate and Au-Ag doré. Plant #2 was purchased by William Resources in 1996. Operations were suspended at both plants in June 2013. In July 2014, Golden Minerals restarted mining operations to feed Plant #1, which started production on November 3, 2014. During the shutdown, Golden Minerals completed several capital projects at Plant #1 prior to its restart including: overhauling the electrical system, installing new concentrate filters, and refurbishing the flotation cells.

Plant #2 has been leased to Hecla until December 31, 2020 for processing oxidized minerals by cyanidation from its own mine located in the area of Guadalupe Victoria, Durango, approximately 90 km from Velardeña, and it is operating at approximately 400 tpd.

A BIOX® circuit would be constructed adjacent to Plant #2. Thickened Fe concentrate from Plant #1 would be trucked to Plant #2, transferred to a stock tank and subsequently to the BIOX® reactors. Oxidized concentrate, BIOX® residue, would be leached in the existing Plant #2 circuit.

1.8 Infrastructure

Infrastructure facilities at the Project include the following:

- Access roads;
- Power line;
- Ancillary buildings; and
- Water wells.

There are no man-camp facilities at the Project site. The Project is in the Mexican state of Durango, approximately 75 km southwest of the city of Torreón and 170 km northeast of the city of Durango. A major 4-lane highway, Highway 40, connects these cities. The Velardeña Plant #1 is located adjacent to the village of Velardeña, which is approximately 500 m west of Highway 40D. The Velardeña mines are located approximately eight kilometers from Plant #1 via a gravel road. Plant #2 is located at approximately 3.5 km from the Velardeña mine, also via gravel roads.

1.9 Market Studies & Contracts

Mill operations, which last operated between November 2014 and December 2015, produce Pb, Zn and pyrite concentrates. Markets for the Pb and Zn concentrates include metal brokers and direct sales to Pb and Zn smelters. Pyrite concentrates produced will be used as feed for the bio-oxidation circuit and then processed through an agitation leach circuit utilizing Merrill-Crowe to produce a gold and silver rich doré. The concentrates and doré produced are typical within the Mexican mining industry and the concentrate and doré markets within Mexico and worldwide are liquid. For purposes of the PEA, it is assumed that Golden Minerals will be successful in securing buyers for its concentrates and doré.

1.10 Environmental Permitting

Minera William's Velardeña Operations consist of existing underground mining and surface mineral processing facilities located on controlled property. Numerous historical mining operations have occurred at the Velardeña Operations with records showing the existence of mining in the district dating back to the early 1800s. Significant mining activity began in 1902 and included mining of both the Terneras and Santa Juana veins and the construction and operation of a 2,500 t/d smelter. Other small-scale development was conducted by local miners throughout most of the 1900's. Many of these historical operations have been incorporated into the current mining activities, while others remain inactive and separate from the current activities. Minera William anticipates continued mining operations at this location.

In early 2012, Golden Minerals applied for and was accepted into the Mexican National Environmental Auditing Program ("NEAP"). Under NEAP, Golden Minerals participated in an audit program to verify compliance with existing regulations and identify non-regulated potential issues that could result in environmental contingencies. Golden Minerals holds various permits required for conducting their current operations at the Velardeña Properties, and their participation in NEAP allows them to continue their current operations during the remediation of any potential non-compliance matters. This program was in play for Plant #1 until the spring of 2014, at which time Golden Minerals had achieved 85% compliance, but the plant was placed on care and maintenance.

Golden Minerals is required to update their environmental licenses and environmental impact assessments for any expansion of or modification to any of the existing two plants. The construction of new infrastructure beyond the current plant facilities would require additional permitting, including environmental impact assessments and possibly land use permits. Golden Minerals does not expect to have difficulty obtaining additional permits or environmental impact assessments.

Tetra Tech is unaware of any outstanding environmental liabilities attached to the Velardeña Properties and is unable to comment on any remediation which may have been undertaken by previous companies.

1.11 Capital & Operating Costs

Capital costs for the Velardeña Mine are summarized in **Table 1-4**.

Table 1-4: Capital Costs

Capital Costs	Preproduction	LOM	Full LOM
Mine Development	$869	$9,922	**$10,791**
Process Plant	$7,410	$2,050	**$9,460**
Infrastructure	$782	$0	**$782**
Other Non-Operating Costs	$1,210	$3,947	**$5,158**
Total (1)	**$10,272**	**$15,919**	**$26,190**

Operating costs for the Velardeña Mine are summarized in **Table 1-5**.

Table 1-5: Operating Costs

.	Total ($000s)	Unit Cost ($/t)
Mining costs	$94,303	$82.87
Milling costs	$54,241	$47.67
Mine & Process	**$148,545**	**$130.54**
Treatment and Refining costs	$45,440	$39.93
Contigency	$15,656	$13.76
G&A	$8,014	$7.04
Royalty	$1,651	$1.45
Total Operating	**$219,306**	**$192.72**

1.12 Economic Analysis

Project cost estimates and economics developed in the Technical-Economic Model (TEM) are prepared on an annual basis for the LoM. The following PEA includes Measured, Indicated and Inferred Mineral Resources; Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. This PEA also includes Inferred Mineral Resources that are too speculative for use in defining Reserves. Based upon design criteria presented in this report, the level of accuracy of the estimate is considered ±35%. Economic results are summarized in **Table 22-1**. The analysis suggests the following conclusions, assuming no debt:

- Remaining Mine Life: ten years;
- Pre-Tax NPV8%: US$85.91million, IRR: 139%;
- Payback: One year; and
- Federal Precious Metal Royalty: US$1.65 million.

1.13 Interpretations & Conclusions

1.13.1 Geology & Resources

Drill hole and channel samples have been collected and analyzed using industry standard methods and practices and are sufficient to support the characterization of grade and thickness and to further support the estimation of Measured, Indicated and Inferred Mineral Resources.

1.13.2 Mining

Results of the PEA indicate mining is potentially economically viable based on the inclusion of Inferred Mineral Resources. However, due to the nature of the mineralization and the scale of the operations, extensive Resource drilling of the deposit is not planned. For this reason, detailed mine plans and schedules are not expected to be produced for the deposit. The consequence of this is that residual risk remains for mining of the project and planning of grades and stope tonnages can only be completed on a short-term basis.

The success of the proposed plan is sensitive to mining dilution, which could increase the costs of saleable products, but also provides opportunity as any potential reductions in dilution from the mining would greatly benefit the project. Recent test mining at the site has confirmed a minimum selective mining width of 0.7 m is achievable, which can contribute to reducing dilution.

1.13.3 Metallurgy & Process

There are no geological, lithological or mineralogical changes in the process plant feed anticipated for the envisaged potential future production as compared to previous operations. Existing legacy operational data fully supports the existing process flow sheet for future production at the Plant #1.

The use of existing and refurbished equipment within the pre-existing facilities, and the production of marketable concentrates, is Golden Minerals' preferred method of treating potential future production. A new BIOX® plant will be constructed at the site to treat the concentrates, as testing has indicated this will be a valid method.

1.14 Recommendations

1.14.1 Geology & Resources

- Continue to collect specific gravity measurements and refine current estimations of specific gravity. Additional measurement should ideally be made with a paraffin wax or epoxy coating;
- Implement procedures of duplicate channel sampling of drifts by secondary sampling teams prior to stope development to ensure grade and thickness characteristics and to serve as field duplication of channel samples;
- Setup of strict control sample review procedures and tolerances involving review of control sample failure on receipt of each batch's results, and automatic triggering of batch reanalysis immediately after being alerted to failures;
- Improve sample data transcription methods to reduce control sample labeling errors and immediately resolve errors when encountered;

- Perform a detailed model reconciliation on a completed stope early in the proposed mine life and alter the estimation methods if the result of the reconciliation suggest refinements should be made;
- Continue to advance exploration drilling down dip of current Inferred Resources as new levels are established. Preferentially target the Terneras, San Mateo, Roca Negra and A4 veins; and
- The costs for additional drilling have not been included in the LoM PEA but any further Resource expansion would be dependent on additional drilling.

1.14.2 Mining

It is recommended that Golden Minerals implements cut and fill mining where waste and vein material are blasted separately in order to reduce ore dilution. This practice would consider more total tonnes blasted in each section. Vein tonnes would be reduced, but the resulting grade would be higher. Recent tests on selective mining widths of 0.7 meters has proven to be achievable. Because this practice requires an efficient operations control, Tetra Tech recommends having detailed control in drilling and blasting.

The mine plan developed for the PEA should be optimized and undertaken at a more detailed level, which will enable a greater understanding of mining constraints, costs and resulting mill feed.

1.14.3 Metallurgy & Process

Antimony and Arsenic are penalty elements in the Lead and Zinc concentrates and need to be added to the database and spatially modeled. Additional metallurgical test work is recommended to investigate the depression of Antimony and Arsenic from the final Lead and Zinc concentrates, and Zinc from the pyrite concentrate. After the startup of the BIOX® plant, it is recommended to focus on the performance of the plant and adjust the process as necessary to achieve the best results.

1.14.4 Economic Analysis

It is recommended that future studies include a more in depth look at the reclamation costs. Currently, it is anticipated that the salvage sale of equipment will cover the reclamation costs. Due to changing parameters in the mine life and size, it is recommended to review this assumption in the future.

2 INTRODUCTION

This report has been prepared for Golden Minerals Company (Golden Minerals) for the Velardeña Project (the Project) held by Minera William S.A. de R.L. de C.V. (Minera William) a wholly owned Mexican subsidiary of Golden Minerals. Minera William holds the title to the oxide mill and processing facility. Additionally, Minera Labri, S.A. de C.V. (Minera Labri), is a wholly-owned Mexican subsidiary company of Golden Minerals,, which holds title to the sulfide mill and floation processing facility, Plant #1. Plant #1 and Plant #2 are constructed and operate within private property.

This Updated PEA Technical Report was prepared to fulfill Golden Minerals' obligation to file a Technical Report in accordance with Section 4.2(1)(j)(ii) of Canadian National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101). This report has been prepared by Qualified Persons employed by Tetra Tech.

This Updated PEA TR has been prepared for the purposes of detailing database work, which includes an updated Mineral Resource, and well as additional processing test data completed by Minera William since the previous Technical Report. This TR details the methods and results of updated Mineral Resource estimation; and presents a Preliminary Economic Assessment (PEA) based on updated Mineral Resources. The Updated PEA investigates the potential mineability of Measured, Indicated and Inferred sulfide Resources for the primary veins, which include the following four areas: Santa Juana (A4, CC, C1, F1, and G1 veins), San Mateo, Terneras (Terneras, Hiletas, and Roca Negra veins), and Chicago (Chicago and Escondida veins).

Golden Minerals is a Delaware corporation based in Golden, Colorado, USA. Golden Minerals' shares are listed on the NYSE American and the Toronto Stock Exchange under the symbol AUMN.

2.1 Sources of Information

Sections 4.0 to **11.0** as described in the previous technical report on the property by Chlumsky, Armbrust and Meyer, L.L.C. (CAM) in 2012 have been utilized in this report where appropriate and applicable. Content in these sections has been collated by various authors contributing to previously issued reports, prepared by CAM 2012, Micon International Limited (Micon) (2009, 2008, 2006, and 1998), Roscoe Postle and Associates Inc. (RPA, 2005), Broad Oak Associates (Broad Oak, 2006), Tetra Tech (2015), and in various government and other publications listed in **Section 27**.

Golden Minerals and Minera William staff have contributed the following data and inputs in support of this Technical Report:

- Drill hole and channel database information;
- Initial interpretations of veins;
- Geologic and vein level and surface maps;
- Existing ramp and level development and mined cavities;
- Production reports;
- Mill cost reports;
- Mineral processing flowsheets, equipment list, and facility layouts for both Plants #1 and #2;
- Freshwater well and infrastructure data;

- Smelting and refining contract terms for Pb, Zn and pyrite concentrates;
- Tax rates and royalty terms for economic analysis;
- Affected environment and baseline data, current permit status and requirements, and the environmental monitoring program; and
- Community relations and social responsibility.

2.2 Property Inspection

Property inspections have been completed by Qualified Persons (QP) Dante Ramírez, Leonel López, Randolph Schneider, and Kira Johnson.

Dr. Ramírez, Mr. López, Ms. Johnson, and Mr. Schneider visited the site on December 10, 2019. The inspections included observations of geologic interpretations, mining, exploration drilling, channel sample locations, survey locations, underground mine accesses, Santa Juana vein (San Mateo Ramp), Chicago Veins (Chicago Ramp), drifts and stopes, stockpiled material, processing Plants # 1 and #2, Golden Minerals Laboratory, and discussions with the mine staff regarding past estimation methods, database structure and vein interpretations.

Tt's QP's also visited and observed surface infrastructure, development, San Mateo ramp, Chicago Ramp, adits, stockpiles, current conditions of the Plant sites #1 and #2, the Velardeña Laboratory, and Golden Minerals headquarters in Torreón, Coahuila.

2.3 Units of Measure

All references to dollars in this report are to US dollars (US$) unless otherwise noted. Distances, areas, volumes, and masses are expressed in the metric system unless indicated otherwise.

For the purpose of this report, common measurements are given in metric units. All tonnages shown are in tonnes of 1,000 kilograms, and precious metal grade values are given in grams per tonne (g/t), precious metal quantity values are given in troy ounces (oz).

3 RELIANCE ON OTHER EXPERTS

The authors of this report are relying on statements and information provided by the issuer concerning legal, environmental, tax and royalty matters included in **Sections 4.0, 19.0, 20.0, and 22.0** of this Technical Report.

The authors of the Technical Report are relying on statements and documents provided by Warren Rehn, Chief Executive Officer, Robert Vogels, Chief Financial Officer, and Aaron Amoroso, Senior Exploration Geologist, of Golden Minerals:

- Standing of environmental permits and compliance;
- Standing of mining, exploration and associated permits to continue mining operations;
- Surface access agreements;
- Smelting terms and contracts;
- Leasing, royalty and purchase agreements relating to the claims; and
- Operating and Capital costs.

A Title Opinion was provided by VHG, Servicios Legales, S.C. Tetra Tech is relying on this Title Opinion for information regarding the location of claims and their standing.

The author of **Section 20.0** of this report is relying on a number of reports and information provided regarding environmental matters:

- Environmental Impact Statement (Manifestación de Impacto Ambiental - MIA), Velardeña Mine Exploration and Mining Project Located in the Town of Cuencamé de Ceniceros, Durango; April 2013, prepared by consultants of Minera William located at Río Támesis 2505, Col. Magdalenas, Torreón, Coahuila, 27010, and has summarized the relevant conclusions of the report.
- Conceptual Closure Plan, Golden Minerals Company, Velardeña Operations prepared by Kermit C. Behnke, July 2012, in its entirety.
- Statements and documents provided by Warren Rehn, Chief Executive Officer of Golden Minerals, particularly in the area regarding community agreements and social impacts.

4 PROPERTY DESCRIPTION AND LOCATION

The Project includes 28 mining concessions covering the Velardeña and Chicago mines controlled by Golden Minerals through its Mexican subsidiary Minera William, and located within the Velardeña mining district. Processing Plants #1 and #2 are located within land owned by Golden Minerals. Surrounding eiido-owned land contains some of the associated installations and infrastructure.

4.1 Location

The Project is located in the Velardeña mining district, within the municipality of Cuencamé, in the central-eastern portion of the State of Durango, México. The property is situated at approximately 75 km southwest of the city of Torreón, and 170 km northeast of the city of Durango. The location of the Project is shown in **Figure 4-1**.

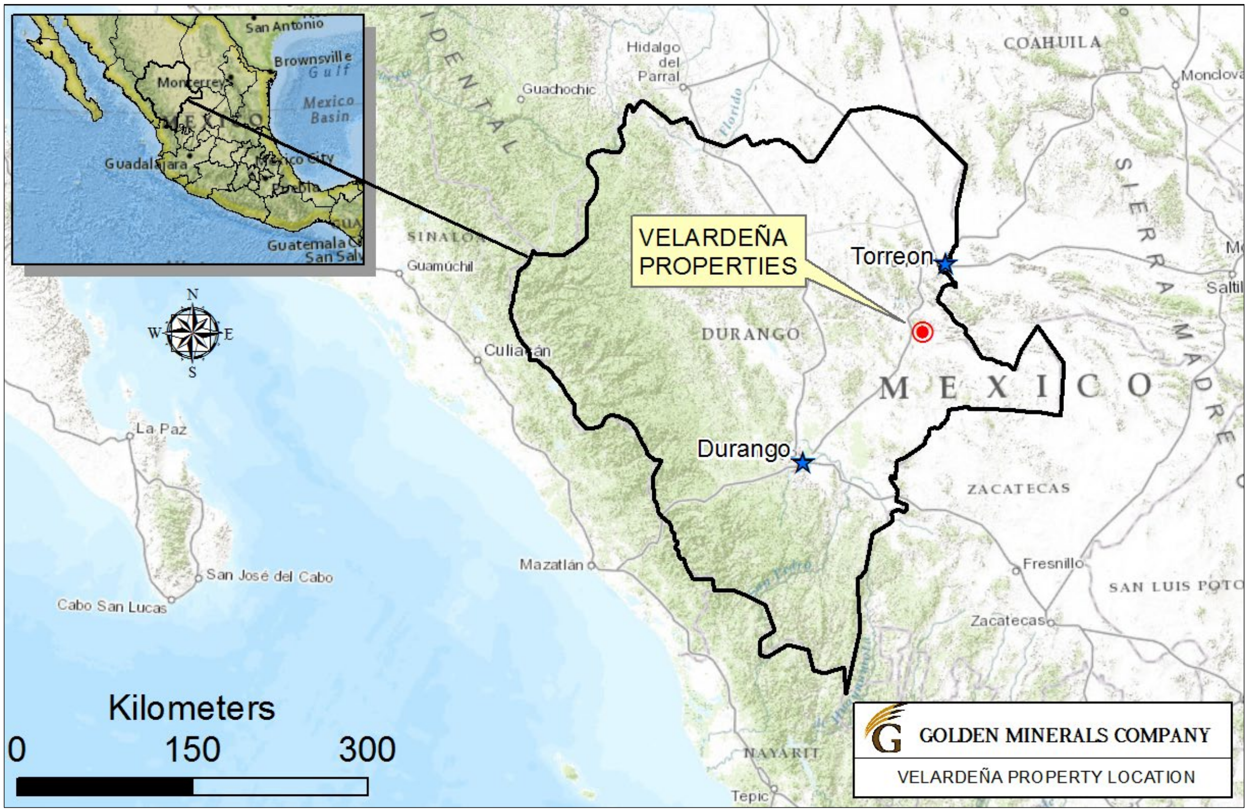


Figure 4-1: General Location Map

4.2 Property Description

The Project is comprised of two properties:

- The Velardeña property is centered on UTM grid coordinates 2774300 N and 632200 E (WGS 84 datum, zone 13). This property contains the Santa Juana mine which has been the focus of mining efforts since 1995, as well as the historical Terneras, San Juanes, and San Mateo mines.
- The Chicago property is located approximately 2 km south of the Velardeña property and is centered at UTM grid coordinates 2772480 N and 631867 E (WGS 84 datum, zone 13). This property contains the historical Los Muertos-Chicago mine.

4.3 Mineral Tenures

4.3.1 Claims

The Project consists of 28 claims covering the Velardeña and Chicago properties controlled by Golden Minerals through its Mexican subsidiary Minera William. Golden Minerals holds 315.51 hectares within all the concessions. **Table 4-1** details the list of concessions, title numbers, dates of registration and expiration and their respective areas.

Table 4-1: Project Mineral Concessions

Location	Claim Name	Claim Owner	Concessions Title Nos.	Issue Date	Expiration Date	Concessions Area (Has)
Velardeña	AMPL. DEL ÁGUILA MEXICANA	Minera William	85580	10/13/1936	10/12/2061	19.86
Velardeña	ÁGUILA MEXICANA	Minera William	168290	4/2/1981	4/1/2031	18.94
Velardeña	LA CUBANA	Minera William	168291	4/2/1981	4/1/2031	2.55
Velardeña	TORNASOL	Minera William	168292	4/2/1981	4/1/2031	4
Velardeña	SAN MATEO NUEVO	Minera William	171981	9/21/1983	9/20/2033	8
Velardeña	SAN MATEO	Minera William	171982	9/21/1983	9/20/2033	4.61
Velardeña	RECUERDO	Minera William	171983	9/21/1983	9/20/2033	8.23
Velardeña	SAN LUIS	Minera William	171984	9/21/1983	9/20/2033	2.4
Velardeña	LA NUEVA ESPERANZA	Minera William	171985	9/21/1983	9/20/2033	9.93
Velardeña	LA PEQUEÑA	Minera William	171988	9/21/1983	9/20/2033	1
Velardeña	BUEN RETIRO	Minera William	172014	9/21/1983	9/21/2033	6.09
Velardeña	UNIFICACIÓN SAN JUAN EVANGELISTA	Minera William	172737	6/28/1984	6/27/2034	13.94
Velardeña	UNIFICACIÓN VIBORILLAS	Minera William	185900	12/14/1989	12/13/2039	46.43
Velardeña	BUENAVENTURA No. 3	Minera William	188507	11/29/1990	11/28/2040	6.01
Velardeña	EL PÁJARO AZÚL	Minera William	188508	11/29/1990	11/28/2040	15
Velardeña	BUENAVENTURA 2	Minera William	191305	12/20/1991	12/19/2041	5.37
Velardeña	BUENAVENTURA	Minera William	192126	12/19/1991	12/18/2041	30

Location	Claim Name	Claim Owner	Concessions Title Nos.	Issue Date	Expiration Date	Concessions Area (Has)
Velardeña	LOS DOS AMIGOS	Minera William	193481	12/19/1991	12/18/2041	25.33
Velardeña	VIBORILLAS NO. 2	Minera William	211544	5/31/2000	5/30/2050	1.6
Velardeña	KELLY	Minera William	218681	12/3/2002	12/2/2052	3.91
Chicago	SANTA TERESA	Minera William	171326	9/20/1982	9/19/2032	22.34
Chicago	SAN JUAN	Minera William	171332	9/20/1982	9/19/2032	8.17
Chicago	LOS MUERTOS	Minera William	171986	9/21/1983	9/20/2033	3.53
Chicago	EL GAMBUSINO	Minera William	171987	9/21/1983	9/20/2033	6.65
Chicago	AMPLIACIÓN SAN JUAN	Minera William	183883	11/23/1988	11/22/2038	10.8
Chicago	MUÑEQUITA	Minera William	196313	7/16/1993	7/15/2043	15.45
Chicago	SAN AGUSTÍN	Minera William	210764	11/26/1999	11/25/2049	7.46
Chicago	LA CRUZ	Minera William	189474	12/6/1990	12/5/2040	7.91

4.3.2 Surface Rights, Agreements & Obligations

The surface rights are owned by either private persons or Ejidos (rural co-operative communities). Typically, a written document with certified authorization is granted for prospecting and sampling and a letter agreement or contract will be used for drilling, trenching, basic road building and similar exploration activities. This document is executed in writing and signed in front of Public Notary to protect the Company's rights and is required by SEMARNAT. A monetary consideration and/or some obligations are usually required to perform any extensive work programs and the landholders must also be compensated should the land be required for development.

Golden Minerals reports that it has valid agreements with two local Ejidos that control surface rights over the claims. The contract with Ejido Velardeña is for ten years and was signed in 2012, which provides surface rights to certain roads and other infrastructure at the Velardeña property of the Project. As part of the contract the company makes payments of US$2,000 every quarter.

The Chicago property is part of the Vista Hermosa Ejido which controls the surface rights. Golden Minerals and the Ejido have signed an agreement regarding surface rights and access. The contract with Ejido Vista Hermosa is for 25 years and was signed in March 2013; it provides exploration access and access rights for roads and utilities for the Chicago area of the Velardeña project properties. As part of the contract the company makes a payment of $400,000 Mexican Pesos (MXN) plus IVA by March 24th of every year.

Golden Minerals has acquired the surface rights for the land underlying the oxide mill from Vista Hermosa Ejido in addition to the land it already owned in the area of surface installations at the entrance of the Terneras mine (San Mateo ramp), the sulfide plant, the area containing the concentrates warehouse, and where one of the wells used by the mill is located.

4.3.3 Royalties and Tax

A royalty of 0.5% on precious metal production (Au, Ag) is paid yearly to the Federal government for the purposes of returning a portion of production to the local community. There are no other known production royalties. In addition, all operations in Mexico are required to pay a 7.5% special mining tax.

4.3.4 Mineral Property Encumbrances

The history of consolidation of the Velardeña concessions and operation of the mine is long and complex, and no attempt has been made to conduct specific due diligence into the status of underlying agreements. It is, however, the author's understanding that the previous royalty provisions attached to acquisition of individual properties have now been extinguished, and all properties are now controlled 100% by Golden Minerals.

In late 2004, a predecessor company purchased an interest in a flotation mill located in the town of Velardeña to treat sulfide production from the Project. Since that time Golden Minerals has increased its ownership interest in the holding company, Minera Labri S.A. de C.V., (Minera Labri) to 100%.

4.4 Environmental Liabilities

This information was sourced from CAM (2012).

In early 2012, Golden Minerals applied for and was accepted into the Mexican National Environmental Auditing Program ("NEAP"). Under NEAP, Golden Minerals will participate in an audit program to verify compliance with existing regulations and identify non-regulated potential issues that could result in environmental contingencies. After the operation was temporarily closed in 2013, the enrollment in NEAP was suspended. Golden Minerals holds various permits required for conducting their current operations, and their participation in NEAP allows them to continue their current operations during the remediation of any potential non-compliance matters.

Golden Minerals is required to update their environmental licenses and environmental impact assessments for any expansion of or modification to any of the existing plants. The construction of new infrastructure beyond the current plant facilities would require additional permitting, including environmental impact assessments and possibly land use permits. Golden Minerals does not expect to have difficulty obtaining additional permits or environmental impact assessments.

Tetra Tech is unaware of any outstanding environmental liabilities attached to the Velardeña Properties and is unable to comment on any remediation which may have been undertaken by previous companies.

4.5 Permitting

This information was sourced in part from CAM (2012).

Mining projects in México are subject to Mexican federal, state and municipal environmental laws and regulations for the protection of the environment. The principal legislation applicable to mining projects in México is the federal General Law of Ecological Balance and Environmental Protection, which is enforced by the Federal Bureau of Environmental Protection, commonly known as PROFEPA.

In order to begin an exploration program on a concession upon which no substantial mining has been conducted, Golden Minerals would have to comply with the Mexican Official Standard: NOM-120-SEMARNAT-1997, which provides, among other things, that mining exploration activities to be carried out must be conducted in accordance with the environmental standards set forth in NOM-120-SEMARNAT-1997; otherwise, concession holders are required to file a preventive report or an environmental impact study prior to the commencement of the exploration program. However, an environmental impact study may not be necessary if the concessionaire files an application with the environmental authorities confirming the concessionaire's commitment to observe and comply with NOM-120-SEMARNAT-1997. If the exploration program requires the removal of vegetation, a permit to change the land use will also be required.

Golden Minerals reports it has obtained and maintained other permits and agreements that include an explosive use permit (from the Secretaría de Defensa Nacional) subject to renewal each year, surface land use agreement (“ocupación temporal” agreement with the Ejido Velardeña), water use permit (Comisión Nacional del Agua) and other environmental permits from the Secretaría del Medio Ambiente y Recursos Naturales (SEMARNAT).

4.6 Significant Risk Factors

Access to the project is granted through agreements with two Ejidos, Vista Hermosa and Velardeña. The relationship between Golden Minerals and the Ejidos is positive; however, should Golden Minerals fail to make payments to the Ejidos, access to the property could be affected.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

This information was sourced from NI 43-101 Technical Report Velardeña Project, Durango, State, México dated on January 01, 2012 prepared by CAM.

5.1 Accessibility

The Project is located in the State of Durango approximately 65 km southwest of the city of Torreón, in the State of Coahuila, and 150 km northeast of the city of Durango, in the State of Durango. A four-lane toll highway connecting Torreón and Durango passes approximately 500 m east of the village of Velardeña. The village is connected to the mine site via a seven km gravel road. An aerial view of the Velardeña Project area is presented below (**Figure 5-1**).

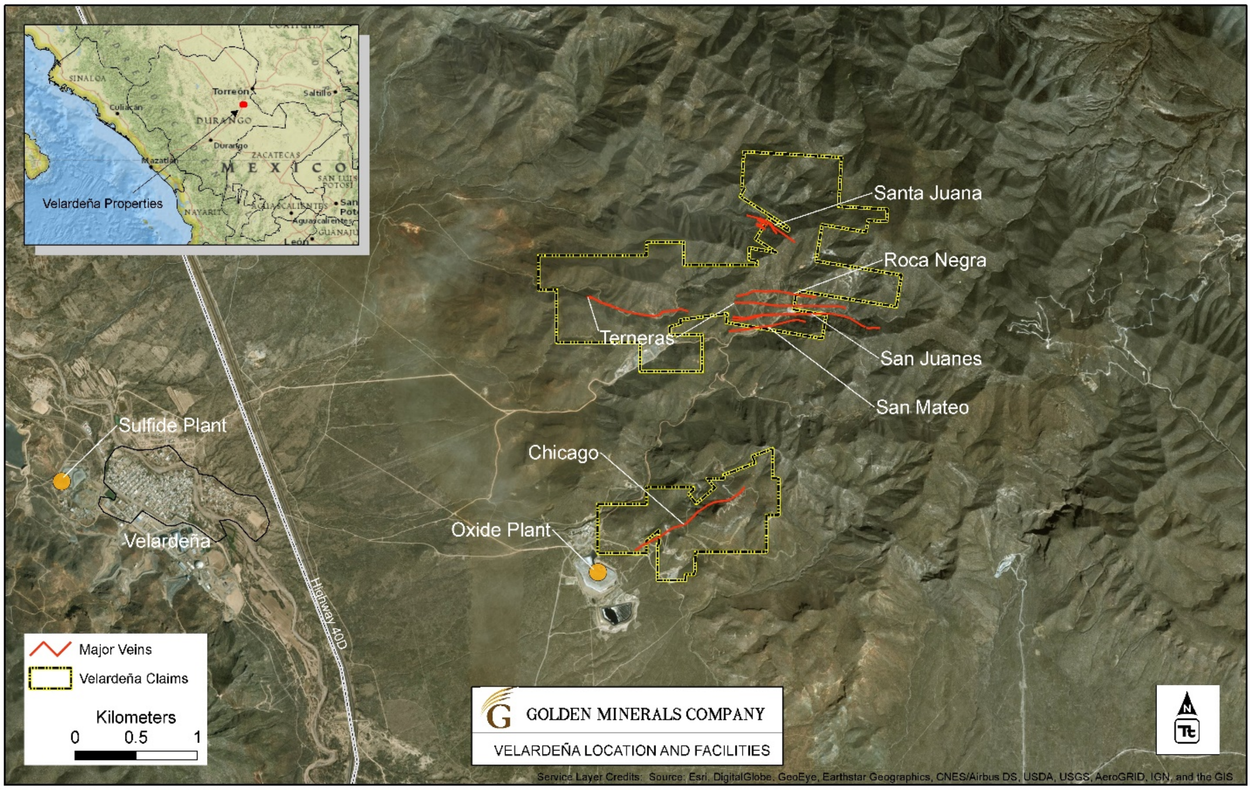


Figure 5-1: Aerial View of the Project

5.2 Climate, Vegetation, Soils and Land Use

The area in which the Project is situated is semi-arid with a climate predominantly warm and dry,according to the Köppen climate classification (Bs 1 hw climate), with a mean annual temperature of 21.1° C and rainfall averaging 243.7 millimeters per year (mm/yr). Temperatures can drop below freezing in the winter and commonly reach the high 30°C's, from July through September. The predominant winds are northeast-southwest, with speeds of 8 to 22 km/hr, or 2.1 to 6.0 m/s. The operating season is year-round.

According to INEGI's classification, the type of vegetation where the project is located corresponds to a vegetation type known as Desert Shrubland rosetophilous (rosette-forming vegetation) and sub montane scrub.

There are 106 recorded animal species in the State of Durango: 35 mammals, 13 species of reptiles; and 58 species of birds. The fauna present in the State of Durango represent 19% of the total Mexican fauna, the aviary species represent 32% and the reptilian fauna represent 19% of the total species registered for the country.

5.3 Infrastructure

Torreón was founded in 1907 by cotton growers and cattle ranchers. It is now a major industrial center which is host to a Peñoles smelter, a coal-fired electricity plant and one of the largest dairy industries in Mexico. The Francisco Sarabia airport, located in Torreón, is one of the many international airports in Mexico with flights not only to major national hubs, but to some international destinations, including Dallas, Houston and Los Angeles. Torreón has a population of approximately 680,000 according to the 2015 census.

Adjacent to Torreón, within the State of Durango, are the cities of Gómez Palacio with 342,000 (census 2015) and Lerdo de Tejada (141,000 census 2015), which have an extensive industry of manufacturing mining equipment and metallurgical processing plants.

Fresh water for the Velardeña project is sourced from wells which tap local aquifers. Golden Minerals has a total of six wells - three located near Process Plant #1 and three near Process Plant #2. These wells are authorized, regulated and permitted by the Mexican Comisión Nacional Del Agua (CONAGUA). Golden Minerals installed a five-kilometer, 4-inch diameter pipeline for shipping water from Plant #2 to Plant #1; however, this was removed when operations were stopped, and the pipes are in storage until operations are resumed at Plant # 1.

Golden Minerals owns a 300 t/d flotation mill (Plant #1), which produces three concentrates of Pb, Zn and pyrite. This plant is situated near the town of Velardeña and was upgraded in 2014, including the overhaul of the electrical system, installing new concentrate filters and refurbishing the flotation cells. The tailings from the pyrite flotation section represents the final plant tailings which are pumped to Tailings Dam #3, which is located adjacent to Plant #1. Tailings Dam #3 has enough capacity to hold the produced tailings from Plant #1. Plant #1 also contains a fully functioning analytical laboratory. 100% of the saleable metals produced in 2014-2015 were processed at this mill.

The company also owns a 550 t/d processing plant with two circuits for separately treating oxide and sulfide Au-Ag ores. This plant is currently on leased to Hecla mining company through December 2020. The main buildings hosting this equipment are open to the environment because of the mild climate, and they have ancillary buildings for warehousing, offices and maintenance. The Velardeña mines, Plants #1 and #2 are connected to the national grid of Comisión Federal de Electricidad (CFE) via a substation located near the entrances of Plant #1 and Peñoles' Velardeña mine.

5.4 Personnel

An experienced labor force is available in the village of Velardeña and in several nearby communities. The properties are supplied with electric power via the national grid. Supplies and equipment are directly available from Torreón, Monterrey, Chihuahua, and Durango, as well as from specialized suppliers elsewhere in Mexico, Canada and the USA.

5.5 Physiography

The Project's physiographic characteristics are mature with a mixed topography. Streams within the area drain either to internal drainage systems or tributaries of the Nazas and Aguanaval rivers which are connected to the Laguna de Mayrán. All of the streams are intermittent and short lived during the rainy season. A series of water dams were built over the years to control water flows from the two rivers for irrigation and water management purposes. The Francisco Zarco dam, located 25 km to the west, is the closest to the Velardeña Properties.

The geomorphology shows characteristics typical of a cycle of arid to semi-arid areas. There is an abundance of valleys and flat alluvial plains variably filled with erosional debris derived from adjacent highlands. The drainage systems are generally dendritic and poorly defined; many channels disappear when they reach the valley floor due to infiltration into poorly consolidated alluvial sediments.

The Project is located on the northwestern edge of the Meseta Central physiographical province, within the Sierras Transversas sub-province, on the eastern flank of the Sierra Madre Occidental mountain range. The village of Velardeña is in the valley floor set between two northwest trending ranges. To the west is the Sierra Santa María which rises approximately 300 m above the valley floor and, to the east, is the Sierra San Lorenzo rising approximately 750 m from the valley, which hosts the Velardeña and Chicago Properties.

6 HISTORY

6.1 Early History of the Velardeña District

Mining in the greater Velardeña District reportedly dates from the late 15th to early 16th century, primarily based on exploitation of oxide mineralization from outcropping or near surface mineralized structures such as the Santa María dome (Gilmer et al, 1988).

In 1888, the Velardeña Mining and Smelting Company was formed, a smelter was installed, and larger scale production began. At this time many of the smaller operations were consolidated within the larger group. According to Pinet (2000), a report written in 1913 recorded that in four years commencing in 1888, the Velardeña District in Durango produced 120,000 kilograms (kg) of Ag, 19,000 t of Pb and 519 kg of Au.

In 1902, the American Smelting and Refining Company (ASARCO) gained control of the operations and installed a new smelter processing 2,500 t/d, principally from the Santa María, Terneras and Reina del Cobre mines. Both the Terneras and Santa Juana veins were mined on a significant scale by ASARCO during the period from 1902 to 1926. The San Mateo vein supported a small-scale operation by an independent company at about the same time. Several other smaller mining companies were also active in the area such as Salida Mining Co., America Mexico Mining and Development Co. and Mexico Texas Co. (Mexican Mining Journal, 1909).

ASARCO and independent operators worked the mines continuously until 1926, when low metal prices and an unstable political environment contributed to their closure. In addition, the softer oxide ore was diminishing with depth, and the operations were encountering harder sulfide ore that made mining more difficult. After the mine closures, the smelter was dismantled and moved to San Luis Potosí (Pinet, 2000). Old reports indicate that early in the twentieth century, the average grade of the Terneras mineralization was 3.5 g/t Au, 835 g/t Ag and 3.85% Pb. Production statistics for the years 1920 to 1924 show that the Terneras mine produced 138,331 t with an average grade of 4.0 g/t Au, 419.7 g/t Ag, 2.1% Pb, 0.3% Cu and 2.5% Zn (Pinet, 2000, and references therein). In 1924, the Terneras shaft was sunk to the 14th level, and a cross-cut was driven to intersect the Santa Juana vein. Also, in 1924, it was reported that production from the Santa Juana vein totaled 37,000 t (in excess of 100 t/d), with an average grade of 5.9 g/t Au and 573 g/t Ag. Lesser production was also reported from the Santa Isabel chimney zone (562 t grading 0.6 g/t Au and 401 g/t Ag) and from the Industrial Minera de Mexico S.A. (IMMSA) controlled, El Pilar zone (863 t grading 2.3 g/t Au and 162 g/t Ag) (Pinet, 2000, and references therein).

After 1926, the mines in the district were worked on a small scale by local miners until the advent in 1961 of nationalization by the Mexican government, which precluded foreign ownership of the majority of shares in mining ventures. Therefore, ASARCO became a minority shareholder in IMMSA, who revived their interest in the area and IMMSA consolidated mineral concessions in two areas in the district. Exploration and development work recommenced in the Santa María and Reina del Cobre mines in 1968, and approximately 300,000 t/y were processed by IMMSA in their plant through 2002.

In 1969, IMMSA abandoned several mineral concession blocks, including those underlying the Terneras and San Diego mines. These were acquired by a consortium of individuals headed by Alejandro Gaitán of Torreón. During the 1970's through the late 1980's several mines in the district were exploited by gambusinos for direct shipping of the Au/Ag ores. Operations by the Gaitán Group on the project area consisted of the removal of material from the old waste dumps and several thousand tonnes of fill left in the stopes from earlier mining. This material was processed in a mill approximately 100 km from the mines. In 1990, Mr. Gaitán purchased a 50 t/d flotation mill located approximately 13 km from the mines. Ores from several veins within the Santa Juana mine were extracted and processed through the mill at a reported average grade of 396 g/t Ag, 5.9% Pb, 7.6% Zn and a mean grade of 4 g/t Au. The mill was operated intermittently at a low throughput due to a lack of mill-feed. The mines and mill were idled in early 1992.

6.2 Mining & Exploration Activities to 2011

In 1994, William Resources acquired the concessions owned by the Gaitán consortium via their Mexican affiliate BLM Minera. During that year, they carried out a Feasibility Study at the Velardeña Mine and commenced pre-production development and mine construction in July 1995 (Duke et al. 1995). From 1995 to 1997, William Resources carried out a surface mapping and sampling program on the various concessions, as well as an underground sampling program, principally on the Santa Juana vein system. William Resources also drove the Terneras adit, providing access to the 6th level of the Terneras mine, which in turn allowed access to the 12th level of the Santa Juana Mine via a pre-existing cross-cut. The Santa Juana winze was deepened 42 m to the 15.5 level, and a ramp was driven to the 17th level.

In 1996, William Resources purchased a 600 t/d processing plant and located it 3.5 km from the mine site. In May of that year they commenced treatment of dump material, which was mixed with minor quantities of development muck from the Santa Juana mine. William Resources ceased operation in mid-1997. Exploration from 1995 to 1997, apart from underground sampling and drilling, included general geological mapping, sampling of the veins exposed on surface and limited surface drilling.

In December 1997, ECU Gold (the predecessor company of ECU) purchased 93.48% of BLM Minera and 100% of Minera William from William Resources.

ECU restarted operations at Velardeña in January 1998 and the mine has been producing intermittently since that date. Production figures for the period 1998-2008 are summarized in Micon (2009); relevant data for the period 2009-2011 are tabulated below (**Table 6-1**).

In 2011, Golden Minerals merged with ECU. As a consequence, both Minera William and BLM are now wholly owned subsidiaries of Golden Minerals.

Table 6-1: Summary of Production by Mine Area – Velardeña Property (2009-2011)

Mine Area	Tonnes	Au g/t	Ag g/t	Pb %	Zn %	Cu %	Au Oz	Ag Oz	Ag Eq Oz
Chicago Oxide	39,788	1.66	59.78	1.01	0.92	0.06	2,119	76,477	182,445
San Juanes Oxide	34,344	1.49	135.97	0.52	0.46	0.09	1,650	150,140	232,658
Santa Juana Oxide	209,534	3.61	153.52	0.56	0.54	0.16	24,349	1,034,193	2,251,656
San Mateo Oxide	7,494	2.78	133.07	0.59	0.91	0.06	669	32,060	65,499
Terneras Oxide	13,318	1.05	155.35	1.50	0.57	0.03	448	66,518	88,899
Total Oxides	**304,478**	**2.99**	**138.87**	**0.66**	**0.59**	**0.13**	**29,235**	**1,359,388**	**2,821,156**

Mine Area	Tonnes	Au g/t	Ag g/t	Pb %	Zn %	Cu %	Au Oz	Ag Oz	Ag Eq Oz
Chicago Sulfide	20,982	1.82	82.28	1.67	1.87	0.07	1,227	55,508	116,845
San Juanes Sulfide	17,195	1.44	210.50	1.71	0.53	0.05	797	116,373	156,231
Santa Juana Sulfide	50,652	3.58	185.53	0.71	0.86	0.22	5,834	302,137	593,813
San Mateo Sulfide	756	2.35	156.35	0.65	1.06	0.06	57	3,800	6,660
Terneras Sulfide	3,303	0.66	203.42	1.05	0.84	0.04	70	21,602	25,103
Total Sulfides	**92,888**	**2.67**	**167.23**	**1.12**	**1.03**	**0.15**	**7,985**	**499,420**	**898,652**

Source: Velardeña Mine Geology Department

Exploration statistics for the period 1995-2008 are summarized in **Table 6-2**. These results have been summarized by Micon (2009) and have not been independently verified by the author.

Table 6-2: Summary of Historic Drilling on the Velardeña Properties (1995-2008)

Company	Target Area	Drill Program	# Holes	Total Length m
William Resources	Santa Juana	Underground	94	6,438
William Resources	San Juanes	Surface	6	973
William Resources	Terneras	Surface	3	282
William Resources	Other	Surface	6	750
	Total		**109**	**8,443**
ECU	Chicago	Surface	14	8,709
ECU	Santa Juana, Terneras	Underground BQ	11	5,533
ECU	various	Underground EX	59	2,750
	Total		**130**	**16,992**

Data taken and modified from Micon 2009 report.

6.2.1 Exploration Drilling (2009-2011)

Compared to previous exploration programs, relatively little drilling was completed during the period 2009-2011 (**Table 6-3**). The objectives were similar to previous programs; namely to confirm the continuity of the known veins, to discover new veins, and to test for deep projections of massive sulfide veins in the Santa Juana area. Based on a review of drill cores and data on-site, these objectives were at least partially achieved, notably with the discovery of deep, massive sulfide mineralization down dip of the A4 vein structure (**Figure 6-1**).

Table 6-3: Summary of ECU's Drilling Programs (2009-2011)

Description	Number of Drill Holes	Total Meters	Total Number of Samples
Surface	0	0	0
Underground (NQ)	3	1,234.6	483
Underground (EX)	35	1,212.1	214
Total	38	2,446.7	697

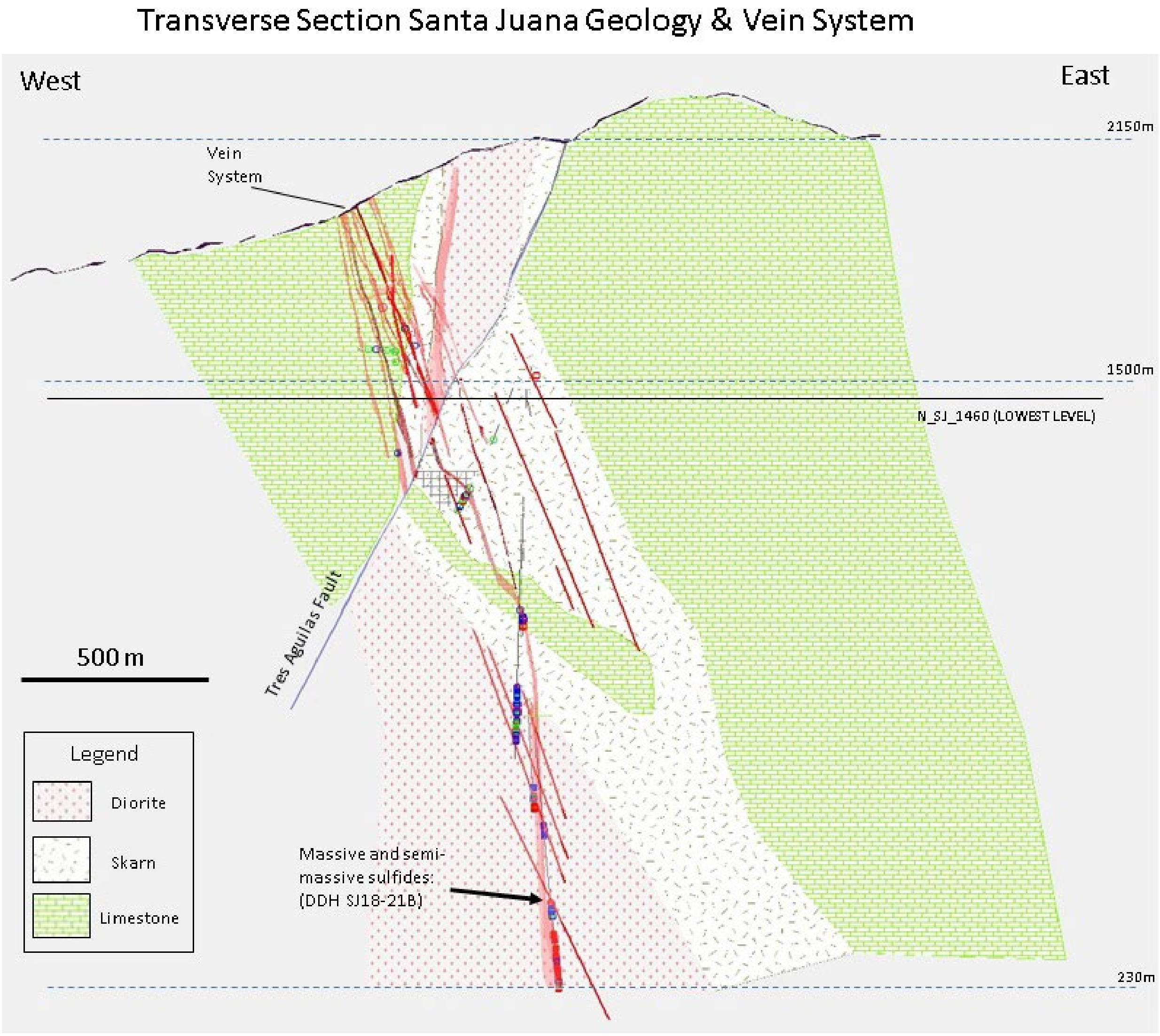


Figure 6-1: Transverse Section – Santa Juana

6.2.2 Underground Development (2009-2011)

In addition to drilling programs, ECU drove underground drifts, ramps and raises to develop the mineralization as well as explore the extent of the mineralization. **Table 6-4** summarizes the underground drifting, ramping and raising completed from 2009 to 2011.

Table 6-4: Summary of the Underground Drifting, Ramping and Raising (2009 to 2011)

Year	Drifts & Ramps m	Raises m
2009	3,368.6	770.0
2010	4,42.8	1,381.0
2011	4,218.9	1,457.5

6.3 Production From 2012 to 2014

After the merger with ECU in 2011, Golden Minerals continued production from the Santa Juana, San Juanes, Chicago, and San Mateo mine areas at an approximate rate of 400 tpd operating both Plant #1 and Plant #2 to treat sulfide, oxide, and mixed mineral types.

As a result of the substantial declines in silver prices in early 2013, Golden Minerals decided to temporarily cease mining and processing operations at the end of June 2013. Just prior to ending operations, the San Mateo ramp was successfully connected with the Santa Juana workings on the Santa Juana 20 level, allowing full access to the deeper portions of the Santa Juana vein system without using the internal Santa Juana winze.

During the shutdown period in 2013 and the first half of 2014, exploration drilling was completed from the San Mateo ramp to test the Terneras and San Mateo vein systems. Mining restarted July 1, 2014 with the commissioning of a new ramp to access deeper levels on the Terneras and San Mateo veins, as well as the Roca Negra vein. Mine production rates ramped up over the second half of 2014 approaching the target rate of 285 tpd by year end. Mined material was stockpiled through October 2014 and mill processing recommenced in November 2014 at the newly refurbished Plant #1 flotation mill. **Table 6-5** summarizes yearly production from the Project as presented in the company's annual 10K filings, with adjusted Ag equivalent values for consistency with the Golden Minerals most recent filings.

Table 6-5: Summary of Production by Year – Velardeña Property (2012-2015)

Year	Tonnes	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Payable Oz Au	Payable Oz Ag	Payable Oz Ag Eq
2012	185,907	2.02	125	0.28	0.41	6,435	457,265	907,715
2013	72,063	2.56	163	0.36	0.53	2,349	252,256	416,686
2014	14,322	1.57	119	0.76	1.08	194	28,746	42,326
2015	80,736	2.63	160	0.89	1.19	1,976	326,651	464,971

Source: Year-end 10k Filings and Company Documents
Notes: Ag:Au = 70:1

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological & Structural Setting

This information was sourced from NI 43-101 Technical Report Velardeña Project, Durango, State, México dated on January 01, 2012 prepared by CAM, including modified minor details.

The Project is located at the easternmost limit of the Sierra Madre Occidental, near its boundary with the Sierra Madre Oriental (Mesa Central sub-province). The deposits of the Sierra de Santa María and Sierra San Lorenzo, like many other polymetallic, hydrothermal deposits in northern Mexico, are situated along this fundamental boundary which separates thick Tertiary volcanic sequences with Mesozoic basement rocks to the west from Mesozoic carbonates with Paleozoic and older basement to the east.

The regional geology is characterized by a thick sequence of limestone and minor, calcareous clastic sediment of Cretaceous age, intruded by Tertiary plutons mostly of felsic to intermediate composition. During the Laramide event, sediments were subject to an initial stage of compression which resulted in formation of large amplitude, upright to overturned folds generating the distinctive strike ridges of limestone which dominate topography. Fold axes trend northerly in the northern part of the region but are warped or deflected to west northwest azimuths in the south. The northeast trending hinge line or deflection which controls this fundamental change in strike passes through the Velardeña district (**Figure 7-1**).

Tertiary volcanic and semi-consolidated alluvial sediments survive as erosional remnants on earlier basement rocks. The volcanic rocks may have been derived from an eruptive center located west of the pueblo of Velardeña where andesites have yielded age dates of 45 million years ago (Mya) (Gilmer et al, 1988).

Tertiary stocks intruded the Cretaceous sediments in the Velardeña area along an axis subparallel to the hinge line described above, resulting in formation of a series of complex limestone domes cored by the younger intrusive rocks (i.e. the Sierra de Santa María, Sierra de San Lorenzo, and San Diego Dome). The Santa María quartz latite porphyry intrusion, west of the village of Velardeña, has yielded a potassium-argon (K-Ar) date of 33.1 Mya (Gilmer et al, 1988 and references therein).

Intrusions range in composition from mafic diorite to felsic alaskite and rhyolite. Thermal metamorphism of sediments at and near intrusive contacts is widespread, generating calc-silicates, hornfels, and bleached/marbleized limestone. Higher temperature, calc-silicate rocks are characterized by the prograde assemblage garnet - wollastonite and by the absence of pyroxene. The various mineral deposits of the Velardeña District occur in close proximity to intrusive centers, contact aureoles, and accompanying alteration zones. Mineralization has been dated at approximately 31 Mya (Gilmer et al, 1988), suggesting a genetic as well as spatial association with the intrusions.

Multiple, high angle, northwest trending faults have been mapped throughout the district; these are sub-parallel to the terrain boundary described above and are therefore likely candidates for deep, basement-penetrating structures which may have served as magma conduits (Clark et al, 1988). Reactivation of the northwest structures and formation of northeast trending faults resulted in a grid of younger, calcite-filled structures which off-set mineralized veins. This is well demonstrated at the Terneras mine where the northeast trending Tres Aguilas fault offsets the mineralized northwest trending Santa Juana veins.

Figure 7-1 illustrates the location of the Velardeña mining district with respect to regional lithologic and structural features.

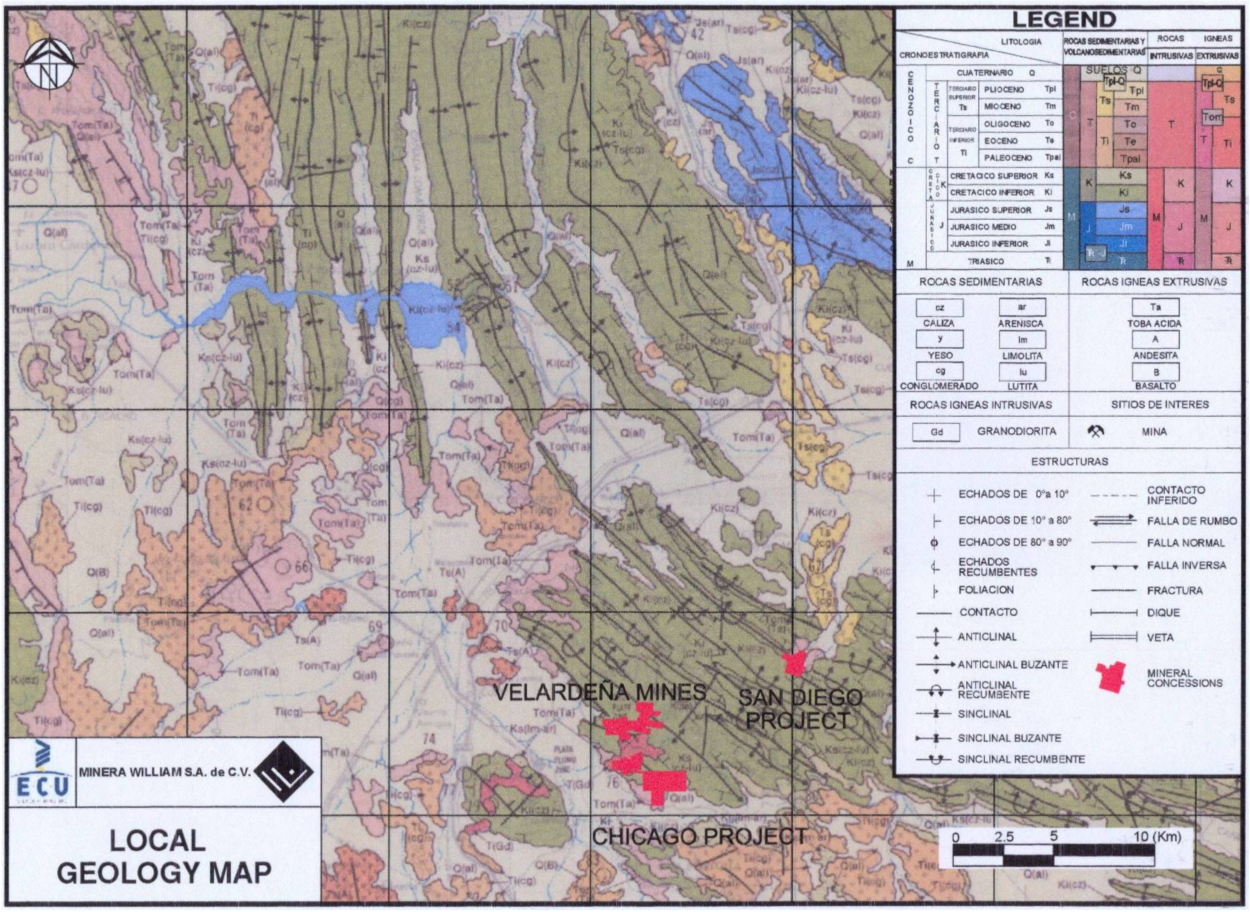


Figure 7-1: Local Geology Map

7.2 Property Geology

7.2.1 Velardeña Property

Medium to thick-bedded limestone of the Cretaceous Aurora Formation comprises basement rocks in the project area. Limestone was first folded then intruded by the multiphase diorite/monzo-diorite Terneras stock and related dikes of Tertiary age that outcrop over a strike length of approximately 2.5 km. In detail, intrusive contacts range from sharp to broad zones characterized by the presence of numerous large, partially metamorphosed blocks of limestone. Alteration of host carbonates consists of a broad front of bleaching and marble formation, with more localized calc-silicate and hornfels. Although intrusive rocks appear fresh in general, alteration and local endoskarn development occurs near contacts. The diorite stock and the contact zone between limestone and intrusive rock primarily host the veins of the Santa Juana, Terneras, San Juanes, and San Mateo deposits (**Figure 7-2**). Veins extend into relatively unaltered limestone especially in the northwestern portion of the Santa Juana veins and eastern portion of the San Juanes vein.

The Velardeña property is transected by a series of northeast to northwest striking, west dipping, post-mineral normal faults. From east to west these are the Tres Aguilas, Los Bancos, Buenaventura and Ordenanza faults which are generally characterized by meters-thick banded calcite vein filling. These normal faults demonstrate west-side-down displacements with the result that the western blocks expose higher portions of the hydrothermal system, have a higher calcite content and generally lower precious metal contents.

Two main vein systems are present on the Velardeña property. The first is the northwest striking system found in the Santa Juana deposit, while the second is the east-west trending vein array which includes the Terneras, San Juanes, Roca Negra and San Mateo deposits. In **Figure 7-2** vein traces are projected to surface and do not cut alluvium.

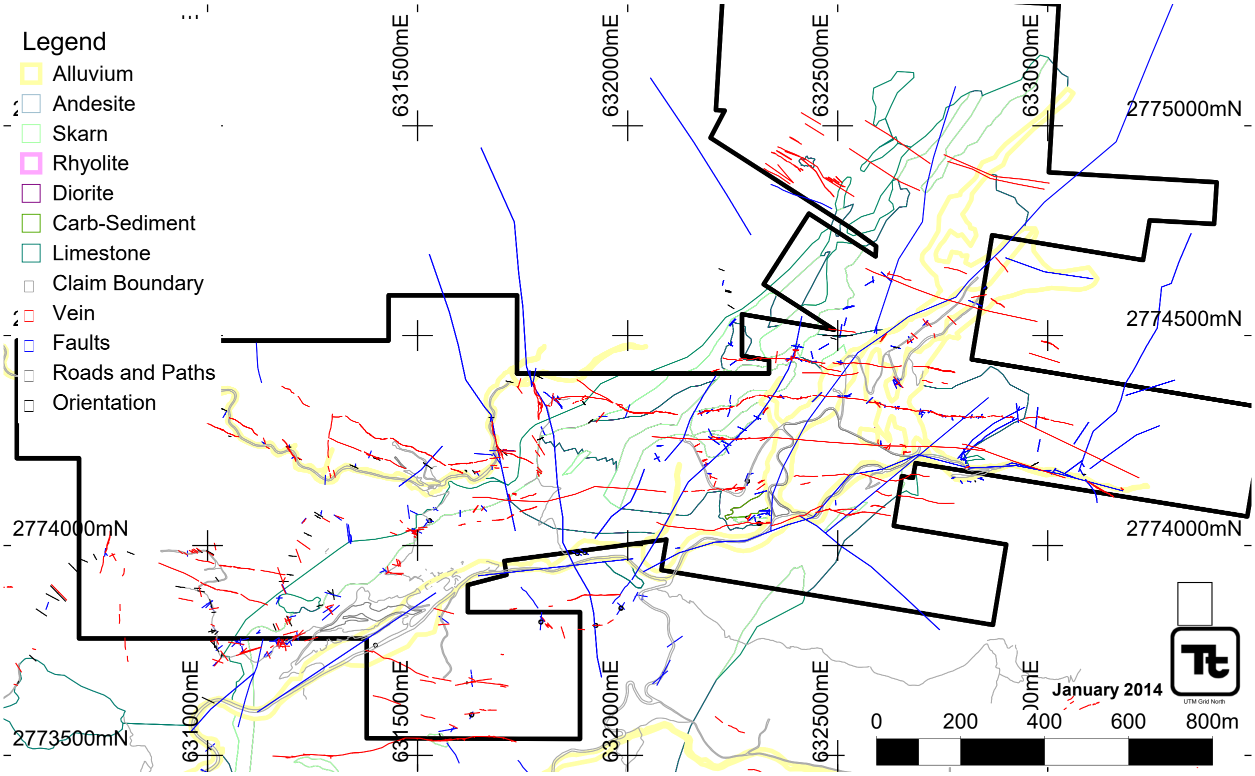


Figure 7-2: Velardeña Property Geology Map

7.2.2 Chicago Property

The geologic setting of the Chicago property is very similar to that at Velardeña (**Figure 7-3**). The oldest rocks outcropping at Chicago are folded limestone of the Aurora Formation which were intruded by Tertiary diorite stocks and dikes. Intrusive rocks occupy the western portion of the property with a northeast orientation. The limestone-diorite contact exhibits widespread recrystallization and marble formation overprinted by a distinctive green calc-silicate alteration dominated by grossular garnet and lesser wollastonite.

As at Velardeña, a system of post-mineralization faults striking northwest-southeast cuts and locally displaces mineralized structures. These faults are normally filled with calcite and can have widths up to 10 m near surface.

In the Chicago mine, rhyolitic volcanic rocks and calcareous conglomerate of the Ahuichila Formation unconformably overlie the mineralized sequence across the eastern half of the area. Mineralization is similar to that encountered at Santa Juana mine in terms of mineralogy, host rocks, geometry of the structures and vein continuity. The difference between the two is orientation: northwest strike, dipping to the northeast for the Santa Juana system; instead of northeast strike, dipping to the southeast for the Chicago system. **Figure 7-3** shows the geology of the Chicago area with vein traces projected to their assumed surface intersection, veins are not hosted in alluvial material.

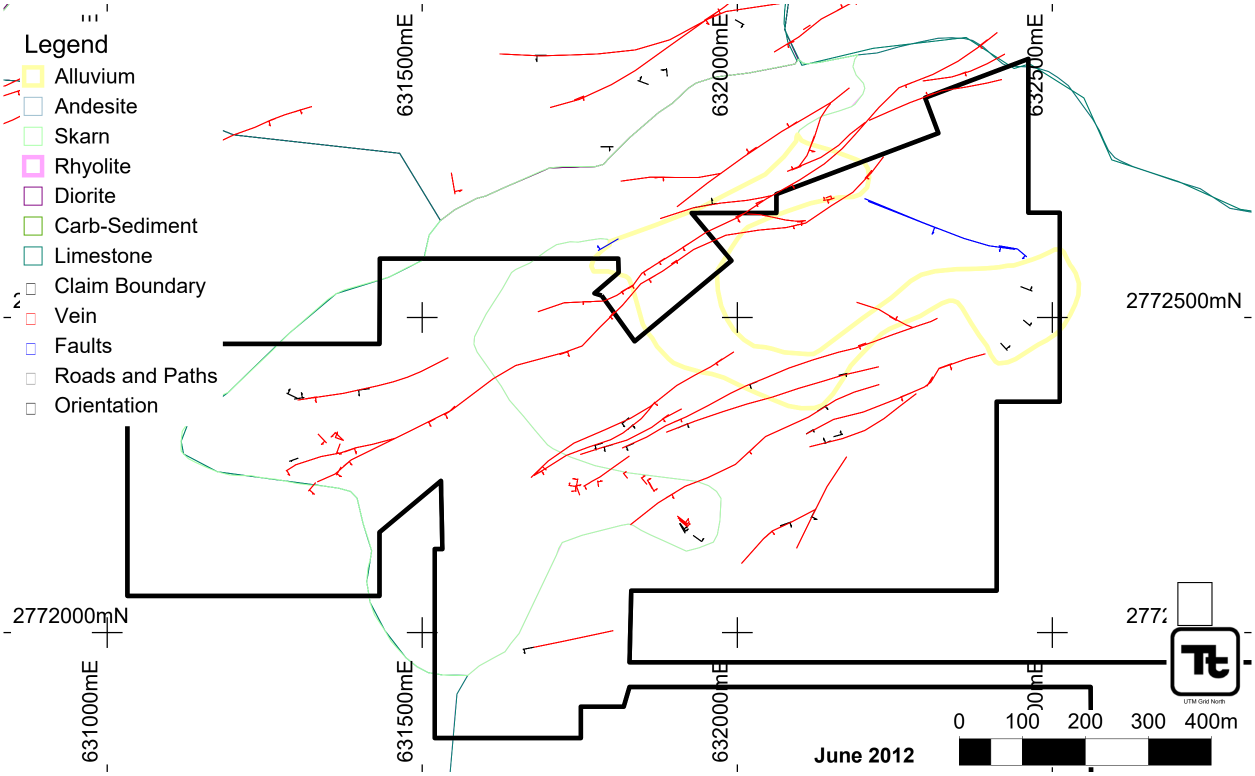


Figure 7-3: Chicago Property Geology Map

7.3 Mineralization

7.3.1 Regional Setting

The Velardeña mineralization system sits near the northern end of the Mexican Silver Belt, a 1,200 km long, northwest trending corridor of Au-Ag and Ag-Pb-Zn vein deposits. Within the belt, the Fresnillo, Guanajuato, Zacatecas and San Francisco del Oro-Santa Barbara districts have all produced in excess of 10,000 t of Ag. Currently Mexico's largest Ag production comes from the Santo Niño and San Carlos veins developed by Peñoles in the Fresnillo district, which have been traced for over 4 km along strike and 500 m in depth with widths up to 4 m (Micon, 2009; Trejo, 2001).

In addition to the mines and prospects which are the subject of this report, numerous other historic workings exist within the Velardeña District (**Table 7-1**).

Table 7-1: Geologic Characteristics and Grades for Deposits of the Velardeña District

Mine	Deposit Type / Genesis
Los Azules	Sulfide bodies /felsic intrusive contacts (mesothermal)
San Nicolás	Breccia hosted / felsic intrusive related (mesothermal)
Industria	Veins / limestone, intrusive hosted (epithermal)
Santa María	Massive sulfide replacement bodies along dike contacts (mesothermal)

Gilmer et al, 1988

Mineralization is variously described as "mesothermal" or "epithermal" based on temperature determinations from fluid inclusion analyses (Gilmer et al, 1988). Mesothermal deposits are typically sulfide-rich, lens-like masses dominated variously by pyrite, arsenopyrite, or pyrrhotite. Epithermal deposits in the Industria Mine are located distal to the intrusive core and consist of tabular veins exhibiting banding, crustification and open-space filling textures. In all cases, there is a persistent association of mineralization with intrusive rocks and with contacts between felsic intrusive rocks and (altered) carbonate host rocks.

7.3.2 Mineralization at Velardeña

Mineralization consists primarily of calcite-quartz veins with minor calc-silicate hosted ("skarn") and massive sulfide replacement bodies. All mineralization is essentially polymetallic, Ag, Au, Pb, Zn plus or minus Cu. Individual veins are usually thin (0.2 m to 0.5 m) but remarkably consistent along strike and down dip. Coxcomb and rhythmically banded textures are common in some vein exposures. Historical production in the district has been primarily from the oxide portions of the veins that can extend to depths of several hundred meters. Previous workers have suggested a vertical zonation with increasing Au:Ag and Cu:Pb with depth (Pinet, 2000). Physical characteristics of the main vein sets are summarized below as **Table 7-2**.

Table 7-2: Physical Characteristics of Select Veins and Vein Sets – Velardeña Mine

Vein	Orientation	Width	Minimum Dimensions Strike m x Vertical m	Host Rocks
Santa Juana Series				
NW Subset 1 (Santa Juana, A 5-7)	NW curvilinear	0.2 - 1.0	350 x 400	limestone, intrusive, skarn
NW Subset 2 (CO, CC, C1, G1, A 1-4, B's, D1, DD, E)	NW linear	0.2 - 1.0	Variable by vein, up to 600 x 1200 (CC)	limestone, intrusive, skarn
Trans Set	EW/steep S	0.2 - 1.0	100 x 600	limestone, intrusive skarn
Terneras	EW/70-85N	0.3-2	1500 x 650	Intrusive>limestone
San Juanes	EW/85N	0.05-1.9	950 x 600	limestone, intrusive, skarn
San Mateo	EW/75N	0.4-0.5	700 x 500	intrusive, skarn>limestone
Roca Negra	EW/75N	0.15 - 1.15	500 x 600	intrusive, skarn

The mineralization at the Chicago property is similar to that encountered at the Santa Juana mine in terms of mineralogy, host rocks, geometry of the structures and continuity at depth and laterally. The difference between the two is geometric, northwest dipping to the northeast instead of northeast dipping to the southeast.

Characteristics of the mineralization are summarized below based on previous reports and the author's observations:

- Veins occur in limestone, marble, calc-silicate and intrusive host rocks. The geometry of the veins is typically wider but more irregular in the limestone. In addition to being more consistent in width, veins within skarn and intrusive rocks tend to be narrower but higher grade with respect to precious metals. Skarn is the least favorable vein host.
- Although individual veins are typically narrow, zones of vein intersections and certain contacts between intrusions and limestone have focused brecciation and silicification, yielding mineralized chimneys which can reach 7 m in width and extend for tens of meters vertically.
- Within the Santa Juana sector, a zone of sheeted veins has been discovered near the intersection of northwest and east west trending veins. The overall dimensions of this corridor are approximately 500 m along strike and 250 m vertically (level 12 to level 18), with widths up to 100 m.
- Gangue minerals consist of calcite and quartz, which generally represent less than 20% of the volume of individual veins. Higher grade segments of veins generally conform to areas dominated by quartz or quartz-calcite mixtures; calcite rich zones are generally low grade. There is a distinct tendency for the upper portions of many of the veins to be calcite dominant, hence lower grades. Lateral changes in the gangue mineral composition have been observed, suggesting controls other than elevation are at work.
- Depth of oxidation is quite variable and the distribution of oxide and mixed mineral types complex. Within limestone host rocks, the veins are oxidized down to depths of up to 450 m. Oxides are rare in intrusive and calc-silicate host rocks, reportedly encountered only near the Tres Aguilas and Los Bancos faults, due to increased fracture-controlled permeability and fluid flow.
- The alteration zone along vein margins is generally less than 10 centimeters (cm) and is comprised of argillic alteration and silicification of the intrusive and skarn host rocks, and localized silicification and recrystallization of limestone. While precious and base metal mineralization is generally confined to the veins, sulfide stringers were observed extending outwards along bedding planes within altered limestone.
- Underground drifting and drilling suggest many of the veins are open at depth below the 19th level.

7.3.3 Mineralogy & Paragenesis

Little detailed work has been carried out on the mineralogy of the veins. The sulfide assemblages are quite diverse within the zone of hypogene mineralization and mineralogy is even more so within the zone of partial oxidation. Accurate mapping of this transition has important implications with respect to metallurgical recoveries, vein density and metal grade.

7.3.3.1 Oxide Mineralogy

The oxide portions of the veins are composed of oxides, halides, carbonates and remnants of sulfide minerals. Concentrations of Cu oxides and carbonates are commonly seen along vein selvages in underground workings.

7.3.3.2 Sulfide Mineralogy

Within the sulfide zone, mineralization consists primarily of galena and sphalerite with lesser amounts of chalcopyrite, tetrahedrite, freibergite, and sulfosalts. Accessory sulfides including arsenopyrite, stibnite, pyrite, and pyrrhotite are locally abundant. Free gold or electrum is rarely seen as microscopic inclusions in pyrite and arsenopyrite.

Disseminated and stringer pyrite is very common in all rock types below 500 m depth and persists to much shallower levels within intrusive and calc-silicate host rocks.

7.3.4 Controls on Mineralization

7.3.4.1 Stratigraphic Controls

As described previously, veins in the district are localized in intrusive rocks and near contacts between intrusions and thermally metamorphosed country rocks but extend up to one kilometer away from these contacts. In detail, however, veins do not conform to these contacts, but in many cases cross at high angles and mineralize limestone, skarn/marble, and intrusive hosts. Observations summarized above suggest that, on average, veins within intrusive are narrower, more regular in form, and higher grade than those in limestone. Skarn is typically a poor vein host with widths and grades less than in diorite or limestone hosts. Observations by mine geological staff suggest that veins may be genetically related to the intrusion of late, felsic porphyries (**Figure 7-4**).

Although data are sparse, it seems likely that at least some of the deeper, massive sulfide mineralization intersected in past drilling will possess more obvious control by stratigraphy, particularly skarn assemblages, than is typical at shallower levels. This would in turn suggest the possible presence of larger mineralized bodies such as have been exploited around the Santa María dome.

7.3.4.2 Structural Controls

Observations underground confirm that at least some veins show an intimate relationship with brittle faulting. In the Santa Juana deposit, two main fracture sets are observed. The most economically significant is a steeply dipping, northwest-trending set that has created dilatant zones that acted as a major control for vein emplacement. A second more spatially extensive fracture swarm trends 110° and, although less obvious, appears to control the orientation of the Trans veins. These veins dip steeply south and, where they intersect the northwest-trending vein set, produce broader stockwork or breccia zones which can be up to seven meters in width. The east-west fracture set also controlled the localization of the parallel Terneras, San Juanes, San Mateo, and Roca Negra veins.

Cross-cutting relationships between the two vein systems are ambiguous, indicating that the two vein sets probably formed contemporaneously as part of a conjugate fault system. A similar structural setting is reported to occur in the Santa María mine.

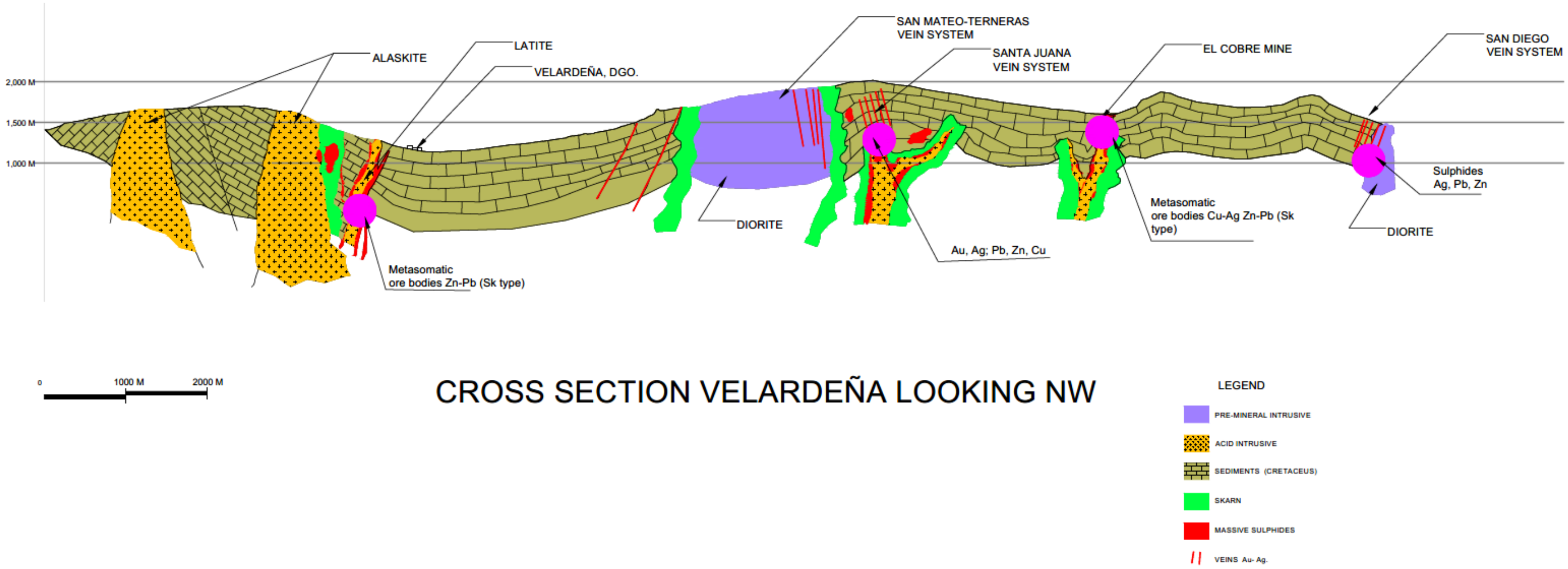


Figure 7-4: Velardeña Section Looking Northwest

8 DEPOSIT TYPES

Although detailed petrologic studies of veins in the Velardeña property have not been completed, individual deposits within the nearby Santa María dome have been studied in some detail and found to correspond to both shallow epithermal and deeper-seated mesothermal styles of mineralization. Epithermal veins, often displaying banded and open-space-filling quartz, occur at the Industria Mine where they are clearly distal to the main intrusive mass (Gilmer et al., 2008). The higher-level veins at Velardeña appear to be of this type. Many veins, especially at deeper levels in the Santa Juana and Terneras mines, are dominated by high modal percentages of coarse and fine grained, polymetallic sulfides, have little silicate gangue, and occupy a position within and proximal to intrusions and their thermally metamorphosed aureoles.

True epithermal veins occur at Velardeña, but at depth the majority of veins, breccias, and massive sulfide replacements are mesothermal in character, commonly contain arsenopyrite, and may be related to a deeper intrusive source.

Exploration strategies at the Project are informed by the above model concepts. Current exploration models explore the deposit in the context of vein controls. It has been recently discovered that east-west trending vein sets, in general, tend to have lower concentrations of deleterious elements (As and Sb) and slightly thicker true widths. Drill hole exploration continues to target these veins down dip of current development as the mineralization style transitions away from more typical epithermal Ag-Au veins to deeper-seated mineralization elevated in Au and base metals.

9 EXPLORATION

The Project has been extensively explored from the surface using geologic mapping, vein mapping, and vein sampling. Underground exploration consisted of diamond drilling, geologic level mapping, vein level mapping, vein sampling, drift and stope development. The author is unaware of any geophysical surveys completed on the property.

9.1 Recent Underground Development

In addition to exploration drilling programs, Golden Minerals has driven underground drifts, ramps and raises to develop the mine as well as explore the extent of the mineralization. **Table 9-1** summarizes the underground drifting, ramping and raising completed from 2012 to 2014 since the last Technical Report has been published.

Table 9-1: Summary of the Underground Drifting, Ramping and Raising (2012 to 2014)

Year	Drifts & Ramps m	Raises m
2012	5,995	1,630
2013	1,991	221
2014	2,136	427

Source: Mine Engineering Department

9.2 Sampling Methods & Approach

Channel samples are taken at drift faces, cross-cuts and stope walls and/or backs according to the following guidelines:

- During level mapping, geologists paint sample locations on the back or development face to guide samplers.
- Samples are collected by chiseling out the painted area, ideally cutting a 5-7 cm wide sample. Often this is not achievable due to rock hardness.
- The sample widths range from 0.2 m to 2.5 m.
- The sample's weight is usually between two kg and five kg. The sample contains a minimum of ten rock pieces (<20 cm in size) as well as fine material.
- Sampling is carried out as perpendicular to the vein strike as possible and the true width is measured by sighting the vein dip and tilting the measuring tape accordingly.
- Stope and face samples are collected at 3 m intervals across strike. Wall rock and vein material are sampled separately. When dictated by geological features, samples are taken at closer intervals.
- Sampling along cross-cuts is carried out continuously.

The locations of the samples are initially determined by means of sighting and taping from established survey markers and annotated on the level plan. The locations are subsequently corrected by the installment of a new survey marker when the drift has been developed completely.

Channel sampling is subject to numerous sources of error, particularly relating to the differential hardness of material being sampled, and the tendency to include a disproportionate volume of softer rock. Diligent and systematic collection of channel samples generates a very large data set which in most cases is statistically representative, but never completely free of errors or potential bias.

The previous author observed the collection of several channel samples in the Chicago mine and noted that the procedures used conformed to those outlined above and follow accepted engineering practices for channel sampling. The current author has not observed the collection of channels at the Project but has spot checked sample locations throughout the mine and thoroughly discussed procedure with the mine staff. The author concludes channel sampling procedures used at the Project result in samples which are reasonably representative of the mineralization and meet industry best practice guidelines for this type of sampling. The resulting data is sufficient to support the estimation of Resources.

9.2.1 Significant Results

The channel database contains 32,006 sample intervals, of which 14,534 intervals have been interpreted as intersecting a named vein. **Table 9-2** shows grade statistics for channel intervals within the database and those identified as on-vein.

Table 9-2: Channel Sample Data Statistics

Dataset	Selection	Count	Mean Ag g/t	Mean Au g/t	Mean Pb%	Mean Zn%	Mean Apparent Thickness
Channel	All	32,006	281	5.1	1.6	1.6	0.66
Channel	On Vein	14,534	518	9.2	2.8	2.7	0.47

9.3 Exploration Potential

Strike extents for most known veins have been identified by exploration but in many cases mineralized shoots at depth have not yet been defined nor have the down dip extensions been condemned. It is likely that as deeper levels are developed additional mineralized shoots will be identified.

The current exploration strategy is focused on the potential of the deeper sulfide portions of the Terneras, San Mateo, Roca Negra, Santa Juana (A1, A4, C1, CC) and Chicago (Chicago and Escondida) veins.

Exploration potential in the Santa Juana area in the near term is focused on developing the A4 vein in the Tres Aguilas southeast fault block where the A4 vein appears to have a greater and more sustained thickness than the other Santa Juana veins. Long term exploration potential in the Santa Juana area is indicated by deep wedge drilling also under the Tres Aguilas southeast fault block where encouraging intervals of massive sulfide mineralization hosted within skarn have been observed (**Figure 9-1**).

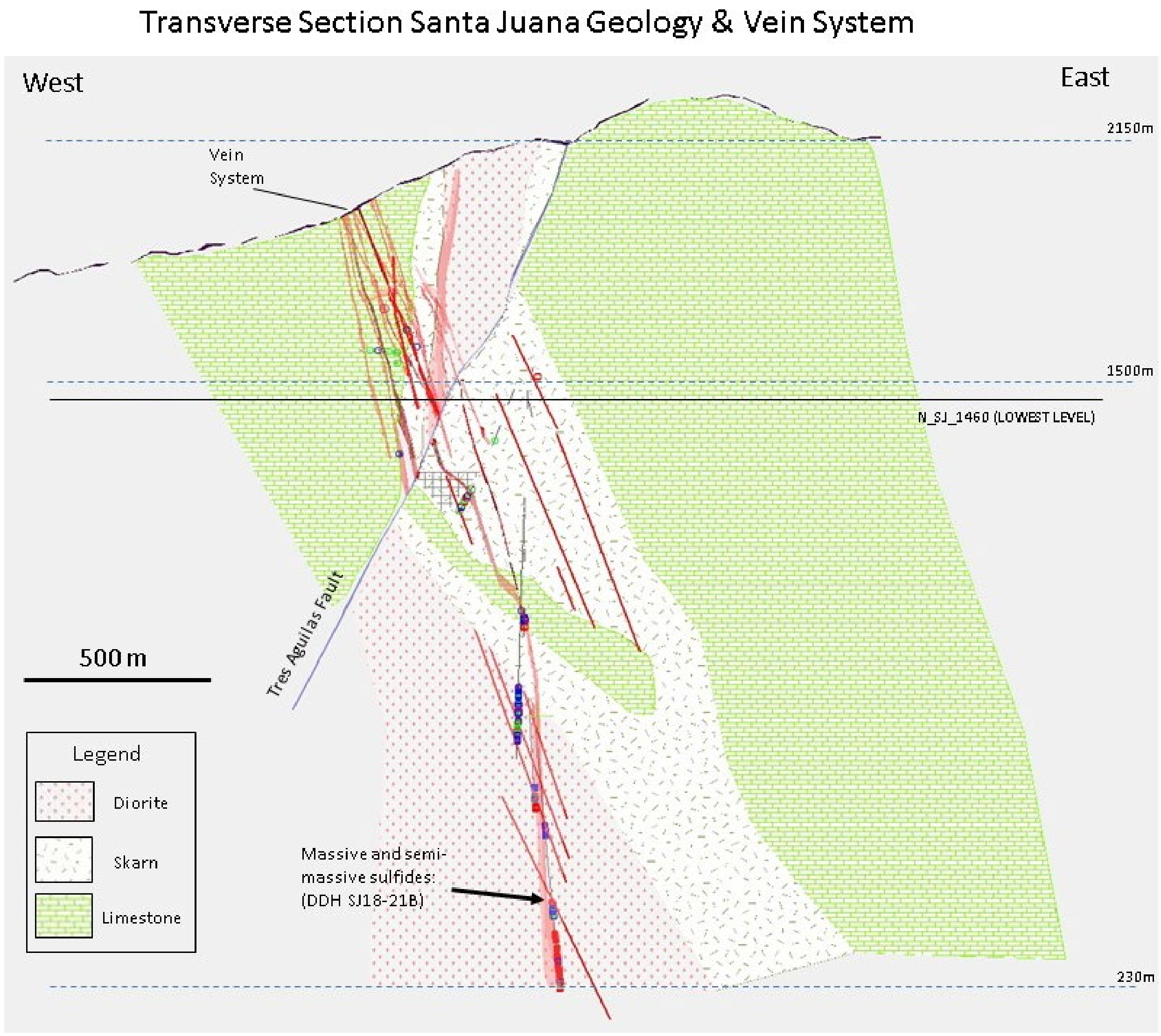


Figure 9-1: Santa Juana Section

10 DRILLING

Since the 2012 technical report, Golden Minerals completed 47 diamond core holes. The majority were completed in 2014 and drilled from underground targeting: the San Mateo, Terneras, and Roca Negra veins. Four holes were drilled from underground in the Santa Juana area targeting primarily the A4 vein.

Table 10-1: Drilling 2013-2014

Year	Area	Number of Holes	Length m
2012	Santa Juana	4	186
2014	San Mateo, Terneras, Santa Juana	43	8,875
	Total	**47**	**9,062**

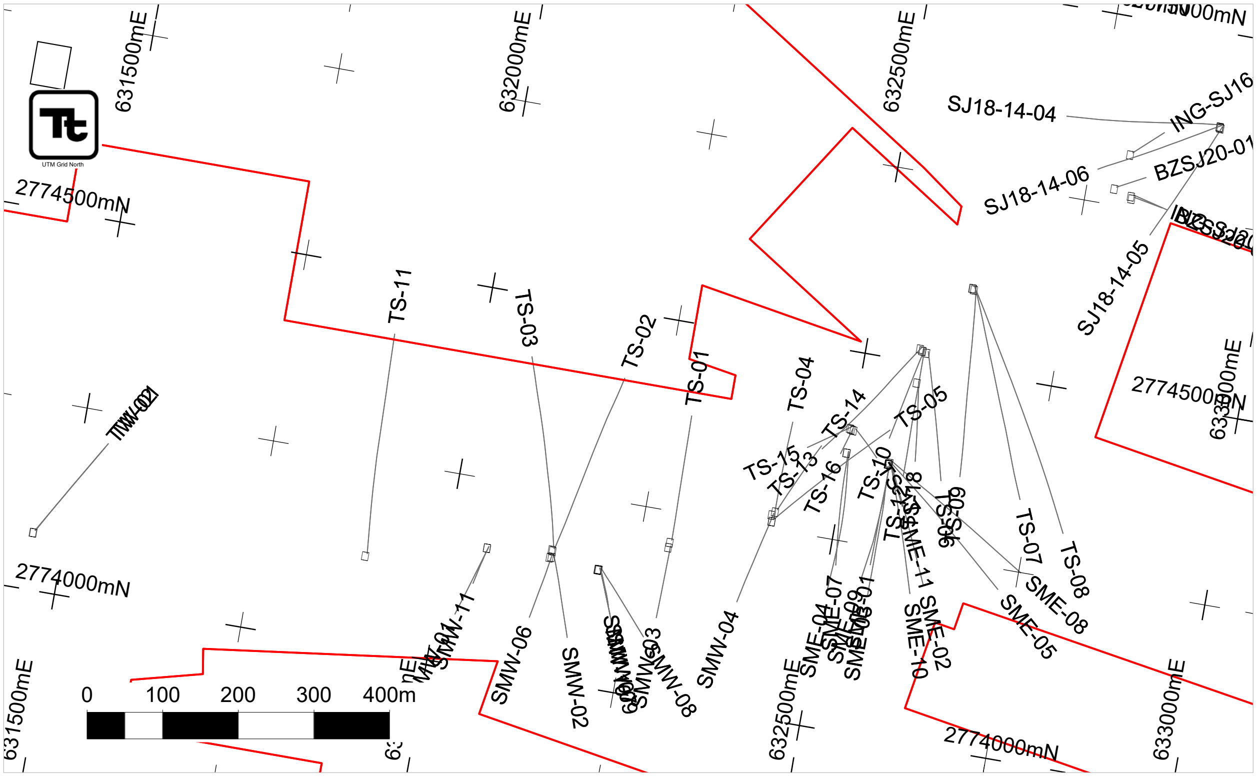


Figure 10-1: Drill Hole Location Map 2013-2014

10.1 Sampling Methods

Diamond drill core samples are taken according to the following criteria:

- Drill core is split using a manual rock splitting device or using a core saw.
- Samples are taken from core sections with visible evidence of mineralization and from 1.5 to 2.0 m of surrounding wall rock.
- Wall rock between two veins is sampled when the distance is less than 6 m.

- The information recorded in the drill logs for each sample includes depth, length, core angle and rock/ore type.

Mineralized sample intervals for either the NQ or BQ size core have a minimum core length of 20 cm and a maximum length of 1 m. For areas sampled outside of the mineralization the maximum sample length for the NQ core is 1.20 m and for BQ core the maximum sample length is 1.50 m. In general, the maximum sample length is 1.50 m except for those areas in which two veins can be joined together in which case the maximum sample length is 2 m.

Sampling was conducted on core not only with visible evidence of mineralization, such as veins and stringers, but also on barren core to preserve the sampling continuity in between mineralized zones and to test for broad zones of lower grade material as well. The sampling of the wall rock next to the zone of mineralization also assists in understanding the grade of the external dilution associated with mining some of the mineralized zones on the Velardeña properties.

Manual splitting of the core can be subject to several sampling biases based usually on the hardness of the material being split. In the case of very hard core, the core may twist in the splitter which may result in uneven core fragments and in a slightly greater split than 50% being sent to the assay laboratory or left in the box as a representative sample. In the case of soft core, the core may crumble when being split or may split along natural fracture lines which again results in uneven core representation. Also, to prevent contamination, the splitter and pans used to collect the samples must be cleaned after each sample. Despite the potential to introduce a bias into the sampling procedure as a result of uneven sample sizes, the splitting of drill core continues to remain a common practice in the exploration and mining industries.

Bazooka drilling is undertaken from the development headings in order to identify the width of a zone where the hanging wall is not visible or where a secondary mineralized system is suspected as in the case of the sheeted veins. Cores obtained from these programs are not split and are sampled completely.

These drill core sampling procedures are consistent with commonly practiced procedures used throughout the mineral industry. Along with in-house standards, blanks, and duplicates included in the sample stream, routine check assays are conducted on the samples by a second laboratory as well.

Drill core sampling practices are consistent with industry standards adequate for use in preparing a Mineral Resource estimate.

10.2 Core Recovery

In the case of large diameter core (HQ, NX, BX), recoveries were reported to average around 60% in oxide mineralization and 90% - 97% in the sulfides. For the smaller Bazooka (EX) drill cores, overall recoveries ranged from 30% - 40%. Recovery for Bazooka cores are poor and may result in underestimation of mineralized widths and grades. In the case of bazooka drilling, drifting is usually conducted afterward to identify the true nature of the mineralization, especially if a secondary zone or vein is suspected. Typically, chip samples from such drifts result in higher grades than initially indicated by the bazooka drilling.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section is sourced from CAM's TR, 2012.

Sample preparation, analyses and security procedures followed by Minera William meet industry common practice standards and are adequate to support the estimation of Resources. The quality control (QC) sampling results throughout the campaigns and laboratories are typical of an operation given the amount of throughput and data handling. Current drill hole analyses are completed by ALS Chemex in Vancouver, Canada (ALS Chemex) and mine channel and mill samples are tested at the on-site Labri laboratory facility (Labri), constructed in 2013. A review of QC samples analyzed since 2012 suggests the on-site laboratory could benefit from further improvements and increased real-time review of performance. Based on the QC sample review, the analytical results determined by the on-site laboratory are less accurate and less precise than those determined by ALS Chemex, however the sampling density is much greater than the drill hole samples and in general the extrapolation is less and therefore the determinations are adequate and do not require the same precision and accuracy as the drill hole samples.

Previous quality control procedures and results have been reviewed by previous authors and those reviews have resulted in improved protocols and performance, but ultimately previous authors concluded the data is sufficient to support estimation of Resources. The drill hole and channel analytical databases are extensive and include results from several campaigns and laboratories. **Table 11-1** details when each laboratory has been used, and the accompanying umpire laboratory.

Table 11-2 details the accreditation and the relationship to Golden Minerals of each laboratory used. Data within both databases, regardless of testing laboratory, is considered current and equivalent.

Table 11-1: Analytical Laboratory Listing

Time Period	Laboratory Used	Umpire Laboratory Used
Pre-2009	Labri (on-site), Ensayes y Representaciones, S.A. (ERSA)	Servicio Geológico Mexicano (SGM), ALS Chemex
2009 to 2013	Labri (on-site), Ensayes y Representaciones, S.A. (ERSA), SGS	SGS
2013 to Present	Labri (on-site), ALS Chemex	Pulp Duplicate Resubmittal to ALS Chemex

Table 11-2: Laboratory Accreditation and Independence

Laboratory	Accreditation	Relationship
Labri	Not Accredited	Not independent, operated by Golden Minerals
SGM	Not Accredited	Independent of Golden Minerals
ERSA	Not Accredited	Independent of Golden Minerals
SGS	ISO 17025	Independent of Golden Minerals
ALS Chemex	ISO 17025	Independent of Golden Minerals

Current drill hole analyses are completed by ALS Chemex and channel samples are tested on-site at the Labri laboratory. ALS Chemex is independent of the issuer and is ISO 17025 accredited: the accreditation of ALS Vancouver encompasses preparation processes completed at ALS Chihuahua. The on-site laboratory is not independent of the issuer and is not accredited. Tetra Tech inspected the on-site laboratory in December 2019 and found the facility and the procedures followed to be of adequate standard.

11.1 Sample Preparation

11.1.1 Diamond Drill Core Samples

Drill hole samples are prepared by splitting the core with a manual rock splitting device or core saw using personnel who have been hired by Minera William for this purpose. The Minera William personnel who conduct the core splitting and sampling are supervised by Minera William's geological staff in order to ensure the integrity of the core splitting and sampling procedures. Half of the core remains in the core box with its identifying ticket while the other half is bagged with a matching ticket. The samples are delivered by mine staff to ALS Chemex's preparation laboratory in Chihuahua or Zacatecas where they are shipped to ALS Chemex in Vancouver for analysis.

11.1.2 Underground Chip Samples

Development chip samples are collected by sampling support staff who are instructed to chip away sample transects painted by the geologist. Sampling is observed by geologic staff. Samples are bagged and transported to the on-site laboratory for preparation and analysis.

11.2 Security, Storage & Transport

11.2.1 Core, Pulp & Reject Storage

The core is stored at the Santa Juana mine site in either a closed building, a shed, or on a prepared uncovered area (in which case durable plastic covering is provided) behind a fence. In each case the core remains in a securely locked area. Pulps and rejects are stored in closed areas and are individually packed in plastic bags in order to avoid contamination. The mine facility is guarded by security personnel 24 hours a day.

11.2.2 Underground Chip, Pulp & Reject Storage

The chip sampling pulps and rejects are obtained from the assay laboratory and are stored in a secured area at the Santa Juana mine site in either a closed building or a shed. The chip sample rejects and pulps remain in a securely locked area.

11.3 Analyses for Drill Hole Samples

Drill hole samples are analyzed by ALS Chemex initially for Au using fire assay with atomic absorption spectroscopy finish (AA24) with re-run for values exceeding 10 g/t Au using fire assay with gravimetric finish (GRA22).

Samples are initially analyzed for Ag, Pb, Zn, Cu, and 32 additional elements using aqua regia inductively coupled plasma - atomic emission spectroscopy (ICP41) with re-run for values exceeding 100 g/t Ag, and

1% Pb, Zn, or Cu using aqua regia digestion and inductively coupled plasma - atomic emission spectroscopy (OG46).

11.4 Analyses for Channel Samples

Channel samples are prepared and then analyzed by the on-site facility for Au, Ag, Pb, Zn, Cu and As. Gravimetric fire assay is used to determine Au and Ag grade. Pb, Zn, Cu and As are analyzed by atomic absorption spectroscopy with hydrochloric and nitric acid digestion.

11.5 QA/QC Program

As a result of the CAM Quality Assurance and Quality Control (QA/QC) review performed in 2012 QA/QC procedures were refined. Within both the drill hole and channel sampling programs standards, blanks and duplicates are inserted in the sample stream. Quality control samples are inserted in a repeating order depending on the last digit of the sample identification (ID). The effective QC submittal for the drill core and channel campaign is approximately one control sample for ten collected samples. **Table 11-3** details the QC sample submittals for the 2014 drilling and 2013-2014 channel campaigns.

Table 11-3: In-Stream Quality Control Samples

QC Sample Type	Drill Hole Stream	Channel Stream
Blank	23	134
Pulp Duplicate	44	197
Standards	51	183
Combined	118	514
QC % of Samples	9%	~10%

11.5.1 Standards

In 2014, 27 low grade standard samples along with 24 high grade standard samples were analyzed in the drill hole sample stream. The high- and low-grade standards are custom made and tested by SGS. The standard results were reviewed and demonstrate adequate performance. Few errors exist that are most likely attributed to sample ID mislabeling and should be addressed prior to performance analysis. It is suggested that in future campaigns the effectiveness of monitoring be improved, and standards should be sourced that more closely bound the actual data averages. **Table 11-4** shows the standards insert during the drilling campaign.

Table 11-4 Custom Standard Reference Material for 2014 Drill Hole Stream

Standard Name	Mean Au ppm	Mean Ag ppm	Mean Pb %	Mean Zn %	Standard Deviation Au	Standard Deviation Ag	Standard Deviation Pb	Standard Deviation Zn
M-4 87438	1.239	1.78	0.0083	0.0194	0.032	0.11	0.00812	0.021
M-3 87427	17.38	1503	2.71	1.29	0.330	14.55	0.10	0.06

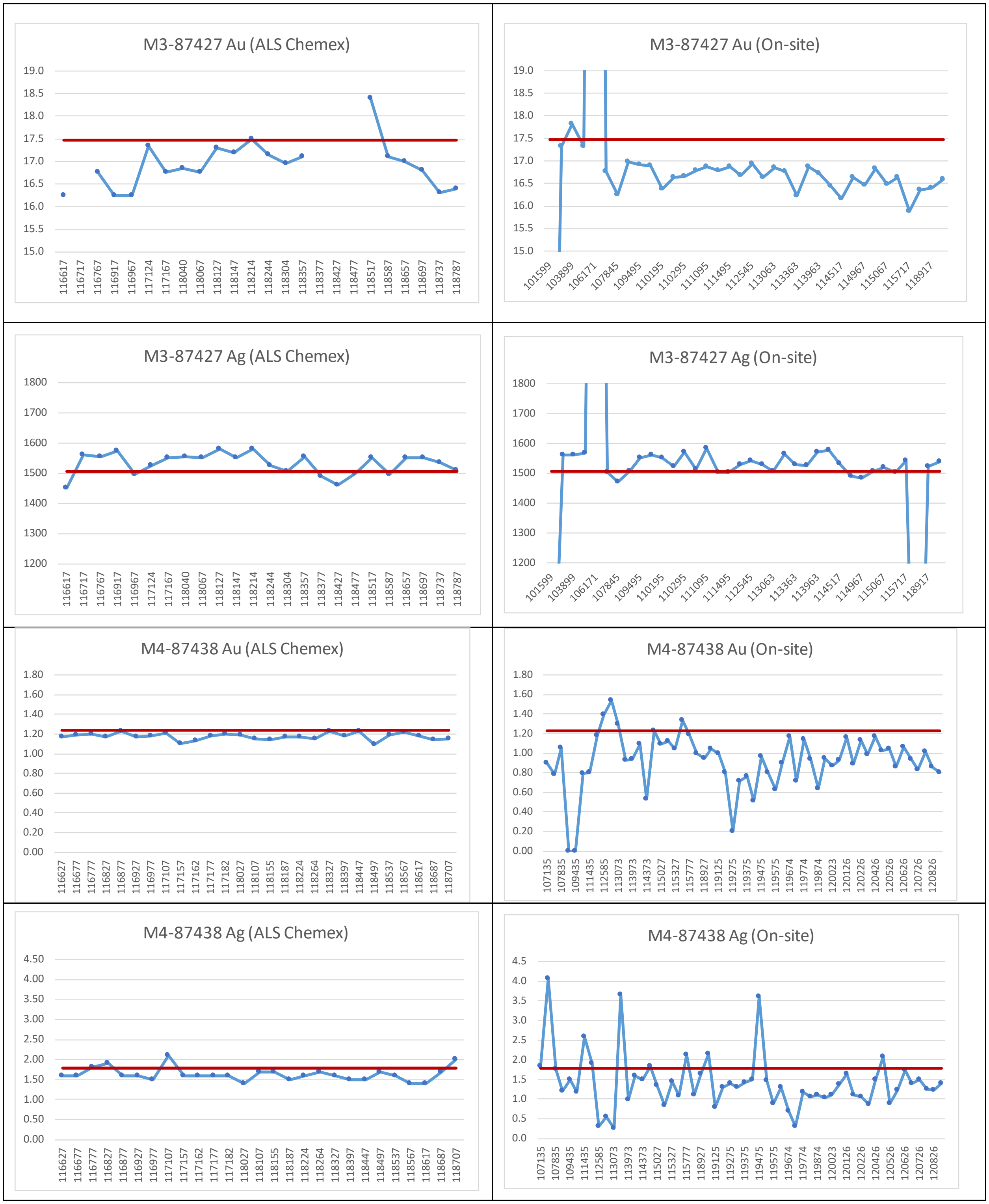
From the time of 2012 report to the end of 2014, 197 standard samples were analyzed in the channel sample stream at the on-site laboratory. The high- and low-grade standards are custom made and tested by SGS. Two of the standards used in the drill hole stream are used in the channel sample stream as well, which provides a check of both labs. The standard results were reviewed and demonstrate reasonable performance but suggest additional improvements should be made. **Table 11-5** shows the standards inserted during the channel sampling campaign.

Table 11-5 Custom Standard Reference Material for Channel Stream

Standard Name	Mean Au ppm	Mean Ag ppm	Mean Pb %	Mean Zn %	Standard Deviation Au	Standard Deviation Ag	Standard Deviation Pb	Standard Deviation Zn
M-1 87440	0.961	8.7	0.73	0.16	0.015	0.19	0.037	0.007
M-2 87439	9.06	379	3.18	4.50	0.029	6.50	0.03	0.04
M-3 87427	17.38	1503	2.71	1.29	0.330	14.55	0.10	0.06
M-4 87438	1.239	1.78	0.0083	0.0194	0.032	0.11	0.00812	0.021

Few noticeable errors exist in the testing of the high-grade standards where significantly higher or lower grades are reported for singular metals or Au and Ag together. The nature of the results indicates the execution of the gravimetric process protocols at the on-site laboratory could be tightened and refined.

The results of the analysis of M-4 87438 at both the on-site laboratory and ALS suggests the on-site laboratory provides more variable results at the low-grade end for Au and Ag, often under-reporting the concentration of Au and Ag. This is most likely attributed to issues with the use of Ag in quart technique modifications to the gravimetric procedures for lower grade analysis. **Figure 11-1** compares standards M-3 87427 and M-4 87438 for Au and Ag, which were both tested at the on-site laboratory and ALS Chemex. ALS Chemex testing is shown on the left and the on-site laboratory is shown on the right. Except for the few noticeable issues, the testing of the high-grade standard at ALS Chemex and the on-site facility are similar, the results of the lower grade standard show the on-site laboratory is less precise and less accurate compared to ALS Chemex.


M3-87427 Au (ALS Chemex)

M3-87427 Au (On-site)

M3-87427 Ag (ALS Chemex)

M3-87427 Ag (On-site)

M4-87438 Au (ALS Chemex)

M4-87438 Au (On-site)

M4-87438 Ag (ALS Chemex)

M4-87438 Ag (On-site)

Figure 11-1: Standard Performance Comparison

11.5.2 Duplicates

In 2014, 44 pulp duplicates were analyzed within the drill hole sample stream. Review of the duplicates indicates good reproducibility. **Figure 11-2** shows Au, Ag, Pb, and Zn in a single log ten transformed scatter plot.

Figure 11-3 shows in-stream pulp duplicates tested at the on-site laboratory also in a single log ten transformed scatter plot with Au, Ag, Pb, and Zn. In general, the sample pair fit for each element is good with few examples for high-grade Ag duplication issues, suggesting, as mentioned above in the review of the standard analysis, that gravimetric process protocols could be improved. The duplicate analyses also indicate that error bias could be positive. Based on visual inspection of the scatter plot, noted duplication errors do not appear to be balanced on either side of the 1:1 fit line. The noted issues in the standards and duplicates suggest improvements but are infrequent and do not suggest invalidation of the results obtained from the onsite facility.

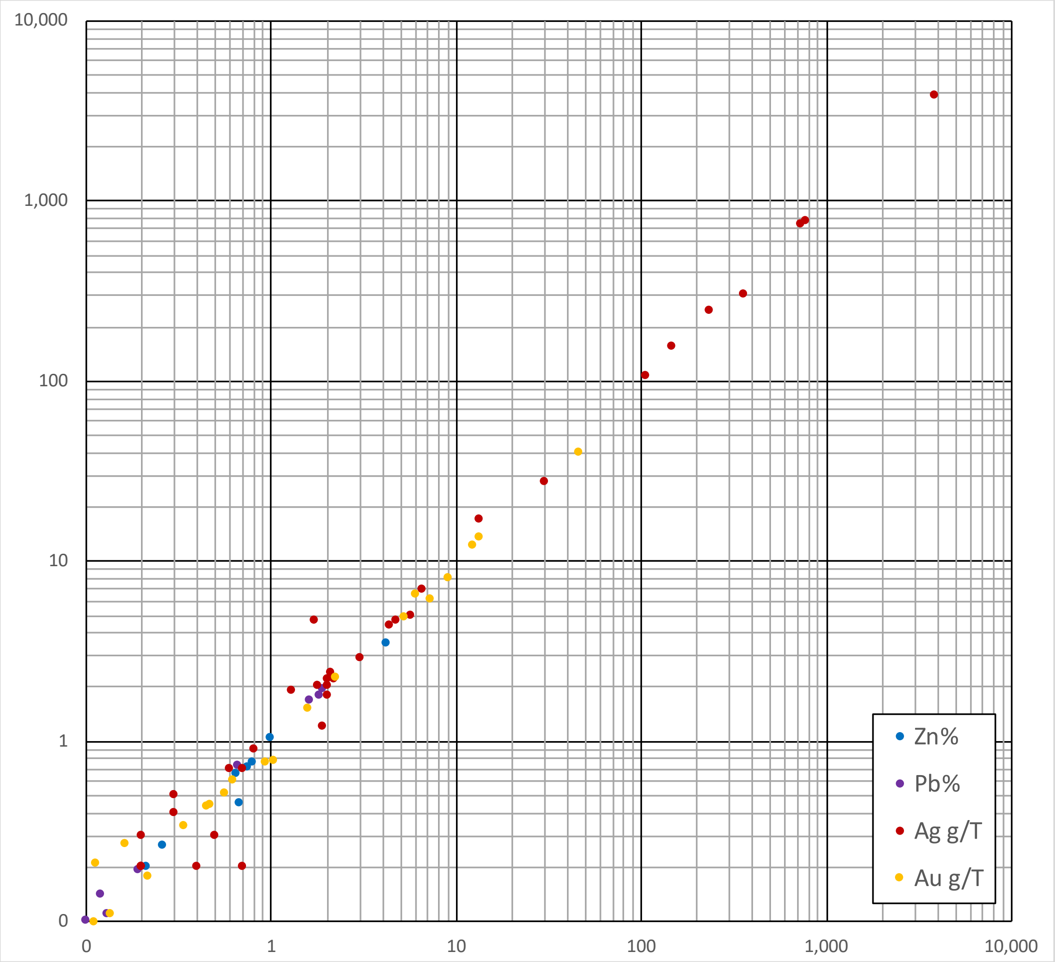


Figure 11-2: 2014 Drill Hole ALS Chemex Pulp Duplicates

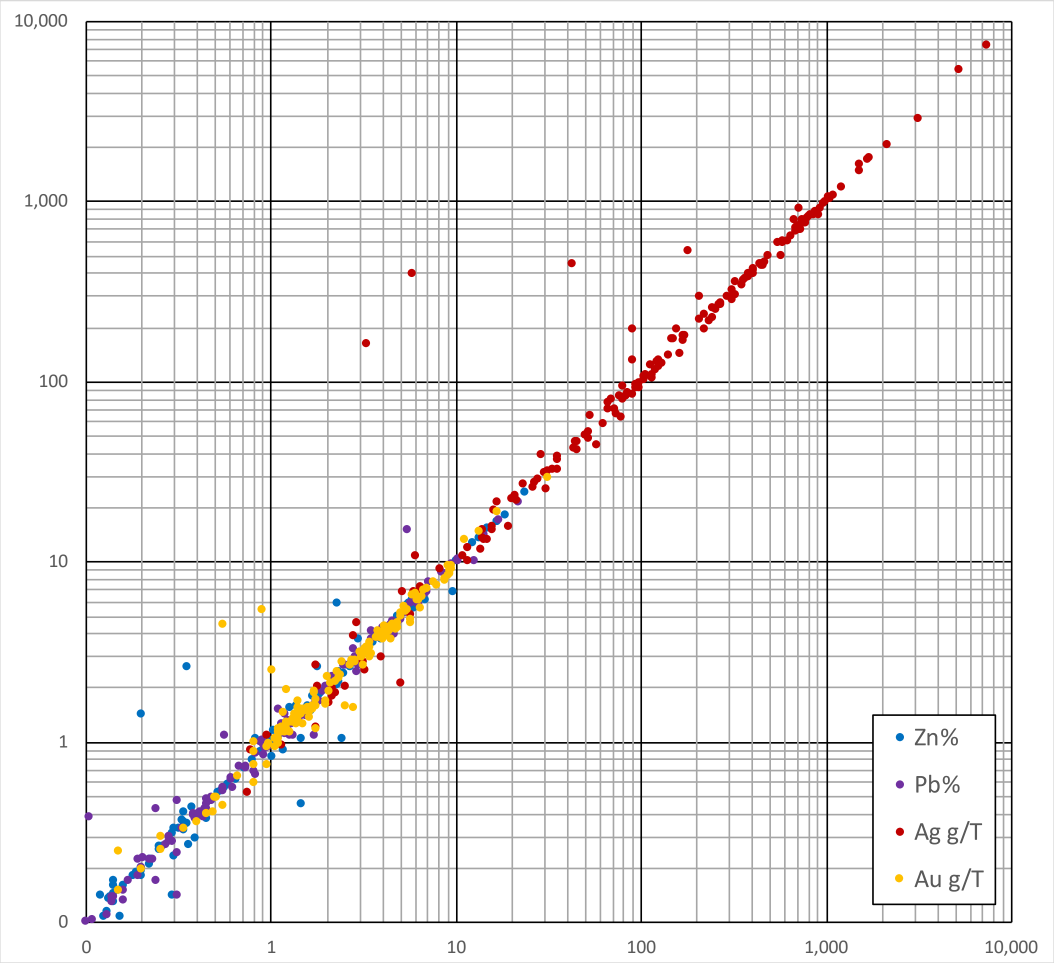


Figure 11-3: On-site Channel Sample Pulp Duplicates

11.5.3 Blanks

In 2014, 23 blanks were analyzed within the drill hole sample stream and performed well for Au, Pb and Zn but suggest the sourced blank material is contaminated with low grade Ag, **Figure 11-4**. Samples showing elevated Ag have been individually investiagted and are preceeded by near dectection limit samples which rules out laboratory contamination. Testing of the low-grade standard M-4 87438, which has a near dectection Ag value, supports the conclusion that instrumentation error or laboratoy contamination are not the likely source of the issue.

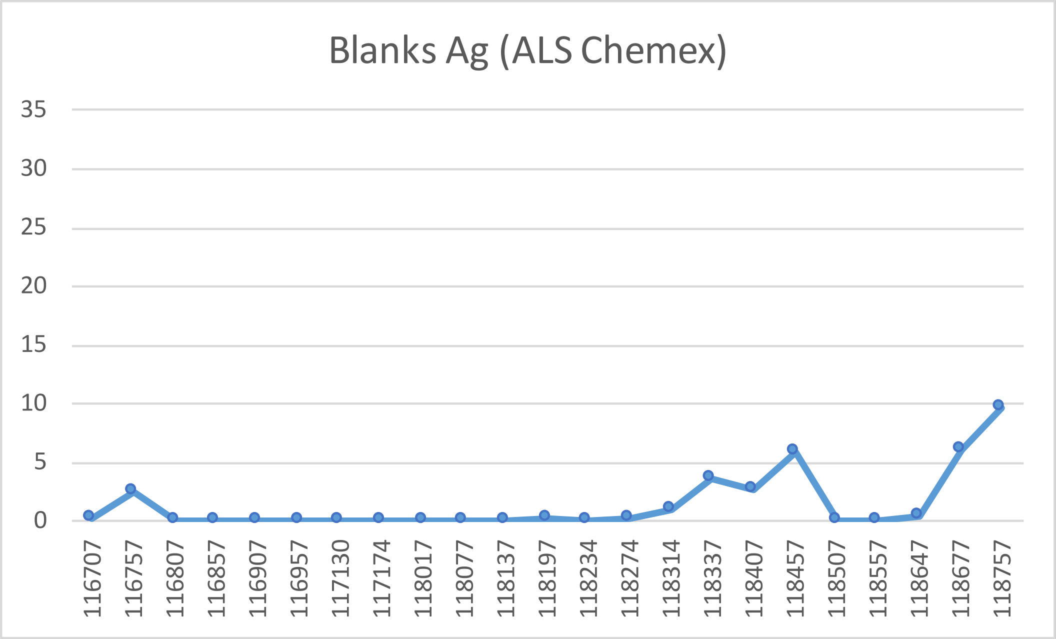


Figure 11-4: 2014 Ag Blanks

The performance of channel sample stream blank material, sourced separately from drill hole blank material, was reviewed and is reasonable for Au, showing few noted errors that are typically found in a QA/QC program but the performance of the blank material for Ag, Pb and Zn is not good.

Poor blank performance for Ag, Pb, and Zn suggests the blank material is contaminated and should be replaced immediately. The poor performance could also indicate low levels of laboratory contamination and low precision at lower grade analysis, but it cannot be definitely determined because of the likely contamination of the source blank material.

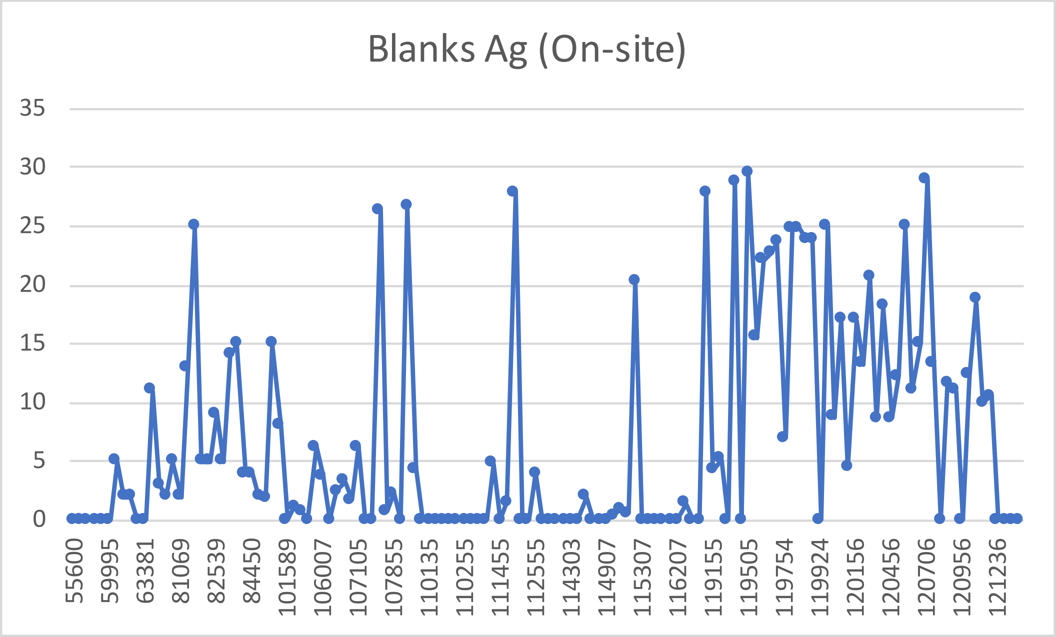


Figure 11-5: On-site Channel Sample Ag Blanks

11.6 QA/QC Recommendations

Although QA/QC samples are being inserted and generally reviewed, it is recommended that stricter monitoring and action protocols are designed and implemented to identify issues and potential tolerance exceedance as soon as results from each batch are returned. The most effective QA/QC monitoring would involve a first pass screening of labeling mix-ups and a second pass of technical review of laboratory performance. The project database has been migrated to the database management software GeoBank™, which has the capabilities to support these recommendations. In addition, the most optimized QA/QC program would involve standards closer to the actual grades of interest than those that are currently being used. If additional custom standards are envisioned for multiple element purposes it is suggested that certification of the standards involve more tests than the five at SGS for the current standards, and an additional laboratory is included in the certification of the values. The current blank material should be replaced with certified blank material that has been subjected to extensive pre-analysis to confirm the absence of Au, Ag, Pb and Zn. The current blanks have not proven useful for their intended purpose and have contributed more to doubt than certainty.

11.7 Analysis Pre-2009 Methodology (Micon)

The following has been sourced from Micon 2009.

11.7.1 Laboratories, Methods & Procedures

ECU used the Ensayes y Representaciones, S.A. (ERSA) laboratory in Torreón, Coahuila as its primary laboratory. The ERSA laboratory is an independent laboratory in Mexico which is currently in the process of acquiring its ISO 9002 certification.

Analyses performed at the laboratory are based on international and certified standards for Au, Ag, Cu and Zn in high concentrations as well as for atomic absorption and plasma. The laboratory is also in charge of the assay interchange with the smelters where they perform assays as part of round robin exercises every six months. In this respect, the ERSA laboratory is no different from any number of small independent commercial laboratories which are operated in Canada and which participate in round robins but do not have ISO certification. While the assay laboratory in Torreón is currently not certified, Micon believes that the laboratory has sufficient experience and QA/QC procedures in place for it to serve as ECU's primary assay laboratory in Mexico.

At the ERSA laboratory, a sample is first reduced to quarter inch fragments using a jaw crusher. It is further reduced to ten mesh size using a small cone crusher. The material is then split in a Jones-type splitter until 300 g of the sample is collected. The split sample is then pulverized to minus 150 mesh, homogenized for about three minutes and placed in a tagged plastic bag. When the sample contains moisture, it is dried at 110° C for approximately four hours prior to the process.

11.8 Quality Control Pre-2009 (Micon Assessment)

The following has been sourced from Micon 2009.

Quality control procedures conducted include the routine incorporation of certified geochemical standards, blanks and sample duplicates, according to the following protocol:

- Diamond Drilling: alternate insertion of a laboratory certified laboratory standard or blank for every 10th sample.
- RC Drilling: For every alternate 10th sample, a duplicate sample of the preceding interval was taken as a field duplicate, or a certified laboratory check standard or blank sample was submitted respectively.
- Trenching: For every alternate 10th sample, a duplicate sample of the preceding interval was taken as a field duplicate, or a certified laboratory check standard or blank sample was submitted respectively.

11.8.1 In-House Reference Material

ECU's on-site reference material consists of finely ground material from the Santa Juana mine. The material was collected, crushed and mixed at one time. Several samples have been assayed at various laboratories to assure constant values prior to their use as an on-site reference. One on-site reference sample for every twenty samples is sent to the laboratory. The statistics regarding the on-site reference sample are summarized in **Table 11-6**.

Table 11-6: Summary of the In-House Reference Material for the Velardeña and Chicago Properties

Element	Units	Average Value	Minimum Value	Maximum Value	Standard Deviation
Au	g/t	9.6	0.7	13.1	2.2
Ag	g/t	1,094	215	2,241	258
Pb	%	8.01	0.82	14.52	2.37
Zn	%	13.15	1.34	19.46	4.49
Cu	%	1.24	0.01	2.04	0.35
As	%	4.16	1.13	5.44	0.72
Fe	%	14.74	1.45	22.35	4.66

The on-site standard was derived from a total of 51 samples which were collected, crushed and mixed together. Table provided by ECU Silver Mining Inc.

ECU created its on-site reference samples as there were no low- or high-grade Ag standard reference standards available on the market. While this is not always the recommended course of action in a QA/QC program, it is preferable to not including a reference sample. If a commercially available high- or low-grade Ag reference sample becomes available with similar characteristics to the mineralization located on the Velardeña and Chicago projects, Micon would recommend that ECU purchase it for its use. It could be argued that any commercially available standard polymetallic reference sample would be preferable to the use of an on-site reference sample; however, given the relatively high grade Ag encountered at the Velardeña and Chicago properties, the use of a standard reference sample without a high grade Ag component almost ensures that the assay methods used by ECU's various assay laboratories to assay for high grade Ag would go unchecked. Micon therefore believes that in this case it is preferable to use an on-site reference sample as a check against the assaying procedures.

11.8.2 Blanks

The material used for blank samples is barren limestone collected from a nearby location for which reasonably constant values have been previously tested in several laboratories. One blank for every 20 samples is sent to the laboratory. The statistics regarding ECU's blanks are summarized in **Table 11-7**.

Table 11-7: Summary of the Blank Material for the Velardeña and Chicago Properties

Element	Average Value	Minimum Value	Maximum Value	Standard Deviation	Units
Au	0.16	0	2.6	0.4	g/t
Ag	12	0	110	19.04	g/t
Pb	0.2	0.00	2.85	0.48	%
Zn	0.51	0.00	19.15	2.75	%
Cu	0.01	0.00	0.16	0.02	%
As	0.14	0.00	0.87	0.18	%
Fe	0.92	0.09	4.6	0.85	%

Total samples =51. Table provided by ECU Silver Mining Inc.

11.8.3 Duplicate Samples

Field duplicates from the stopes and mill have been assayed occasionally in order to test ECU's mill laboratory (LABRI), comparing it against the results from the regular ERSA laboratory. The on-site laboratory was used to verify stope sampling and mill assays. The on-site mill laboratory has not been used to assay the samples contained in the database.

Underground samples collected by ECU between 1997 and 2001 were assayed at the previous mill laboratory which was sold to Hecla Mining Company in 2001. The use of an on-site laboratory to assay the development sampling at a mine is a common practice in the mining industry and in the majority of cases this database is also used in the Resource estimate. This sampling is considered historical and the development sampling dating to this period is contained for the most part in mined out areas. Thus, these assays were not used in the current Resource estimate.

Duplicate sampling using the rejects is not conducted on the core or development chip samples. Micon recommends that ECU begin to conduct some duplicate assaying on the core and development chip rejects. The assaying of a portion of the reject core will assist in checking both the variance in the mineralization and the sample preparation procedures and the assay laboratories.

11.8.4 Re-Assays

Re-assaying has been systematically carried out on batches of pulp samples. In the case of pulp samples these are conducted on a random grouping of samples approximately every three months.

Each pulp sample selected for re-assaying is re-assayed at the following two laboratories.

Sample pulps are sent to the ALS Chemex laboratory facilities in Guadalajara (Mexico) and Vancouver (Canada). The ALS Chemex Guadalajara laboratory conducted the sample preparation and then sent the pulps by plane to the Vancouver laboratory for assaying. ALS Chemex is committed to having ISO

9001:2001 certification at all locations. The assaying procedures used by ALS Chemex for these samples have been described earlier in this section.

Sample pulps are sent to the Servicio Geológico Mexicano (SGM) laboratory in Chihuahua, Mexico. This laboratory is a Mexican government facility which services the mining industry and University of Chihuahua. This laboratory conducts both assaying and mineralogical testwork. This government facility is an ISO 9001:2000 and BS EN ISO 9001:2000 certified laboratory.

ECU has conducted a number of comparisons using the assay results from these laboratories against the original assays conducted at ERSA. An example of these comparisons made on the average grades for 113 samples from the three assay laboratories is summarized in **Table 11-8**.

Table 11-8: Summary of the Assay Laboratory Comparisons for the Average Grades Based on 113 Samples

	Averages					
Assay Laboratories	Au g/t	Ag g/t	Pb %	Zn %	Cu %	As %
ERSA	2.82	178.9	0.68	0.58	0.22	1.08
Chemex	2.91	153.7	0.67	0.57	0.21	1.14
SGM	2.14	168.0	0.80	0.71	0.20	1.15
	Differences (%)					
	Au	Ag	Pb	Zn	Cu	As
ERSA/ALS Chemex	-3.2	14.0	2.4	1.2	4.8	-5.4
ERSA/SGM	24.0	6.0	-17.0	-22.0	6.0	-6.0
ALS Chemex/SGM	24.0	-9.0	-20.0	-24.0	2.0	-1.0
	Correlation					
	Au	Ag	Pb	Zn	Cu	As
ERSA/ALS Chemex	0.98	0.95	1.00	0.99	1.00	0.95
ERSA/SGM	0.63	0.17	1.00	0.84	1.00	0.93
ALS Chemex/SGM	0.62	0.18	1.00	0.84	1.00	0.98

Table provided by ECU Silver Mining Inc.

As indicated by **Table 11-8** the correlation between ERSA and ALS Chemex for the pulp samples is very good. For the assay correlations between ERSA and SGM as well as ALS Chemex and SGM, the correlation for the base metals ranges from fair in the case of Zn to good for Pb, Cu and As. However, for the precious metals it is poor for Au and nonexistent for Ag.

Since one of the samples (No. 43293) included in the batch of 113 samples sent to the other two laboratories was very high in Ag, this sample was removed and a comparison of the average grades for the remaining 112 samples was conducted. The comparison conducted on the 112 samples is summarized in **Table 11-9**.

Table 11-9: Summary of the Assay Laboratory Comparisons for the Average Grades Based on 112 Samples

Assay Laboratories	Averages					
	Gold (g/t)	Silver (g/t)	Lead (%)	Zinc (%)	Copper (%)	Arsenic (%)
ERSA	2.75	128.2	0.68	0.58	0.16	1.05
Chemex	2.83	123.9	0.67	0.57	0.15	1.11
SGM	2.13	155.1	0.80	0.71	0.15	1.12
	Differences (%)					
	Gold	Silver	Lead	Zinc	Copper	Arsenic
ERSA/ALS Chemex	-2.8	3.0	2.5	1.2	2.3	-5.4
ERSA/SGM	23.0	-21.0	-17.0	-22.0	2.0	-7.0
ALS Chemex/SGM	25.0	-25.0	-20.0	-24.0	0.0	-2.0
	Correlation					
	Gold	Silver	Lead	Zinc	Copper	Arsenic
ERSA/ALS Chemex	0.99	0.99	1.00	0.99	1.00	0.95
ERSA/SGM	0.63	0.04	1.00	0.84	1.00	0.93
ALS Chemex/SGM	0.63	0.09	1.00	0.84	1.00	0.98

Table provided by ECU Silver Mining Inc.

As indicated by **Table 11-9,** the correlation between ERSA and ALS Chemex for pulp samples is very good. For the assay correlations between ERSA and SGM as well as ALS Chemex and SGM, the correlation for the base metals ranges from fair in the case of Zn, to good for Pb, Cu and As. However, for the precious metals it is again poor for Au and nonexistent for Ag.

11.9 2009 to 2012 Sample Preparation & Assaying (CAM Assessment)

The following has been sourced from CAM 2012.

A significant proportion of samples from the latest exploration program were analyzed at the ERSA laboratory which is not certified. Additionally, the pulps and coarse rejects from this work were lost. Analysis of the available QA/QC data for these assays indicated the quality of the ERSA assays was not satisfactory. For these reasons CAM recommended, and Golden Minerals agreed, that these samples be redone at a certified laboratory. Because the pulps and coarse rejects were lost this required a re-sampling, prep and assay according to the following steps:

1) A re-split of the remaining samples at the Velardeña site.
2) Insertion of QA/QC samples into the sample stream and an approximate rate of one in 15.
3) Transport of the samples to the SGS laboratory in Durango.

4) Dry, crush to 75% passing through a two-millimeter (mm) screen, subsample 250 g and pulverize to 85% passing through a 75 micrometer (µm) screen.
5) Assay using standard SGS procedures. Because of detection limits multiple assays were sometimes run on the same sample pulps.

11.9.1 General QA/QC

There are several types of QA/QC samples which are regularly inserted into the sample stream to assure that the results obtained are representative of the samples and are correct in terms of contained metal.

Types of QA/QC samples include:

- **Standards.** Standards are samples, usually pulps that have a known value. Standards may be purchased commercially and have an accepted value provided by the commercial entity preparing the standard for at least some elements. Internal standards are prepared by the company (often using contractors because of the larger volumes desirable for standards) with the accepted value being estimated as a result of a series of round-robin assays at various labs. Commercial standards have the advantage that at least some assay values are known "a priori". Commercial standards may have the disadvantage that not all elements of interest for given operation are provided by the commercial vendor. For example, in a polymetallic deposit like Velardeña, several different standards are required from a commercial lab to cover all the elements of interest. Internal standards are less independent than commercial standards but have the advantage that they tend to be more representative of the mineralogy of the specific deposit and are more likely to cover the elements of interest in more reasonable grade ranges.
- **Blanks.** Blanks may be considered a special type of standard with elemental values of zero. However, blanks, particularly prep blanks, are extremely useful in detecting cross-contamination between samples. Generally, blanks are prepared by the company.
- **Replicates.** Replicates are repeats of prior assay values and for most operations are duplicates. However, the term replicate is used because in some cases multiple analyses for the same element by the same method are run on the same pulp. There are several types of replicate samples including a prep replicate, which is a re-preparation of the course rejects after the first size reduction step. Prep replicates are also called coarse replicates, if multiple pulp envelopes are available the pulp replicates may be submissions of the same envelope or another envelope. For any reputable laboratory there are also replicates out of the same pulp envelope. These are typically used by the laboratory for internal QA/QC.

11.9.2 QA/QC SGS Re-Assays

A total of 10,755 assay records in 197 batches from SGS were provided to CAM in tab delimited spreadsheets by Golden Minerals; in addition to these assay records 561 internal lab duplicates by SGS were provided.

Relative to non-QA/QC samples this represents a QA/QC sample rate of almost 15% which is consistent with best industry practice. This 15% does not include the internal SGS duplicates.

Velardeña is transitioning from manual and labor-intensive data handling and Resource estimation techniques to computerized data handling methods, and CAM encountered several cases of misspellings, inconsistent use of blank, "_", and "-" in evaluating the QA/QC data. All of these inconsistencies were reported to Golden Minerals and there was significant improvement over the several months while the assays were in progress.

SGS provided assays on 35 different elements by 47 different assay methods (multiple assay methods are required for some elements because of the detection capabilities of the analytical equipment). CAM only reviewed QA/QC on the elements impacting NSR which were Au, Ag, Cu, Pb, Zn, As and antimony (Sb). QA/QC review of all of these elements were reported to Golden Minerals; however, for this report only QA/QC data for Au and Ag are discussed. These results are representative of assay results for the other elements.

A detailed table of QA/QC samples is given in **Table 11-10**.

Table 11-10: Count of Velardeña QA/QC Samples by Type

Type	Count
Blank	252
Pulp Duplicate	222
Coarse Duplicate	343
STANDARDS	
STD 1-87440	54
STD 3-87427	60
STD Au 0.0849 ppm	40
STD Au 4.086 ppm	35
STD CDN-ME-18	173
STD CDN-ME-4	169
STD OREAS 134a	26
Total Standards	**557**

Graphical review of QA/QC data is the easiest way to determine if there are issues in QA/QC. Figures of QA/QC results are shown in **Figure 11-6** through **Figure 11-15**. Brief comments are presented following the figures.

Figure 11-6 shows two anomalous values of over one g/t but is still less than a 1% anomaly rate. **Figure 11-7** shows only one anomalous value.

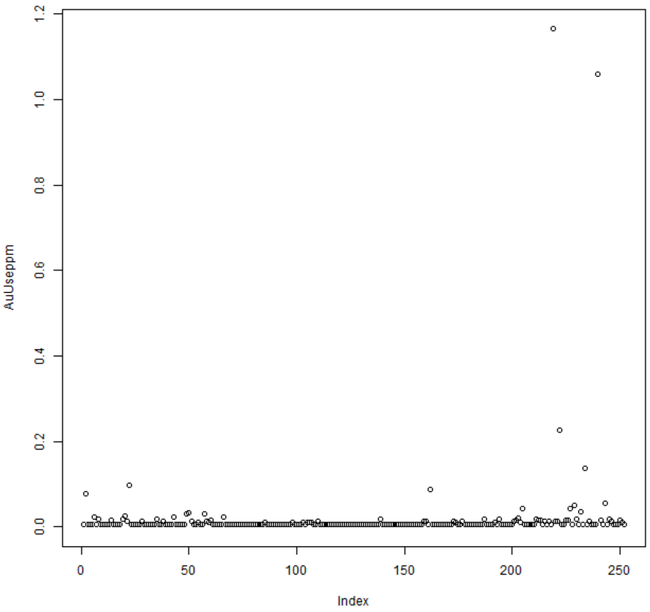


Figure 11-6: Gold Blanks

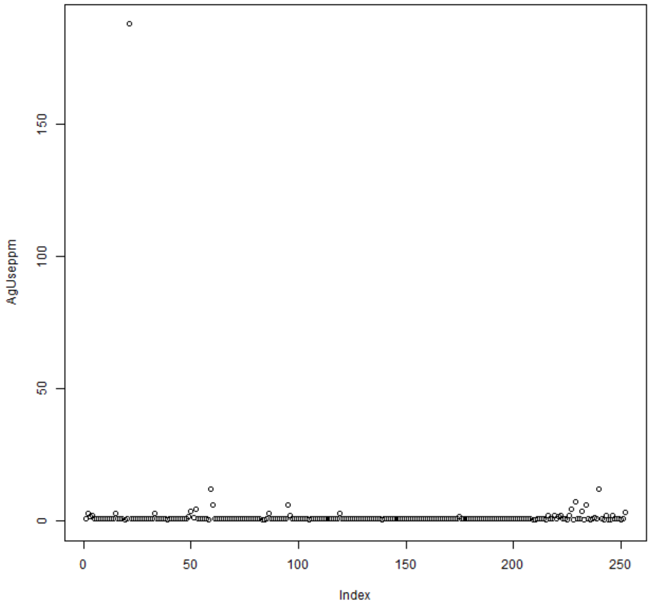


Figure 11-7: Silver Blanks

No anomalous values are shown in **Figure 11-8**. Two anomalous values are shown in **Figure 11-9**, with an anomaly rate of just over 1%.

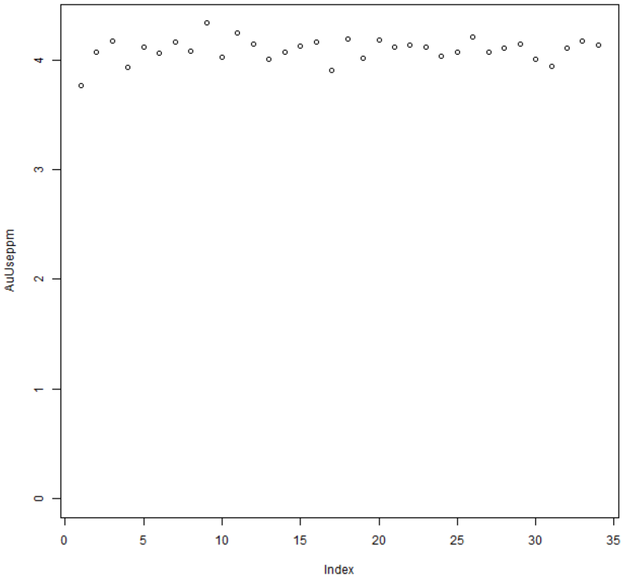


Figure 11-8: Typical Gold Standard

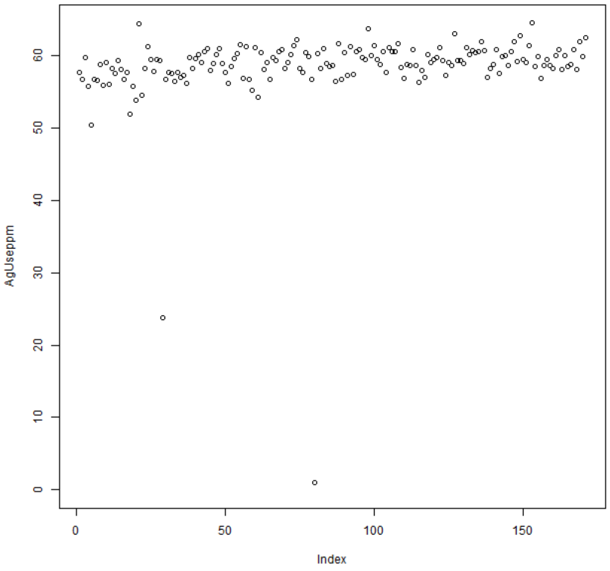


Figure 11-9: Typical Silver Standard

Figure 11-10 shows four or five anomalous points; a higher anomaly rate is typical for duplicates than for blanks and standards. **Figure 11-11** shows two or three anomalous points; a higher anomaly rate is typical for coarse duplicates than for fine duplicates.

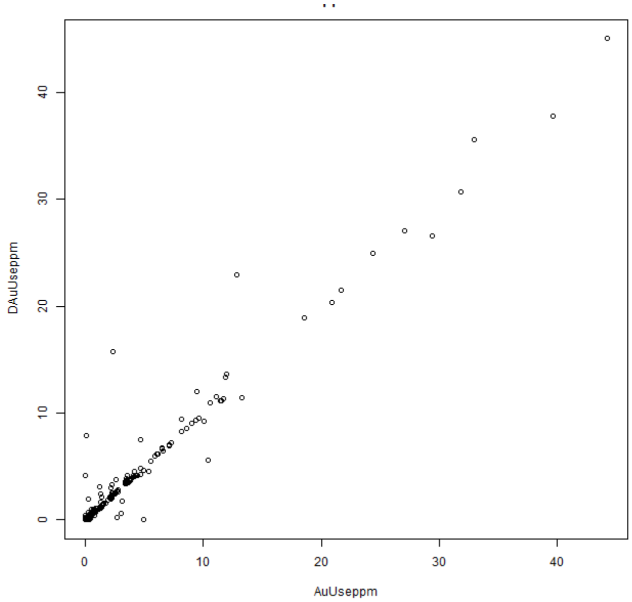


Figure 11-10: Gold Coarse Duplicates

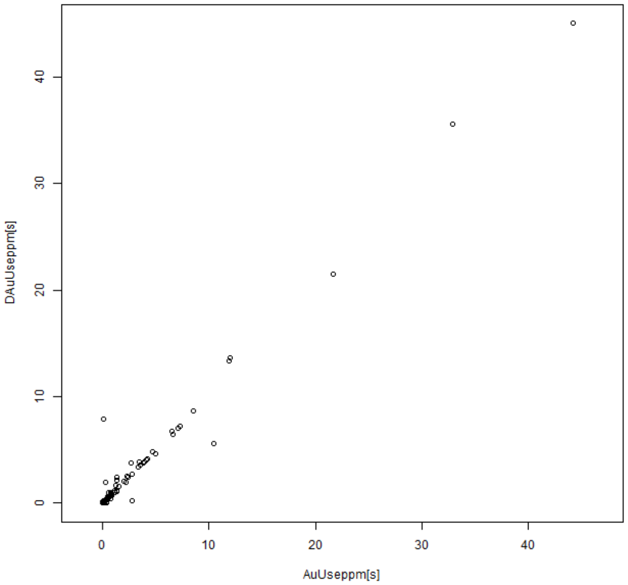


Figure 11-11: Gold Fine Duplicates

Figure 11-12 shows three or four anomalous points; as with Au the anomaly rate for duplicates is higher than for blanks and standards. **Figure 11-13** shows one or two anomalous points; as with Au the anomaly rate for fine duplicates is lower than for coarse duplicates.

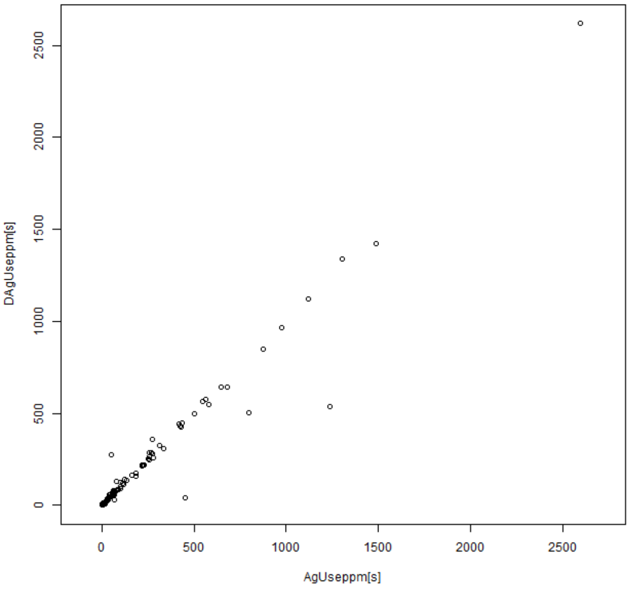


Figure 11-12: Silver Coarse Duplicates

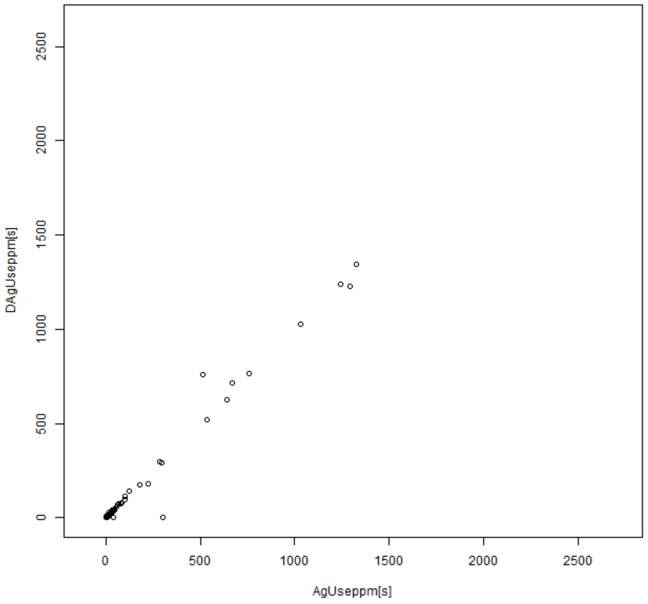


Figure 11-13: Silver Pulp Duplicates

Figure 11-14 shows one anomalous data point, which is unusual for internal duplicates because normally batches containing these are re-run and never reported to the client. CAM recommended Golden Minerals review SGS reporting standards to determine why this anomaly occurred. In this same context, internal standards for SGS reruns should be disclosed to the client.

There are no immediately obvious anomalous points in **Figure 11-15**. However, it appears the scatter on the low end is somewhat higher than for the Au duplicates in terms of percentage. Since Ag is a significant revenue element, routinely including log-log plots of replicates is suggested.

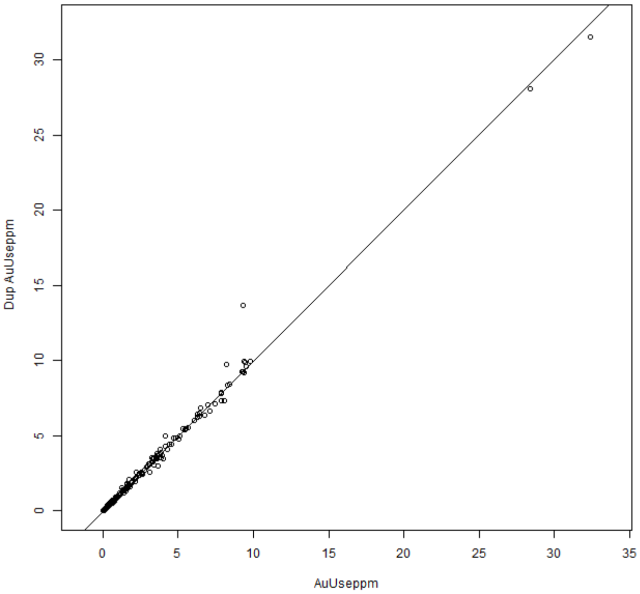


Figure 11-14: SGS Gold Internal Duplicates

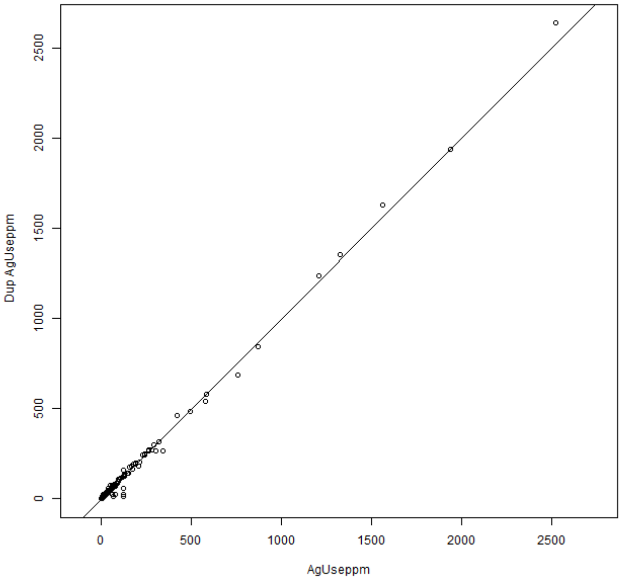


Figure 11-15: SGS Internal Silver Duplicates

11.9.2.1 Conclusions on the SGS Re-Assay Dataset

Minera William has followed best practices in terms of the number of QA/QC samples and the number of statistically anomalous values observed in the QA/QC charts occur at a rate of less than 2%. Hence, CAM concluded the SGS re-assay dataset is suitable for use in calculation of Resources.

11.10 Specific Gravity Determinations

Specific gravity measurements have been made on chip and core samples for varying lithological units and mineral types (oxide, mixed and sulfide) present at each area of the Project (Santa Juana, Terneras, and Chicago). Samples were selected to represent the major lithology, alteration, and mineralization types.

Several thousand density samples were available from water-immersion measurements on core samples and some hand specimens. The data are of sufficient accuracy for use in Resources estimation, but new samples should be routinely collected and used for Resource estimation.

Specific gravity is calculated by the following formula:

$$\text{Specific Gravity} = \text{weight dry} / (\text{weight dry} - \text{weight submerged}).$$

Table 11-11 lists the averages by process type and **Table 11-12** lists the averages by vein by process type.

Table 11-11: Velardeña Average Densities by Mineral Type

Mixed	Oxide	Sulfide
3.34	2.82	3.57

Table 11-12: Velardeña Average Density by Vein and Process Type

Vein	Ore Type	Average Density
A1	Oxide	2.39
	Mixed	3.38
A2	Sulfide	3.65
A4	Oxide	2.97
	Mixed	3.76
	Sulfide	3.55
Bs	Oxide	2.88
	Mixed	3.57
CO	Oxide	3.00
SJ	Sulfide	3.81
C1	Oxide	2.90
CC	Oxide	2.98
	Sulfide	4.03
E	Sulfide	3.50
E1	Mixed	2.86
G1	Oxide	2.77
Nueva	Oxide	2.89
	Mixed	3.59
	Sulfide	3.59
San Juanes	Sulfide	3.81
San Mateo	Oxide	2.67
	Sulfide	3.41
Oriente	Sulfide	3.21
Terneras	Oxide	2.81
Flechas	Oxide	2.55
Roca Negra	Sulfide	3.18
Trans	Oxide	2.85
Gambusino	Oxide	2.94
	Mixed	3.05
Escondida	Mixed	3.19

11.10.1 Comparing Specific Gravity Datasets

There is a very high level of variability in sulfide densities due to highly variable sulfide assemblages and modal proportions. Conclusions regarding densities include:

- Oxides are generally consistent.
- Mixed does not appear in the 2005 dataset and only in the Santa Juana set for 2011. Within the latter, densities are consistent.
- Densities for diorite, limestone and skarn are generally consistent - locally high densities on limestone presumably represent a calc-silicate component.

The variation in total sulfide and sulfide mineral assemblages within sulfide mineral types are the main source of density variations and overshadow relatively subtle differences in host rock. It is important to accurately estimate volumes of vein versus host rock mined as this will have significant impacts on density/tonnage factors as well as grade dilution.

12 DATA VERIFICATION

The data collected by the mine staff is in support of operations planning and many of the data inputs provided by Golden Minerals are supported by historic and current production actuals and through this activity have been verified. Additional verification procedures are described below.

12.1 Geologic Data Inputs

To verify geologic data inputs the qualified person reviewed provided digital data in context of other data provided along with physical observations while on site. For example: the level mapping was reviewed alongside selected vein samples, geologic mapping was reviewed in conjunction with drill hole geologic interval logging, on-vein development was compared to sample locations, mine stopes were compared to development and channel sampling.

Traditional drill hole database validation checks were run on the drill hole and channel database and errors were provided to Minera William staff for correction. Each provided on-vein interval for every modeled vein was reviewed in three-dimensional (3D) view, level plan, and in section during model construction and was checked for consistency of location and grade in context of nearby samples.

The quantity of data provided is immense and is not free of errors and omissions. Data provided often required additional organization and in some instances alterations to be internally consistent with respect to location. Location inconsistencies are often related to levels not being perfectly flat and often partially following ramps; two-dimensional (2D) data representation without regard to vertical position; and 2D data representation with respect to idealized vein planarity.

Since the 2015 PEA was released, Minera William has undertaken an effort to examine database intervals that intercept the vein. Each interval was examined alongside with the mine level maps, as well as existing wireframes. If it was deemed that the vein code was not correct, the database was corrected. Special attention was also given to intervals and whether they contain dilution or not in the sampling. This recoding of intervals was used for the Resource update in this TR.

The geologic data provided is adequate for the purposes used in the technical report.

12.2 Mine Planning Data Inputs

Tetra Tech conducted a site visit to the Velardeña mine to verify that parameters used in mine planning are adequate for use in the PEA. This included visiting underground workings, as well as test mining areas. This site visit allowed for verification of mining parameters used in the PEA, confirming that the parameters are adequate.

12.3 Mineral Processing Data Inputs

Technical and cost data were obtained during the Project site visit and in subsequent communications with Golden Minerals personnel at the Velardeña site and in Golden Minerals' Golden, Colorado office. The data provided by Golden Minerals conforms to industry standards and is considered to be within the accuracy of this PEA study and verified for use in this PEA study.

At no time was there any limitation to, or failure to provide the requested technical and cost data for the processing plants or infrastructure to Tetra Tech's metallurgical or infrastructure personnel during or subsequent to the Project site visit.

The technical and cost data for the processing plants and infrastructure collected during the site visit to Velardeña and subsequent communications with Golden Minerals are adequate for the assemblage and production of this PEA study.

12.4 Economic Data Inputs

A technical economic model template and cost data were obtained in subsequent communications with Golden Minerals. The data provided by Golden Minerals conforms to industry standards and is considered to be within the accuracy of this PEA study and verified for use in this PEA study.

At no time was there any limitation to, or failure to provide the requested technical and cost data for the economic model to Tetra Tech.

The technical and cost data for the economic model are adequate for the assemblage and production of this PEA study.

12.5 Environmental Information

A list of current permits was obtained from Golden Minerals. The information provided by Golden Minerals conforms to the requirements of Mexican environmental regulations; however, no information regarding an environmental monitoring program or adherence thereto was reviewed and the waste rock area permits will need to be updated before mining recommences.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

There are two processing plants at the Project. Plant #1 is designed to treat sulfide material by conventional crush, grind, and differential flotation to produce Pb, Zn and pyrite concentrates. Process Plant #2 has two production circuits for separately processing oxide and sulfide Au-Ag material for the production of Au-Ag doré by cyanide leach/Merrill-Crowe and a bulk Au-Ag rich sulfide concentrate by flotation, respectively.

Operation of Plant #1 was discontinued in late 2015 due to a combination of low metal prices, dilution and metallurgical challenges. Plant #2 was leased to Hecla Mining Company in July 2015 and is operating. The lease is in place through December 2020.

Because of the historical production for Plant #1, the liberation characteristics of the material and their response to differential flotation are within typical design criteria and known by the operations personnel. There are no geological, lithological or mineralogical changes in the process plant feed anticipated for the envisaged future production as compared to previous operations. Existing legacy operational data supports the existing process flowsheet for potential future production at Plant #1. Further, the use of existing and refurbished equipment within the pre-existing facilities is Golden Minerals' preferred method of future treatment.

In 2007 the potential to increase gold recovery from Plant #1 and improve project economics by installing a bio-oxidation circuit to treat pyrite (Fe) concentrates on site and recover gold and silver to doré was explored by sending samples to SGS in South Africa for testwork. Treatment on site would allow operating the flotation circuit to pull more mass to the pyrite circuit containing gold and silver previously lost to tailings. The testwork indicated Velardeña pyrite concentrate could be successfully oxidized with the BIOX® process prior to cyanidation. Golden Minerals sent additional samples for testing at Outotec facilities in 2019 that confirmed the initial findings.

An abbreviated Outotec description of the process follows:

- The BIOX® process was developed for the pre-treatment of sulfide refractory ores and concentrates ahead of a conventional cyanide leach for gold recovery. The gold in these ores is encapsulated in sulfide minerals such as pyrite, arsenopyrite and pyrrhotite thus preventing the gold from being leached by cyanide. The BIOX® process destroys the sulfide minerals and exposes the gold for subsequent cyanidation, thereby increasing the overall gold recovery that can be achieved.
- The heart of the BIOX® process is a mixed culture of naturally occurring microbes which, under controlled conditions, are able to oxidize gold-bearing sulfide ores or concentrates due to a chemolithotrophic mode of metabolism. This means that they require inorganic compounds for the acquisition of both energy and carbon.
- The carbon requirements of the microbes for biosynthesis of cellular biomass are met by CO_2 in the atmosphere or from the dissolution of carbonate minerals in the ore.
- The microbial culture in the BIOX® reactors is not controlled but rather allowed to adapt to the concentrate and operating conditions.
- The species, viz. *Acidithiobacillus* ferrooxidans, *Leptospirillum* ferrooxidans, *Leptospirillum* ferriphilum, *Ferroplasma* cupricumulans as well as many archaea species make up the dominant species of the BIOX® microbial consortia. Detailed laboratory and pilot plant studies have indicated that the microbes require a very acidic environment (pH 1.1 to 1.7), a

temperature of between 35°C and 45°C and a steady supply of oxygen and carbon dioxide for optimum growth and activity. The unusual operating conditions, which are optimal for the BIOX® microbes, are not favorable for the growth of most other microbes, thus eliminating the need for sterility during the BIOX® process. The BIOX® microbes are non-pathogenic and incapable of causing disease. The microbes employed in the BIOX® process do not, therefore, pose a health risk to humans, animals, or plant life.

- The oxidation reactions are also highly exothermic. In addition to the direct oxidation of sulfide minerals, several indirect chemical and microbial assisted reactions occur.

An additional benefit of oxidizing Fe concentrates on site would be the ability to precipitate arsenic solubilized as arsenic acid in the BIOX liquor into a stable ferric arsenate compound suitable for disposal in a tailings dam.

Two series of Batch Amenability Tests (BAT) were performed on different samples of Fe concentrate; in 2007 at SGS LAKEFIELD Research Africa and in 2019 at Outotec BIOMIN (Pty) Ltd RSA. Copies of the reports are available in **Appendix A.** Based on results of the testwork, both reports concluded the refractory Velardeña Fe concentrate is amenable to bio-oxidation treatment. The oxidized sulfides yielded improved gold and silver dissolutions in a cyanide leach from single digits before treatment to greater than 90% for Au, and from less than 40% before to 50 - 95% for Ag.

Operating conditions, reagent consumptions and results for a BIOX® BAT treatment period of 24 days was chosen for evaluation in this assessment. **Figure 13-1** shows the increase in oxidation with time for the two series of BATs. Gold dissolution approaching 90% was achieved in both test reports after sulfide oxidation to near 60%, corresponding to plateauing of the Gold Dissolution vs. Sulfide Oxidation curve **Figure 13-2.**

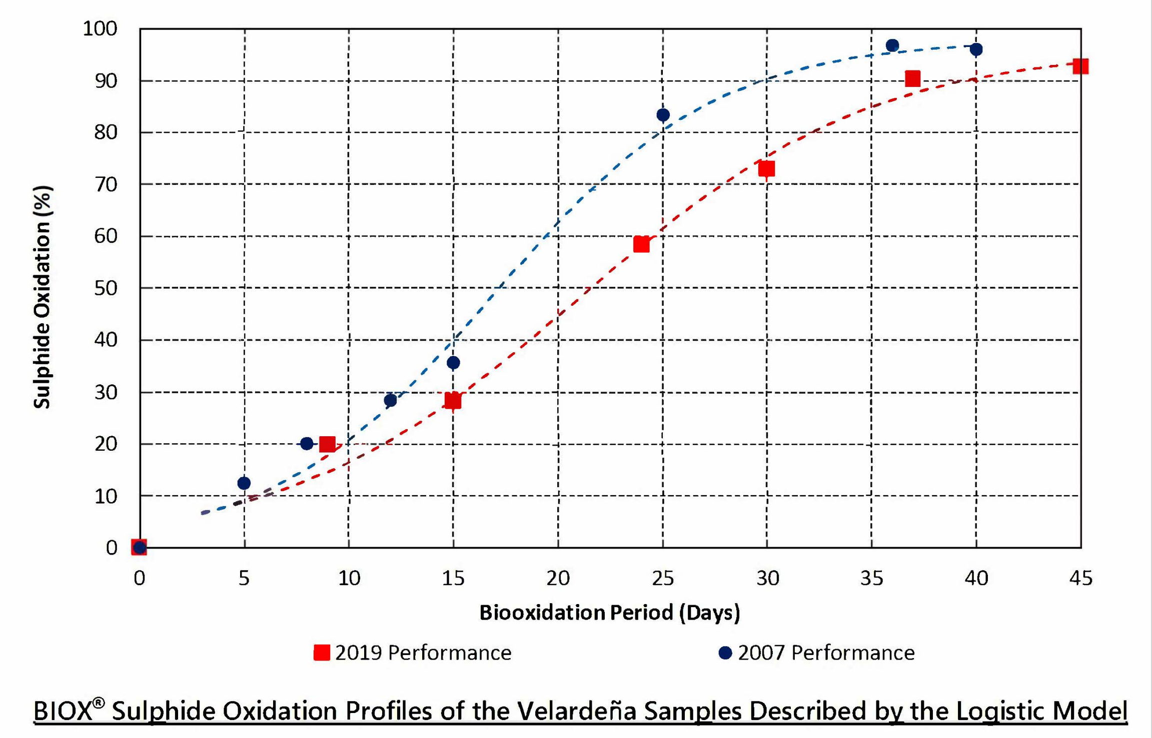


Figure 13-1: Oxidation vs. Time

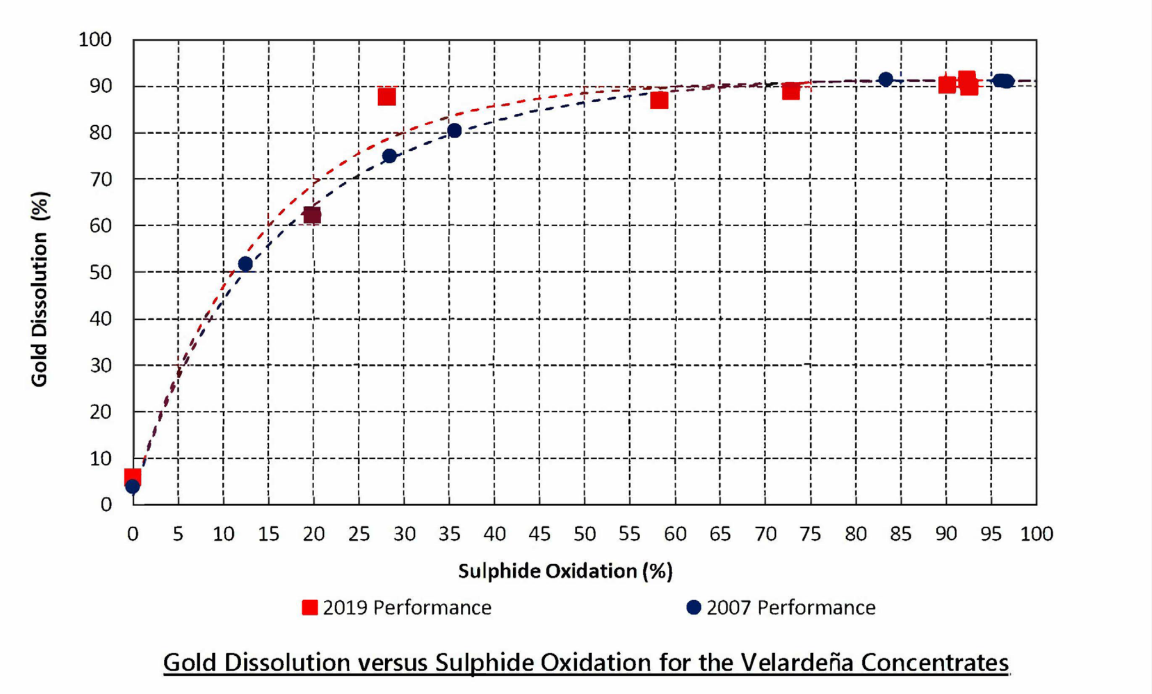


Figure 13-2: Dissolution vs. Oxidation

Neutralization of the acidic ferric arsenate BIOX® solution with a two-stage treatment with limestone and lime produced environmentally stable arsenic compound residues fully compliant with the United States of America's Environmental Protection Agency's (USA EPA) guidelines for tailings disposal. These guidelines call for arsenic re-solubilization of tailings residues to be < 5 mg/L. For the Velardeña neutralization tests, arsenic values of 0.06 and 0.07 mg/L were obtained.

With the success of the testing programs, this updated PEA includes a BIOX® circuit to oxidize the Fe concentrate for recovery of the contained gold and silver to doré on site.

Table 13-1 summarizes recent production data for Process Plant #1 for November and December 2014 and forecast production for the projected LoM of the Project.

Table 13-1: Historical Production Data for Plant #1, BAT Test Samples and Forecast Project Production w/BIOX® Plant for Years 1 - 11

Description	Units	Nov 2014 Actual (2)	Dec 2014 Actual (2)	BAT 2007	BAT 2019	Year 1 (3)	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Material Processed	Tonnes	6,127	8,195			113,150	113,150	113,151	113,150	113,150	113,150	113,150	113,150	113,150	113,150	6448
Head Grade																
Au	g/t	1.55	1.78			3.98	4.68	4.42	7.01	5.40	4.30	5.01	5.65	5.81	5.27	3.81
Ag	g/t	90	127			317	388	323	398	459	335	361	223	264	310	148
Pb	%	0.72	0.79			2.18	1.36	1.40	1.55	1.30	1.37	1.20	0.95	0.94	0.99	0.98
Zn	%	0.96	1.16			2.41	2.58	2.02	2.16	1.47	1.68	1.22	0.89	0.67	1.24	1.41
As (estimated)	%					1.00	0.80	1.18	1.65	1.23	1.11	1.08	1.23	1.25	0.95	0.37
Lead Concentrate	Tonnes	117.45	151.0			5254	3050	2712	3375	3092	2779	2829	1869	2164	2267	148
Au	g/t	18.79	27.41			17.1	49.8	30.9	41.2	34.2	26.5	39.2	65.0	52.7	65.9	66.1
Ag	g/t	4 458	5,631			5185	10,601	9628	9607	11,657	9640	10,099	9648	9400	10,659	5500
Pb	%	28.07	33.95			36.4	37.6	38.8	37.2	34.1	36.9	33.6	38.8	33.1	37.7	40.3
Zn	%	5.00	4.93													
As	%	NR	0.89													
Sb	%	NR	2.56													
Zinc Concentrate	Tonnes	75.53	126			4825	5148	3949	4325	2877	3155	2460	1689	1176	2569	176
Au	g/t	2.19	1.66			3.49	2.99	4.01	4.91	5.28	4.17	5.84	8.79	11.9	5.85	4.24
Ag	g/t	365	422			586	769	907	828	1756	1145	1762	1704	3199	1457	274
Pb	%	1.07	1.69													
Zn	%	50.30	55.86			43.4	43.6	44.7	43.7	43.5	45.2	43.7	45.2	44.9	42.8	42.0
As	%	NR	0.22													
Sb	%	NR	0.10													
Pyrite (Fe) Concentrate	Tonnes	385.68	137.0			16,061	24,246	18,961	31,539	24,657	18,810	23,515	27,356	27,609	26,983	1259
Au	g/t	7.84	10.55	23.6	18.5	13.6	10.4	12.8	12.2	12.0	12.6	11.7	11.3	11.6	10.58	9.11
Ag	g/t	147	217	198	124	180	154	181	139	219	197	176	80.1	111	145	42.6
Pb	%	0.59	0.98													
Zn	%	1.24	1.43													
As				11.0	4.4											
Recoveries																
Lead Concentrate	% of Au	28.28	28.4			20	29	17	18	17	15	20	19	17	25	40
	% of Ag	78.42	82.2			76	74	71	72	69	71	70	71	68	69	85
	% of Pb	75.23	79.2			78	75	66	72	72	66	70	68	68	76	94
	% of Zn	9.95	7.88			NDA										
Zinc Concentrate	% of Au	2.12	1.4			3.7	2.9	3.2	2.7	2.5	2.7	2.5	2.3	2.1	2.5	3.0
	% of Ag	4.13	5.1			7.9	9.0	9.8	7.9	9.7	9.5	11	11	13	11	5.0
	% of Pb	1.84	3.29			NDA										
	% of Zn	64.29	74.4			77	77	77	77	75	75	78	75	70	78	81
Pyrite (Fe) Concentrate	% of Au	38.73	9.9			48	48	48	48	49	49	48	48	49	48	47
	% of Ag	8.51	2.86			8.1	8.5	9.4	9.8	10	9.8	10	8.7	10	11	5.6
	% of Pb	5.22	2.06			---										
	% of Zn	8.07	2.05			---										
Total	% of Au	69	40			72	79	68	69	68	66	70	70	68	75	89
	% of Ag	91	90			92	92	91	91	90	90	91	92	91	91	95
	% of Pb	82	85			NDA										
	% of Zn	82	84			NDA										

Source: Golden Minerals; November-December 2014 data: (1) November 3-30, 2014, (2) December 1-31, 2014, (3) Steady state conditions, after ramping up

14 MINERAL RESOURCE ESTIMATES

Resources have been estimated independently for 39 known veins. Estimation was completed using vein wireframes for the primary veins, and centerline guiding surfaces for the secondary veins. A combination of variable thickness block models and block factor models were used for the veins. Block attributes have been estimated in three passes, using inverse distance to a power of 2.5.

Estimated Mineral Resources for the Velardeña project are shown in **Table 14-1** below, as well as the mineral type portions for each Resource class.

Table 14-1: Velardeña Project Resources

Classification	Mineral Type	NSR Cutoff	Tonnes	Grade Ag g/t	Grade Au g/t	Grade Pb%	Grade Zn%	Ag oz	Au oz	Pb lb	Zn lb
Measured	Oxide	125	135,000	260	5.55	1.72	1.54	1,130,000	24,000	5,120,000	4,570,000
Indicated	Oxide	125	301,000	250	4.89	1.70	1.47	2,420,000	47,000	11,300,000	9,750,000
Measured + Indicated	Oxide	125	436,000	253	5.10	1.71	1.49	3,550,000	71,000	16,430,000	14,310,000
Inferred	Oxide	125	372,000	399	4.82	2.52	1.46	4,770,000	58,000	20,680,000	11,950,000
Measured	Sulfide	125	269,000	346	5.38	1.53	1.88	3,000,000	47,000	9,100,000	11,140,000
Indicated	Sulfide	125	645,000	327	4.62	1.43	1.94	6,790,000	96,000	20,300,000	27,530,000
Measured + Indicated	Sulfide	125	915,000	333	4.84	1.46	1.92	9,790,000	142,000	29,410,000	38,670,000
Inferred	Sulfide	125	1,393,000	342	4.70	1.51	1.97	15,320,000	211,000	46,380,000	60,400,000
Measured	All	125	404,000	317	5.43	1.60	1.76	4,120,000	71,000	14,220,000	15,710,000
Indicated	All	125	946,000	303	4.71	1.52	1.79	9,220,000	143,000	31,610,000	37,280,000
Measured + Indicated	All	125	1,351,000	307	4.92	1.54	1.78	13,340,000	214,000	45,830,000	52,990,000
Inferred	All	125	1,765,000	354	4.73	1.72	1.86	20,080,000	268,000	67,060,000	72,350,000

Notes:

(1) Reported Measured and Indicated Resources are equivalent to mineralized material under SEC Industry Guide 7
(2) Inferred Resource is not a recognized category under SEC Industry Guide 7
(3) Resources are reported as diluted Tonnes and grade to 0.7 m fixed width
(4) Metal prices for NSR cutoff are three-year trailing average as of December 2019: US$16.30/troy ounce Ag, US$1,305/troy ounce Au, US$0.99/lb Pb, and US$1.27/lb Zn
(5) Columns may not total due to rounding

14.1 Input Data

The project database contains 10,597 assayed drill hole and 33,233 channel sample intervals. Of those, 734 drill hole intervals and 15,131 channel intervals have been interpreted as intersecting a named vein and subsequently used for Resource modeling. **Table 14-2** shows grade statistics for intervals within the overall project database and those selected for Resource modeling.

Table 14-2: Input Data Statistics

Dataset	Selection	Count	Mean Ag g/t	Mean Au g/t	Mean Pb%	Mean Zn%
Channel	All	33,233	284	5.14	1.61	1.64
Drill hole	All	10,597	45	0.97	0.16	0.29
All	All	43,845	226	4.13	1.26	1.32
Channel	On Vein	15,131	514	9.09	2.81	2.74
Drill hole	On Vein	734	283	5.04	0.72	1.29
All	On Vein	15,865	503	9.0	2.71	2.67

14.2 Compositing

Each drill hole and channel that intersected the vein was composited into one variable length composite and a generated centroid coordinate. Each composite represents an accumulation of the intervals from the hanging wall to footwall of the vein. A channel sample set, or drill hole was permitted to have only one composite per vein. There was no predetermined interval length for the composites.

14.3 Grade Capping

Assay intervals from the combined drill hole and channel sample database that were identified as being on-vein were analyzed as a natural log transformed population to determine upper grade limits. Upper limits were applied to composited vein intervals. The upper limit chosen for Ag was 4,000 g/t and 55 g/t for Au, and 20% for both Pb and Zn. **Table 14-3** shows capping statistics and the effects on the population mean **Figure 14-1** shows probability plots for Ag, Au, Pb, and Zn. A traditional interpretation of the probability plots, shown in **Figure 14-1** could conclude higher capping limits are justified, however limits higher than selected would not be supported by observed vein variability.

Table 14-3: Capping Statistics

Element	Uncapped Mean g/T	Upper Limit g/T	Number Capped	Capped Mean g/T
Ag	513	4000	149	491
Au	9.3	55	163	9.0
Pb	2.7	20	181	2.64
Zn	2.67	20	204	2.60

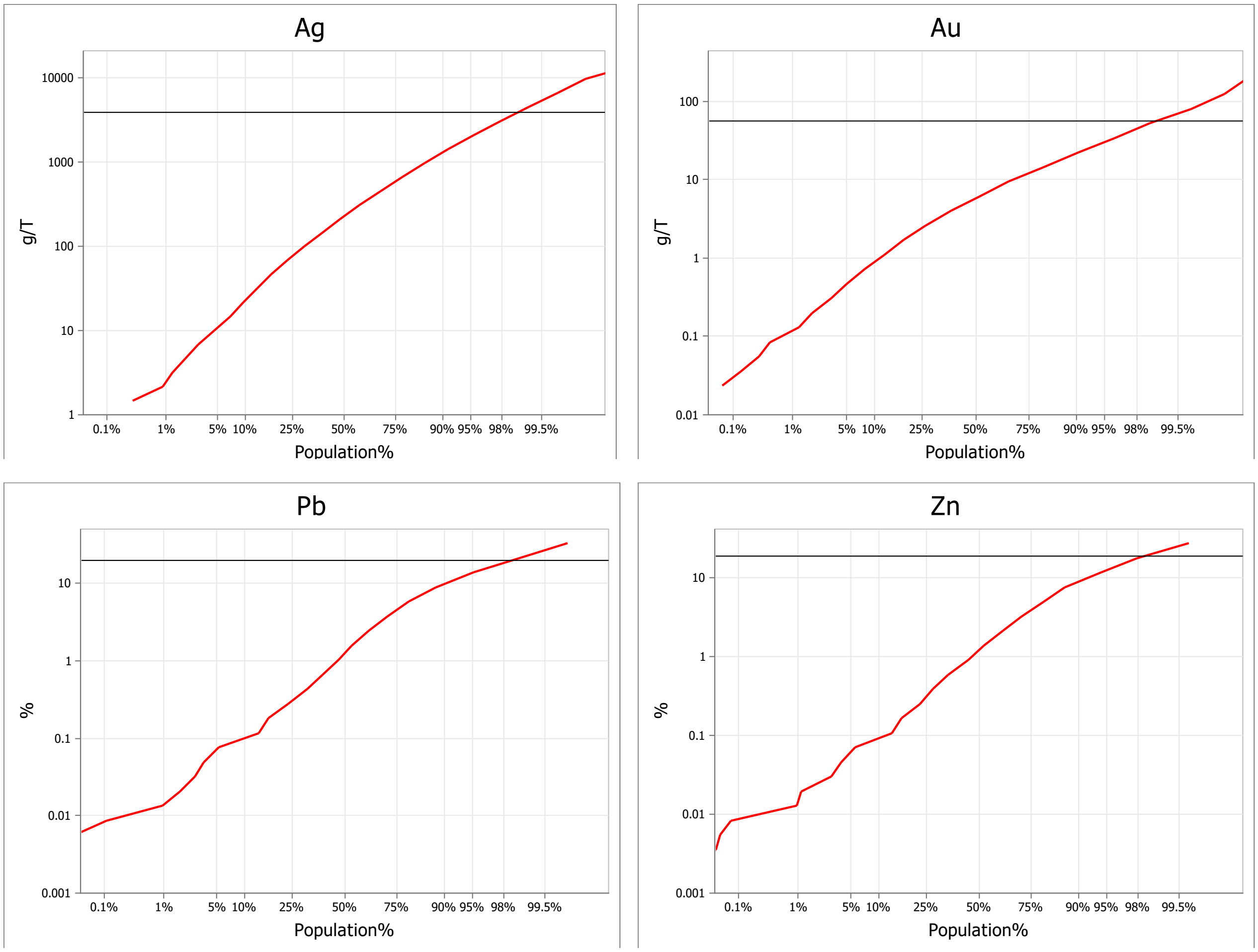


Figure 14-1: Upper Limit Analysis Probability Plots

14.4 Vein Modeling

The veins at the Project are interpreted to be epithermal type formed by fluids that have flooded relatively narrow pre-existing structurally prepared zones. Initial vein intervals were provided by Golden Minerals as an attribute in the project database along with indicative vein surface models. These initial picks were reviewed in 3D in context of the vein mapping and underground development mapping provided by the company.

14.4.1 Primary Veins

Following the 2015 report, additional work on the database was done by Minera William to verify and refine the intervals in the drill holes. Each interval that was examined with level mapping and vein mapping to define whether it was assigned the appropriate vein coding. There was also investigation into the diluted vs undiluted samples in the database. The diluted and undiluted samples were originally mixed and both used for estimation purposes, adding additional dilution to the estimate. The undiluted samples were used for the database modification exercise and was completed for the principal veins at the site, which include CC, C1, A4, F1, G1, San Mateo, Roca Negra, Hiletas, Terneras, Chicago, and Escondida. The new database information was utilized by Golden Minerals to create updated models for these veins. The updated interval data was fed into Leapfrog software and new wireframes were created that honored the intervals in the database, using only on vein, undiluted intervals. The new wireframes were brought into Micromine and used to create and estimate Resource models. The wireframe models are shown below with the existing development in **Figure 14-2**, **Figure 14-3**, and **Figure 14-4.**

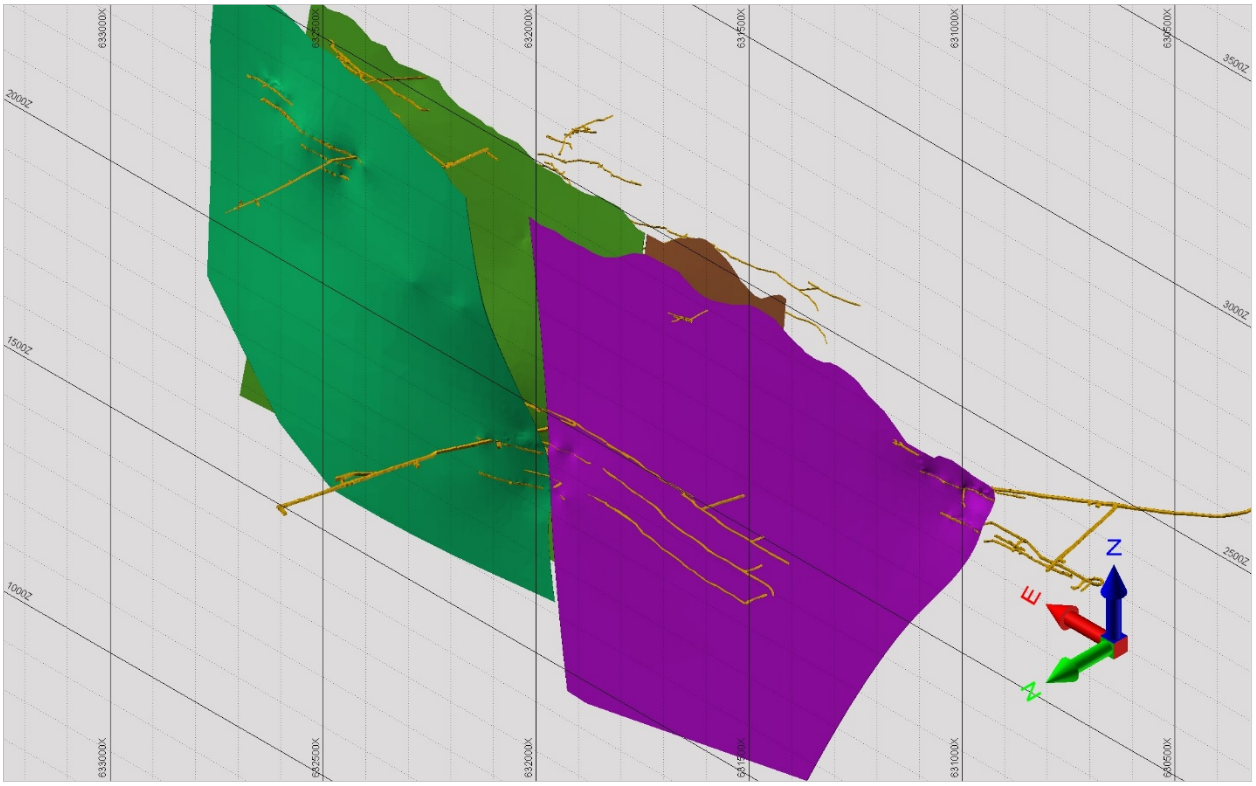


Figure 14-2: 3D View of the Wireframes from the Terneras area

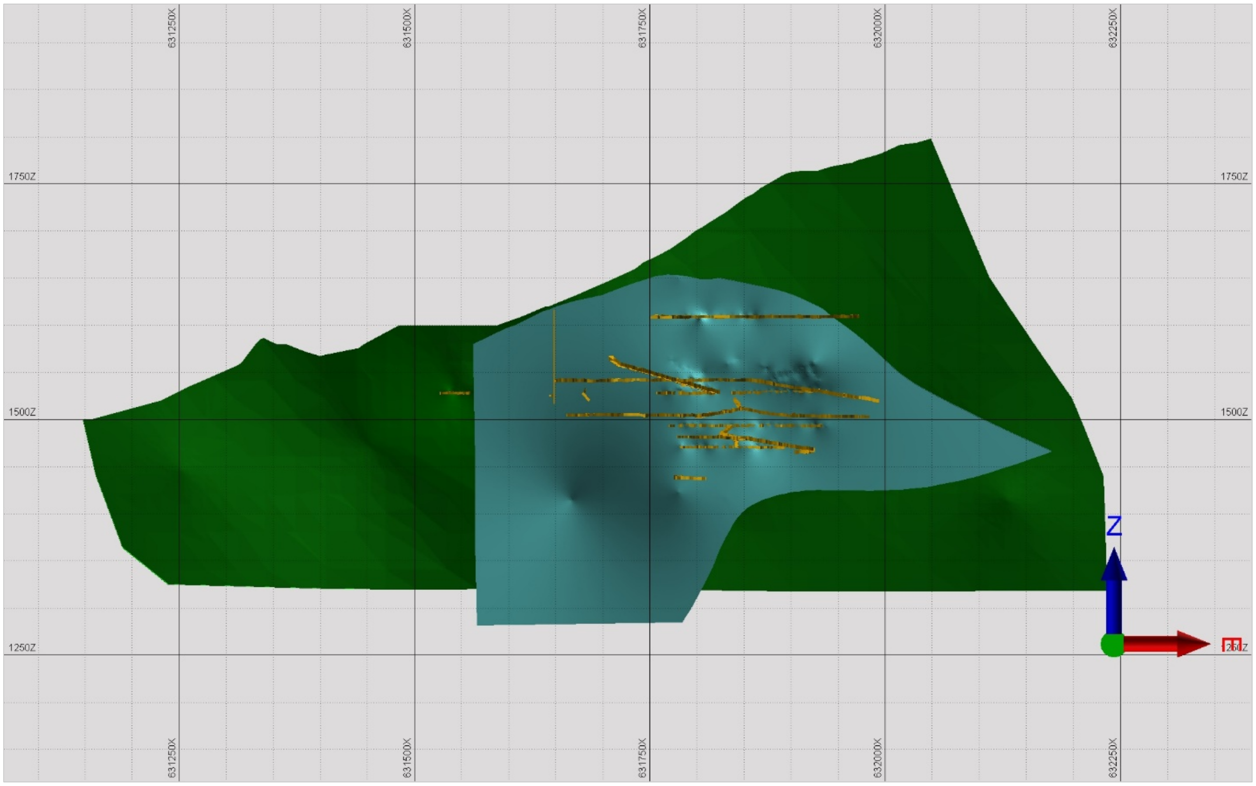


Figure 14-3: 3D View of the Chicago Area Wireframes, looking North

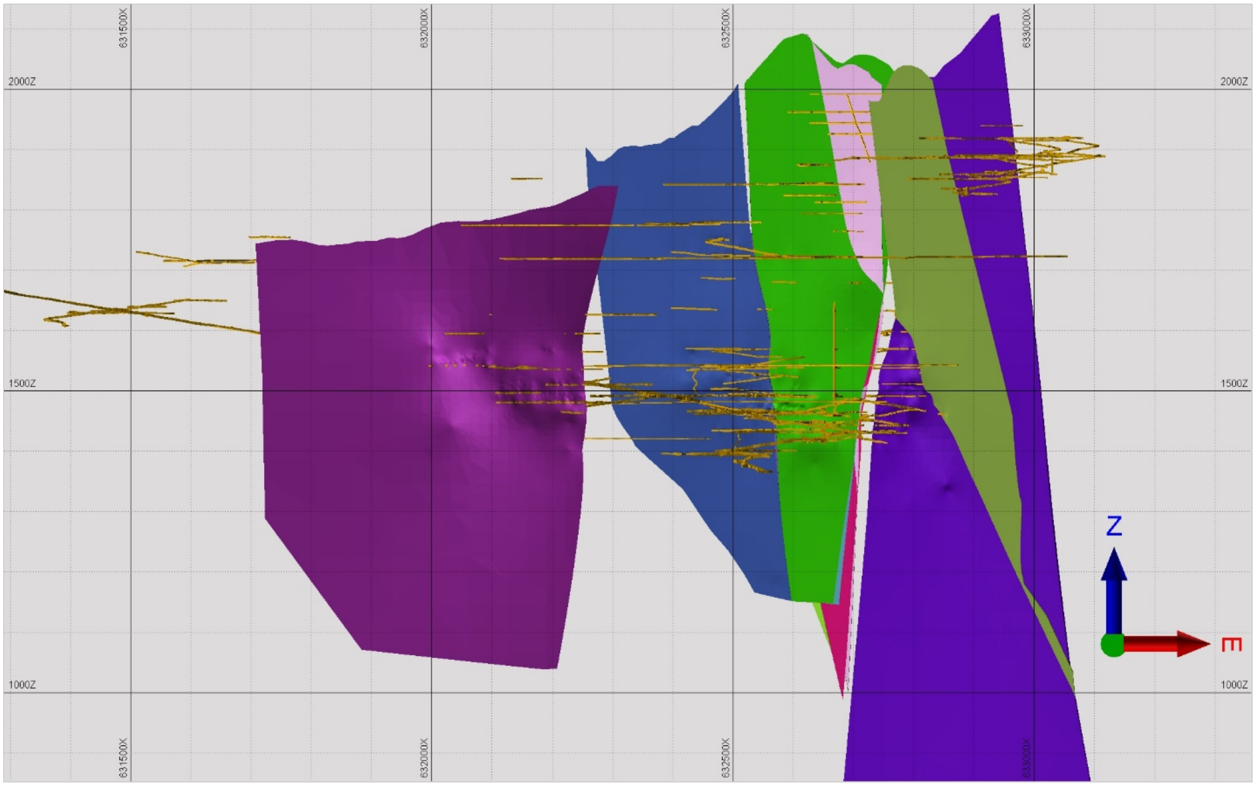


Figure 14-4: 3D View looking North of the Wireframes from the Santa Juana Area (north) and San Mateo Area (South)

14.4.2 Secondary Veins

For the secondary veins, initial vein intervals were used to generate a vein centerline surface. Intervals were adjusted to construct the most spatially probable vein surface. The highest-grade vein interval was not necessarily chosen when fitting the vein centerline surface. Gentle curvature in surfaces were common as veins extended away from ideal host lithologies and became distal to the features that prepared the host structures. The vein model proposed assumes a continuous traceable vein structure as suggested in the level mapping. Vein splays were captured and assigned subordinate identifiers where they could be traced along dip and strike. Each interval interpreted to be on-vein was given a numeric vein code, position code (center, hanging wall, or footwall), and a fault side indicator where offsets existed. Following vein assignment centerline surfaces were generated for each unique combination of identifiers. Combinations without extensive sample support or spatial continuity were not generated into centerline surfaces.

The centerline surface was interpolated using implicit modeling on a fixed 4 m by 4 m grid. The surfaces are not “snapped” to the vein intervals due to resistance in the implicit modeling algorithm and insignificant localized positioning error. Where initial vein extrapolations deviated greatly additional spatial control points were added to guide the model. The extrapolation of the vein surfaces was limited by spherical buffers, the topography, and known termination points. Each vein surface was pierced by each drill hole in the database. If a vein was pierced by a drill hole and a composite was not previously identified for that drill hole, a composite was inserted with a given thickness of 0.001 m and half detection limit grade attributes. Very low values were used instead of “0” because of the issue caused with “0” when estimating grade multiplied by thickness and converting back to grade. The effect of the inserted composites was block model attributes that pinched in grade and thickness at pierce points that were not flagged as on-vein. **Figure 14-5** and **Figure 14-6** show 3D views of modeled vein centerline surfaces for the Chicago and Santa Juana areas.

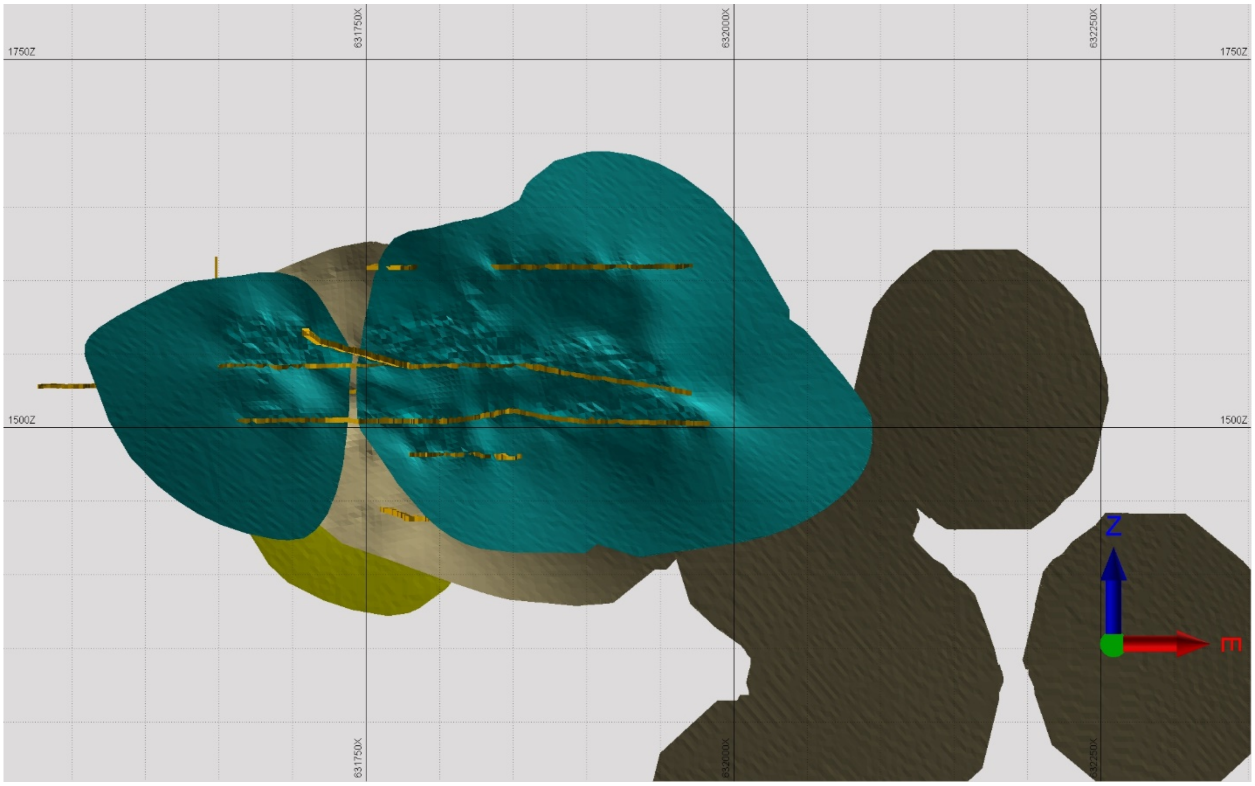


Figure 14-5: Surfaces of Secondary veins in the Chicago area, looking North

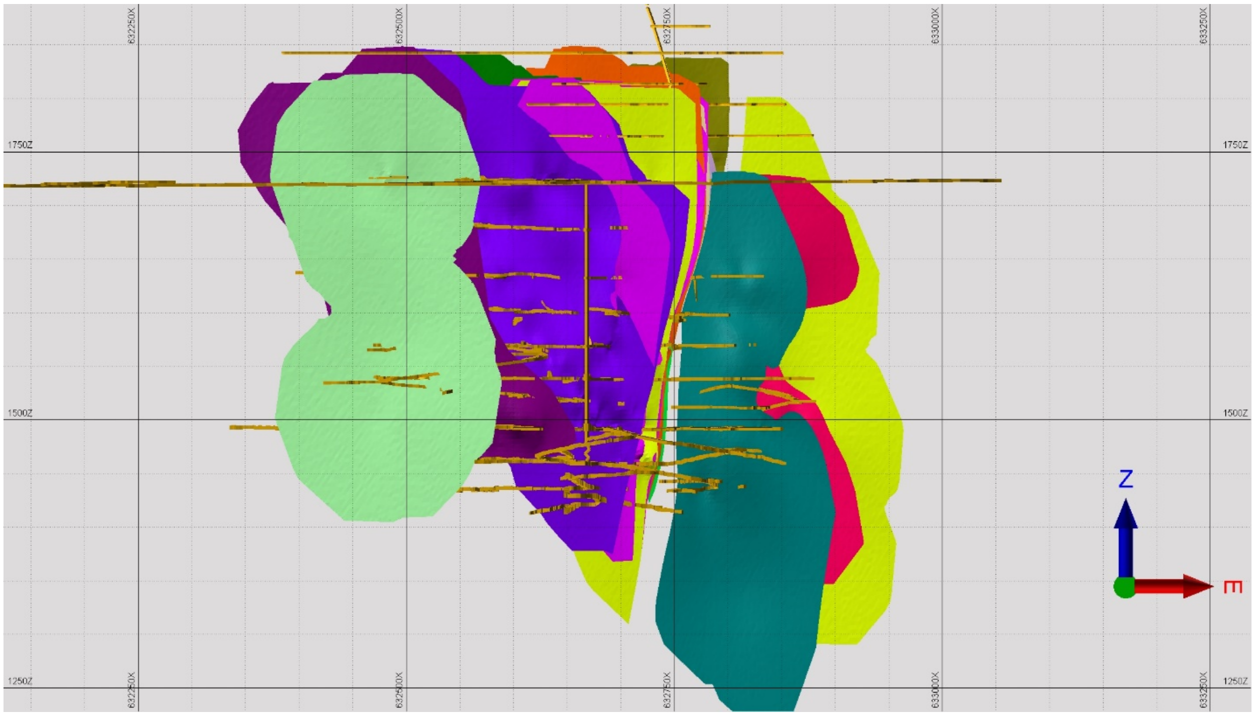


Figure 14-6: Surfaces of Secondary veins in the Santa Juana area, looking North

14.4.3 Mineral Type Boundaries

Mineral types have been flagged as attributes in the block model along with grade attributes. Strings were created to model the oxide, sulfide, and mixed (transition) boundaries. These boundaries have been updated since the 2015 report and are included in the Resource model.

14.4.4 Boundary Exclusions

Each block has been flagged by historic and recent mine cavity (stope) polygons and assigned a mined-out code of “1”. In addition, Resource reporting has been limited to within the claim boundary by adding an on-claim designation to the block model.

Mined out shapes have been provided by the mine site staff. Due to the three dimensionality of the current block model and the mine working primarily in 2D, and often in false coordinate space, the depletion boundaries are not exact when reviewed in context of the channel positions. The current model and the composite positions are relative to each other in as close to true 3D as possible.

14.4.5 Density Determination

Golden Minerals' geologists have made several thousand measurements on core and hand samples collected from underground workings using the water immersion method. Samples were collected where accessible and were not collected on all veins. Where measurements were not made, values associated with nearby veins or the default values based on mineral types were used.

Vein density values for oxide, mixed and sulfide mineral types were assigned to blocks of the same vein and mineral type. For purposes of dilution, waste was assigned a density of 2.6 g/cm^3. **Table 14-4** details the density values used for each vein.

Table 14-4: Vein Density Used in Model

Vein Name	Oxide	Mixed	Sulfide	Vein Name	Oxide	Mixed	Sulfide
A4	2.97	3.76	3.55	C2_NW	2.82	3.34	3.57
CC	2.98	3.34	4.03	D0_NW	2.82	3.34	3.57
C1	2.9	3.34	3.57	D1_NW	2.82	3.34	3.57
G1	2.77	3.34	3.57	DD_NW	2.82	3.34	3.57
F1	2.82	3.34	3.57	Ds_NW	2.82	3.34	3.57
Escondida	2.82	3.19	3.57	E_NW	2.82	3.34	3.5
Chicago	2.82	3.34	3.57	E1_NW	2.82	2.86	3.57
Hiletas	2.82	3.34	3.57	EE_NW	2.82	3.34	3.57
Roca Negra	2.82	3.34	3.18	Trans_NW	2.85	3.34	3.57
San Mateo	2.67	3.34	3.41	VetaOriente_NW	2.82	3.34	3.21
Terneras	2.81	3.34	3.57	A1_SE	2.39	3.38	3.57
D2_NW	2.82	3.34	3.57	A2_SE	2.82	3.34	3.65
SantaJuana_NW	2.82	3.34	3.81	AA4SE_A4Alto	2.82	3.34	3.57
SantaJuana_SE	2.82	3.34	3.81	D1_SE	2.82	3.34	3.57
SantaJuanaFW1_SE	2.82	3.34	3.81	Chicago NE	2.82	3.34	3.57
Trans_Alto_NW	2.85	3.34	3.57	Gambusino_NE	2.94	3.05	3.57
A1_NWHW1	2.39	3.38	3.57	Gambusino_SW	2.94	3.05	3.57
A2_NW	2.82	3.34	3.65	Nueva	2.89	3.59	3.59
A3_NW	2.82	3.34	3.57	Brenda	2.82	3.34	3.57
Bs_NW	2.88	3.57	3.57	LosMuertos	2.82	3.34	3.57
Bs_NWhw	2.88	3.57	3.57	San Juanes	2.82	3.34	3.81
G2_NW	2.82	3.34	3.57	San Juanes_fw1	2.82	3.34	3.81
C0_NW	3	3.34	3.57	Estrato_Chicago	2.81	3.34	3.57

14.5 Estimation Methods and Parameters

Resources have been estimated for each named vein using a variable thickness block model oriented in the best fit plane of the vein for the secondary veins. Primary veins have been updated in a 4x4x4 meter, block factored block model. Block attributes have been estimated using inverse distance weighting to a power of 2.5.

14.5.1 Variography and Search

The grade distance relationship was investigated for Ag, Au, Pb, and Zn using natural log transformed omni-directional variography on composited vein intervals. The entire composite dataset contributed to the variogram model without cross-vein influence. Composites from each vein were shifted to false space by a separation distance of the maximum range of the analysis. This enabled the mineralized system to be analyzed all at once.

Experimental and modeled variograms are shown in **Figure 14-7**, and **Table 14-5** details the modeled components. Nugget and sill portions have not been made relative to a total sill of one or 100% to correspond with the graphical output presented in **Figure 14-7**. Each variogram was well formed with Ag and Au having better-formed experimental variograms than Pb and Zn.

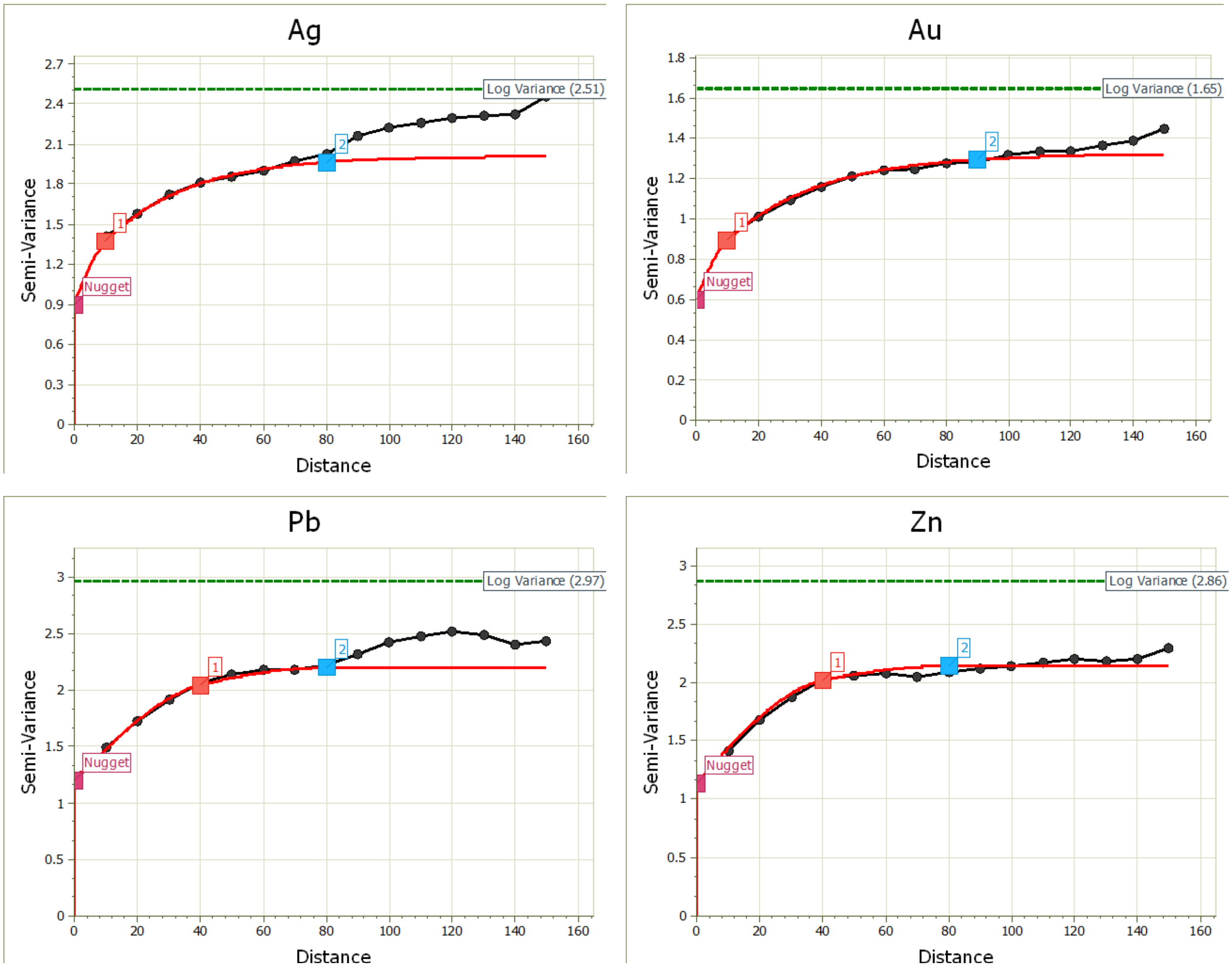


Figure 14-7: Natural Log Transformed Omni-Directional Variography

Table 14-5: Modeled Variograms

Element	Nugget	C1 Partial	C1 Range m	C2 Partial Sill	C2 Range m	Total Sill
Ag	0.9	0.18	10	1	85	2.1
Au	0.6	0.125	10	0.6	90	1.3
Pb	1.2	0.5	40	0.5	80	2.2
Zn	1.14	0.6	40	0.4	80	2.1

Although grade-distance relationships were investigated, the ultimate search distances, classifications, orientations and anisotropies implemented were guided by a combination of the results of the Ag and Au variography, visual review of the vein and professional judgment.

14.5.1.1 Primary Vein Estimation

A 3D block model was created for each of the primary veins, with block sizes of 4x4x4 meters. Block percent on vein was assigned to an attribute within the block using the vein wireframe models. Primary vein block models utilized dynamic anisotropy for each block in the model. A direction was assigned to each block by Micromine using the wireframe vein models. The search ellipse was then oriented based on this information in each block, providing a dynamic search that follows the curves of the vein as modeled.

14.5.1.2 Secondary Vein Estimation

A tilted and rotated point model was generated from the nodes of the modeled vein centerline surface for the secondary veins. This was possible because each surface was triangulated on a grid with 4 m by 4 m cells. The point models generated are not typical fixed array block models. The location of each node is dependent on the interpolated surface and not centered at typical block model centroids. Each point node has been assigned the vein ID and estimated grade and thickness attributes for the secondary veins, Search orientation, anisotropies, and maximum ranges are shown in **Table 14-6**.

Table 14-6: Vein Estimation Parameters for Secondary Veins

Vein Name Character	Dip+	Dip Direction+	Radius Pass1	Axis1 Azi	Axis1 Plunge	Axis2 Plunge	Vein Name Character	Dip+	Dip Direction+	Radius Pass1	Axis1 Azi	Axis1 Plunge	Axis2 Plunge
D2_NW	76	57	75	57	76	0	**E1_NW**	72	46	75	46	72	0
SantaJuana_NW	75	58	75	58	75	0	**EE_NW**	72	66	75	66	72	0
SantaJuana_SE	75	49	75	97	70	-14	**Trans_NW**	70	182	75	130	53	30
SantaJuanaFW1_SE	77	51	75	97	70	-14	**VetaOriente_NW**	54	15	75	338	51	19
Trans_Alto_NW	70	182	75	130	53	30	**A1_SE**	75	49	75	140	58	32
A1_NWHW1	76	57	75	65	79	0	**A2_SE**	72	35	75	55	80	0
A2_NW	76	57	75	68	79	0	**AA4SE_A4Alto**	77	50	75	54	77	0
A3_NW	76	57	75	68	79	0	**D1_SE**	77	47	75	47	77	0
Bs_NW	74	58	75	65	79	-5	**ChicagoNE**	72	140	75	140	72	0
Bs_NWhw	73	40	75	65	79	-5	**Gambusino_NE**	65	140	75	140	65	0
G2_NW	77	35	75	35	77	0	**Gambusino_SW**	60	150	75	150	60	0
C0_NW	84	40	75	112	73	-16	**Nueva**	74	160	75	160	74	0
C2_NW	78	34	75	40	73	1	**Brenda**	79	148	75	148	79	0
D0_NW	76	57	75	57	76	0	**LosMuertos**	86	150	75	258	83	7
D1_NW	76	57	75	57	76	0	**San Juanes**	60	3	75	25	60	-7
DD_NW	76	57	75	57	76	0	**San Juanes_fw1**	60	325	25	25	60	-7
Ds_NW	75	48	75	48	75	0	**Estrato_Chicago**	82	168	75	168	82	0
E_NW	60	51	75	51	60	0							

14.5.2 Resource Classification

Block attributes were estimated in three passes from small to large. **Table 14-7** details the search ellipse sizes and orientations along with sample selection criteria and classification. Resource classification was assessed by pass (maximum search), number of samples, and the nearest composite and average distance. Maximum extrapolation and therefore total potential was limited by the extent of the modeled vein surface, see column 'Buffer' in **Table 14-6**, as well as the clipping limitations where veins were known to be limited. Measured or Indicated classification was only permitted in pass one, 75 m maximum search, and was primarily but not exclusively defined within blocks haloing the existing drifts and stopes.

Table 14-7: Pass Parameters and Classification

Pass	Method	Max Search	Ratio 1st:2nd:3rd	Sectors	Max Per Sector	Comp Min	Comp Max	Classification
First	IDW 2.5	75	See vein parameter table	4	2	1	8	Inferred, Indicated if; comps >=3 and nearest comp <= 50m, Measured if; comps >=4 and nearest comp <= 16m and average comp distance <= 25
Second	IDW 2.5	150	1:0.25:0.5	1	2	1	2	Not classified, Inferred if; nearest comp <= 125m
Third	IDW 2.5	200	1:0.5:0.5	1	2	1	2	Not Classified

14.5.3 Dilution

Grade and thickness estimations were completed as undiluted. Diluted thickness and grades were calculated after estimation. The dilution assumes a minimum mining width of 0.70 m, and has accounted for hanging wall and footwall waste where true thickness was less than 0.70 m. If a block is estimated to have a true thickness less than 0.70 meter, the diluted thickness is 0.70 m. If a block is estimated to have a true thickness greater than or equal to 0.70 m, the diluted thickness is equal to true thickness. Grades and tonnes were diluted by the ratio of true thickness to diluted thickness. Variable vein density was used for the true thickness vein portion and the waste portion was assigned a density of 2.6 g/cm^3.

14.5.4 Cutoff Grade and NSR Calculation

Resources have been tabulated using a US$125/t NSR cutoff grade for each 4 m by 4 m block with the price assumptions shown in **Table 14-8**. The Resource tabulation is presented as the three-year trailing average prices as of December 2019. The prices used are US$16.30/troy ounce Ag, US$1,305/troy ounce Au, US$0.99/lb Pb, and US$1.27/lb Zn.

Table 14-8: Cutoff Price Assumptions

Assumption	Value
Ag Price $/oz	16.30
Au Price $/oz	1,305
Pb Price $/lb	0.99
Zn Price $/lb	1.27

NSR has been calculated with concentrate characteristics and marketing terms supplied by Golden Minerals. Metal contributions are dependent on the concentrate and mineral type, and the overall recoveries are shown in **Table 14-9**.

Table 14-9: NSR Metallurgical Recovery Assumptions

Metal	Sulfide Metallurgical Recovery %
Au	67
Ag	90
Pb	72
Zn	77

For the oxide and mixed NSR equations the payable terms were combined as single factors with the recoveries and were provided by Golden Minerals. Oxide and mixed mineral types are not the subject of the subsequent sections of this report that assess preliminary economics. Independent NSR cutoff calculations have been applied to oxide and mixed mineral types but the tabulated Resources have been grouped in the oxide category. The sulfide NSR equation has been updated for proposed mining areas that are the subject of this PEA and is based on metallurgical testing from that area.

14.6 Deleterious Elements

Deleterious elements that are relevant to the potential extraction of Resources are As and Sb. The Project database has inconsistent coverage for these elements. Recently collected channel and drill hole samples have the most complete information for As, but Sb is limited to recent drilling. Concentrate characteristics for As and Sb, determined through testing and mill actuals, have been used for purposes of Resource NSR cutoff calculations, instead of the incomplete project drill hole and channel database. This approach is the best alternative because the presence of As and Sb is most relevant in achieved concentrates; however, it does not account for spatial variability.

14.7 Statement of Resources

Resources at a $125 NSR cutoff are shown in **Table 14-8.** **Figure 14-8** to **Figure 14-15** show the grade and tonnage relationship at a range of NSR cutoffs using the base case price inputs.

Table 14-10: Velardeña Project Resources

Classification	Mineral Type	NSR Cutoff	Tonnes	Grade Ag g/t	Grade Au g/t	Grade Pb%	Grade Zn%	Ag oz	Au oz	Pb lb	Zn lb
Measured	Oxide	125	135,000	260	5.55	1.72	1.54	1,130,000	24,000	5,120,000	4,570,000
Indicated	Oxide	125	301,000	250	4.89	1.70	1.47	2,420,000	47,000	11,300,000	9,750,000
Measured + Indicated	Oxide	125	436,000	253	5.10	1.71	1.49	3,550,000	71,000	16,430,000	14,310,000
Inferred	Oxide	125	372,000	399	4.82	2.52	1.46	4,770,000	58,000	20,680,000	11,950,000
Measured	Sulfide	125	269,000	346	5.38	1.53	1.88	3,000,000	47,000	9,100,000	11,140,000
Indicated	Sulfide	125	645,000	327	4.62	1.43	1.94	6,790,000	96,000	20,300,000	27,530,000
Measured + Indicated	Sulfide	125	915,000	333	4.84	1.46	1.92	9,790,000	142,000	29,410,000	38,670,000
Inferred	Sulfide	125	1,393,000	342	4.70	1.51	1.97	15,320,000	211,000	46,380,000	60,400,000
Measured	All	125	404,000	317	5.43	1.60	1.76	4,120,000	71,000	14,220,000	15,710,000
Indicated	All	125	946,000	303	4.71	1.52	1.79	9,220,000	143,000	31,610,000	37,280,000
Measured + Indicated	All	125	1,351,000	307	4.92	1.54	1.78	13,340,000	214,000	45,830,000	52,990,000
Inferred	All	125	1,765,000	354	4.73	1.72	1.86	20,080,000	268,000	67,060,000	72,350,000

Notes:

(1) Reported Measured and Indicated Resources are equivalent to mineralized material under SEC Industry Guide 7

(2) Inferred Resource is not a recognized category under SEC Industry Guide 7

(3) Resources are reported as diluted Tonnes and grade to 0.70m fixed width

(4) Metal prices for NSR cutoff are three year trailing average as of December 2019: US$16.30/troy ounce Ag, US$1,305/troy ounce Au, US$0.99/lb Pb, and US$1.27/lb Zn

(5) Columns may not total due to rounding

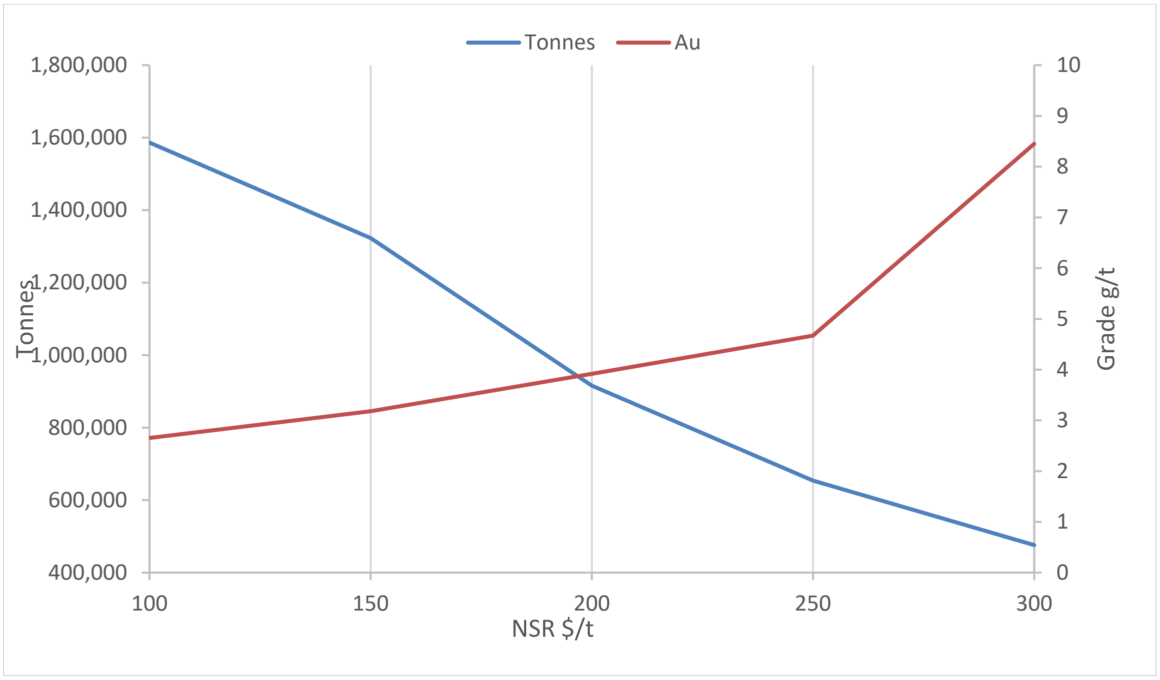


Figure 14-8: Grade tonnage curve, Measured and Indicated, Oxide and Sulfide, Au

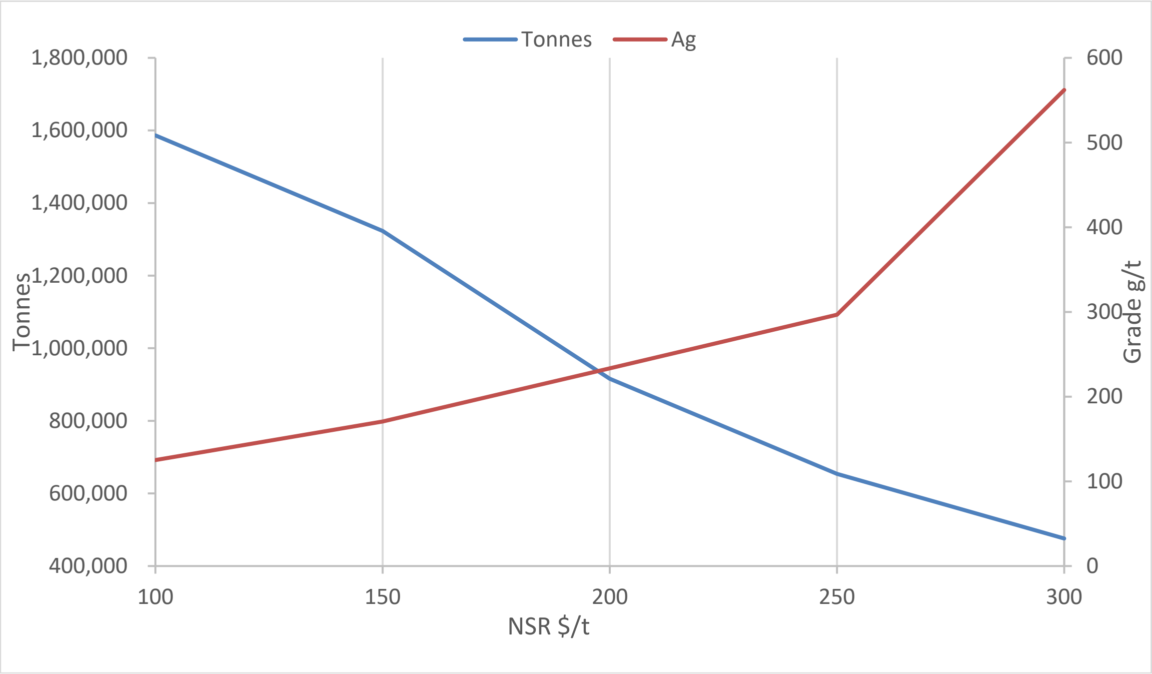


Figure 14-9: Grade tonnage curve, Measured and Indicated, Oxide and Sulfide, Ag

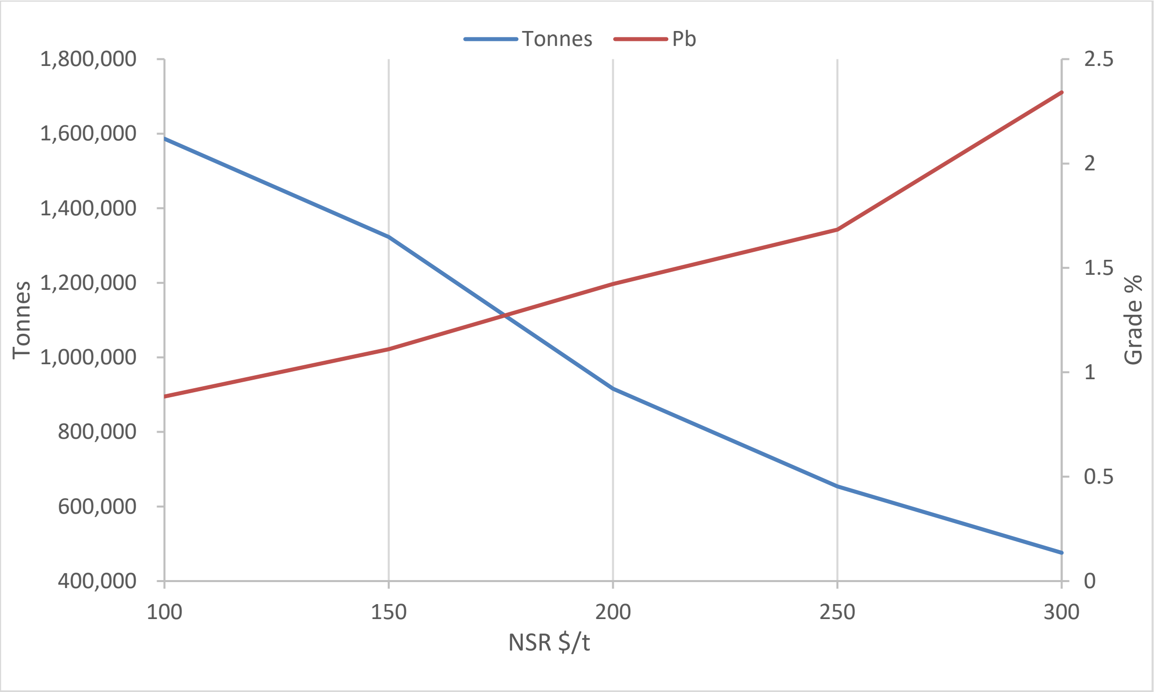


Figure 14-10: Grade tonnage curve, Measured and Indicated, Oxide and Sulfide, Pb

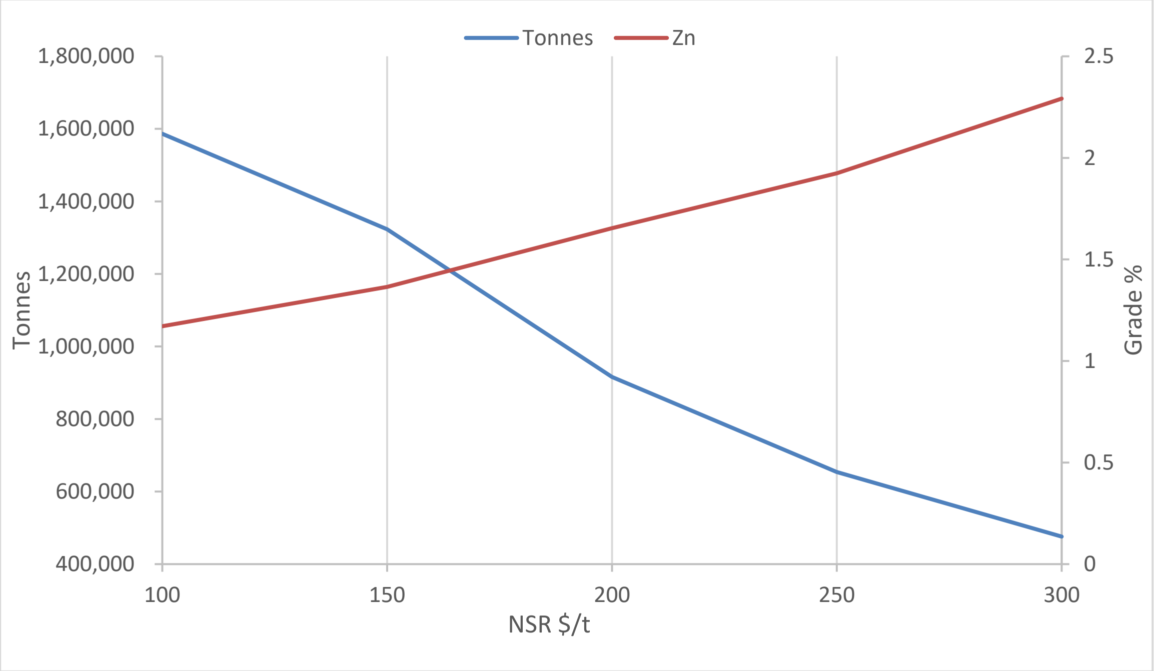


Figure 14-11: Grade tonnage curve, Measured and Indicated, Oxide and Sulfide, Zn

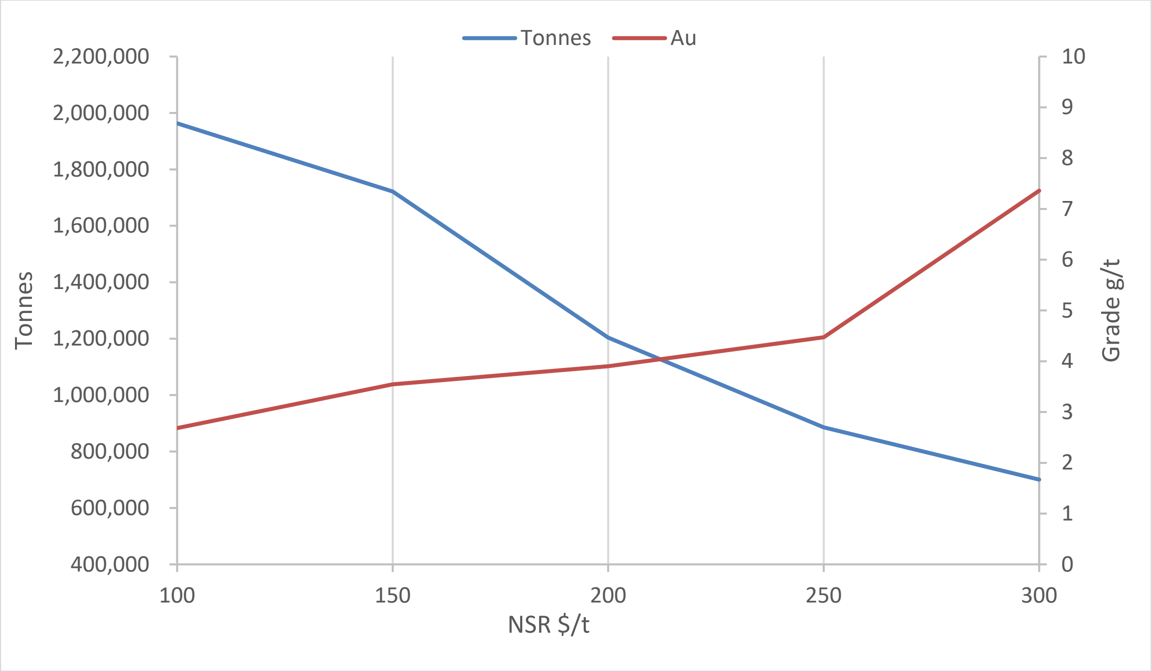


Figure 14-12: Grade tonnage curve, Inferred, Oxide and Sulfide, Au

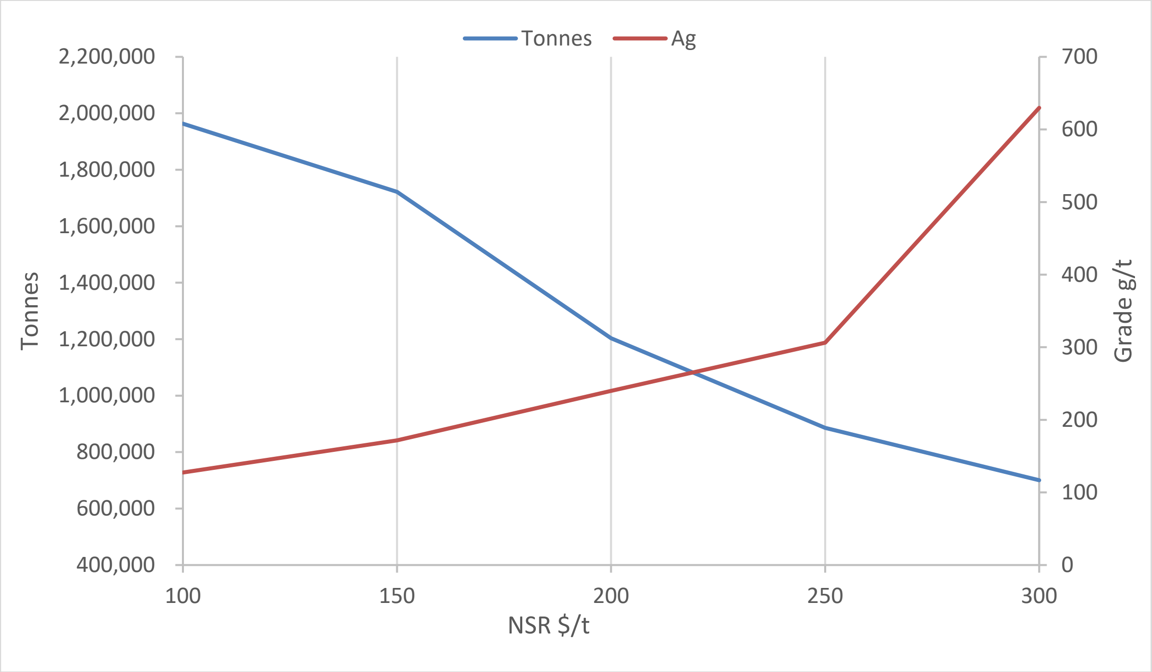


Figure 14-13: Grade tonnage curve, Inferred, Oxide and Sulfide, Ag

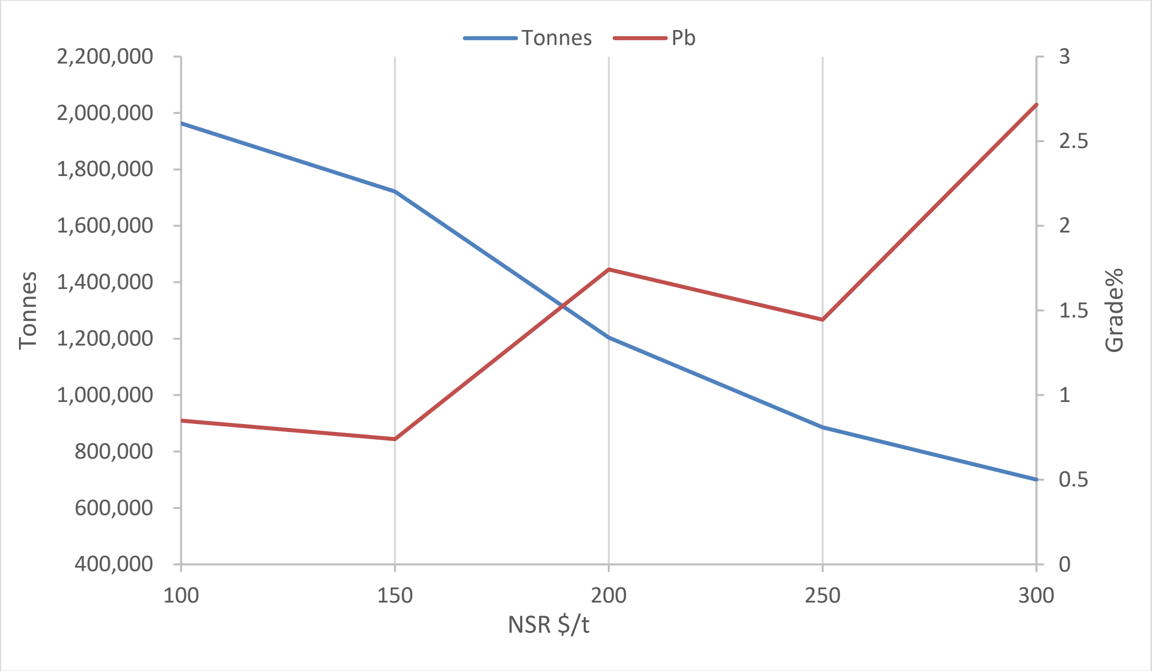


Figure 14-14: Grade tonnage curve, Inferred, Oxide and Sulfide, Pb

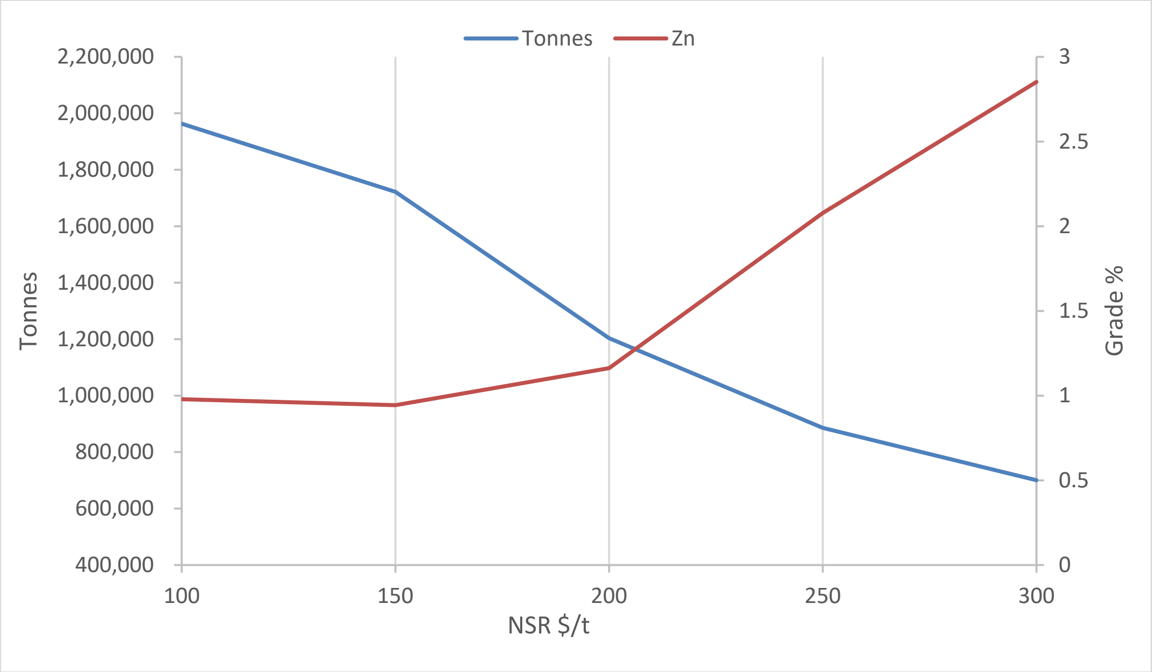


Figure 14-15: Grade tonnage curve, Inferred, Oxide and Sulfide, Zn

14.8 Model Verification

Resource estimations have been verified by visual review and population analysis.

The grade population was tracked for Ag from input assays (drill hole and channel), to composites, and to block grades. The grade progression histograms were compared as population relative as well as log normal transformed. The population comparison shows the means throughout the progression are sufficiently similar and that the high-grade component of the raw assays and composites have been satisfactorily moderated in the block population.

Long-section review of composite samples and block grades verify that the estimation respects the input data well. **Figure 14-16** to **Figure 14-19** is a series of long-sections looking north for the San Mateo vein as an example, showing composite values and resulting block grades for Ag, Au, Pb, and Zn. **Figure 14-20** shows the location of channel samples and the location of drill hole intercepts in relation to blocks classified as Measured, Indicated and Inferred.

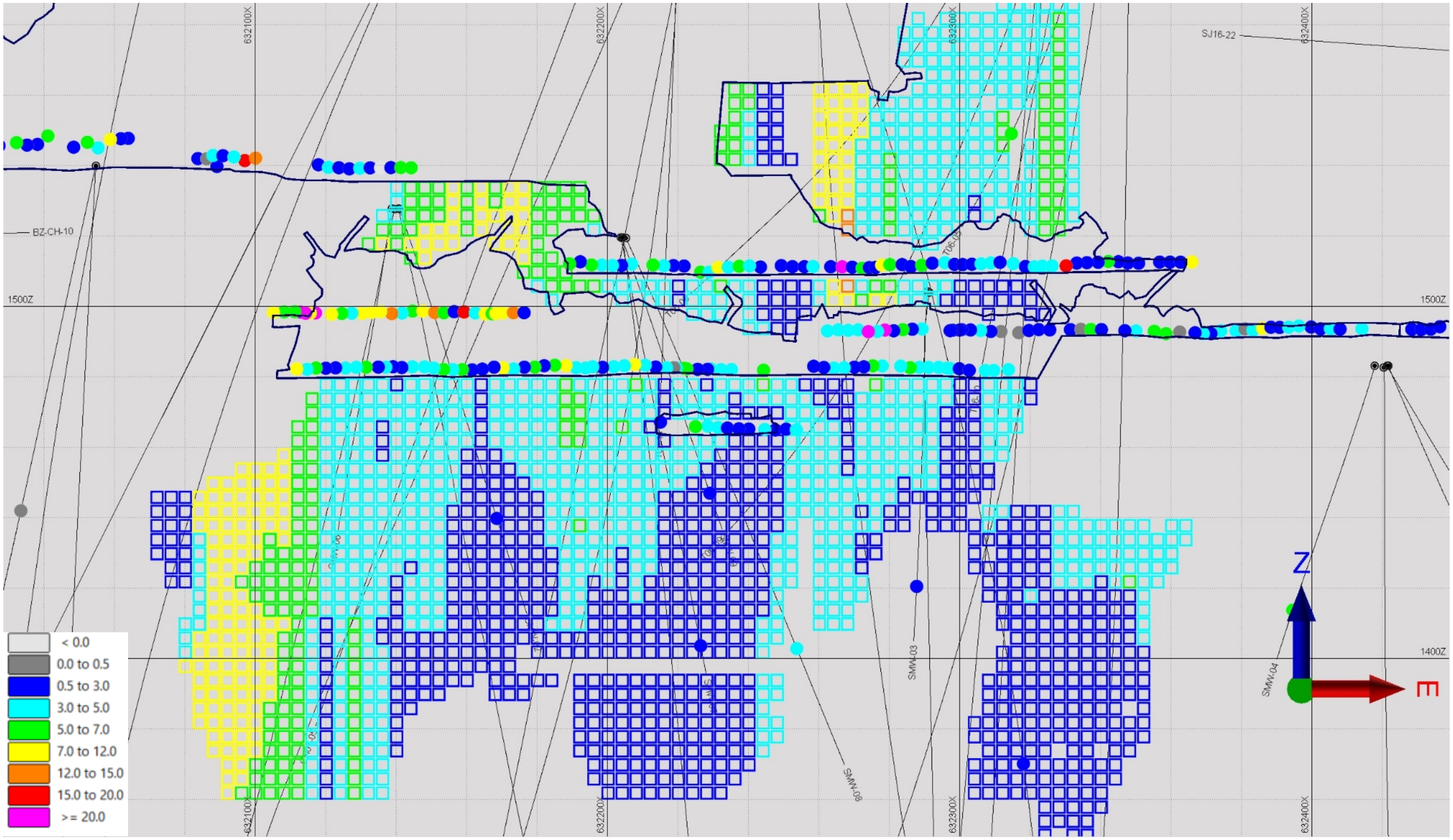


Figure 14-16: Long Section San Mateo Vein Au, Composites and Blocks in g/t.

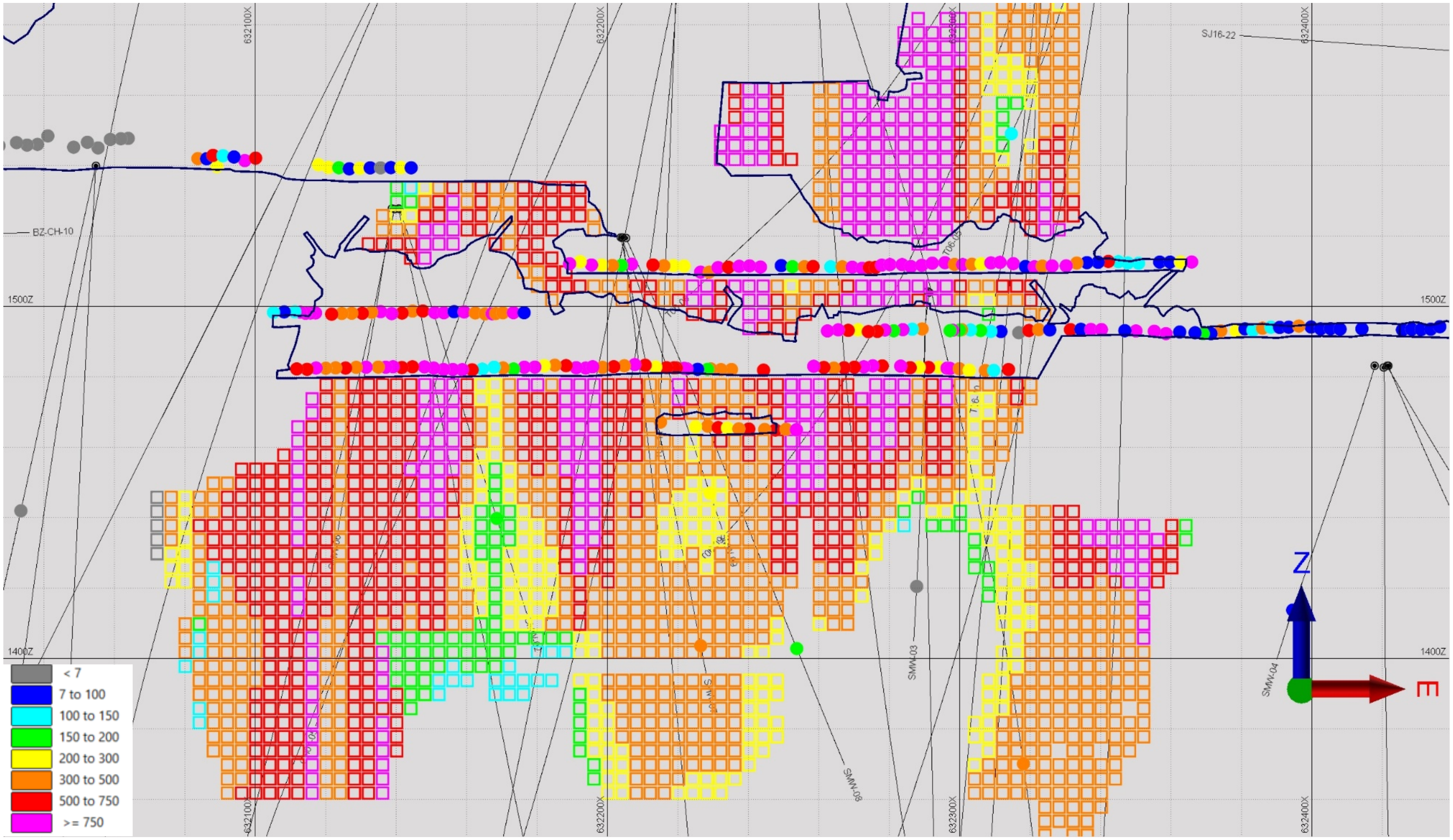


Figure 14-17: Long Section San Mateo Vein Ag, Composites and Blocks in g/t.

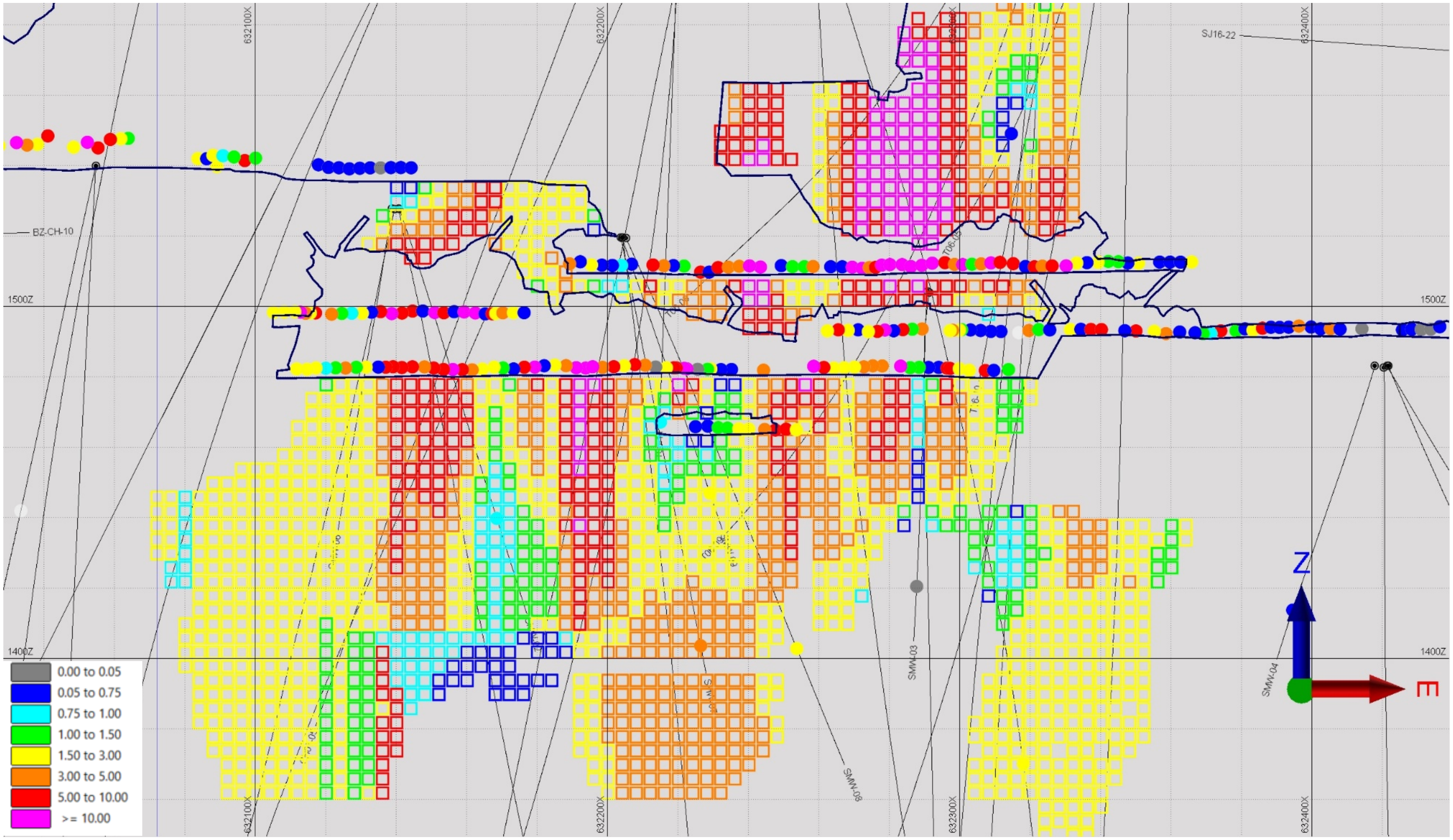


Figure 14-18: Long Section San Mateo Vein Pb%, Composites and Blocks

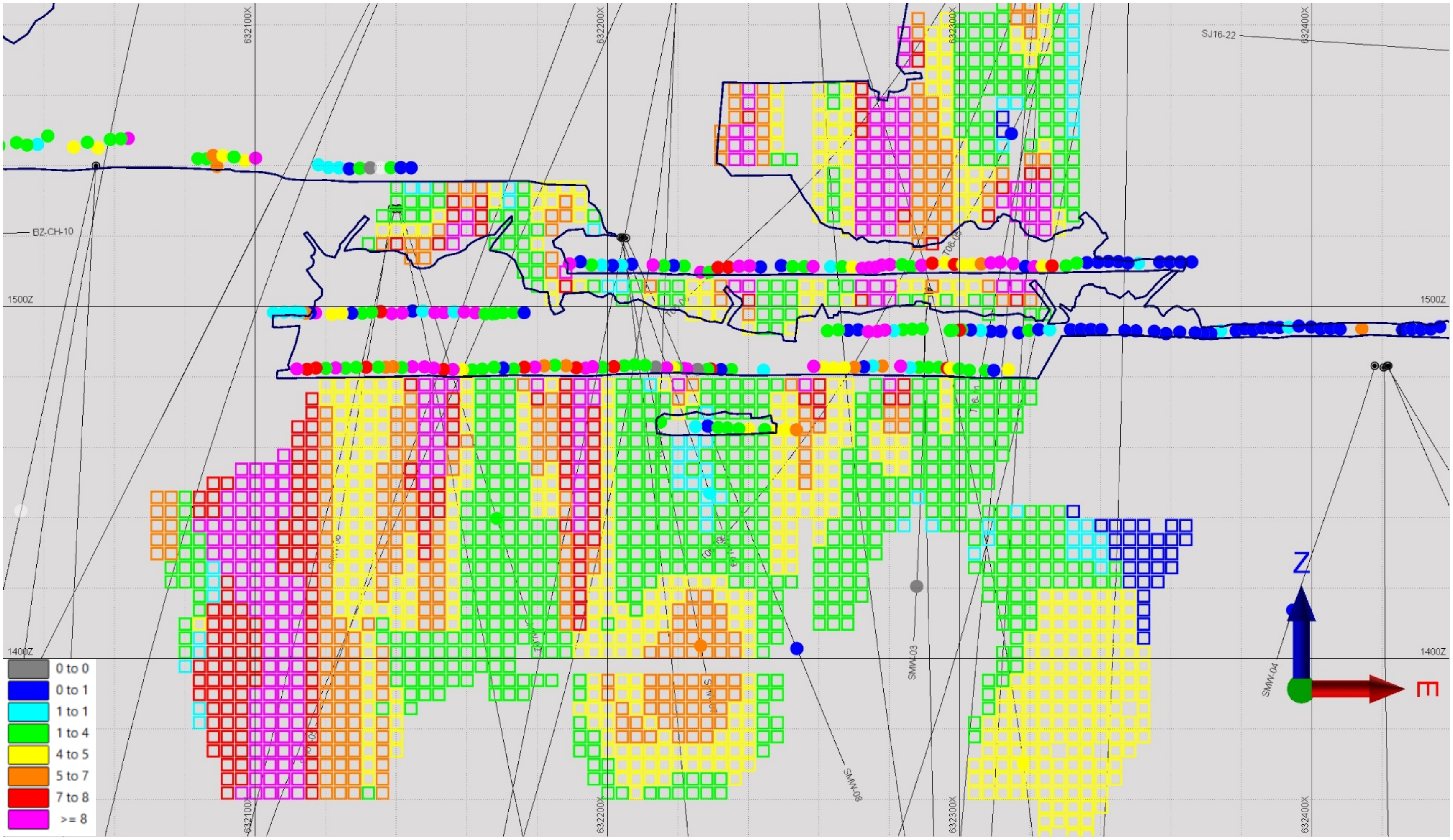


Figure 14-19: Long Section San Mateo Vein Zn%, Composites and Blocks

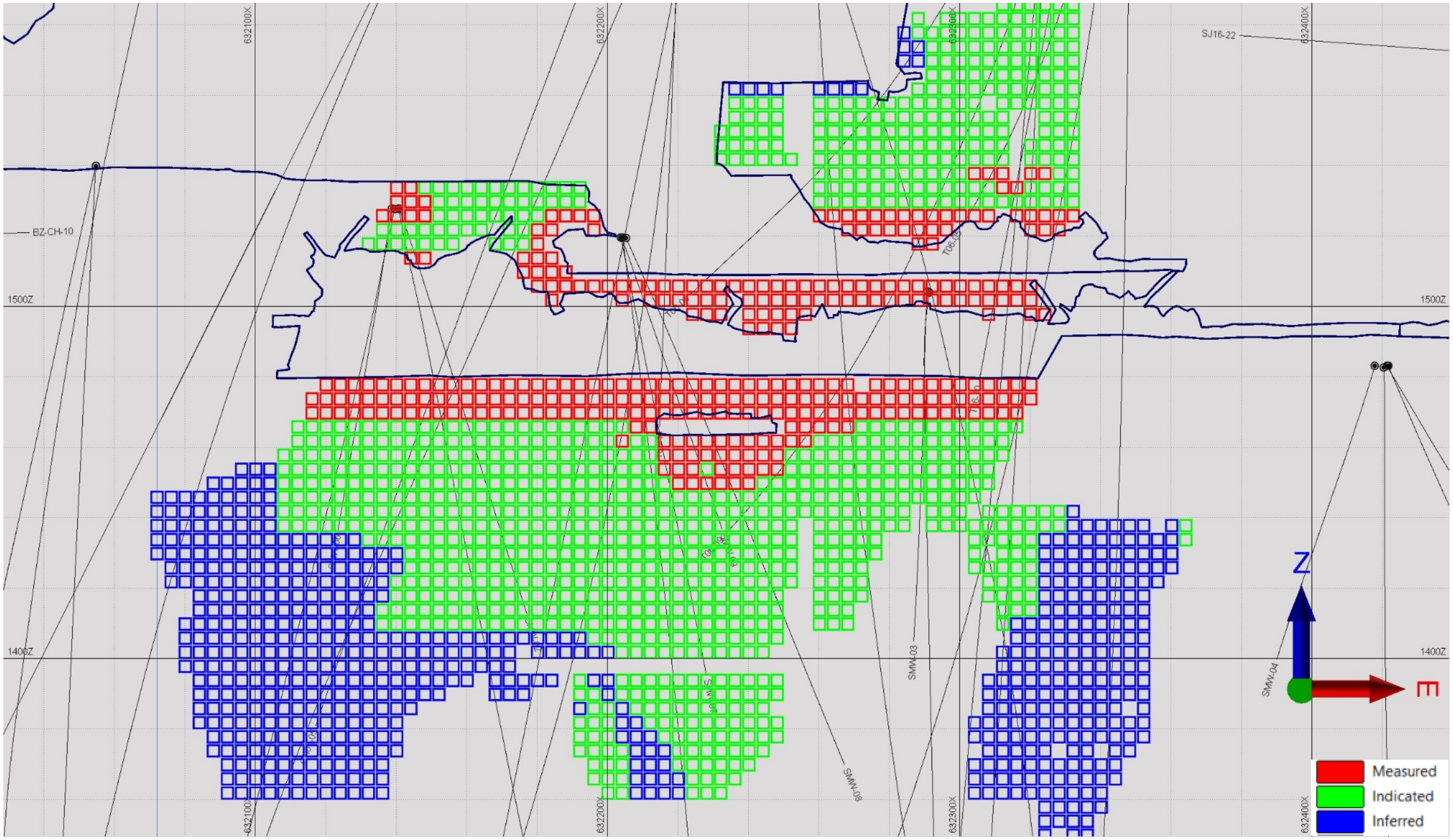


Figure 14-20: Long Section San Mateo Vein Classification.

14.9 Resource Expansion Targets

The following discussion of Resource expansion targets is conceptual in both tonnage and grade. There has been insufficient exploration to define these areas as a Mineral Resource and it is uncertain if further exploration will result in the target being delineated as a Mineral Resource. Quantification of Resource expansion potential is presented in **Table 14-11** and has been limited to the extent of the vein surface buffers and the potential classification estimated within passes two and three. Maximum extrapolation is 150 m and maximum search is 200 m. True Resource expansion potential is most likely much greater but additional quantifications cannot be currently justified. Previous estimates for potential were extrapolated 300 m from data points excluding the tonnage where Resources were classified; whereas this estimate for potential only represents the distance between the currently defined Resources and the maximum vein buffer, 105 m to 150 m, with the classified Resource excluded. In comparison to the previous, this estimate for potential is substantially more conservative.

Table 14-11: Quantifiable Resource Expansion Targets

Mineral Type	Cutoff NSR	Tonnes	Grade Ag g/t	Grade Au g/t	Grade Pb%	Grade Zn%
All	50	1,000,000 to 1,500,000	175 to 225	3 to 4	1 to 1.5	0.5 to 1

Notes:
1. *Resource expansion potential is not Resource and is not a recognized Resource category under SEC Industry Guide 7 or NI 43-101*
2. *Resource expansion potential is reported as diluted tonnes and grade*
3. *There is no guarantee or expectation that the above tonnage can be demonstrated or upgraded to a recognized Resource classification*

Most of the known veins' strike extents have been identified by exploration but in many cases mineralized shoots at depth have not yet been defined nor have the down dip extensions been condemned. It is likely that as deeper levels are developed additional mineralized shoots will be identified and enrich the Resource base. Demonstrating Measured and Indicated Resources below existing development levels is particularly difficult in a mine of this mineralization style. Resource expansion is unlikely to outpace level development due to the cost of drilling versus the achieved sample spacing.

Deep wedge drilling under the Tres Aguilas southeast fault block in the Santa Juana Mine area shows encouraging intercepts well below the deepest quantified potential. These intercepts have not been included in resource estimation because sufficient information regarding mineralizing style and orientation is not known. They do however suggest the system is mineralized well below the current Resource area.

14.10 Relevant Factors

If subsequently converted to Reserves and mined, the inability to precisely predict the true shape and orientation of mineralized shoots could materially affect the Mineral Resources. The geologic controls dictating the extents of the mineralized shoots are not currently known in much of the Inferred Resource areas. Interpolation and extrapolation of channel and drill hole samples represents an unbiased approximation of mineralized shoot shape but will fall short of predicting the shape exactly.

NSR calculations are based on reasonable price and contracts assumptions. The inability to market concentrates or changes in prices or contract terms could materially affect the quantified Resources in relation to the NSR cutoff. The estimation of in situ tonnage and grade attributes estimated would not be affected.

There are no additional environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that the author of this report is aware of that could materially affect the Mineral Resource estimate. The property has been in operation and many of the above factors have been studied in detail and addressed in the initial permitting process and have not affected the Resource estimates to date. It is possible complications with any or all of the above-mentioned factors could arise in the future, but currently no material complications are known.

Once the mine starts operations, it is recommended to closely check the productivity of the new bio-oxidation plant to verify if the expected results are obtained.

15 MINERAL RESERVE ESTIMATES

Though Measured and Indicated Resources have been estimated at the Velardeña mine, this preliminary economic assessment includes Inferred Mineral Resources that are too speculative for use in defining Reserves. Standalone economics have not been undertaken for the Measured and Indicated Resources and as such no Mineral Reserves have been estimated for the Project.

16 MINING METHODS

The Project is planned to be operated as an exclusively underground operation. The current mine plan includes only the sulfide material from the primary veins, which includes veins CC, C1, A4, F1, G1, San Mateo, Roca Negra, Hiletas, Terneras, Chicago, and Escondida.

Tetra Tech has conceptually planned stopes for mining based on Measured, Indicated and Inferred Resources, which total 1.38 million tonnes for mining over 10 years, from stopes and stope development. This tonnage is a subset of the total Resource that includes only sulfide material within the primary veins.

A site visit was conducted on December 10, 2019. The past extraction methods observed during the visit were Mechanized Cut and Fill stoping and Mechanized Resuing Cut and Fill stoping. These two techniques are considered for the PEA and are discussed below. These methods are suitable to the steeply dipping veins found at the Project.

16.1 Resue Cut and Fill Stoping

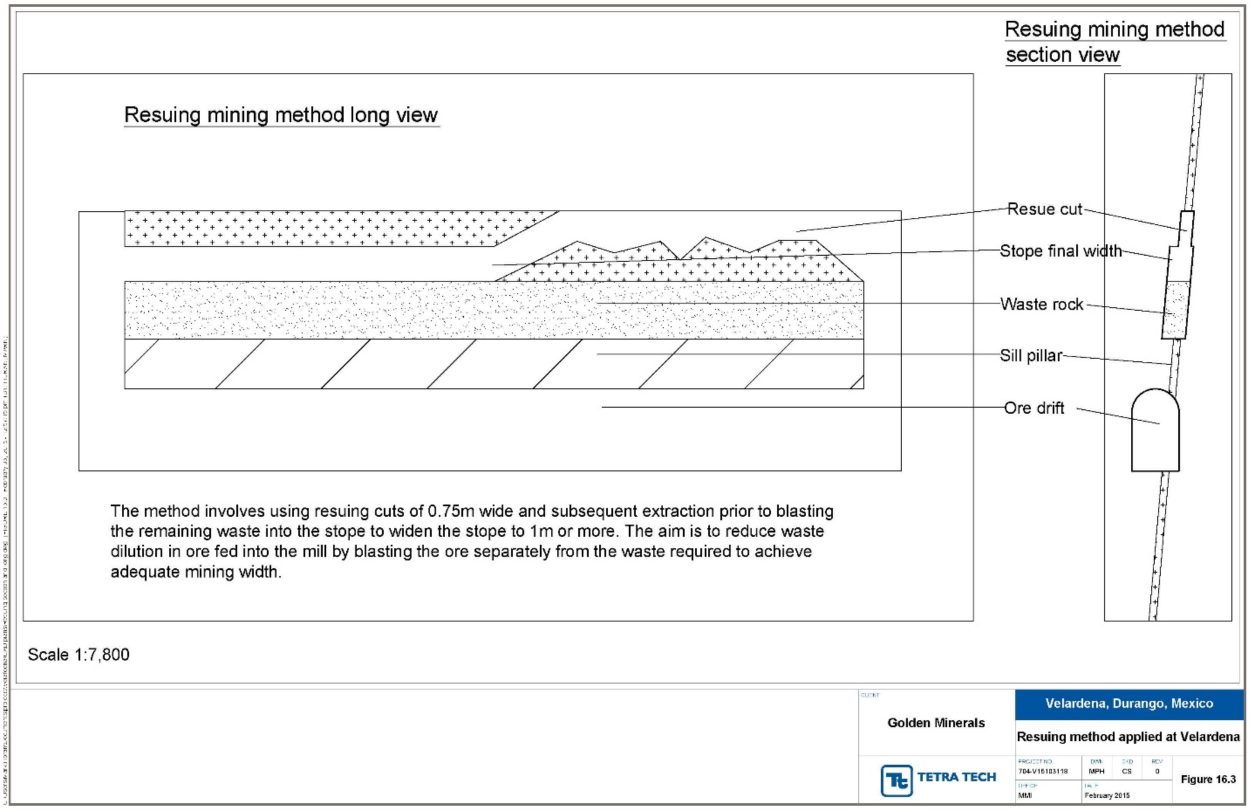
Mechanized Resue Cut and Fill Stoping is used when a vein system contains high grade material in a narrow width, less than one meter in width for the Velardeña vein packages. The following sequence outlines the stope life:

- Resue stoping begins by accessing the vein from a development drift with an attack ramp that enters the vein at a near perpendicular angle. The slope of the attack ramps ranges from -27% (initial) to +27% (final) to keep the attack development length to a minimum.
- The excavation of the stope's sublevel is driven away from the attack ramp intersection on vein. The sublevel is excavated using a horizontal resue technique; the footwall side of the face is excavated to one meter width for a single drill-round, followed by a slab waste round on the hanging wall side that brings the sublevel's final width for the single drill round to +/-2.0 meters. The single round horizontal resue technique is continued until the longitudinal boundaries of the process grade material is reached. Internal waste chutes encountered in the sublevel are excavated to the final sublevel width, +/-2.0 meters, with a single drill round. The sublevel is also taken in a single drill round when the mineralized material is greater than the resue cutoff width.
- The first cut will start once the sublevel is complete; the process grade material in the back will be drilled with vertical holes outlining the width of the process grade material; the minimum width is 0.7 meters. The drilled process grade material will be blasted and mucked from the stope before the widening of the excavation is started. The waste material will be blasted to bring the excavation to a final width of +/-2.0 meters, the blasted waste material will not be removed from the stope.
- The attack ramp will be slabbed, to enable smooth access to the stope excavation, with each resue cut.
- Waste from mine development areas or the surface will be hauled into the stope as fill to bring the sill level up to approximately 2.2 open vertical meters (the fill is the floor for working on the next cut).
- Tailings from Plant #1 will be spread in the stope on top of the course fill to reduce dilution and losses.

- The excavation of cuts continue until the top of the stope block is reached, 30 meters for the Velardeña mine plan.

The top of the stope block usually has a three-meter crown pillar that will be left to support the fill in the stope above. An engineered crown pillar can be constructed in the sill above the block so that the entire 30 vertical meters can be extracted; however, this normally occurs in areas of very high grade. The typical daily production for resue stoping at Velardeña is 20 to 30 tonnes per day of process grade material; the stope unit rate production is highly dependent of stope length. **Figure 16-1** displays a typical Resue Cut and Fill stope.

Ventilation of the stope is gained by installing an Axivane fan in the development that the attack ramp is started in. Brattice vent line, usually 24 to 30 inches, extends from the fan to the working faces in the stope. The development also contains main piping for compressed air, drill water, and a drainage. Typically, HDPE lines are extended from the main lines located in the development to the working faces. The brattice vent line and HDPE service lines are removed and re-installed each time the attack ramp is slabbed.



The method involves using resuing cuts of 0.75m wide and subsequent extraction prior to blasting the remaining waste into the stope to widen the stope to 1m or more. The aim is to reduce waste dilution in ore fed into the mill by blasting the ore separately from the waste required to achieve adequate mining width.

Figure 16-1: Illustration of Resuing Mining Method as Applied at Velardeña

16.2 Mechanized Cut & Fill Mining

Mechanized Cut and Fill Stoping is used when a vein system contains process grade material in a width greater than 1.8 meters; or when an area can be mined without the resue technique because the material with dilution is above the cutoff grade. The following sequence outlines the stope lifecycle:

- Cut and Fill stoping begins by accessing the vein from a development drift with an attack ramp that enters the vein at a near perpendicular angle. The slope of the attack ramps ranges from -27% (initial) to +27% (final) to keep the attack development length to a minimum.
- The excavation of the stope's sublevel is driven away from the attack ramp intersection on vein developing the sublevel at the width of the process grade material or the minimum width.
- The first cut will start once the sublevel is complete; the back will be drilled with vertical holes at the width of the process grade material or the minimum width. The drilled material will be blasted and mucked from the stope.
- The attack ramp will be slabbed, to enable smooth access to the stope excavation, with each resue cut.
- Waste from mine development areas or the surface will be hauled into the stope as fill to bring the sill level up to approximately 2.2 open vertical meters (the fill is the floor for working on the next cut).
- Tailings from Plant #1 will be spread in the stope on top of the course fill to reduce dilution and losses.
- The excavation of cuts continue until the top of the stope block is reached, 30 meters for the Velardeña mine plan.
- The top of the stope block usually has a three-meter crown pillar that will be left to support the fill in the stope above. An engineered crown pillar can be constructed in the sill above the block so that the entire 30 vertical meters can be extracted; however, this normally occurs in areas of very high grade. The typical daily production for stopeing at Velardeña is 40 to 50 tonnes per day of process grade material; the stope unit rate production is highly dependent of stope length.

16.3 Geotechnical Analysis

A geotechnical analysis for the Project has not been conducted or reviewed by the author. The mine is currently operating without significant underground support. Several areas of the underground workings were inspected during the site visit and it was observed that the rock mass is competent and self-supporting. No areas of concern were noted. It is recommended that, for mining at depth greater than the current, the services of rock engineering firm are engaged to provide expertise on stope layout and future potential rock mass stability concerns that may arise due to increased stress.

16.4 Dilution

Due to the narrow vein widths, waste dilution is greater than typical for underground mining operations. **Table 16-1** outlines the estimated dilution methods for the PEA. In addition, a minimum drift width of 2.5 m has been estimated for stope development.

Stopes have been planned to use a weighted average vein width of 0.67 m, with a weighted average stoping width of 0.98 m, resulting in an average dilution of 42% for stoping and 171% for development.

Table 16-1: Mining Dilution Estimation Parameters

Mining Method	Minimum Mining Widths	Dilution Applied
Resue	0.7 m	Vein width less than 0.5 - mining width is estimated at 0.7 Vein widths above 0.5 - mining width is estimated as vein width plus 0.2 m
Cut and fill	3 m	Vein width less than 2.6 m - mining width is estimated at 3 m Vein widths above 2.6 m - mining width is estimated as vein width plus 0.4 m

16.5 Mining Extraction & Recovery

Overall extraction of planned stopes has been estimated at 87%.

For the purpose of this PEA, the loss of Resources available to mining through mining extraction losses has been considered. The considerations include stoping with both shrinkage and resue mining which require the leaving of rib, sill and crown pillars. For the PEA rib, sill, and crown pillars have been included as 3 m in width.

A mining loss of 5% has been included, which accounts for blasted material left *in situ* in stopes, above pillars and in stope drifts after stope completion.

16.6 Mining Equipment

Table 16-2 shows the list of equipment available at the Project as provided by Golden Minerals. The key pieces of equipment required for mining are the scoop-trams, underground trucks and drilling jumbos. The current equipment fleet is expected to be adequate to achieve the 310 t/d of mill feed for processing and, as such, no additional equipment is expected to be purchased. Not listed here, but owned by Golden Minerals, are jacklegs required for stoping and underground development (narrow drifts) and ventilation equipment in use underground.

Table 16-2: Velardeña Equipment List

List of Equipment used by Minera William, S.A. de C.V.					
#	Tag #	Model	Manufac.	Series	Motor
Scoop Trams					
1	ST-1	MTI-270	MTI	3215	DEUTZ F5L912W
2	ST-04	EJC 100A	EMCO JARVIS CLARK	9171808	DEUTZ F6L413FW
3	ST-8	LT-125	MTI	509	DEUTZ F4L912W
4	ST-13	LT-270	MTI	9171808	DEUTZ F5L912W
5	ST-10	EJC-100	SANDVIK	O8861795	DEUTZ F6L914
6	ST-11	ST-2D	Wagner	RBO42009	1RBO42009
7	ST-17	ST 1030	ATLAS COPCO	AVO 11X265/8997 3178 00	Cummins QSL
8	ST-18	LT-250	JCI	67695	DEUTZ F6L914
9	ST-19	LH-203	SANDVIK	L203D767	DEUTZ BF6L914
10	ST-20	LH-203	SANDVIK	L103D778	DEUTZ BF6L914
11	ST-23	LT-210	MTI	4314	DEUTZ F4L912W
12	ST-24	LT-210	MTI	4313	DEUTZ F4L912W
13	ST-25	ST 1030	ATLAS COPCO	AVO 07X430/8997 149900	Cummins QSL
14	ST-26	LH-203	SANDVIK	L003D685	DEUTZ BF6L914
15	ST-27	LH-203	SANDVIK	L103D787	DEUTZ BF6L914
16	ST-28	LH-203	SANDVIK	L007D303	DEUTZ BF6L914
17	ST-29	LH 307	SANDVIK	L007D303	MB OM906LA
18	ST-30	LH-203	SANDVIK	L203D790	DEUTZ BF6L914
19	ST-31	50M	JCI	87388	
Drilling Jumbos					
1	JB-01	BOOMER S1D	ATLAS COPCO	AVO 11A239/8991894700	
2	JB-03	BOOMER S1D	ATLAS COPCO	AVO 08A640/8991 7 74400	DEUTZ D914L04
3	JB-02	BOOMER T1D	ATLAS COPCO	AVO11A362/8991895700	

List of Equipment used by Minera William, S.A. de C.V.					
#	Tag #	Model	Manufac.	Series	Motor
Motor Grader					
1	MOTO-01	CAT 140M	CATERPILLAR	CAT0140MLB9D02937	
Personnel Transport					
1	KU-01	RTV 900	KUBOTA	A5KB1FDACBG0C4080	
2	KU-02	RTV 900	KUBOTA	A5KB1FDACBG0C4078	
3	KU-03	RTV 900	KUBOTA	A5KB1FDAHBG0C6068	
4	KU-04	RTV 900	KUBOTA	A5KB1FDAKBG0C7535	
5	KU-06	RTV 900	KUBOTA	A5KB1FDAPCG0D4307	
6	KU-07	RTV 900	KUBOTA	A5KB1FDAACG0D1107	KUBOTA D902-ET03
7	KU-08	RTV 900	KUBOTA	A5KB1FDACG0D3167	
8	KU-08	RTV 900	KUBOTA	A5KB1FDACG0D3167	
9	KU-10	RTV 1140	KUBOTA	A5KB1FDAHCG0D6374	
Underground Trucks					
1	CBP-01	JCI 704	MTI		DEUTZ F6L914
2	CBP-02	JCI 704	MTI		DEUTZ F6L914
3	CBP-05	JCI 704	MTI	RB-148-0812	DEUTZ F6L914
4	CBP-06	JCI 704	MTI	RB-149-0812	DEUTZ F6L914
5	CBP-07	MT 431B (264)	ATLAS COPCO	AVO 12X463/8997 4225 00	DETROIT S-60
6	CBP-08	MT 431B (265)	ATLAS COPCO	AVO12X513	DETROIT S-60
7	CBP-09	TH-320	SANDVIK	4565	MERCEDES BENZ
8	CBP-10	TH-320	SANDVIK	4649	MERCEDES BENZ
Front End Loader					
1		916	CATERPILLAR	2XB01887	
2		930G	CATERPILLAR	CAT0930GETWR02020	
3		930G	CATERPILLAR	CAT0930GHTWR01237	

List of Equipment used by Minera William, S.A. de C.V.					
#	Tag #	Model	Manufac.	Series	Motor
Telehandler					
1	TH-01	TH 580 B	CATERPILLAR	CATTH580JSLH01098	
TLB (tractor, loader, backhoe)					
1		420E	CATERPILLAR	CAT0420ELKMW01116	
2		416E	CATERPILLAR	CAT0420ELKMW1116	
3		416D	CATERPILLAR	CAT0416DAB2D00688	
Bobcat					
1		236B	CATERPILLAR		
Compressors and Generators					
1	COM	TS-20-250-60	SULLAIR	9963	
2	COM	SSR-EPE300	INGERSOLLRAND	E1241U94053	
3	COM	EAU99P	GARDNER DENVER	S290593	
4	COM	SSR-XF100	INGERSOLLRAND	F8769U94104	
5	COM	267913U66327	INGERSOLLRAND	185WJD-196-D	
6	COM	P375WCU	INGERSOLLRAND	309961UCK413	
7	COM	9185WJD	INGERSOLLRAND	347689UG0221	
8	COM	P185WJD	INGERSOLLRAND	267913UGG327	
9	COM	ZR-4	ATLAS COPCO		
10	GEN	432R5L2014A-L00W		UH3509556	
Tractors					
1	TR-01	2635	MASSEY FERGUSON	FX729539	TSJ436E 05190 / MF 2635 4WD STD2
2	TR-02	2635	MASSEY FERGUSON	FX729535	MF 2635 4WD STD2
3	TR-03	2635	MASSEY FERGUSON	FX752999	MF2635 /MF 2635 4WD STD2
4	TR-04	2635	MASSEY FERGUSON	FX777239	MF2635 /MF 2635 4WD STD2

List of Equipment used by Minera William, S.A. de C.V.					
#	Tag #	Model	Manufac.	Series	Motor
Vehicles for Transporting Personnel and Cargo					
14	EX65140	International	CHASIS CABINA TANDEM 740		
15	EX01679	International	CHASIS C 7400-300 CAMION		
18	EX01622	International	CHASIS C 4400-250 CAMION		
33	EX05301	International	CHASIS CORAZA 3300 210 CE		
34	EX01616	International	CHASIS CABINA 7400 310		
35	EX01625	International	CHASIS CABINA 7400 310		
4	EX05302	International	AUTOBUS 4700 22 FE		

16.7 Waste Rock

Waste rock from the underground mine consists of tonnage from the ramp and lateral development. Since the mining methods include cut and fill, the waste from the stopes would either be stored underground in mined out stopes or transported to the mill with the diluted mined material. Currently limited cut and fill mining is undertaken and, as such, most of the waste rock is planned for surface storage.

The waste rock that would not be stored underground would be contained along the valley between the San Mateo adit and the Santa Juana adit.

16.8 Tailings

The dry tailings placed near Plant #1 are suitable for spreading on the fill of each cut to eliminate the dilution and losses associated with blasting process grade material on course placed fill. Tailings will be hauled from Plant #1 to the active mine and dumped at a centralized area. Trucks will then haul the tailings underground to a stope area where an LHD will spread the material on top of the recently placed course fill, a cover of approximately 15 centimeters. The planning and calculated production rates used in this estimate contain time for placing the tailings cover..

16.9 Dewatering

Neither a water balance nor dewatering investigations were performed for this PEA. The water handing system currently in place relies on a chain of submersible dirty water pumps to evacuate the inflow from the mine. No significant water infiltration was noted at the underground mine site during the site visit. Seepage and dewatering are not expected to be of concern; it is not anticipated that excessive dewatering costs will be incurred during the life of mine, but further studies are recommended to confirm this.

16.10 Ventilation

The current underground workings at the Project are naturally ventilated, with the main ramp used as an intake airway and the old Santa Juana mining areas and shafts for exhausting air. However, Golden Minerals is installing a booster fan which will force air from the San Mateo and Terneras areas, down the main adit and ultimately out of the old Santa Juana mining areas (reference Figure 16-2).

Access to the old shafts within the Santa Juana Mine is still possible and provides access for inspections to ensure that the old excavations remain open to provide exhaust.

Ventilation circuits are created in stoping areas through forced ventilation, via fans and ducting of various sizes. Stopes are set up to have a minimum of two entrances, which when connected provide for thorough ventilation.

No further evaluation on the ventilation has been performed but it is expected that the main booster fan, once installed, will be adequate for mine ventilation.

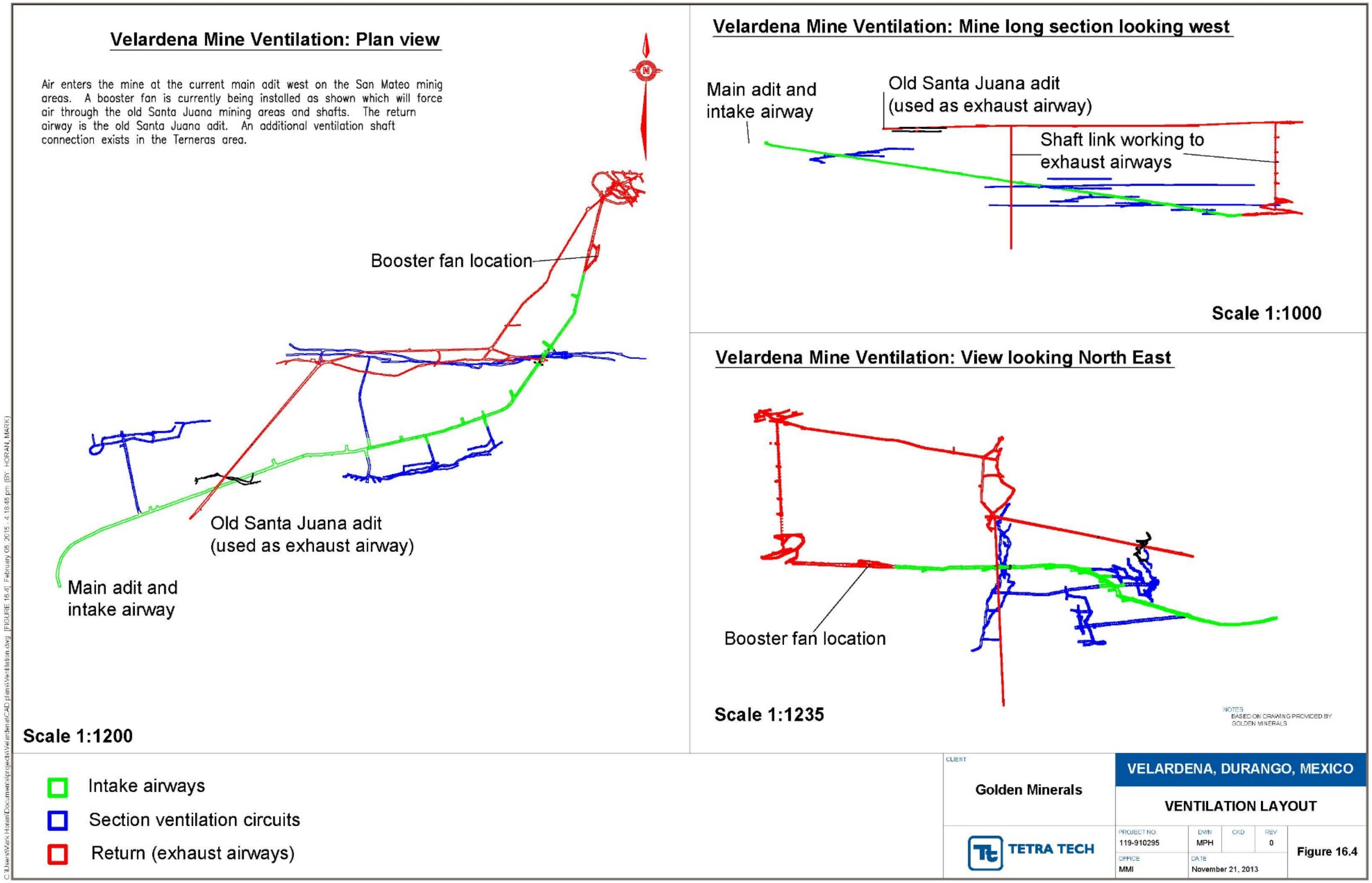


Figure 16-2: Ventilation Layout of the Velardeña Min

16.11 Power

The underground power is available from a primary substation located at the portal. The power taken into the mine is stepped down at the substation to 4160 volts. The 4160 is stepped down to a typical working voltage of 440 volts using mobile mine load centers or pad mount transformers set on concrete. The power is stepped down to 120/240 single phase in many locations at the load centers. The mine power system was modernized in 2011.

16.12 Mine Plan

In order to plan stopes for the PEA, areas were selected where: the estimated diluted NSR for a minimum of a 0.7 m mining width exceeds US$125 within the primary veins. **Table 16-3** shows the overall tonnage and the tonnage ultimately included for scheduling.

Table 16-3: Summary of Tonnes & Grade Included in the Proposed Mine Plan

Category	Total/Avg
Tonnes	1,137,949
NSR	295
Ag (gpt)	337
Ag oz	12,325,300
Au (gpt)	5.15
Au (oz)	188,250
Pb (%)	1.32
Pb (lbs)	33,096,126
Zn (%)	1.63
Zn(lbs)	40,886,729

16.12.1 Stope Layout

Resources were selected with a diluted NSR above US$125 and conceptual 2D stope shapes created for these areas. Existing development was considered for the mine plan. **Figure 16-3** to **Figure 16-6** shows the 2D stope plans with the existing development. The stope shapes were then used to flag the Resource blocks within the stope, and calculate attributes of each stope. Mining, dilution, extraction, and stope development parameters were applied to each individual stope in order to estimate grades and tonnages of potential mill feed from each stope.

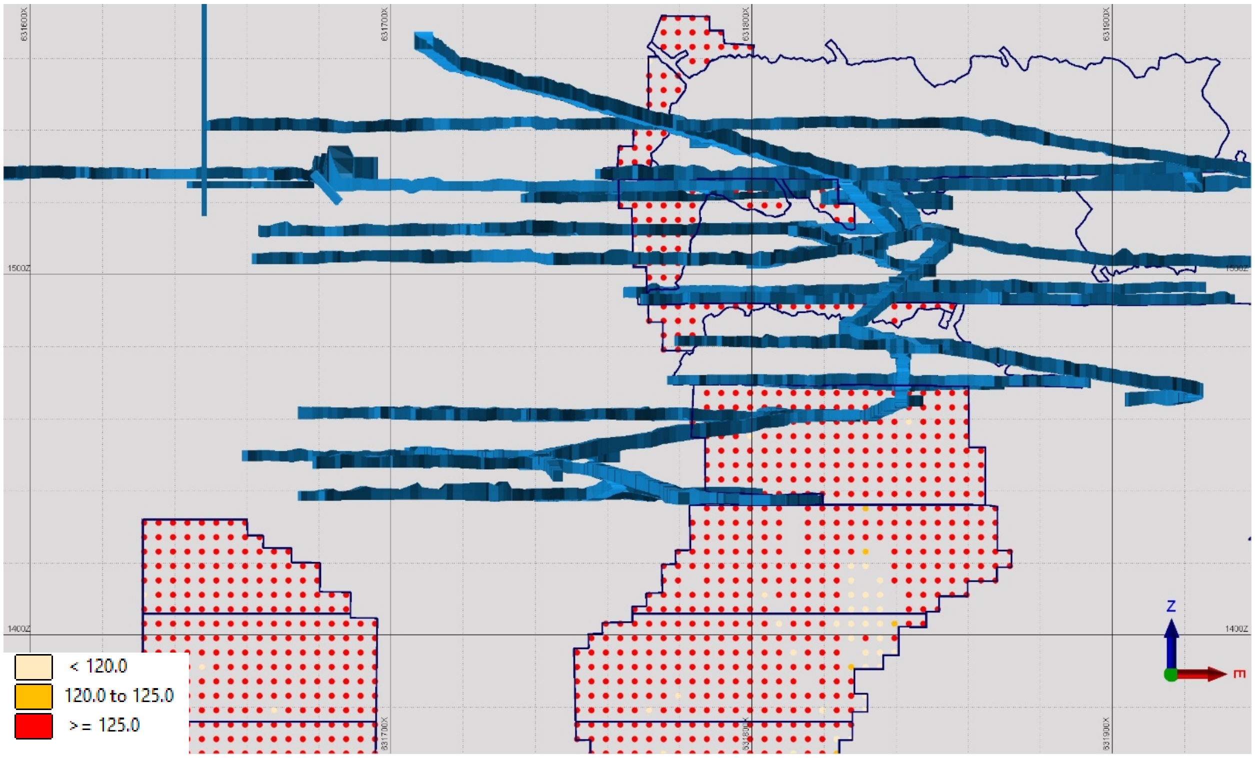


Figure 16-3: Example of detailed view of Chicago area, Escondida vein, stope and existing development

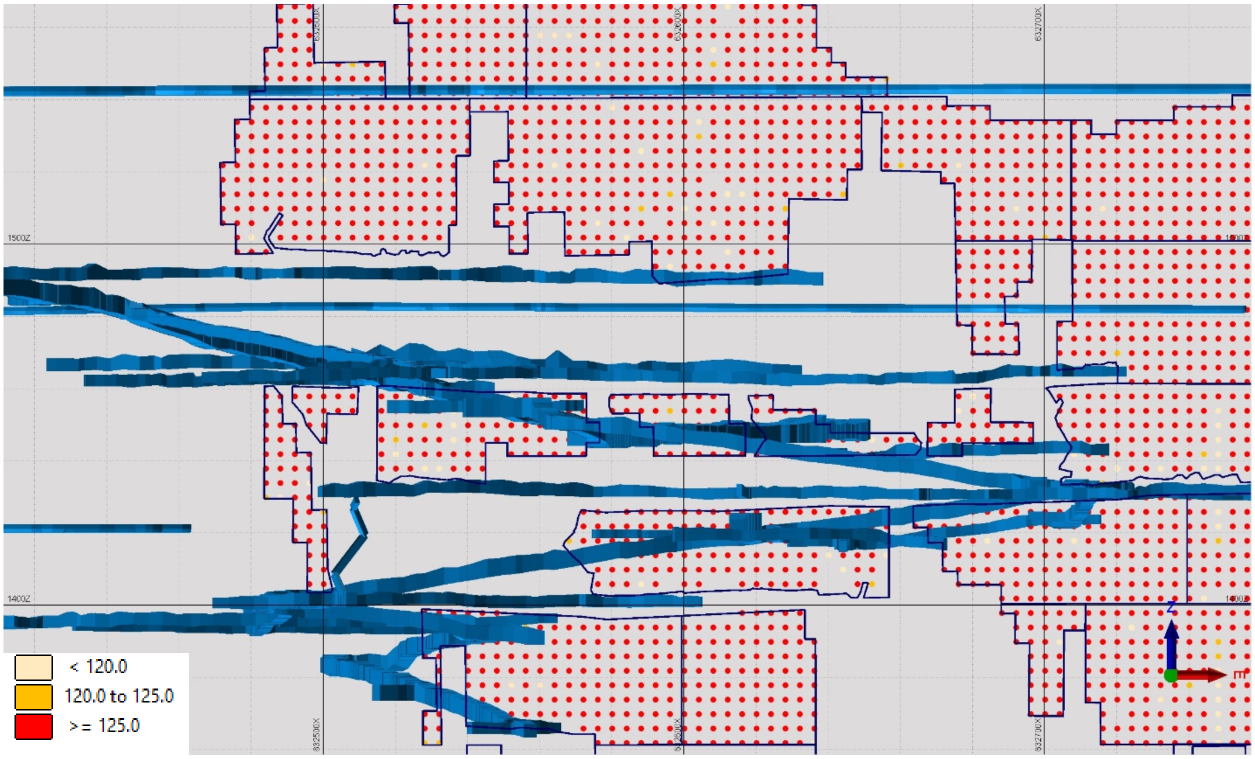


Figure 16-4: Example of detailed view of San Mateo area, stope and existing development

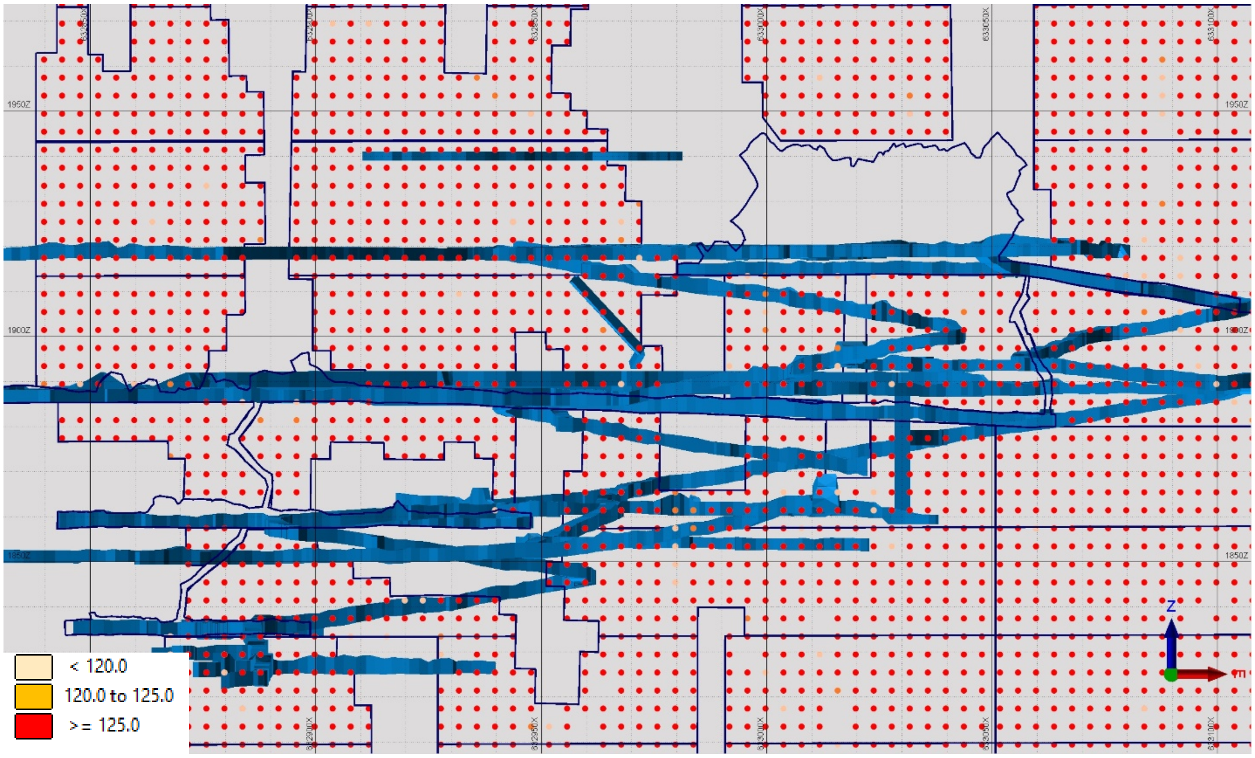


Figure 16-5: Example of detailed view of Ternarnes area, Roca Negra vein, stope and existing development

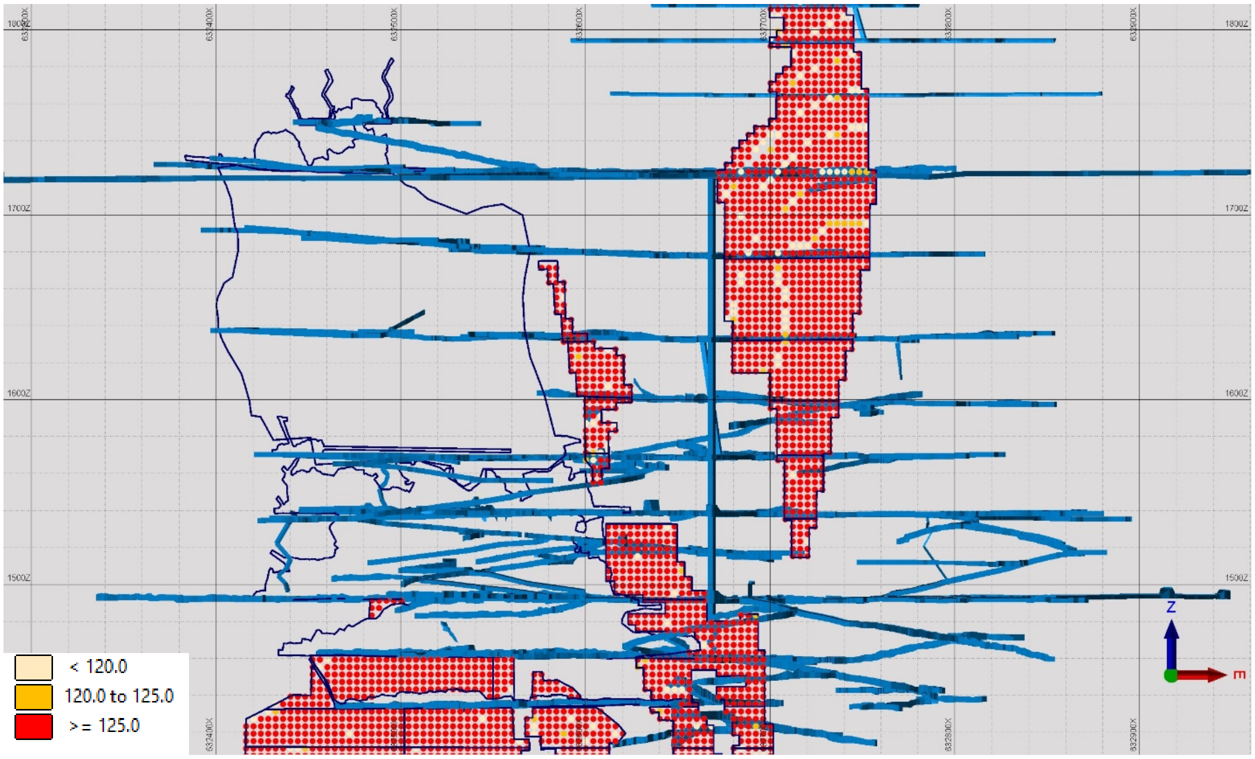


Figure 16-6: Example of detailed view of Santa Juana area, CC vein, stope and existing development

16.12.2 Main Access Ramps

The main access ramps are 4 meters high by 4 meters wide. The ramps are driven at slopes no greater than -15% (+15%). The ramps are equipped with HDPE lines carrying compressed air, drill water, and mine water drainage. The Velardeña advance rate for ramps is typically 3.9 meters per day. Single boom hydraulic jumbos are used to drill and six LCY LHD units are used to muck. An AWG 0 600-volt rated three phase conductor is installed in the ramps to provide power for the jumbos, pumps, and other equipment.

16.12.3 Cross Cuts & Footwall Development

Cross cuts and footwall development required to access each stope for mining were examined. On vein development was utilized where possible. Stope designs wherein the cost of development resulted in negative cash flow have not been included in the schedule. This mine plan includes some stopes already in production, and as such, additional development was not required for access.

16.12.4 Production Schedule

A mining schedule was developed based on the stopes described above. The scheduling parameters are listed below.

- Maximum annual production of 113,150 Tonnes;
- Maximum number of stopes running in one period of 15;
- Maximum development meters for stoping and crosscut - 90 m per month; and
- Maximum ramp development is 3.9 meters per day.

Table 16-4 shows the annualized mining schedule.

Table 16-4: Annual Mining Schedule

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Tonnes	113,150	113,150	113,151	113,150	113,150	113,150	113,150	113,150	113,150	113,150	6,448	1,137,949
NSR	275	315	276	374	351	275	297	249	268	280	179	295
Ag (gpt)	317	389	323	398	459	335	361	223	264	309	148	337
Ag oz	1,154,247	1,154,247	1,413,373	1,175,363	1,449,640	1,670,226	1,219,864	1,312,874	811,598	961,471	1,125,893	30,751
Au (gpt)	3.98	4.68	4.42	7.01	5.40	4.30	5.01	5.65	5.81	5.27	3.81	5.15
Au (oz)	14,484	17,012	16,071	25,486	19,644	15,641	18,225	20,571	21,149	19,176	789	188,250
Pb (%)	2.18	1.36	1.40	1.55	1.30	1.37	1.20	0.95	0.94	0.99	0.98	1.32
Pb (lbs)	5,422,580	3,385,212	3,490,686	3,860,058	3,236,666	3,418,863	2,983,773	2,358,765	2,330,574	2,469,588	139,361	33,096,126
Zn (%)	2.41	2.58	2.02	2.16	1.47	1.68	1.22	0.89	0.67	1.24	1.41	1.63
Zn(lbs)	6,002,134	6,419,914	5,019,968	5,370,485	3,666,065	4,174,794	3,047,480	2,226,872	1,662,967	3,095,819	200,230	40,886,729

**Note: Columns may not total due to rounding*

17 RECOVERY METHODS

There are two existing process plants, Plant #1 and Plant #2, at the Project. Plant #1 is designed to treat sulfide material for the production of Pb, Zn and pyrite concentrates and is located near the village of Velardeña, approximately eight kilometers from the mining operations. Plant #1 has an operating capacity of 350 tpd with effective capacity of 300 tpd at 95% operating time, equal to 113,150 tonnes per year (tpy) on a 350-day schedule. Plant #2 is a process plant for treating Au-Ag material, to produce Au-Ag flotation concentrate and Au-Ag doré. Plant #2 was purchased by William Resources in 1996. Operations were suspended at both plants in June 2013. In July 2014, Golden Minerals restarted mining operations to feed Plant #1, which started production on November 3, 2014. During the shutdown, Golden Minerals completed several capital projects at Plant #1 prior to restart including: overhauling the electrical system, installing new concentrate filters, and refurbishing the flotation cells. Operation of Plant #1 was discontinued in late 2015 due to a combination of low metal prices, dilution and metallurgical challenges. Plant #2 was leased to Hecla Mining Company in July 2015 and is operating. The lease is in place through December 2020.

Plant #1 is designed to process sulfide material in a conventional flow sheet of crushing, grinding, and differential flotation to produce three separate concentrates - Pb-Ag, Zn, and pyrite.

Figure 17-1 shows the processing flow sheet for Plant #1, and **Figure 17-2** shows a layout of Plant #1 and the tailings dams. **Table 17-1** lists the major equipment at Plant #1. Reagents used in Plant #1 include lime, collectors, depressants and frothers.

Run of Mine (RoM) material is received from the underground mines by truck and unloaded onto a small area near the Plant #1 crushing circuit. The RoM material is reclaimed by a front-end loader and fed to a jaw crusher for primary crushing. The primary crushed material is sized by a vibrating screen operating in closed-circuit with a secondary cone crusher. The crushed fine material is conveyed to a 350-t fine ore bin ahead of the grinding circuit. The fine material is ground in two ball mills operating in parallel. The ball mill discharge is classified by cyclones, with the cyclone oversize returned back to the ball mills and the cyclone undersize, at 80% minus 200 mesh, advances to a conditioning tank ahead of Pb flotation. After conditioning, the slurry is fed to the Pb flotation circuit comprised of rougher, scavenger, and three stages of cleaner cells. The Pb concentrate from the cleaner cells represents the final Pb concentrate, which is then thickened and filtered to a moisture content of 10-12%, by weight, for shipment. The final Pb concentrate has a low projected grade of 35-40% Pb, which is rich in Au and Ag byproducts. The Pb and Ag recoveries to the Pb concentrate are projected to be over 65% and about 70% respectively.

The tailings from the Pb flotation circuit are fed to a conditioning tank ahead of the Zn flotation circuit. The conditioned slurry is fed to the Zn flotation circuit comprised of rougher, scavenger, and three stages of cleaner cells. The Zn concentrate from the cleaner cells represents the final Zn concentrate, which is then thickened and filtered to a moisture of 10-12%, by weight, for shipment. The final Zn concentrate is projected to contain over 40% Zn. The Zn recovery to the Zn concentrate is projected to be over 70%. Both the Pb and Zn concentrates contain levels of As and Sb impurities (see **Table 13-1**).

The tailings from the Zn flotation circuit are fed to a conditioning tank ahead of the pyrite flotation circuit. The conditioned slurry advances to the pyrite flotation circuit comprised of rougher, scavenger, and two stages of cleaner cells. The pyrite concentrate from the cleaners represents the final pyrite (Fe) concentrate, which contains high Au and Ag values, would then be thickened for transport as a slurry to the BIOX® plant for oxidization, leaching and recovery of the precious metals into doré.

The tailings from pyrite flotation represents the final flotation plant tailings that are pumped to the Tailings Dam #3 located adjacent to Plant #2 (**Figure 17-2**). Tailings Dam #3 has sufficient capacity to hold 3.9 years of tailings from Plant #1. Any additional capacity in Tailings Dam #3 would need to be permitted.

Plant #1 obtains power from the national Comisión Federal de Electricidad (CFE) power grid. The nominal electrical consumption for Plant #1 is approximately 33 kWh/t of material processed. Fresh water for Plant #1 is obtained from existing water wells located near Plant #1 and Plant #2 at an average consumption rate of 184 cubic meters per day. Historically, some fresh water has been trucked from Plant #2 to Plant #1 during periods of insufficient water flow. Golden Minerals plans to construct a 4-in. diameter water line from Plant #2 to Plant #1, a distance of about five kilometers.

Table 17-1: Major Process Plant Equipment List for Plant #1

Description	Quantity	Function
Coarse Ore Bin; 120 t Capacity	1	ROM Feed Ore Bin
Jaw Crusher; 10 in. by 30 in.; 100 HP	1	Primary Crusher
Cone Crusher; Sandvik Model H3800; 200 HP	1	Secondary Crusher
Vibrating Screen; FIMSA 4 ft by 6 ft; 10 HP	1	Size Classification
Fine Ore Bin; 350 t Capacity	1	Surge Capacity
Ball Mill #1; FIMSA; 7 ft by 10 ft; 200 HP	1	Ore Grinding
Ball Mill #2: MERCY; 5 ft by 8 ft; 125 HP	1	Ore Grinding
Cyclones; D6	3	Size Classification
Lead Conditioning Tank; 6 ft by 6 ft; 10 HP	1	Conditioning
Lead Rougher Flotation Cells; FIMSA; 100 cu ft; 60 HP	4	Lead Rougher Flotation
Lead Scavenger Flotation Cells; FIMSA; 100 cu ft; 20/30 HP	4	Lead Scavenger Flotation
Lead Cleaner Flotation Cells; FIMSA; 3 stages; 24 cu ft; 7.5/10 HP	6	Lead Cleaner Flotation
Lead Concentrate Thickener; 25 ft diameter; 2 HP	1	Thicken Final Lead Concentrate
Lead Concentrate Filter; SEW; 6 ft diameter; 3 Discs; 2 HP	1	Filter Lead Concentrate
Zinc Conditioning Tank; 6 ft by 6 ft; 10 HP	1	Conditioning
Zinc Rougher Flotation Cells; Denver; 100 cu ft; 15 HP	6	Zinc Rougher Flotation
Zinc Primary Scavenger Flotation Cells; Denver; 50 cu ft; 15 HP	6	Zinc Scavenger Flotation
Zinc Secondary Scavenger Flotation Cells; Denver; 50 cu ft; 15 HP	4	Zinc Scavenger Flotation
Zinc Cleaner Flotation Cells; Denver; 3 stages; 24 cu ft; 7.5 HP	6	Zinc Cleaner Flotation
Zinc Concentrate Thickener; 25 ft diameter; 2 HP	1	Thicken Final Zinc Concentrate
Zinc Concentrate Filter; Filter Press; 0.25 HP	1	Filter Zinc Concentrate
Pyrite Conditioning Tank; 6 ft by 6 ft; 10 HP	1	Conditioning
Pyrite Rougher Flotation Cells; MINPRO; 100 cu ft; 30 HP	4	Pyrite Rougher Flotation
Pyrite Scavenger Flotation Cells; Denver; 50 cu ft; 25/30 HP	5	Pyrite Scavenger Flotation
Pyrite Cleaner Flotation Cells; Denver; 2 stages; 25 cu ft; 7.5 HP	8	Pyrite Cleaner Flotation
Pyrite Concentrate Thickener; 25 ft diameter; 2 HP	1	Thicken Final Pyrite Concentrate
Pyrite Concentrate Filter; 0.25 HP	1	Filter Pyrite Concentrate

Table 17-2: Process Materials for Plant #1

Process Materials	Consumption Rate (kg/t processed)
Grinding Balls - 2.5 in. diameter	0.83
Grinding Balls - 2 in. diameter	0.72
Grinding Balls - 1.5 in. diameter	0.17
Lime	1.16
Sodium Cyanide	0.07
Sulfate	0.88
Xanthate 350	0.8505
Aeropromoter 211	0.02
Aeropromoter 3416	0.0675
Aerofloat 31	0.054
Frother 1065	0.0945
Aerofloat 70	0.01
P404	0.03
P242	0.04
Copper Sulfate	0.92

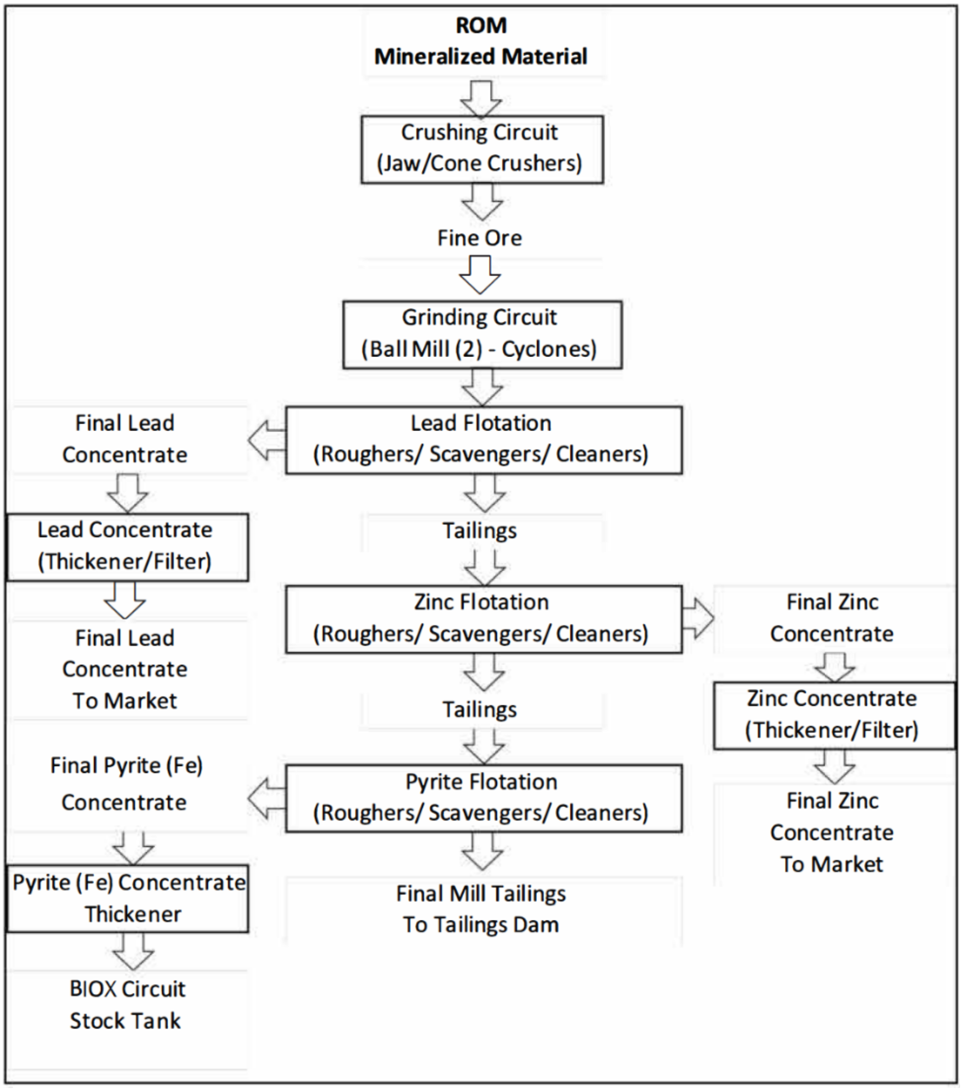


Figure 17-1: Process Plant Flow Sheet for Plant #1

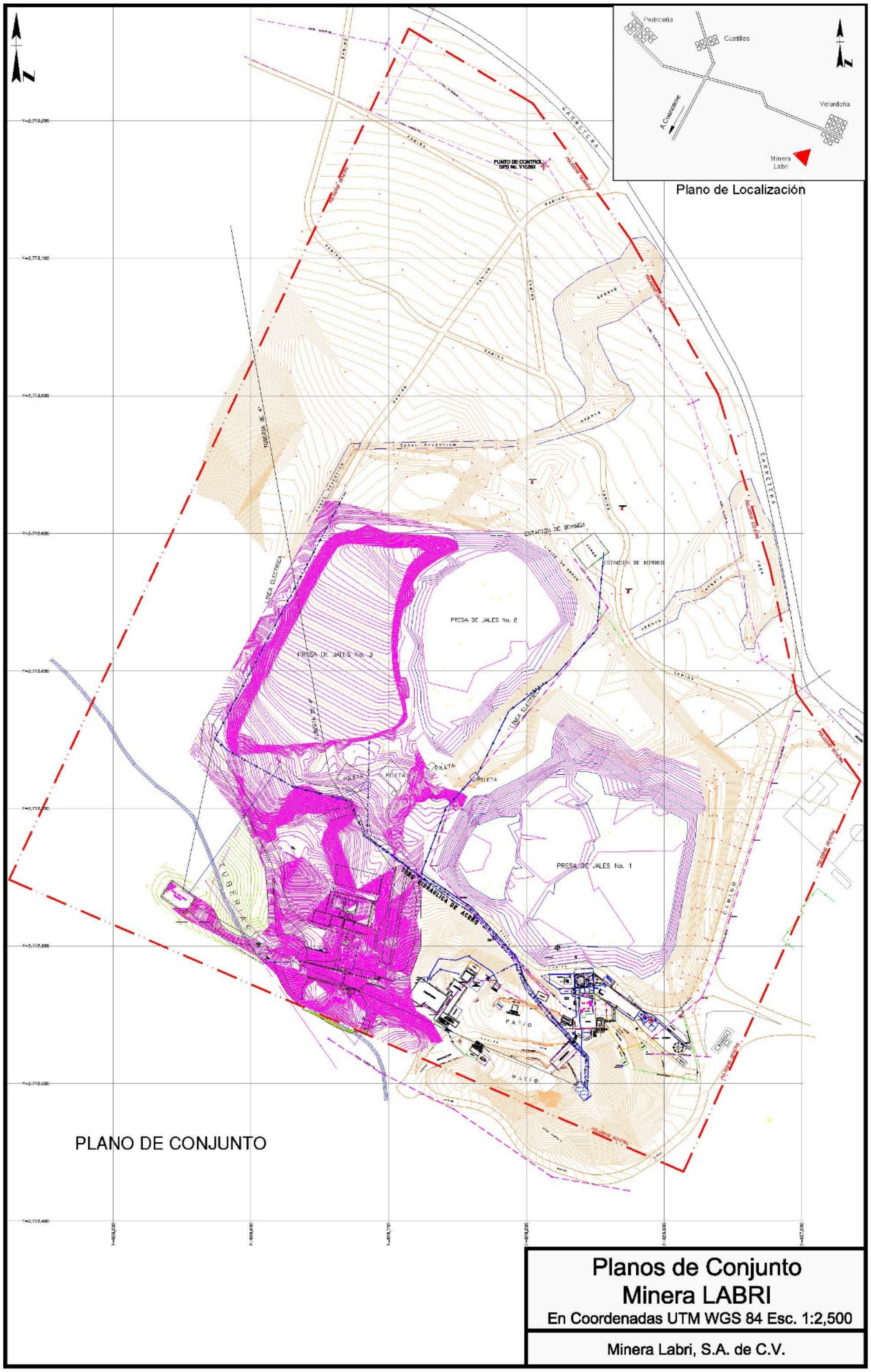


Figure 17-2: Site Layout for Process Plant #1

Plant #2 is capable of separating sulfide material from mixed from mixed material in two conventional flowsheets for the production of Au-Ag doré for oxide material and a bulk Au-Ag rich sulfide flotation concentrate for sulfide material. Plant #2 has an operating capacity of up to 550 tpd depending on material hardness of grind size requirements. **Figure 17-3** shows the processing flow sheet for Plant #2, and **Figure 17-4** shows a layout for Plant #2 and the tailings dams. **Table 17-3** lists the major equipment at Plant #2. Plant #2 is located approximately 3.5 kilometers from the mining operations. RoM material is received from the underground mines by truck and unloaded in an area near the Plant #2 crushing circuit. The RoM material is reclaimed by a front-end loader and fed onto an apron feeder to a jaw crusher for primary crushing. The primary crushed material is sized by a vibrating screen operating in closed-circuit with a secondary standard cone crusher. The crushed fine material is conveyed to a 500-t fine ore bin ahead of grinding. The fine material is ground in a ball mill operating in closed circuit with cyclones for size classification. The cyclone undersize is returned back to the ball mill and the cyclone oversize to a thickener for oxide material or a conditioning tank for sulfide material ahead of flotation. For sulfide material, the cyclone overflow is fed to the sulfide flotation circuit comprised of roughers, scavengers and cleaner cells (two stages). The sulfide concentrate from the cleaners represents the final sulfide concentrate, which is rich in Au and Ag. The final sulfide concentrate is thickened and filtered for shipment. The tailings from the sulfide flotation circuit represents the final Plant #2 sulfide tailings that are pumped to the nearby tailing. Reagents used in Plant #2 are cyanide, Zn dust, diatomaceous earth, flocculants, lime, collectors and frothers.

Table 17-3: Major Process Plant Equipment List for Plant #2

Description	Number	Function
Coarse Ore Bin; 7 ft by 11 ft by 14 ft; 50 t Capacity	1	ROM Feed Ore Bin
Coarse Ore Apron Feeder; 4 ft by 17 ft; 3 HP	1	Feed Jaw Crusher
Jaw Crusher; 24 in. by 36 in.; Allis-Chalmers; 100 HP	1	Primary Crusher
Cone Crusher; 4 ft diameter Standard; Symons; 100 HP	1	Secondary Crusher
Vibrating Screen; Double-Deck; TYLER, 6 ft by 10 ft; 20 HP	1	Size Classification
Fine Ore Bin; 8 m by 9 m; 500 t Capacity	1	Surge Capacity
Ball Mill; Allis-Chalmers; 10.5 ft by 13 ft; 800 HP	1	Ore Grinding
Cyclones; Krebs D6	10	Size Classification
Primary Thickener; 16 m diameter by 3 m high; 3 HP	1	Thicken Cyclone Overflow
Leach Tanks; Agitated; 8 m by 8.5 m; 25 HP	8	Cyanide leach Au and Ag
CCD Thickeners; 60 ft diameter; 5 HP	4	Solid-liquid separation; PLS
PLS Tank; 8 m diameter by 4 m high	1	PLS Surge Tank
Clarifiers; 52 sq. m; Diatomaceous Earth; 1.5 HP	2	Clarify PLS
Clarified PLS Tank; 5 m diameter by 6 m high	1	Clarified PLS Surge Tank
Zinc Filter Presses; 1.84 m diameter by 1.84 m high; 4.89 cu m	2	Filter Zinc Precipitate
Primary Flotation Cells; WEMCO; 75 cu m; 15 HP	3	Au-Ag rougher flotation
Cleaner Flotation Cells; First Stage; DENVER; 25 cu ft; 5 HP	2	First stage cleaners
Cleaner Flotation Cells; Second Stage; DENVER; 45 cu ft; 10 HP	4	Second stage cleaners
Conditioner; 1.7 m diameter by 2 m high; 25 HP	1	Conditioning
Concentrate Thickener; 6.84 m diameter by 2.45 m high; 3 HP	1	Thicken Final Concentrate
Concentrate Filter; Vacuum; Komline-Sanderson	1	Concentrate Filter

Description	Number	Function
Smelting Furnace; IDUCTOHERM, 650 kg charge; 150 KW	1	Precipitate Smelting
Filter Presses; DURCO/PERRIN/HYSTAR	3	Filter

Table 17-4: Process Reagents for the Plant #2 Leach Circuit

Process Materials	Consumption Rate (kg/t processed)
Lime	4.4
Sodium Cyanide	7.5
Flocculant	0.05

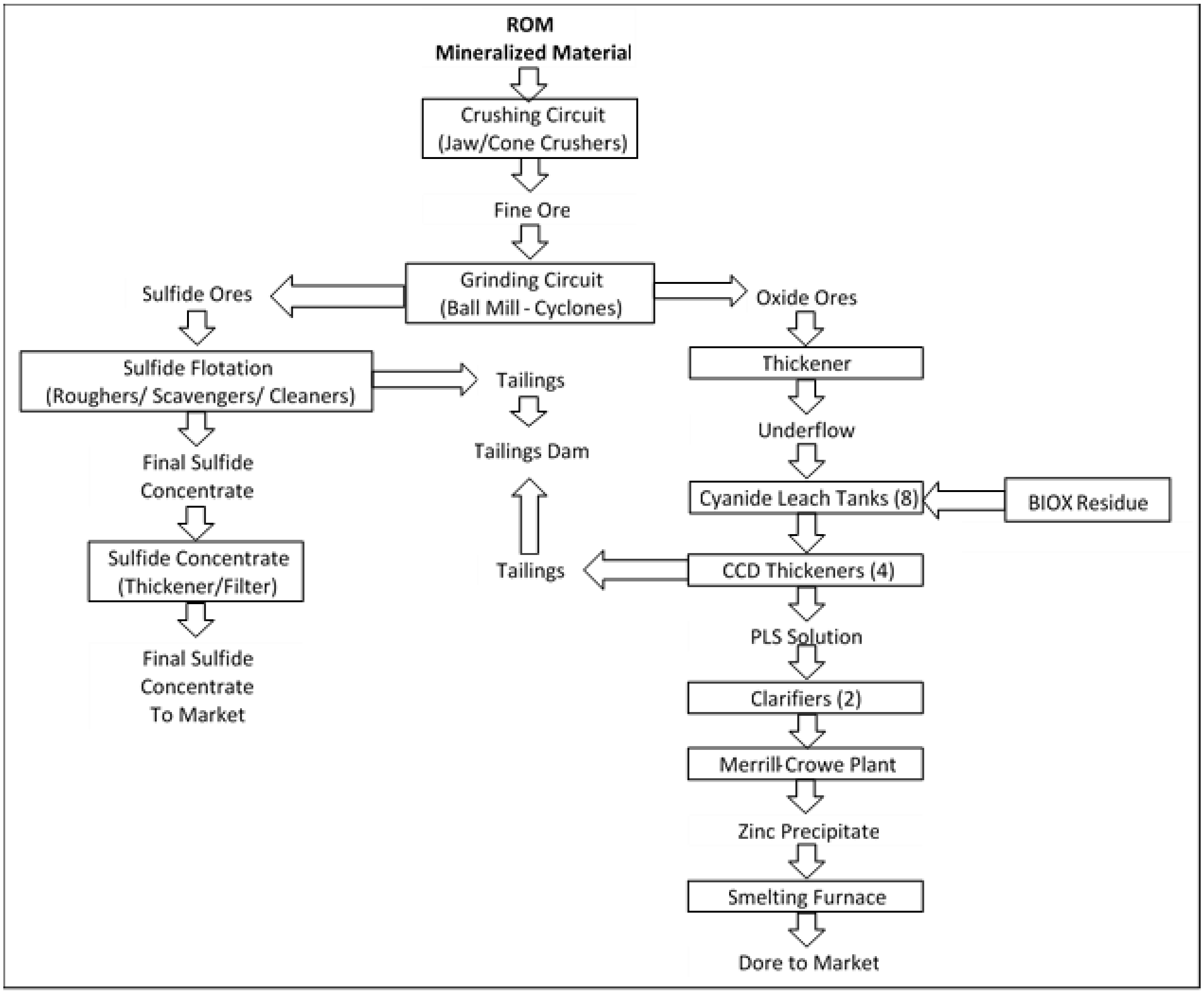


Figure 17-3: Process Plant Flow Sheet for Plant #2

When processing oxide material, the cyclone overflow is sent to a thickener with the thickener underflow fed to the cyanide leach tanks. Discharge from the cyanide leach tanks advances to a four-stage CCD thickener circuit. The solution from the first stage is Au-Ag rich solution, termed pregnant liquor solution (PLS). The PLS is clarified of any fine particles and pumped to a Merrill-Crowe circuit where Zn dust is added to precipitate the contained Au and Ag from the PLS. The resulting Zn precipitate is smelted in an induction furnace for the production of doré that is shipped to refineries. The underflow from the last CCD thickener represents the final tailings from the oxide circuit that are pumped to the nearby tailings dam.

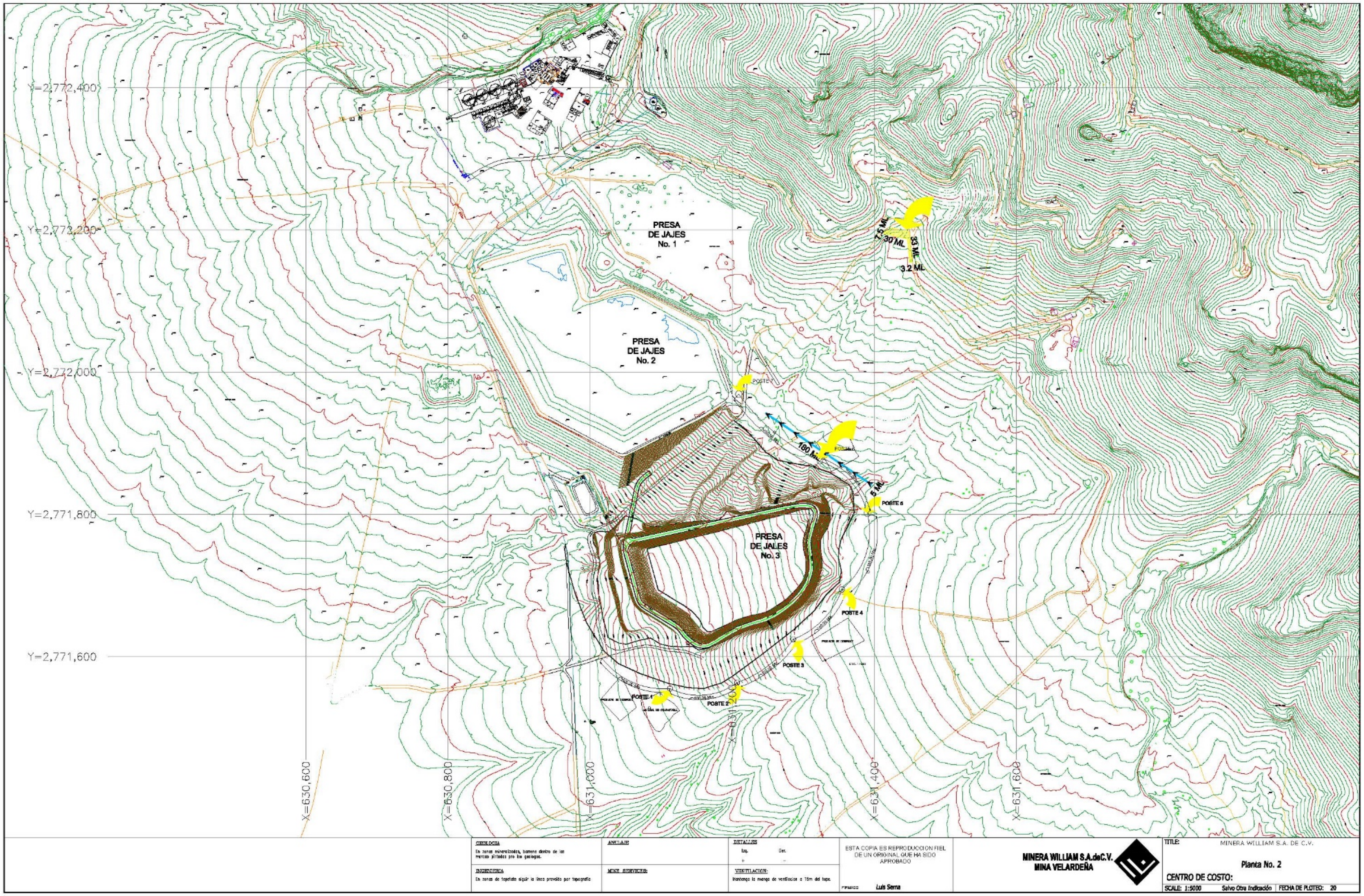


Figure 17-4: Site Layout for Process Plant #2

The flowsheet for the BIOX® circuit that would be located adjacent to Plant #2 is shown in Fig 17-5. Major equipment for the BIOX® plant is listed in Table 17-5. Thickened Fe concentrate from Plant #1 would be trucked to Plant #2, transferred to a stock tank and subsequently to the BIOX® reactors along with nutrients for the microbes from the nutrient make-up tank.

The BIOX® circuit would be comprised of six agitated and cooled reactors in two stages providing 6.6 days of residence time for the project average 2.7 tph of solids. Stage 1 would include three primary reactors in parallel followed by Stage 2 with three secondary reactors in series. Cooling water would be circulated through a cooling tower to maintain reactor temperatures at 40°C.

Oxidized concentrate would advance from the BIOX® reactors to a three-stage CCD thickener circuit to wash the acidic liquor from the solids. Underflow from thickener #3 would advance to a conditioning tank for pH correction with lime before being pumped to the existing Plant #2 agitated leach circuit available for oxidized concentrates, along with the Merrill-Crowe circuit and refinery. The existing leach tanks are sized for larger tonnage whole ore leaching.

There may be the opportunity to blend the BIOX® residue with whole oxide ore for leaching.

CCD thickener circuit overflow would flow to the neutralization circuit comprised of six agitated tanks in series. Limestone and lime would be added to increase the pH to 7 before advancing the neutralized slurry to a thickener to recover water before discharge of the underflow, containing precipitates including ferric sulfate, to the tailings dam.

Table 17-5: Major Process Plant Equipment List for the BIOX® Plant

Description	Estimated Size	Number	Function
Stock Tank; agitated	6 m ∅ x 5.3 m	1	Hold Fe concentrate slurry
BIOX® Reactors; SST, agitated	7.7 m ∅ x 7.7 m	6	Oxidize sulfides
Blowers		4	Provide oxygen to reactors
Cooling Towers		4	Cool reactors
CCD Thickeners; SST	6.9 m ∅	3	Separate BIOX® residue from liquor
Leach Conditioning Tank: agitated	2.9 m ∅ x 1.9 m	1	Adjust BIOX® residue pH
Neutralization Tanks; SST, agitated	3.2 m ∅ x 4.3 m	6	Raise BIOX® liquor pH
Water Recovery Thickener	6.9 m ∅	1	Recycle process water

Table 17-6: Process Materials for the BIOX® Circuit

Process Materials	Consumption Rate (kg/t processed)
Nutrients	TBD
Lime	4.4
Sulfuric Acid	TBD
Flocculant	0.05

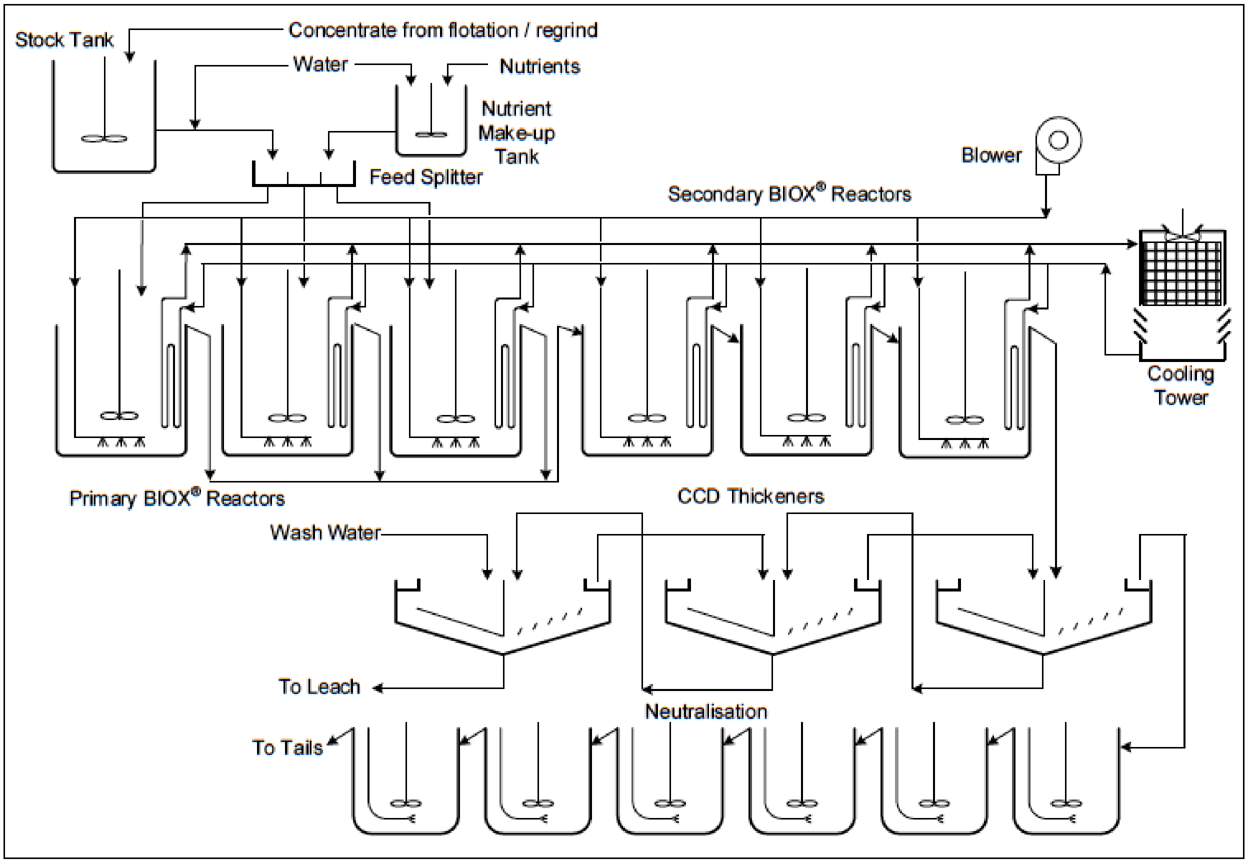


Figure 17-5: BIOX® Circuit Flow Sheet

18 PROJECT INFRASTRUCTURE

Infrastructure facilities at the Project (Velardeña Operations) include the following:

- Access roads;
- Power line;
- Ancillary buildings; and
- Water wells.

There are no man-camp facilities at the Project site.

18.1 Access Roads

The Project is located in the Mexican state of Durango, approximately 65 kilometers southwest of the city of Torreón and 150 kilometers northeast of the city of Durango. A major 4-lane highway, Highway 40, connects these cities. The Velardeña Plant #1 is located adjacent to the village of Velardeña which is approximately 500 meters west of Highway 40D. The Velardeña mines are located about eight kilometers from Plant #1 via a gravel road. Plant #2 is located approximately 3.5 kilometers from the Velardeña mine, also via gravel roads.

18.2 Power Lines

The mines, Plants #1 and #2 are connected to the national grid of CFE via a substation located near the entrances of Golden Minerals' Plant #1 and Peñoles' Velardeña mine. The total installed capacity of Plant #1 in two transformers of 500 kilovolt amps (KVA) is 34,500 with a step down to 480 volts (V).

18.3 Ancillary Buildings

Ancillary buildings for the Project include administration buildings, warehouses, maintenance shops, offices, a metallurgical laboratory and an analytical laboratory for the preparation and assaying of mine and plant samples. Security gates are located at the entrances of the mines and both Plant #1 and #2.

18.4 Water Wells

There are six existing water wells (three at Plant #1 and three at Plant #2) for extracting water from local aquifers. These wells are authorized, regulated and permitted by CONAGUA, the Mexican *Comisión Nacional del Agua*. **Table 18-1** and **Table 18-2** summarize the data for these wells.

Table 18-1: Data for Water Production Wells - Plant #1

Pump	Well Depth (m)	Well Pump Submersible	Authorized Volume	
			m^3/d	m^3/yr
Discordia	25	2-inch	67.53	24,655
Noria	25	2-inch	67.53	24,655
Rancho	200	2-inch	49.165	17,946

Source: Golden Minerals, December 2014 Site Visit

Table 18-2: Data for Water Production Wells - Plant #2

Pump No.	Well Depth (m)	Well Pump Submersible	Flow Rate (L/s)	Authorized Volume	
				m^3/d	m^3/yr
1	220	4-inch	8	460.65	168,192
2	400	4-inch	5	201.5	73,584
3	431	6-inch	11	431.8	157,680

Source: Golden Minerals, December 2014 Site Visit

In 2015, Golden Minerals planned the construction of a 4-inch diameter water line from Plant #2 to Plant #1, five kilometers, at a capital cost US$90,000, which was partially built but since removed until needed again.

19 MARKET STUDIES AND CONTRACTS

Mill operations, which last operated between November 2014 and November 2015, produce Pb, Zn and pyrite concentrates. Markets for the Pb and Zn concentrates include metal brokers and direct sales to Pb and Zn smelters. Pyrite concentrates produced will be used as feed for the bio-oxidation circuit and then processed through an agitation leach circuit utilizing Merrill-Crowe to produce a gold and silver rich doré. The concentrates and doré produced are typical within the Mexican mining industry and the concentrate and doré markets within Mexico and worldwide are liquid. For purposes of the PEA, it is assumed that Golden Minerals will be successful in securing buyers for its concentrates and doré.

19.1 Doré

The Velardeña Operations has a small furnace capable of smelting the precipitates produced in the Merrill-Crowe circuit into a silver/gold doré bar. The bars produced are expected to contain approximately 85-90% silver and 4-6% gold. For purposes of the PEA, it is assumed the Velardeña Operations will be paid for 97% of the contained gold and silver in the doré, with a treatment charge of $5 per kilogram of doré and a refining charge of $6 per ounce of gold and $0.60 per ounce of silver. Marketing studies with potential buyers of doré have not been completed and therefore have not been reviewed by the author of this section.

19.2 Concentrates

The sulfide plant at the Velardeña Operations contains a typical flotation circuit that produces lead, zinc and pyrite concentrate products for sale to customers. Lead and zinc concentrates each comprise approximately 10% of total concentrate production from the sulfide plant. Pyrite concentrates comprise approximately 80% of total concentrate production from the sulfide plant. The pyrite concentrates produced will be used as feed for the bio-oxidation circuit and then processed through an agitation leach circuit utilizing Merrill-Crowe to produce a gold and silver rich doré. The lead and zinc concentrates will be sold to various customers under annual contracts which are generally re-negotiated each calendar year. The concentrate products are generally shipped in covered trucks. The company generally incurs the cost of freight to the customer. Marketing studies with potential buyers for lead and zinc concentrates have not been completed and therefore have not been reviewed by the author of this section.

19.2.1 Lead Concentrates

The lead concentrates have typical assays as follows: 35-40% lead, 8,000-10,000 gpt silver, and 40-50 gpt gold. After metal deductions, the company is generally paid for 90-95% of the contained lead, silver and gold. Concentrate treatment charges are negotiated annually and generally reflect market terms for the industry for similar products. The following treatment charges have been assumed for purposes of the PEA:

- Lead concentrate treatment charge: $200 per dry metric tonne of lead concentrate
- Au refining charge of $15.00 per payable ounce
- Ag refining charge of $0.95 per payable ounce

- Penalties:
  - Arsenic: For As contents less than 0.3% As, there is no penalty. If the As content is greater than 0.3% and less than 0.7%, there is a penalty of US$2.00/t of Pb concentrate for every 0.1% As greater than 0.3%. If the As content is greater than 0.7% but less than 1.0%, there is a penalty of US$3.00/t of Pb concentrate for every 0.1% As greater than 0.7%. For As above 1.0% there is a penalty of US$5.00/t of Pb concentrate for every 0.1% As greater than 1.0%.
  - Antimony: For Sb contents less than 0.3% Sb, there is no penalty. If the Sb content is greater than 0.3% there is a penalty of US$1.50/t of Pb concentrate for every 0.1% Sb greater than 0.3%.
  - Bismuth: For Bi contents less than 500 ppm Bi, there is no penalty. If the Bi content is greater than 500 ppm there is a penalty of US$2.00/t of Pb concentrate for every 100 ppm Bi greater than 500 ppm.

19.2.2 Zinc Concentrates

The zinc concentrates have typical assays as follows: 40-45% zinc, 90-100 gpt silver, and 5-6 gpt gold. After metal deductions, the Company is generally paid for approximately 85% of contained zinc and 60-70% of silver with lesser amounts payable for the contained gold. Concentrate treatment charges are negotiated annually and generally reflect market terms for the industry for similar products. The following treatment charges have been assumed for purposes of the PEA:

- Zinc concentrate treatment charge: $300 per dry metric tonne of lead concentrate
- Penalties:
  - Arsenic and antimony: For As+Sb contents less than 0.5% As+Sb, there is no penalty. If the As+Sb content is greater than 0.5% and less than 1.0%, there is a penalty of US$2.50/t of Zn concentrate for every 0.1% As+Sb greater than 0.5%. If the As+Sb content is greater than 1.0% As+Sb there is a penalty of US$6.00/t of Zn concentrate for every 0.1% As+Sb greater than 1.0%.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project consists of the existing Velardeña and Chicago mines, the Labri Mill sulfide ore processing facility (Plant #1), an oxide processing facility (Plant #2), and related ancillary facilities including tailings impoundments, access roads, storage buildings and water pumping stations. The Project is located within the Municipality of Cuencamé; State of Durango, México.

Environmental studies were prepared by Consultores en Ecología con Visión Integral S.A de C.V.

20.1 Introduction

The Project is a group of properties held by Golden Minerals' wholly owned Mexican subsidiary, Minera William, within the Velardeña mining district of México. The Velardeña mining district is located within the municipality of Cuencamé, in the northeast quadrant of the State of Durango, México. The property is situated approximately 65 km southwest of the city of Torreón in the State of Coahuila and 150 km northeast of the city of Durango, capital of the State of Durango.

Golden Minerals holds the Velardeña Operations through its 100% owned Mexican subsidiary Minera William S.A. de C.V. (Minera William). At present, the properties comprise a total of 28 mineral concessions (20 for the Velardeña Property and 8 for the Chicago Property). The mineral concessions in each of the properties vary in size and the concessions comprising each mineral property are contiguous.

The Minera William operating properties include the Velardeña Mines, the Chicago property, the Plant #2 oxide mill and the Labri mill (held through Minera Labri S.A. de C.V., also a wholly owned Mexican subsidiary of Golden Minerals)and related infrastructure. The Velardeña Mining Complex includes the Santa Juana mine, which has been the focus of ECU's mining efforts since 1995, as well as the Terneras, San Juanes, and San Mateo mines. The Chicago Mining Project is located approximately two km south of the Velardeña Mining Complex. Associated with the Velardeña Mining Complex and Chicago Mine are two primary minerals processing facilities. Plant #1, which is a 300 tonne per day sulfide plant, is located on the west side of the town of Velardeña. Plant #2, located just west of the Chicago Mine, is a 550 t/d oxide processing plant.

20.2 Current Property Status & Environmental Liabilities

Minera William's Velardeña Operations consist of existing underground mining and surface mineral processing facilities located on controlled property. Numerous historical mining operations have occurred at the Velardeña Operations with records showing the existence of mining in the district dating back to the early 1800s. Significant mining activity began in 1902 and included mining of both the Terneras and Santa Juana veins and the construction and operation of a 2,500 t/d smelter. Other small-scale development was conducted by local miners throughout most of the 1900's. Many of these historical operations have been incorporated into the current mining activities while others remain inactive and separate from the current activities. Minera William anticipates continued mining operations at this location.

In early 2012, Golden Minerals applied for, and was accepted into, the Mexican National Environmental Auditing Program ("NEAP"). Under NEAP, Golden Minerals participated in an audit program to verify compliance with existing regulations and identify non-regulated potential issues that could result in environmental contingencies. Golden Minerals holds various permits required for conducting their current operations at the Velardeña operations, and their participation in NEAP allows them to continue their current operations during the remediation of any potential non-compliance matters. This program was in play for Plant #1 until the spring of 2014, at which time Golden Minerals had achieved 85% compliance, but the plant was placed on care and maintenance.

Golden Minerals is required to update their environmental licenses and environmental impact assessments for any expansion of or modification to any of the existing two plants. The construction of new infrastructure beyond the current plant facilities may require additional permitting, possibly including environmental impact assessments and land use permits. Golden Minerals does not expect to have difficulty obtaining additional permits or environmental impact assessments, if required.

Tetra Tech is unaware of any outstanding environmental liabilities attached to the Velardeña properties and is unable to comment on any remediation which may have been undertaken by previous companies.

20.3 Mexican Permitting Framework

Environmental permitting of the mining industry in Mexico is mainly administered by the federal government body SEMARNAT, the federal regulatory agency that establishes the minimum standards for environmental compliance. Guidance for the federal environmental requirements is largely held within the General Law of Ecological Equilibrium and Environmental Protection (Ley General Del Equilibrio Ecológico y la Protección al Ambiente, or LGEEPA). Article 28 of the LGEEPA specifies that SEMARNAT must issue prior approval to parties intending to develop a mine and mineral processing plant. An environmental impact assessment (by Mexican regulations called a Manifestación de Impacto Ambiental, or “MIA”) must be filed with SEMARNAT for its evaluation and, if applicable, further approval by SEMARNAT through the issuance of an Environmental Impact Authorization; the document specifies approval conditions where works or activities have the potential to cause ecological imbalance or have adverse effects on the environment. Further requirements for compliance with Mexican environmental laws and regulations are supported by Article 27 Section IV of the Ley Minera and Articles 23 and 57 of the Reglamento de la Ley Minera. Article 5 Section X of the LGEEPA authorizes SEMARNAT to provide the approvals for the works specified in Article 28. The LGEEPA also contains articles for soil protection, water quality, flora and fauna, noise emissions, air quality, and hazardous waste management.

The Ley de Aguas Nacionales provides authority to CONAGUA, an agency within SEMARNAT, to issue water extraction concessions, and specifies certain requirements to be met by applicants.

Another important piece of environmental legislation is the LGDFS. Article 117 of the LGDFS indicates that authorizations must be granted by SEMARNAT for land use changes to industrial purposes. An application for change in forestry land use CUSF, must be accompanied by a technical study that supports the Technical Justification Study (Estudio Técnico-Justificativo ETJ). In cases requiring a CUSF, a MIA for the change of forestry land use is also required.

Mining projects also must include a Risk Analysis (AR) and an Accident Prevention Plan (PPA) from SEMARNAT.

The Ley General para la Prevención y Gestión Integral de los Residuos (LGPGIR) also regulates the generation and handling of hazardous waste coming from the mining industry. The LGPGIR also regulates the generation and handling of hazardous waste coming from the mining industry. Guidance for the environmental legislation is provided in a series of Norms, (NOM). These regulations provide specific procedures, limits and guidelines and carry the force of law.

20.4 Project Permitting Requirements & Status

There were several environmental permits required to put the project into operation. Most of the mining regulations are at a federal level through SEMARNAT, but there are also a number regulated and approved at state and local level. There are three SEMARNAT permits that were required prior to construction; MIA, CUS and AR, which are described below. A construction permit was also required from the local municipality and an archaeological release letter from the National Institute of Anthropology and History (INAH); and an explosives permit was required from the Ministry of Defense (SEDENA) before construction as well. The key permits and the stage at which they are required are summarized in **Table 20-1**.

- **Environmental Impact Manifest** – Regulations within Mexico require that an MIA be prepared by a third-party contractor for submittal to SEMARNAT. The MIA must include a detailed analysis of climate, air quality, water, soil, vegetation, wildlife, cultural resources and socio-economic impacts.
- **Analysis of Risk** – A second required permit is an AR. A study is developed to obtain this permit. This study identifies potential environmental releases of hazardous substances and evaluates the risks in order to establish methods to prevent, respond to, and control environmental emergencies. In the Project, since no hazardous substances will be used or processed, SEMARNAT will not need an AR to be done for current project conditions.
- **Land Use Change (CUS)** – The third permit is CUS. In Mexico, all land has a designated use. The various areas comprising the project site are designated as forest land, cattle grazing, and agriculture. The CUS is a formal instrument for changing the designation to allow mining on these areas. The CUS study is based on the Forestry Law and its regulations. It requires that an evaluation be made of the existing conditions of the land, including a plant and wildlife study, an evaluation of the current and proposed use of the land and impacts on natural resources and an evaluation of the reclamation and revegetation plans. The establishment of agreements with all affected surface landowners is also required.

A construction permit is required from the local municipality and an archaeological release letter is required from the INAH. An explosives permit is required from the SEDENA before construction begins. Water discharge and usage must be granted by CONAGUA. A project-specific unique environmental license (Licencia Única Ambiental, LAU), which states the operational conditions to be met, is issued by SEMARNAT when the agency has approved the project operations. The key permits and the stages at which they are required are summarized in **Table 20-1**.

Table 20-1: Permitting Requirements

KEY ENVIRONMENTAL PERMITS		
Permit	Mining Stage	Agency
Environmental Impact Assessment - MIA	Construction/Operation/Post-Operation	SEMARNAT
Land Use Change - CUS	Construction/Operation	SEMARNAT
Risk Analysis - RA	Construction/Operation	SEMARNAT
Construction Permit	Construction	Local Municipality
Explosive & Storage Permits	Construction/Operation	SEDENA
Archaeological Release	Construction	INAH
Water Use Concession	Construction/Operation	CONAGUA
Water Discharge Permit	Operation	CONAGUA
Unique Environmental License	Operation	SEMARNAT
Accident Prevention Plan	Operation	SEMARNAT

The project has acquired permits for mineral exploration and construction of the Project works, including water concessions, ramp, hazardous waste generator and the archaeological release. The permitted activities and the corresponding permit numbers are listed in **Table 20-2**.

Golden Minerals personnel report that the Project holds and has retained the necessary permits to operate the mines and plants at Velardeña, and further there are no unresolved issues with the environmental regulatory agencies. They do not anticipate any limitations on the operations due to future inspections or evaluations by the environmental authorities.

The following has been sourced from CAM 2012.

In order to begin an exploration program on a concession upon which no substantial mining has been conducted, Golden Minerals would have to comply with the Mexican Official Norm: NOM-120-SEMARNAT-1997, which provides, among other things, that mining exploration activities to be carried out must be conducted in accordance with the environmental standards set forth in NOM-120-SEMARNAT-1997; otherwise, concession holders are required to file a preventive report or an environmental impact study prior to the commencement of the exploration program. However, an environmental impact study may not be necessary if the concessionaire files an application with the environmental authorities confirming the concessionaire's commitment to observe and comply with NOM-120-SEMARNAT-1997. If the exploration program requires the removal of vegetation, a permit to change the land use will also be required.

Golden Minerals reports that it has obtained and maintained other permits and agreements that include an explosive use permit (from the Secretaría de la Defensa Nacional) renewable each year, surface land use agreement (“ocupación temporal” agreement with the Ejido Velardeña), water use permit (Comisión Nacional del Agua) and other environmental permits from SEMARNAT.

Table 20-2: Permitting Status

No.	Activity	MIA/Permit Number/Date	Comments
	Plant #1 Permitting: Plant #1 and Tailings		
	MIA Expansion of the Tailings Dam at Plant #1 - Environmental Impact Study for tailings for the Plant 1 Expansion	SG/130.2.1.1/001013/11 Dated: June 16, 2011	
	With the above MIA there is also a Technical Justification Study (Estudio Técnico-Justificativo - ETJ)	ETJ SG/130.2.2/001189/11, Dated: June 28, 2011	
1	Site and Construction Preparation		Disturbance = 2.2257 hectares
2	Operation: No extraction, ramps or mining works	Valid from June 2016 through 2018	
3	Abandonment - 45 days prior to renewal.	Valid from June 2016 through 2018	
4	Single Environmental License (LAU)	SG/130.2.1/001312 July 4, 2008	
5	Voluntary National Environmental Auditing Program (NEAP) on hold for Plant 1		Enrolled until spring of 2014 for Plant 1 (achieved 85% compliance) but placed program on hold when plant shut down. Eligible to re-enroll and this effort is in progress.
	Mine Site and Plant #2		
	Environmental Impact Study for Production and Operation of the Velardeña Mines (MIA Explotación y Operación Minas de Velardeña)	SG/130.2.1.1/002384/13 August 29, 2013	Complements previous MIA filed in 1996 which cover much of the previous construction at the mine site and at Plant 2.
	BLM Minera Mexicana, S.A de C.V. for operation activities related to the construction, operation, maintenance and abandonment in an area of 75-00-00 ha. The activity refers to re-opening and operation of the following: San Juan Evangelist, Buen Retiro and Viborillas. The construction and operation of the processing plant and tailings dam were added with Permit No. DOO. DGOEIA04852, October 2, 1996. On May 28, 2001, through Memorandum No. 10.1.MA/243, transfer of environmental impact of BLM MEXICAN MINING LTD A HECLA MINING LTD was authorized.		
	Plant #2 (Minera William Permits): Applicable only if ore screening is performed. Permits for tailings facilities and operation of the plant are listed below:		
a.	Unique Environmental License (LAU) SEMARNAT	10/LU-0310/10/09; SG/130.2.1/002086 dated November 3, 2009; LAU-10 / 035-2009	
b.	Environmental Impact Study MIA-P Mining Project for Minera William Phase III Tailings Project" SEMARNAT	10DU11M0806 August 29, 2013	Filed with Center for Integrated Services (CIS) of the Federal Delegation Durango, under SEMARNAT.
c.	Conditional authorization for the Tailings Dam Construction Minera William. SEMARNAT	SG/130.2.1.1/002292/11 December 7, 2011	

No.	Activity	MIA/Permit Number/Date	Comments
d.	CUS - Tailings Dam Construction III	SG /130.2.2/001291/12 May 21, 2012	
f.	MIA Approved for Beneficiation Plant Expansion of the Minera William II and IV Tailings Dams SEMARNAT	April 23, 2012	

Table provided by Minera William

20.4.1 Environmental Monitoring Program

As part of the MIA for the Project and in compliance with environmental regulations, Minera William has established an Environmental Monitoring Program that identifies potential impacts during each of the phases of the project along with actions to prevent, mitigate and compensate the effects. The program requires internal control and periodic reporting to verify compliance with the program. Golden Minerals has retained an independent consultant to evaluate compliance with current environmental reporting and requirements.

20.5 Environmental Baseline Data

The following has been sourced from the Environmental Impact State for the Velardeña Mine Project Exploration and Mining Operation, MIA (April 2013).

A variety of studies have been completed in order to characterize the natural environment of the area. The Project has completed a number of MIAs, the most recent report completed in April 2013.

20.5.1 Flora & Fauna

20.5.1.1 Flora

According to the classification of INEGI- INE (1996), the type of vegetation where the project is located corresponds to a vegetation type known as Desert Shrubland rosetophilous (rosette-forming vegetation) and sub montane scrub.

The project area is in a transition zone between two types of ecosystems; the desert scrub rosetophilous and the submontane scrub. However, there is no demarcation that determines the separation between the ecosystems, so it is possible to find species from the two ecosystems. In the lower parts of the Project Area, the type of vegetation presented in the middle and lower mountains was the Xeric Scrub, which includes Desert Shrubland rosetophilous and sub montane scrub. This vegetation is sparse in places, while less extensive areas may have higher densities with the presence of shrubs and trees (Prosopis laevigata (Smooth mesquite), Acacia constricta (Whitethorn acacia), Dasylirion palmeri, Yucca carnerosana (Giant Spanish Dagger), Fouqueira splendens (ocotillo) and Flourensia cernua (tarbush).

Based on the results obtained in the field and bibliographic records, on-site and surrounding areas and six sampling sites, a total of 24 plant species were registered. The best represented family was the Cactaceae (cactus) family, represented with a total of eight species, followed by the Fabaceae (legume, pea or bean) family with four species.

Of the 24 species of flora recorded for the Project study area, only one species is reported within a risk category: Mammillaria candida (snowball cactus), falls under the category of endangered according NOM-059- SEMARNAT- 2010.

The densest vegetation is located in the northern study area, where the following species could be found: Acacia farneciana (needlebush), Agave lechuguilla ("lechuguilla"), Jatropha dioica (Leatherstem and Sangre de Drago), Fouquieria splendes (Ocotillo), Opuntia microdasy (Bunny Ears Cactus, Bunny Cactus or Polka-dot Cactus) and Larrea tridentata (Creosote bush).

In the south of the mineral processing plant area the density of the vegetation of Desert Shrubland rosetophilous is low and species like Agave lechuguilla, Jatropha dioica, Fouquieria splendes, Opuntia microdasy, Lippia graveolens (Mexican oregano) and Larrea tridentata are present.

The vegetation in the Project study area is diverse, abundant and has been deteriorated in areas with significant traffic of locals and paths. The arid ecosystem provides for a predominately shrub vegetation cover, mainly by species from Gobernadora (creosote), Ocotillo and Lechuguilla which contribute to soil stability. An indication of the stability maintained in this environmental is shown by the abundance of various cacti species.

Among the species that should be monitored due to their intrinsic biological and ecological characteristics, include the species: conglomeratus Echinocereus cactus (Hedgehog cactus), Mammillaria heyderi (Ball Cactus, Cream Cactus, Cream Pincushion, Flat Cream Pincushion), Mammillaria candida (snowball cactus), Opuntia imbricata (Giant Tree or Cane Cholla) , Opuntia microdasy (Bunny Ears Cactus), Opuntia violácea (Violet Prickly Pear), Opuntia leptocaulis (Desert Christmas cactus), Opuntia humifusa (Creeping Prickly Pear) and Fouquieria Splende (Ocotillo).

The area where the mineral processing plant will be located does not currently have any type of vegetation. Because no roads will be built and existing dirt roads will be used, the disturbance will generally be low in terms relative to the size of the project. Where soil degradation and erosion processes are likely, the affected area will be covered with rock, or other, material to hold the soil in place.

20.5.1.2 Fauna

There are 106 recorded animal species in the State of Durango: 35 mammals; 13 species of reptiles; and 58 species of birds.

The fauna present in the State of Durango represent 19% of the total Mexican fauna, the aviary species represent 32% and the reptilian fauna represent 19% of the total species registered for the country.

The Project area is located within the mammalian fauna area known as Zacatecana. The 35 mammal species identified in the zone are distributed in 26 subgroups and 17 families. Two of the mammal spices are considered threatened: Vulpes Macortis, commonly known as the Kit fox, and Peromyscus Boylii; commonly known as the Brush mouse. One species is considered Endangered: Erethizon dorsatum; commonly known as the North American porcupine.

Within the State of Durango there are 13 areas of importance for the conservation of birds, although; none of the areas is located close to the Velardeña Project. Of the 58 species of birds that were identified in the study area: four of the species are under special protection: Red-tailed hawk, Peregrine falcon, Pine siskin, and Townsend's solitaire; one is an endangered species: Falco mexicanus, commonly known as the Prairie Falcon; and another is considered threatened: Vireo atricapillus, known as the Black-capped vireo.

Finally, in the general area of the Chihuahuan desert that extends to the north in the country, there are 13 species of amphibians and reptiles that have been identified. Two of the species are considered threatened; Coluber constrictor (Black racer) and Masticophis flagellum (Coachwhip snake); and another two are identified under special protection; Cnemidophorus neomexicanus (New Mexico Whiptail), and Crotalus Lepidus (Rock rattlesnake).

20.5.2 Climate, Topography & Vegetation

The following has been sourced from CAM 2012.

The area in which the Velardeña properties are situated is semi-arid with a climate predominantly warm and dry, with a mean annual temperature of 21.1° C and rainfall averaging 243.7 millimeters per year (mm/yr). Temperatures can drop below freezing in the winter and commonly reach the high thirties from July through to September. The predominant winds are northeast-southwest, with speeds of 2.1 to 6.0 meters per second (m/s).

The Velardeña district is located on the northwestern edge of the Meseta Central physiographical province, within the Sierras Transversas sub-province, on the eastern flank of the Sierra Madre Occidental mountain range. The village of Velardeña is located in the valley floor set between two northwest trending ranges. To the west is the Sierra Santa Maria which rises approximately 300 m above the valley floor and, to the east, is the Sierra San Lorenzo rising approximately to 750 m. The Sierra San Lorenzo hosts the Velardeña, Chicago and San Diego properties, the latter being located farthest to the east into the Cerros El Trovador.

In physiographic terms, the zone is mature with a mixed topography. Streams within the area drain either to internal drainage systems or tributaries of the Nazas and Aguanaval rivers which are connected to the Laguna de Mayrán. All of the streams are intermittent and flow only during the rainy season. A series of water dams were built over the years to control water flow from the two rivers for irrigation and water management purposes. The Francisco Zarco dam, located 25 km to the west, is the closest to the Velardeña properties.

The geomorphology shows characteristics typical of a cycle of arid to semi-arid areas. There is an abundance of valleys and flat alluvial plains variably filled with erosional debris derived from adjacent highlands. The drainage systems are generally dendritic and poorly defined; many channels disappear when they reach the valley floor due to infiltration into poorly consolidated alluvial sediments.

20.5.3 Hydrology

The Velardeña project property is located in the Hydrologic Region RH 6 Nazas-Aguanaval on the center-west part of the State of Durango. The Hydrologic Region consists of five basins: R. Aguanaval, R. Nazas-Rodeo, P. Lazaro Cardenas, L. de Mayran y Viesca, and R. Nazas-Torreón.

The project property is located in the Rio Nazas- Torreón basin, next to the Rio Aguanaval basin. The two water bodies are connected by both surface and groundwater. Due to previous mining activity and the nature of the operations, the Project environmental impact is negligible.

20.5.3.1 Surface Water Hydrology

The Cuencamé River is the dominant stream course in the region with the headwaters located in the Sierrilla Atotonilco. There are a number of small springs that are tributary to this river. In the watershed there are 17 washes or stream courses; however, all but two of these are ephemeral and flow only for a brief time after a rainfall event. The two permanents waters are the Aguanaval River located roughly 29 Km, and the Nazas River located 35 km, respectively, from the site.

Two dams were identified, the Francisco Zarco and Las Mercedes located 27.8 km and 29.9 km from the site, respectively. These were constructed in the 1950's to divert flood water and for irrigation water storage and are located north of the Cuencamé Community within the Nazas River basin.

20.5.3.2 Geohydrology

The Project is located in the area containing Pedriceña-Velardeña aquifer (identified with key 1021 according to the Geographic Information of Underground Water). The surface area extent of the aquifer is approximately 3,000 square kilometers (km^2) and is located in middle of the state of Durango.

The majority of the aquifer is located under the municipality of Cuencamé and a small portion lies underneath Peñon Blanco. The aquifer is mainly used for irrigation of crops and a small proportion of the water is used for the urban community. There is no district or irrigation unit, and neither is there a Technical Committee of Underground Water. According to the federal water law, in force since 2008, the municipality of Cuencamé and Peñon Blanco are classified as areas of beneficial use.

20.6 Community Relations & Social Responsibilities

Minera William holds agreements (contracts) with both Ejido communities: Velardeña and Vista Hermosa. The Project plants, as well as the majority of the surface facilities at the mine portals, are on private land owned by Minera William or Minera Labri. Agreements with the Velardeña and Vista Hermosa Ejidos are in place for access easements and surface disturbance outside of these private parcels. Minera William pays an annual rental fee to both Ejidos.

The agreements also state that the mine will preferentially hire residents from the Ejidos and currently approximately 90% of the hourly employees are from Vista Hermosa and Velardeña. When it is not possible to hire from either of the Ejidos, Golden Minerals seeks candidates from the local communities of Cuencamé, Cuatillos or Pedriceña.

The following has been sourced from the Environmental Impact State for the Velardeña Mine Project Exploration and Mining Operation, MIA (April 2013).

According to the results of the 2005 National Census of Population and Housing, the municipality has a total of 53 people who speak an indigenous language. According to the 1995 National Census of Population and Housing conducted by the National Institute of Statistics and Geography (Instituto Nacional de Estadistica y Geografia or INEGI), the total population of the region included 34,660 inhabitants, of which 17,521 are men and 17,139 are women; the growth rate in the period 1990-1995 was 0.02%. The population density is 7.2 inhabitants per square kilometer. The population of the municipality regarding the state represents 2.42%. The largest concentration of population is located in the county seat.

The residents are predominantly Catholic, subsequently followed by the Protestant and Evangelical.

For the provision of educational services at the basic level, there are 36 campuses of pre-school education, 56 primary, 21 secondary schools, two middle and one high school.

Health services in the region are provided by the Ministry of Health and Welfare (SSA), the Institute for Social Security and Services for State Workers (ISSSTE) and the Mexican Social Security Institute (IMSS).

The SSA runs a health center which provides outpatient services and first aid. A laboratory and five hospital rooms are available and are attended by two general physicians who serve an average of 40 patients per day usually with common problems such as colds, stomach ailments and first aid. More serious problems are channeled to the Regional General Hospital of Cuencamé or the city of Gomez Palacio or Guadalupe Victoria.

The Regional Hospital serves the inhabitants of the towns of: Cuencamé, Nazas, Simon Bolivar, Santa Clara, White Rock, Guadalupe Victoria, San Juan de Guadalupe; Juan Aldama and Miguel Auza in the state of Zacatecas; and has general consultation, consultation Obstetrics and Gynecology, Surgery, Dentistry, Pediatrics, Orthopedics, hospitalization, Emergency Medical, Laboratory, X-ray, pharmacy; 13 medical specialists 8 general physicians, 44 nurses, 3 radiology technicians, 3 laboratory technicians, and myriad additional personnel.

The workforce generally consists of a total of 8,500 people, representing 24% of the total population of the area, which is mainly engaged in agriculture, livestock and mining industry. A further breakdown of workforce and the respective industries shows that approximately 2,300 persons, (27% of the total) are engaged in agriculture, forestry, hunting and fishing; 2,100 people (25%) are dedicated to mining, oil and gas, manufacturing, electricity , water and construction; and 4,100 people, (48%), are dedicated to trade, transport , government and other services.

20.7 Closure & Reclamation

In July 2012, a "Conceptual Closure Plan, Golden Minerals Company, Velardeña Operations" was prepared by Kermit Behnke for the Project. The purpose of this Conceptual Closure Plan (CCP) was to present the conceptual plan for closure and reclamation of the Minera William Velardeña Operations located near Cuencamé; State of Durango, Mexico in substantial accordance with international mine closure standards. The Velardeña Operations consist of the Velardeña and Chicago mining facilities, the Labri Mill sulfide ore processing facility (Plant #1), an oxide processing facility (Plant #2), and related ancillary facilities including tailings impoundments, access roads, storage buildings, and water pumping stations.

20.7.1 Reclamation Statement of Responsibility

Golden Minerals has assumed the responsibility for the reclamation of surface disturbances that are attributable to the Velardeña Mineral Properties consistent with the Mining Rights of Public Lands Under Article 27 of the Mexican Constitution (the government may establish compulsory measures for the preservation and restoration of land and ecological equilibrium), Article 98 (land reclamation should consider all the necessary actions to prevent land degradation), and Article 35 (reclamation conditions defined in the approved Environmental Impact Statement are binding) of the General Law of Ecological Equilibrium and the Protection of the Environment. The Mining Law establishes that holders of mining concessions are subject to the compliance of all Mexican general regulations and official norms that relate to the environmental protection and ecological equilibrium. All areas that have been disturbed at the Velardeña Mineral Properties by Golden Minerals and owned predecessors will be reclaimed to a safe and stable condition upon cessation of mining operations.

20.7.2 Reclamation Approach

All mobile and stationary equipment and constructed facilities associated with the mining operations will be removed prior to or during closure and reclamation. All disturbed areas that have been associated with Golden Minerals mining and processing activities will be reclaimed to a safe and stable condition in accordance with the 2012 CCP upon cessation of mining operations.

20.7.3 Description of Facilities

A main highway (Mexico Hwy 40D) and improved gravel roads provide access to the operations from the town of Velardeña. Other off-highway access roads are maintained from the facilities entrance to the office, mine, and plant locations. The mining operations at these facilities include mining the Ag, Au, and associated base metals ore from the underground workings and transporting the ore to the surface metallurgical processing plants to develop a final salable product. The metallurgical operations include the use of equipment typical of the respective type of processing operations and include feeders, crushers, conveyors, screens, fine ore bins, grinding mills, flotation equipment, leach plant, and other associated stationary and mobile equipment in support of mining and processing operations.

Minera William anticipates continued mining operations at this location including the expansion of the disturbance areas as necessary for continued mining and processing activities. This plan addresses only the current disturbance areas and structures as generally described on the attached figures and within the 2012 CCP.

The Velardeña Mine area consists of both the Minera William current operations and the historic mine workings. As depicted on the following figures the current active operations are delineated with orange boundaries, the historic mine disturbance areas excluded from this plan are delineated with a green boundary line, and general existing access roads are depicted with black boundaries:

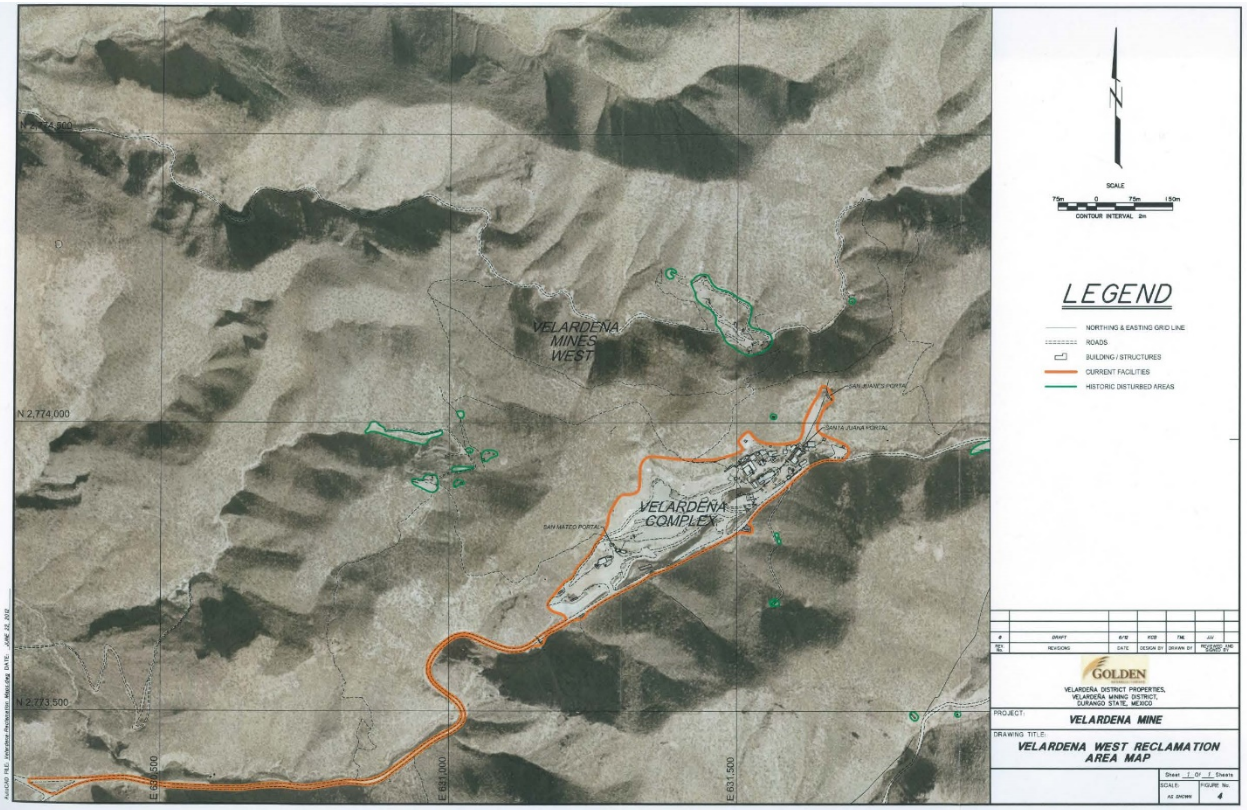


Figure 20-1: Velardeña West Reclamation Area Map

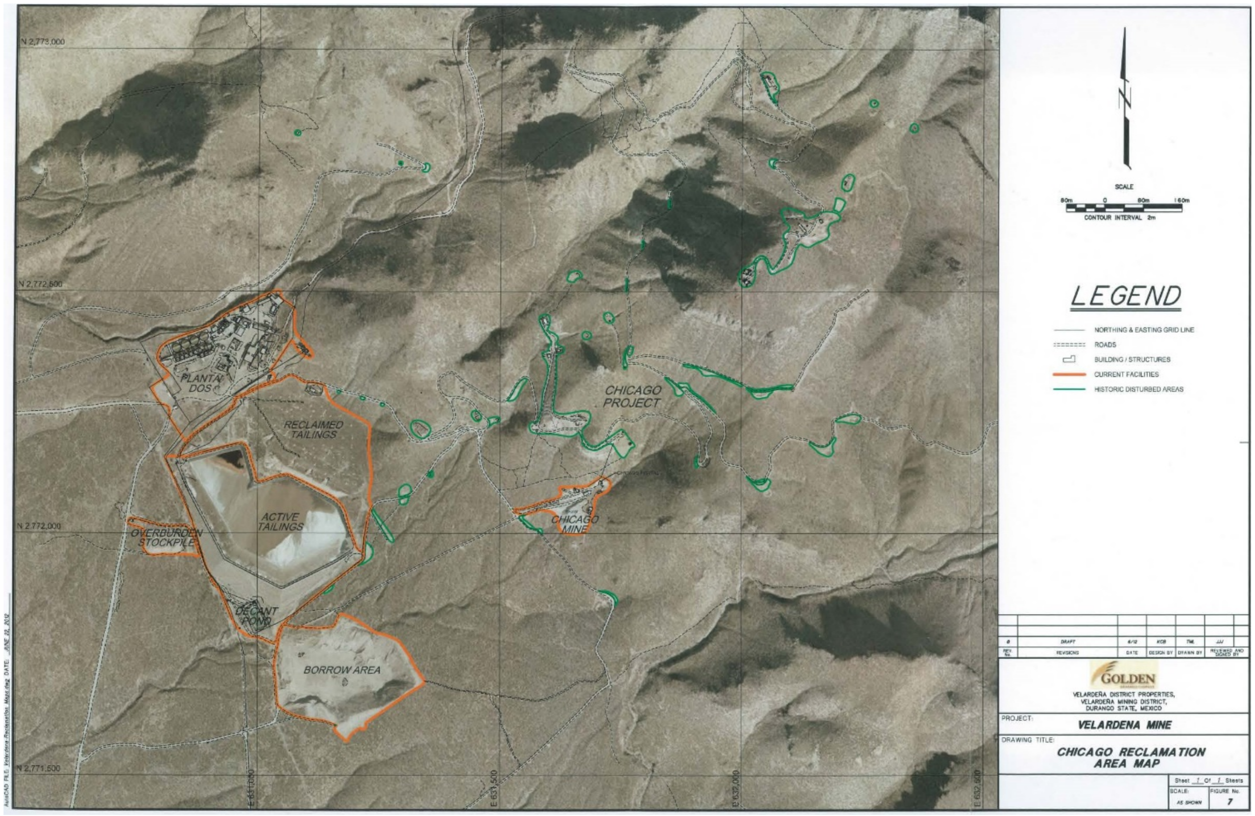


Figure 20-2: Velardeña East Reclamation Area Map

The Velardeña Complex is an active underground mining area and consists of underground access portals including the San Mateo Portal, Santa Juana Portal, and the San Juanes Portal. The surface facilities include security offices, mine offices, maintenance shops, fueling areas, a warehouse, material storage areas, a compressor installation, electrical substations, and other miscellaneous support facilities. Other ground disturbances in the area include primary, secondary, and inter-plant access roads, ore stockpiles, and waste rock dumps.

The Chicago Project area consists of active underground mining and process plant activities in addition to historic mining disturbances. The Chicago Mine consists of access roads, the Chicago Portal, mine support structures, and a waste rock dump. The process area includes Plant #2 (a 550 t/d oxide leach processing facility), an active tailings stockpile with a decant pond system, a reclaimed tailings stockpile, an overburden stockpile, and a borrow area. Existing facility structures located at Plant #2 include security offices, a warehouse, maintenance shops, an ore dump pocket, a crusher, conveyor, grinding mill, flotation mill, numerous leach tanks, and Cu recovery facilities. Other ground disturbances in the Plant #2 area include primary, secondary, and inter-plant access roads, material storage areas, and parking areas.

The Labri Mill (Plant #1) is a 300 t/d sulfide ore flotation plant that is located on the west side of the town of Velardeña. The area consists of the Labri Mill operating facilities inclusive of security offices, a warehouse, maintenance shops, an ore dump pocket, a crusher, conveyor, grinding mill, flotation mill, numerous leach tanks, and Cu recovery facilities. Other facilities include the active tailings stockpile and the inactive tailings stockpile.

20.7.4 Property Acreage

The respective areas of the mining facilities and the current mining disturbances at this operation are further described in **Table 20-3**.

Table 20-3: Reclamation & Disturbance Areas

Area Description	Description	Acres
Reclamation Areas	Areas within the orange boundary lines as depicted on the reclamation maps	104
Historic Mining Disturbances	Areas within the green boundary lines as depicted on the reclamation maps	Not Applicable
Local Roads	Roads not associated with current mining activities as depicted on the reclamation maps with black-font dashed boundaries	Not Applicable

20.7.5 Post-Mining Land Use

Minera William will remove the mining equipment and structures at the facilities that are not required for the Post-Mining Land Use (PMLU) and will regrade all of the applicable disturbed areas of the property to a safe and stable condition suitable for the PMLU upon completion of mining activities as further described herein.

It is anticipated that the PMLU will include:

- Future mining operations;
- Other future industrial use;
- Livestock grazing; and
- Open space.

20.8 Reclamation Approach

The following reclamation activities were adapted from the 2012 CCP and are assumed to meet the requirements necessary for the PMLU discussed in **Section 20.7.5**.

20.8.1 Equipment & Building Removal

All equipment (mobile equipment, feeders, crushers, conveyor belts, screens, stackers, etc.) and structures (offices, shops, tanks, process buildings, fuel and oil tanks, etc.) will be removed from the property upon completion of operations. Structures may remain if requested by the subsequent owner for the PMLU. Any contaminated soils resulting from vehicle traffic and maintenance, or from other processing activities, may be subject to remediation prior to closure according to applicable environmental rules.

20.8.2 Roads, Power Lines, Water Lines & Fences

Any roads constructed or used specifically for mining and processing operations at this facility and not required for the PMLU will be reclaimed. The roads will be regraded and ripped to inhibit erosion and to promote revegetation seed growth as appropriate. Roads required for use during reclamation and closure will be reclaimed upon completion of reclamation and closure activities. Primary power lines and water lines installed on the property for mining and processing operations will generally remain in place as designated for the PMLU.

20.8.3 Area Regrade & Closure

Upon cessation of mining activities, all disturbed areas of the property will be assessed for any hazardous conditions including unstable soils or slopes, hazardous depressions, potential erosion conditions, and drainage requirements. Potentially unstable slopes or hazardous depressions will be regraded to a stable condition. Mine shafts or portals will be plugged, bulkheaded, sealed, or capped. All areas will be regraded to blend with natural topography and ripped, as appropriate, prior to the application of a hydro-mulched seed mix as appropriate.

20.8.4 Slope Stabilization

Final slopes will have a slope of approximately 1(H):1(V) (angle of repose) or less (i.e., flatter) depending upon the characteristics of the material. Slopes deemed unstable at their operational slopes will be regraded to minimize erosion and provide geotechnical stability.

20.8.5 Soil Conservation

Stockpiles of salvaged soil (plant growth material) that may be developed during mining operations may be located on the property as soil stockpiles and maintained for use during reclamation activities. All soil stockpiles will be stabilized as necessary during operations to prevent excessive losses from erosion or fugitive dust emissions. During reclamation activities, areas will be regraded and ripped as necessary to incorporate plant growth materials, to prevent excess compaction, and to achieve a suitable soil zone to enhance voluntary plant growth.

20.8.6 Revegetation

Where surface disturbances result in compaction of the soil, ripping, disking, or other measures will be employed to reduce compaction and establish a suitable root zone to promote the hydro-mulched revegetation growth.

20.8.7 Measures to Achieve Post-Mining Land Use

1) Measures to restrict public access to hazardous surface features:
   a) All equipment and structures not required for the PMLU will be removed from the property. Additionally, all scrap metal, wood, trash, and other debris will be removed, and disturbed areas will be regraded, bermed, and/or fenced to remove any unstable slopes or hazardous depressions and to restrict public access. All portals and shafts associated with the Minera William operations will either be sealed or fenced as discussed in **Section 20.8.3**. Warning signs will be posted in locations where public access is available.
2) Measures will be taken to address erosion control and stability:
   a) The site will be regraded utilizing existing storm water drainage control and will be ripped to promote rainfall infiltration and reduce run-off and erosion. Local storm water containment ponds (charcos) will be constructed and rip-rap will be installed for bank protection, as appropriate.

3) Measures will be taken to address revegetation:
   a) All disturbed areas will be regraded and ripped to promote vegetation growth and a suitable seed mix will be hydro mulched on disturbed areas during reclamation.

20.8.8 Schedule of Reclamation

The anticipated schedule for reclamation is as follows:

- Reclamation will be initiated within one year following cessation of mining activity.
- Minera William may extend the period in which to initiate reclamation at this facility if there is a reasonable likelihood that the facility may resume operations based upon:
  - The existence of additional Ag and Au ore or other mineral commodities;
  - Historical fluctuations in the value of the commodity; or
  - The design life of any process components existing at the facility.
- Once initiated, Minera William will continue the final reclamation measures as discussed until complete unless mining is reinitiated.

20.8.9 Reclamation & Closure Costs

Reclamation costs were adapted from the 2012 CCP. Unit costs applied to the 2012 CPP were verified by comparing equipment and labor costs used in the 2012 CCP to industry standard cost estimating material. Upon review these unit costs were deemed reasonable and reflective of the prescribed reclamation activities. The 2012 reclamation costs were then adjusted to reflect 2015 dollars by accounting for inflation from 2012 to 2015. An average inflation rate from 2012 to 2015 was calculated to be approximately 0.74% based on the following three producer price indexes (PPI) from the US Bureau of Labor Statistics (BLS):

- PPI – Total Manufacturing Industries = 0.72%;
- PPI – All Commodities = 0.38%; and
- PPI – Industrial Commodities = 1.11%.

All reclamation and closure costs will be wholly born by Minera William, with Golden Minerals Company as its parent company. The cost will be covered by the sale of the equipment with salvage value. The total costs as developed on the **Table 20-4** are summarized as:

Table 20-4: Closure Cost Estimate

Activity		Total Cost (US$)
Direct Costs		
Mobilization & Demobilization		$26,000
Infrastructure Removal		$1,002,523
Regrade / Rip		$326,227
Revegetation		$187,259
Total Direct Cost		**$1,542,008**
Site Maintenance Costs		$75,675
Indirect Costs		
Project Engineering, Surveying (8% of Direct Costs)	4.0%	$61,680
Project Management & Owner's Costs (4% of Direct Costs)	8.0%	$123,361
Financial Assurance (3% of Direct Costs)	3.0%	$46,260
Contractor Profit (15% of Total Direct Cost)	15.0%	$231,301
Legal Support (3% of Direct Costs)	3.0%	$46,260
Total Indirect Cost		**$508,863**
Subtotal - Direct; Maintenance; Indirect		*$2,126,547*
Contingency Applied to Total Direct Costs; 25%		$385,502
Total Reclamation Costs		**$2,512,049**
Inflation Adjustment Applied Over Three Years		0.74%
2015 Reclamation Costs		**$2,567,975**

There are no known performance bonds or financial guarantees, either required or posted by Golden Minerals, with respect to closure and reclamation activities.

20.9 Mine Closure

20.9.1 Mining & Processing Areas

Reclamation of the mine disturbances, processing plants, tailing piles, and the associated mining areas as described in this report and in the 2012 CCP will be initiated within one year after cessation of mining activity as the site.

20.9.2 Personnel

Minera William personnel will inspect and maintain this facility as appropriate during the post-closure period until such time as closure and reclamation is complete. Minera William will not have any full-time personnel assigned to this facility following completion of the reclamation project.

20.9.3 Monitoring

The closure of operations at this site will be monitored by Minera William personnel to assure final reclamation is in accordance with the conditions of this report and the 2012 CCP.

21 CAPITAL AND OPERATING COSTS

The Project is currently in operation without quoted Reserves. Many of the costs have been provided to Tetra Tech based on Golden Minerals' past production data and internal forecasts, which Tetra Tech has reviewed and found consistent with a mine of this type.

21.1 Capital Costs

Remaining capital cost expenditures over the LoM are estimated to be US$26.2 million as shown in **Table 21-1**. For the restart of operations, Golden Minerals will spend US$10.3 million in the preproduction year, as indicated in **Table 21-1** below.

Table 21-1: Capital Costs

Capital Costs	Preproduction	LOM	Full LOM
Mine Development	$869	$9,922	$10,791
Process Plant	$7,410	$2,050	$9,460
Infrastructure	$782	$0	$782
Other Non-Operating Costs	$1,210	$3,947	$5,158
Total (1)	**$10,272**	**$15,919**	**$26,190**

(1) Columns may not total due to rounding

The preproduction costs include

- US$6.5 million for the construction of the BIOX® plant to treat sulfides; and
- US$250,000 for the process facility Plant #1 instrumentation and metering, planned for the LoM.

Planned expenditures over the mine life include US$230,000 per year for the tailings storage facility.

21.2 Operating Costs

Operating Costs will average about US$193/t milled over the remaining LoM as shown in **Table 21-2**. This projection is based upon actual operating costs and adjusted, as appropriate to reflect planned activities.

Table 21-2: Operating Costs (2015-2018)

Item	Total ($000s)	Unit Cost ($/t)
Mining costs	$94,303	$82.87
Milling costs	$54,241	$47.67
Mine & Process	**$148,545**	**$130.54**
Treatment and Refining costs	$45,440	$39.93
Contingency	$15,656	$13.76
G&A	$8,014	$7.04
Royalty	$1,651	$1.45
Total Operating	**$219,306**	**$192.72**

22 ECONOMIC ANALYSIS

Project cost estimates and economics developed in the Technical-Economic Model (TEM) are prepared on a monthly basis for the LoM as based on the primary veins from the total project Resource. The following PEA includes Measured, Indicated and Inferred Mineral Resources; Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. This PEA also includes Inferred Mineral Resources that are too speculative for use in defining Reserves. Based upon design criteria presented in this report, the level of accuracy of the estimate is considered ±35%. Economic results are summarized in **Table 22-1**. The analysis suggests the following conclusions, assuming no debt:

- Mine Life: 10 years;
- Pre-Tax NPV8%: US$85.91 million, IRR: 139%;
- Payback : One year; and
- Federal Precious Metal Royalty: US$1.65 million

22.1 Inputs & Assumptions

Technical assumptions used in the economic analysis are presented in **Table 22-1**. All costs are in Q1 2020 US dollars. Market prices are a reflection of current conditions and utilize the 3-year trailing average metal prices. Results reflect an 8% hurdle rate. No debt is assumed.

Table 22-1: Input and Assumptions Summary

Description	Value	Units
Market Prices:		
Gold (Au)	$1,324.00	/oz
Silver (Ag)	$16.23	/oz
Lead (Pb)	$0.90	/lb
Zinc (Zn)	$1.25	/lb
Taxes:		
Federal Precious Metal Royalty	0.50%	-
Special Mining Tax (SMT)	7.50%	-
Financial:		
Discount Rate	8%	-

Mine and process plant production summaries over the LoM are summarized in **Table 22-2** and **Table 22-3**, respectively. These schedules are discussed in detail in other sections of this report.

Table 22-2: RoM Summary

Description	Value	Units
RoM Milled	1,138	kt
RoM Grades		
Gold (Au)	5.1	g/t
Silver (Ag)	337	g/t
Lead (Pb)	1.32%	-
Zinc (Zn)	1.63%	-
Metal Contained in RoM		
Au	188	koz
Ag	12,325	koz
Pb	33,165	klb
Zn	40,972	klb

Pb values in the Zn concentrate as well as Zn values in the Pb concentrate are not paid-for by the smelters and are therefore not reported in the table. Au and Ag pay-fors are different for each concentrate produced and are appropriately accounted for in the TEM.

Table 22-3: Process Summary

Description	Unit	Total	Pb Conc	Zn Conc	Doré (kg)
Concentrate	kt-dry	61,888	29,539	32,350	40,398
Metal Recoveries					
Gold (Au)	%	67%	20%	3%	44%
Silver (Ag)	%	90%	71%	10%	9%
Lead (Pb)	%	71%	71%	0%	0%
Zinc (Zn)	%	76%	0%	76%	0%
Contained Metals					
Gold (Au)	koz	125	37	5	83
Silver (Ag)	koz	11,086	8,773	1,202	1,112
Lead (Pb)	klb	10,754	10,754	0	0
Zinc (Zn)	klb	14,212	0	14,212	0

Payable metals for each concentrate type are derived using the smelter terms as described in **Section 19** and shown in aggregate in **Table 22-4**. These metals account for the gross value of the concentrates.

Table 22-4: Payable Metals

Description	Value	Units
Au	119	koz
Ag	10,214	koz
Pb	9,868	klb
Zn	12,080	klb

22.2 Technical-Economic Results

Technical-economic results are presented in **Table 22-5**. Over the remaining LoM, the Project is projected to return a cash flow, discounted at 8%, of approximately US$130.2 million. Given current conditions, positive cash flow is projected to occur in year one.

Table 22-5: Technical-Economic Results (US$000s)

Item	Total ($000s)	Pb Concentrate	Zn Concentrate	Doré
Gross Payable	**$375,728**	**$201,243**	**$50,112**	**$124,373**
TCs, RCs and penalties	($33,130)	($18,327)	($13,469)	($1,333)
Freight & Insurance (1)	($12,311)	($5,113)	($4,748)	($2,450)
NSR	$330,288	$177,803	$31,896	$120,590
Operating Costs				
Mining costs	($94,303)			
Milling costs	($54,241)			
Contingency	($15,656)			
G&A	($8,014)			
Federal Mining Royalty	($1,651)			
	($173,866)			
Operating Margin	$156,423			
Mine Development	($10,791)			
Process Plant	($9,460)			
Infrastructure	($782)			
Other Non-Operating Costs	($5,158)			
Cash Flow	$130,232			
$NPV_{8\%}$	$85,914			
IRR	139%			
Payback (years)	1			

The LoM cash flow results are presented in **Table 22-6** on an annual basis.

Table 22-6: LoM Cash Flow Results

Item	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
NSR													
Gross Payable	$375,728		$36,361	$42,439	$34,730	$46,735	$42,943	$33,714	$37,114	$31,111	$32,519	$36,495	$1,567
TCs and RCs Doré	($1,333)		($104)	($129)	($120)	($167)	($175)	($125)	($142)	($104)	($125)	($139)	($3)
TCs, RCs and Penalties Pb	($18,327)		($2,589)	($2,028)	($1,703)	($2,134)	($2,129)	($1,741)	($1,827)	($1,204)	($1,366)	($1,528)	($77)
TCs, RCs and Penalties Zn	($13,469)		($2,009)	($2,144)	($1,644)	($1,801)	($1,198)	($1,314)	($1,024)	($703)	($489)	($1,070)	($73)
Freight & Insurance Doré	($2,450)		($190)	($228)	($215)	($319)	($291)	($218)	($250)	($228)	($253)	($251)	($8)
Freight & Insurance Pb	($5,113)		($833)	($545)	($468)	($589)	($550)	($477)	($495)	($336)	($381)	($413)	($25)
Freight & Insurance Zn	($4,748)		($702)	($750)	($578)	($632)	($426)	($463)	($364)	($251)	($178)	($379)	($26)
NSR	**$330,288**		**$29,934**	**$36,615**	**$30,001**	**$41,093**	**$38,175**	**$29,376**	**$33,013**	**$28,286**	**$29,726**	**$32,715**	**$1,355**
Operating Costs													
Mining costs	($94,303)		($9,347)	($9,347)	($9,347)	($9,347)	($9,347)	($9,347)	($9,347)	($9,347)	($9,347)	($9,347)	($830)
Milling costs	($54,241)		($4,488)	($5,409)	($4,814)	($6,230)	($5,455)	($4,797)	($5,327)	($5,759)	($5,788)	($5,717)	($456)
Contingency	($15,656)		($1,462)	($1,554)	($1,494)	($1,636)	($1,558)	($1,493)	($1,546)	($1,589)	($1,592)	($1,585)	($148)
G&A	($8,014)		($782)	($782)	($782)	($782)	($782)	($782)	($782)	($782)	($782)	($782)	($195)
Federal Mining Royalty	($1,651)		($150)	($183)	($150)	($205)	($191)	($147)	($165)	($141)	($149)	($164)	($7)
Operating Costs	**($173,866)**		**($16,229)**	**($17,275)**	**($16,588)**	**($18,201)**	**($17,334)**	**($16,566)**	**($17,167)**	**($17,619)**	**($17,657)**	**($17,595)**	**($1,636)**
Operating Margin	**$156,423**		**$13,706**	**$19,339**	**$13,413**	**$22,892**	**$20,841**	**$12,810**	**$15,846**	**$10,667**	**$12,069**	**$15,121**	**($281)**
Capital Costs													
Mine Development	**($10,791)**	($869)	($603)	($749)	($811)	($1,499)	($1,522)	($1,519)	($1,219)	($874)	($728)	($398)	$0
Process Plant	**($9,460)**	($7,410)	($226)	($226)	($226)	($226)	($226)	($226)	($226)	($226)	($226)	($13)	$0
Infrastructure	**($782)**	($782)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Other Non-Operating Costs	**($5,158)**	($1,210)	($358)	($373)	($379)	($448)	($450)	($450)	($420)	($385)	($370)	($316)	$0
Pre-Tax Cash Flow	$130,232	($10,272)	$12,518	$17,991	$11,997	$20,719	$18,643	$10,615	$13,981	$9,182	$10,744	$14,394	($281)
NPV$_{8\%}$	$85,914												
IRR	139%												
Payback (years)	1												

22.3 Sensitivities

Project sensitivity to market prices, capital and operating costs are shown in **Figure 22-1**. As shown, the project is most sensitive to metal price, and slightly more sensitive to the silver price. Given the low remaining required capital expenditures, it is expected that the Project is least sensitive to capital.

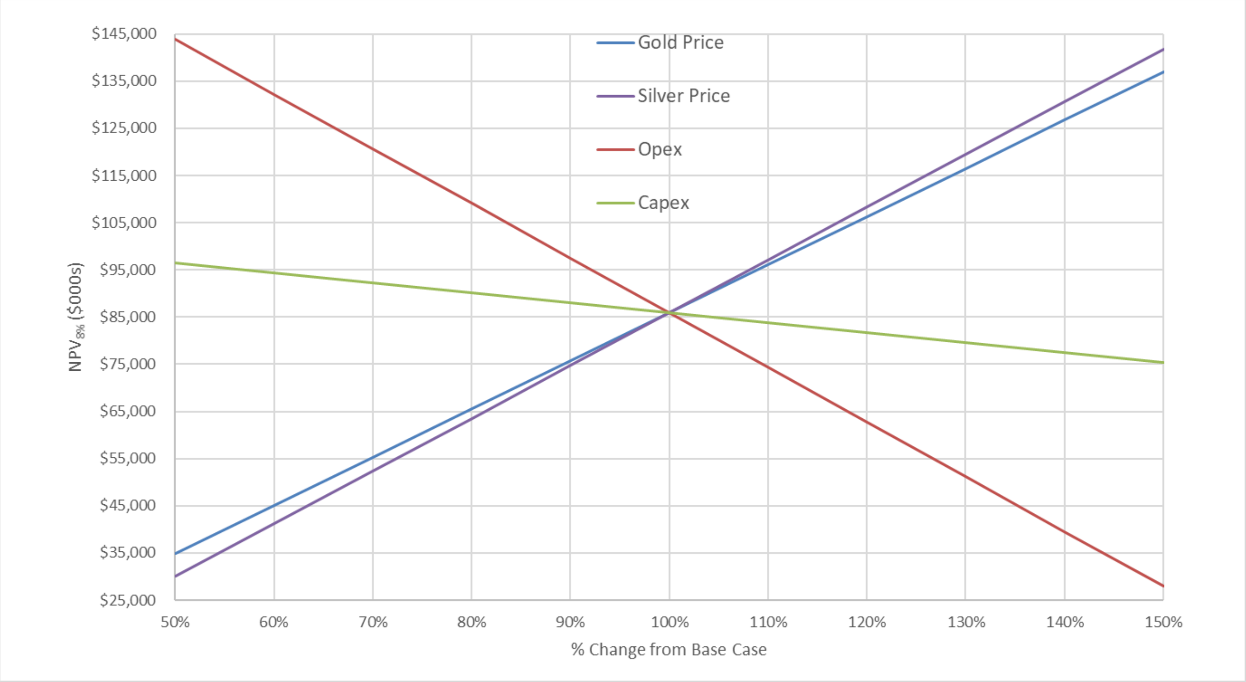


Figure 22-1: Sensitivities

23 ADJACENT PROPERTIES

The Project is surrounded by claims held by various entities, with the most significant holdings controlled by Industrias Peñoles, S.A.B. de C.V. (Peñoles) and Grupo México S.A.B. de C.V. (Grupo Mexico). Publicly available data regarding exploration results, Mineral Resources, and Mineral Reserves for adjacent properties were not located.

As described in **Section 6**, the Velardeña property is located within a broader district of the same name, which is host to a number of significant, past-producing Ag-Au-Pb-Zn mines. The most important of these cluster within the Santa Maria Dome, west of the pueblo of Velardeña, and include the Santa Maria, Industria, San Nicholas, and Los Azules mines. In addition, the San Diego project, located nine kilometers east of Velardeña, is a 50:50 Joint Venture between Golden Minerals and Golden Tag Resources, Ltd.

24 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any additional information for which the exclusion thereof would render this report misleading.

25 INTERPRETATIONS AND CONCLUSIONS

With the inclusion of Measured, Indicated and Inferred Mineral Resources the findings of this PEA suggest the Project is conceptually economically viable. The PEA has been based on Mineral Resources, which are not Mineral Reserves and do not have demonstrated economic viability; therefore, without the definition of Reserves and subsequent feasibility or pre-feasibility studies, the project cannot be determined to be economically viable.

25.1 Geology & Resources

Drill hole and channel samples have been collected and analyzed using industry standard methods and practices and are sufficient to support the characterization of grade and thickness and further support the estimation of Measured, Indicated and Inferred Resources.

25.2 Mining

Results of the PEA indicate mining is potentially economically viable based on the inclusion of Inferred Mineral Resources. However, due to the nature of the mineralization and the scale of the operations, extensive Resource drilling of the deposit is not planned. For this reason, detailed mine plans and schedules are not expected to be produced for the deposit. The consequence of this is that residual risk remains for mining of the project and planning of grades and stope tonnages can only be completed on a short-term basis.

The success of the proposed plan is sensitive to mining dilution, which could increase the costs of saleable products, but also provides opportunity as any potential reductions in dilution from the mining would greatly benefit the project. Recent test mining at the site has confirmed a minimum selective mining width of 0.7 m is achievable, which can contribute to reducing dilution.

25.3 Metallurgy & Process

There are no geological, lithological or mineralogical changes in the process plant feed anticipated for the envisaged potential future production as compared to previous operations. Existing legacy operational data fully supports the existing process flow sheet for future production at the Plant #1.

The use of existing and refurbished equipment within the pre-existing facilities, and the production of a marketable concentrates, is Golden Minerals' preferred method of treating potential future production.

Once the mine starts operations, it is recommended to closely check the productivity of the new bio-oxidation plant to verify if the expected results are obtained.

25.4 Significant Risk Factors

Factors that could affect the potential economic viability of the project could include underestimations of operating capital and declines in any or all of the metal prices. Estimation of Resources could be affected by changes in metal prices and the actual mineralized shoot shapes and orientations. Successful implementation of the proposed mine plan is subject to the successful conversion of Inferred Resources to Indicated or Measured classification as well as conversion of Measured and Indicated Mineral Resources to Mineral Reserves, the prediction of stope layout and shape which is controlled by the actual shape of mineralized shoots and their orientations, and the ability of the mining operations to control waste dilution.

The performance of the BIOX® plant is key to the economics estimated in this PEA. If the expected results are not achieved, the BIOX® process would compromise an important part of the entire process.

Many of the above stated risks are balanced with the opportunity to add potential value in excess of what has been described in this PEA.

26 RECOMMENDATIONS

The following recommendations are made to refine the current operation but are not integral to the implementation of the plan proposed in this PEA. **Table 26-1** outline estimated significant costs if the following recommendations were completed.

Table 26-1: Estimated Costs Associated with Recommendations

Description	$USD
Exploration Drilling ($100/m)[1]	500,000
Mining Trade-off Studies	35,000
Metallurgical Testwork	100,000
Total	**635,000**

Note
1: Assuming 9,000 m drill program.

26.1 Geology & Resources

- Continue to collect specific gravity measurements and refine current estimations of specific gravity. Additional measurement should ideally be made with a paraffin wax or epoxy coating;
- Implement procedures of duplicate channel sampling by secondary sampling teams of drifts prior to stope development to ensure grade and thickness characteristics and to serve as field duplication of channel samples;
- Setup of strict control sample review procedures and tolerances involving review of control sample failure on receipt of each batch's results, and automatic triggering of batch reanalysis immediately after being alerted to failures;
- Improve sample data transcription methods to reduce control sample labeling errors and immediately resolve errors when encountered;
- Perform a detailed model reconciliation on a completed stope early in the proposed mine life and alter the estimation methods if the result of the reconciliation suggest refinements should be made;
- Continue to advance exploration drilling down dip of current Inferred Resources as new levels are established. Preferentially target the Terneras, San Mateo, Roca Negra and A4 veins; and
- The costs for additional drilling have not been included in the LoM PEA but any further Resource expansion would be dependent on additional drilling.

26.2 Mining

It is recommended that Golden Minerals implements cut and fill mining where waste and vein material are blasted separately in order to reduce ore dilution. This practice would consider more total tonnes blasted in each section. Vein tonnes would be reduced, but the resulting grade would be higher. Recent tests on selective mining widths of 0.7 meters has proven to be achievable. Because this practice requires an efficient operations control, Tetra Tech recommends having detailed control in drilling and blasting.

The mine plan developed for the PEA should be optimized and undertaken at a more detailed level, which will enable a greater understanding of mining constraints, costs and resulting mill feed.

26.3 Metallurgy & Process

Antimony and Arsenic are penalty elements in the Lead and Zinc concentrates and need to be added to the database and spatially modeled. Additional metallurgical test work is recommended to investigate the depression of Antimony and Arsenic from the final Lead and Zinc concentrates, and Zinc from the pyrite concentrate. After the startup of the BIOX® plant, it is recommended to focus on the performance of the plant and adjust the process as necessary to achieve the best results.

26.4 Economic Analysis

It is recommended that future studies include a more in-depth look at the reclamation costs. Currently, it is anticipated that the salvage sale of equipment will cover the cost of the reclamation costs. Due to changing parameters in the mine life and size, it is recommended to review this assumption in the future.

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28 DATE AND SIGNATURE PAGE

CERTIFICATE OF AUTHOR

Guillermo Dante Ramírez-Rodríguez, PhD, MMSAQP
Principal Mining Engineer of Tetra Tech
350 Indiana Street, Suite 500
Golden, Colorado 80401
Telephone: (303) 217-5700

I, **Guillermo Dante Ramírez-Rodríguez, PhD, MMSAQP**, of Golden, Colorado do hereby certify:

a) I am a Principle Mining Engineer with Tetra Tech, Inc. with a business address of 350 Indiana St., Suite 500, Golden, CO 80401.

b) This certificate applies to the Technical Report titled "Preliminary Economic Assessment NI 43-101 Technical Report of the Velardeña Project, Durango State, Mexico" with an effective date of April 2020.

c) I have a Bachelor's degree in Mining and Metallurgical Engineering from the University of Zacatecas School of Mines in Mexico, and a Master and Doctorate degrees in Mining and Earth Systems Engineering from the Colorado School of Mines, in the United States of America. I am a QP member for the Mining and Metallurgical Society of America (Member No. 01372QP). I have over 32 years of professional experience since my graduation in 1987. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

d) I visited the property December 10,2019.

e) I am responsible for Sections 16, 19, 21, and 22, as well as portions of Sections 1, 2, 24, 25, 26, and 27.

f) I satisfy all the requirements of independence according to NI 43-101.

g) I have read NI 43-101, Form 43-101 F1, and the Companion Policy to NI 43-101 (43-101 CP) and this Technical Report has been prepared in compliance with NI 43-101, Form 43-101 F1, and 43-101 CP.

h) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

i) I consent to the filing of the Technical Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Technical Report.

Dated: May 8, 2020

"Guillermo Dante Ramírez-Rodríguez PhD, MMSAQP"

SIGNATURE OF QUALIFIED PERSON

Guillermo Dante Ramírez-Rodríguez PhD, MMSAQP

PRINT NAME OF QUALIFIED PERSON

CERTIFICATE OF AUTHOR

Leonel López C.P.G., Associate Principal Geologist of Tetra Tech

Principal Mining Consultant of Tetra Tech
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Telephone: 303-217-5700

I, **Leonel López, CPG, SME-RM**, of Golden, Colorado do hereby certify:

a) I am currently employed as Associate of Tetra Tech located on 350 Indiana Street, Suite 500 Golden, Colorado 80401.
b) This certificate applies to the Technical Report titled “Preliminary Economic Assessment NI 43-101 Technical Report of the Velardeña Project, Durango State, Mexico” with an effective date of April 2020.
c) I am a Professional Geologist (PG-2407) in the State of Wyoming, USA, a Certified Professional Geologist (CPG-08359) in the American Institute of Professional Geologists, an SME Founding Registered Member (#1943910), a registered Geological Engineer (Cédula Profesional #1191) in the Universidad Nacional Autónoma de México, a member of the Society of Economic Geologists, and a member of the Geological Society of America.
d) I graduated from the Universidad Nacional Autónoma de México with the title of Ingeniero Geólogo in 1966 and subsequently have taken numerous short courses in Economic Evaluation and Investment Decision Methods at Colorado School of Mines, and other technical subjects in related professional seminars. I have practiced my profession continuously since 1966.
e) 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 and relevant work experience, I fulfill the requirements of a “qualified person” for the purposes of NI 43-101.
f) I visited the property December 10,2019.
g) I am responsible for Sections 20 and 23, as well as portions of Sections 1-12, 24, 25, 26, and 27 of this Technical Report.
h) I satisfy all the requirements of independence according to NI 43-101.
i) I have read NI 43-101, Form 43-101 F1, and the Companion Policy to NI 43-101 (43-101 CP) and this Technical Report has been prepared in compliance with NI 43-101, Form 43-101 F1, and 43-101 CP.
j) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
k) I consent to the filing of the Technical Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Technical Report.

Dated May 8, 2020

“Leonel López, CPG, SME-RM” - Signed
Signature of Qualified Person

Leonel López, CPG, SME-RM
Print name of Qualified Person

CERTIFICATE OF AUTHOR

Randolph P. Schneider, QP, Associate Metallurgical Engineer of Tetra Tech
Metallurgical Engineer
350 Indiana Street, Suite 500
Golden, Colorado 80401
Telephone: 303-217-5700

I, **Randolph P. Schneider, QP**, of Wellington, Colorado do hereby certify:

a) I am currently employed as subcontractor of Tetra Tech located on 350 Indiana Street, Suite 500 Golden, Colorado 80401.
b) This certificate applies to the Technical Report titled “Preliminary Economic Assessment NI 43-101 Technical Report of the Velardeña Project, Durango State, Mexico” with an effective date of April 2020.
c) I am a Professional Metallurgist and a Registered Member of The Society for Mining, Metallurgy & Exploration, a member of the Canadian Institute of Mining, Metallurgy and Petroleum, a fellow of the Australasian Institute of Mining and Metallurgy, a QP member of Mining & Metallurgical Society of America (Member No. 01330), a member of the Extractive Metallurgy Chapter of Denver, and a member of the Colorado Mining Association.
d) I graduated from the Colorado School of Mines with BSc in Metallurgical Engineering. I have practiced my profession continuously since graduating and have more than 40 years’ experience.
e) 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 and relevant work experience, I fulfill the requirements of a “qualified person” for the purposes of NI 43-101.
f) I visited and inspected the subject property on December 10, 2019.
g) I participated and am responsible for sections 13 and 17, and portions of sections 1, 25, 26 and 27 of this Technical Report.
h) I satisfy all the requirements of independence according to NI 43-101.
i) I have read NI 43-101, Form 43-101 F1, and the Companion Policy to NI 43-101 (43-101 CP) and this Technical Report has been prepared in compliance with NI 43-101, Form 43-101 F1, and 43-101 CP.
j) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
k) I consent to the filing of the Technical Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Technical Report.

Dated May 8, 2020

“Randolph P Schneider, MMSAQP”-Signed
Signature of Qualified Person

Randolph P Schneider, MMSAQP
Print name of Qualified Person

CERTIFICATE OF AUTHOR

Kira Lyn Johnson, MMSAQP
Senior Geological Engineer of Tetra Tech
350 Indiana Street, Suite 500
Golden, Colorado 80401
Telephone: (303) 217-5700

I, **Kira Lyn Johnson, MMSAQP**, of Golden, Colorado do hereby certify:

a) I am a Senior Geological Engineer with Tetra Tech, Inc. with a business address of 350 Indiana St., Suite 500, Golden, CO 80401.

b) This certificate applies to the Technical Report titled “Preliminary Economic Assessment NI 43-101 Technical Report of the Velardeña Project, Durango State, Mexico” with an effective date of April 2020.

c) I have a Bachelor’s degree in Geological Engineering from South Dakota School of Mines and Technology. I am a QP member for the Mining and Metallurgical Society of America (Member No. 01539). I have over 11 years of professional experience. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

d) I inspected the property on December 10, 2019.

e) I am responsible for Section 14 of the report, as well as portions of sections 1-12, 25, 26, and 27.

f) I satisfy all the requirements of independence according to NI 43-101.

g) I have read NI 43-101, Form 43-101 F1, and the Companion Policy to NI 43-101 (43-101 CP) and this Technical Report has been prepared in compliance with NI 43-101, Form 43-101 F1, and 43-101 CP.

h) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

i) I consent to the filing of the Technical Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Technical Report.

Dated May 8, 2020.

“Kira Lyn Johnson, MMSAQP” - Signed
Signature of Qualified Person

Kira Lyn Johnson, MMSAQP
Print name of Qualified Person

APPENDIX A:
Metallurgical Testing

Outotec

Golden Minerals Company

BIOX® Batch Amenability Test Work Report on the Velardena Sample

Colorado; USA

"Outotec RSA (PTY) LTD"

Golden Minerals Company

350 Indian Street, Suite 800

Golden

Colorado

USA, 80401

Attn: Mr Warren Rehn

Re: BIOX® Batch Amenability Test Work on the Velardena Sample

Thank you once again for the opportunity to conduct the BIOX® test work on your 2019 Velardena sample. Please receive this report as an amended final report issued as "*Revision 01".* This report has an amendment in the acid generation characteristics determined for the Velardena 2019 sample which establishes the sample as net acid generating. The net acid generation determination presented in the Revision 00 issue contained an error in its determination. Please accept my apologies for this and I hope it will not cause an inconvenience. The remaining findings still stand and to present an overview:

- The Velardena sample is amenable to the BIOX® process;
- An average maximum sulphide oxidation of 93% has been achieved for the 2019 Velardena concentrate when exposed to a BIOX® batch pre-treatment period of 45 days;
- The Velardena sample is refractory. A gold dissolution of 6% was achieved on direct cyanidation of the concentrate. This was increased to 91% when exposed to BIOX® pre-treatment;
- An averaged batch cyanide consumption of 6 kg NaCN per tonne concentrate produced the averaged gold dissolution listed above;
- A pleasing set of silver dissolutions were also achieved, ranging between 53% to 95%. A silver dissolution of 39% was achieved on direct cyanidation of the concentrate;
- The iron to arsenic molar ratio of the concentrate was determined to be 9.7. This produced an acidic BIOX® liquor molar ratio for neutralisation of 7.3;
- Batch neutralisation of the acidic ferric-arsenate BIOX® solution produced environmentally stable residues fully compliant with the United States of America's Environmental Protection Agency's (USA EPA) guidelines for tailings disposal. These guidelines call for arsenic re-solubilisation of tailings residues to be < 5 mg/L. For the Velardena neutralisation tests, arsenic values of 0.06 and 0.07 mg/L were obtained were obtained. This is an extremely pleasing result;
- The Velardena concentrate was determined to be net acid generating during bio-oxidation with an average final batch lime consumption of 137 kg CaO per tonne of concentrate required for the sulphide oxidation up to 93%;

- Looking back on the 2007 Velardena results, a maximum sulphide oxidation of 96% was achieved with a concomitant maximum gold dissolution of 91% established at this oxidation extent;
- The 2007 sample was also quite refractory, achieving a gold dissolution of 4% on direct cyanidation of the concentrate;
- The silver dissolutions on this 2007 sample ranged from 52% to 96%;

While the 2007 and 2019 Velardena concentrate characteristics are different in that the 2007 sample contained an arsenic level at 11% compared to the 2019 sample arsenic of 4%, a sulphide level of 37% compared to 34% and 2007 gold and silver levels at 24 g/t and 198 g/t compared to 19 g/t and 124 g/t respectively – the overall Velardena campaign results for the two samples are pleasing showing that BIOX® can be a suitable pre-oxidative treatment technology.

We therefore invite you to review this report and welcome an opportunity to discuss this with you once you have perused the information contained herein.

For any additional information, please do not hesitate to contact me on +27 83 387 1959, or via my email at craig.vanbuuren@outotec.com.

Yours truly,
On behalf of
"Outotec RSA (PTY) LTD"

Craig van Buuren
Senior Process Engineer : BIOX®

TABLE OF CONTENTS

REVISION HISTORY

Revision No.	Revision date	Prepared	Revision text
00	30 January 2020	Crabuu	
01	07 February 2020	Crabuu	Acid generating

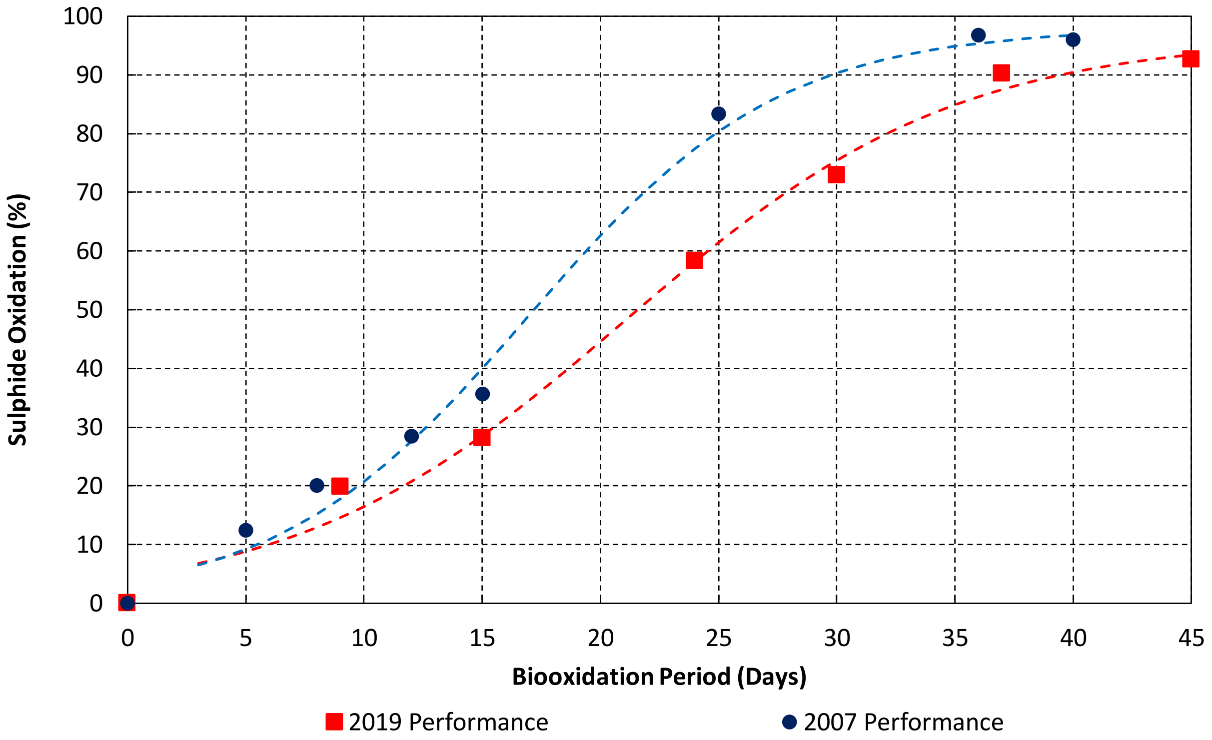
EXECUTIVE SUMMARY

The chemical analyses of the Velardena gold and silver bearing samples are shown in the accompanying table.

Species / Ratios	Unit	Velardena 2019 Sample	Velardena 2007 Sample
Au	g/t	18.5	23.6
Ag	g/t	124.0	197.8
Hg	g/t	1.14	
Si	%	3.45	
Fe	%	31.80	37.20
S(T)	%	37.50	41.40
S^{2-}	%	33.50	37.00
S^{o}	%	0.80	<0.50
SO_4^{2-}	%	0.48	
Sb	%	0.18	
As	%	4.39	11.00
C(T)	%	1.41	0.10
C(org)	%	0.07	0.10
CO_3^{2-}	%	6.48	<0.05
Solid SG		4.26	
Fe / As Molar Ratio		9.72	4.54
Au / S^{2-}		0.55	0.64
S^{2-} / CO_3^{2-}		5.17	> 740

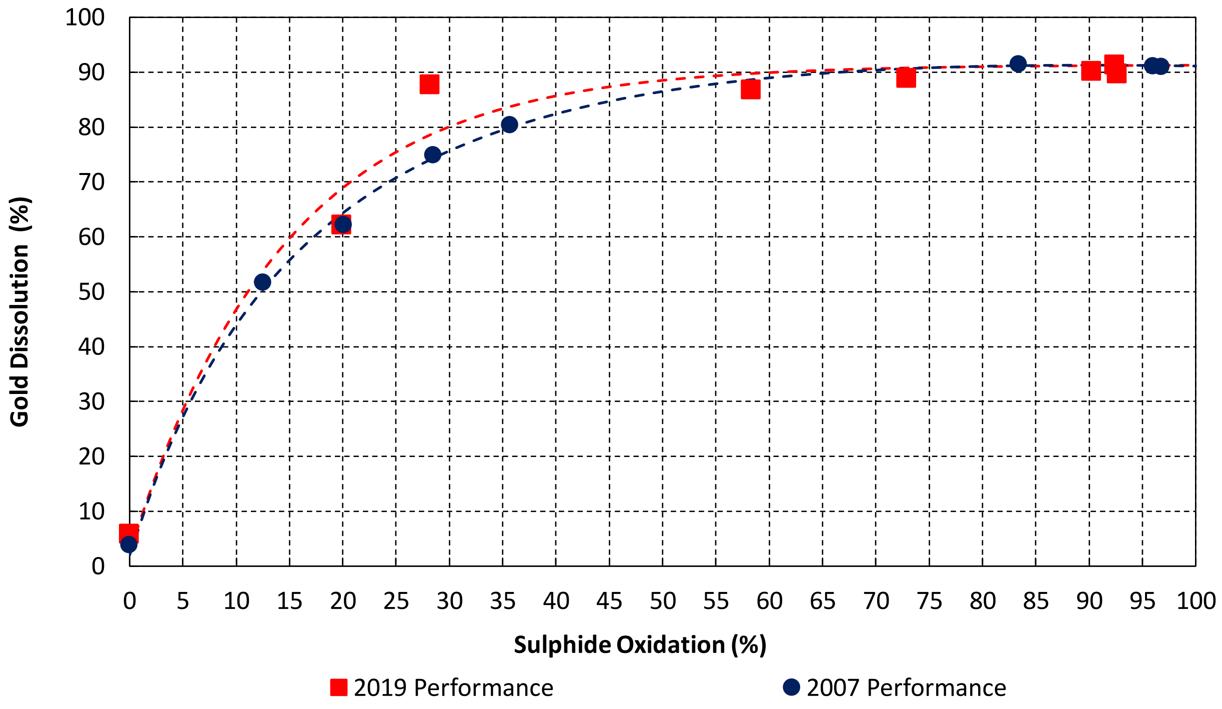
Particle size analyses of the 2019 Velardena sample showed the sample to have a top size of 100% passing 150 microns with 93% passing 75 microns.. The minimum grind for BIOX® is 80% passing 75 microns with < 1% plus 150 microns. The concentrate was treated “as-received”.

The adaptation of the BIOX® culture to the Velardena sample was very successful. The sulphide oxidation profile achieved on the Velardena sample is shown in the accompanying figure.



BIOX® Sulphide Oxidation Profiles of the Velardena Samples Described by the Logistic Model

The bio-oxidation of the Velardena concentrate increased the gold dissolution from 6% on the un-oxidised concentrate to 91% on well oxidised material. Other intermediate gold dissolutions from the leach tests spanned the range 62% to 90% on material with sulphide oxidations spanning the range 20% to 90%. For the Velardena concentrate it is apparent that the extent of sulphide oxidation is not too a significant parameter in maximising the gold dissolution as gold dissolutions already approach 88% at a moderate sulphide oxidation of 28%. The relationship between gold dissolution and extent of sulphide oxidation for the two samples are shown in the accompanying figure.



Gold Dissolution versus Sulphide Oxidation for the Velardena Concentrates

The cyanide and lime consumptions as well as other related performance data derived in the batch leach experiments are shown in the accompanying table.

BIOX® Period (days)	**Sulphide Oxidation (%)**	**Residue Metal Dissolution**		**Consumption**		**Au Accountability (%)**
		Gold (%)	**Silver (%)**	**NaCN (kg NaCN / t Concentrate)**	**Lime (kg CaO / t Concentrate)**	
0	0	5.80	38.62	5.46	0.41	95
9	19.9	62.13	90.52	6.49	2.82	95
15	28.2	87.72	94.44	7.24	7.10	99
24	58.3	86.74	94.98	5.81	4.44	100
30	72.9	88.84	92.03	7.55	4.35	104
37	90.2	90.14	53.22	5.01	5.73	97
45	92.6	89.70	91.25	6.24	4.04	94
45	92.4	91.33	88.51	4.83	4.95	90

The batch neutralisation test work showed that the iron and arsenic in the BIOX® liquor can be successfully removed from solution by a two-stage treatment with AR grade limestone to pH 5 followed by the addition of AR grade lime to pH 7. This could also be achieved by a single stage treatment with AR grade lime to pH 7.

The neutralised effluents in both these tests contained < 0.34 mg/L arsenic, satisfying the limit set by the United States of America's Environmental Protection Agency (USA EPA) which requires neutralized effluent streams to be discharged at 0.5 mg/L arsenic. The toxicity characteristic leach procedure (TCLP) testing of the neutralisation precipitates produced solution extracts containing 0.06 and 0.07 mg/L arsenic, well below the 5 mg/L limit prescribed by the USA EPA. The neutralisation results are shown in the table that follows.

Test Number	**Neutralisation Products**					
	Solids (%)		**Solution (mg/L)**		**TCLP (mg/L)**	
	Fe	**As**	**Fe**	**As**	**Fe**	**As**
Test 1 (AR Grade Limestone + AR Lime)	16.60	2.50	<0.02	<0.34	< 0.05	0.06
Test 2 (AR Grade Lime Only)	15.50	2.48	<0.02	<0.34	< 0.05	0.07

The neutralised precipitates as well as the neutralised solution effluents can therefore be considered suitable for disposal on a tailings storage facility.

Overall, a pleasing set of results have been achieved in the 2019 Velardena BIOX® batch campaign and importantly, the results achieved in this campaign are similar to that achieved in the 2007 test campaign.

The results of the two Velardena BAT campaigns therefore demonstrate that BIOX® can be an effective pre-oxidative treatment technology to improve the gold and silver dissolutions on the Velardena concentrate.

1. INTRODUCTION

At the request of David Dripps, Consulting Metallurgist of Golden Minerals, Outotec BIOMIN (Pty) Limited (Outotec) recently concluded a series of batch tests to evaluate the amenability of the Velardena gold and silver bearing flotation concentrate to the BIOX® technology.

The BIOX® process was developed for the pre-treatment of sulphide refractory ores and concentrates ahead of a conventional cyanide leach for gold recovery. The gold in these ores is encapsulated in sulphide minerals such as pyrite, arsenopyrite and pyrrhotite thus preventing the gold from being leached by cyanide. The BIOX® process destroys the sulphide minerals and exposes the gold for subsequent cyanidation, thereby increasing the overall gold recovery that can be achieved.

The BIOX® process is one of the three main refractory gold treatment technologies offered by Outotec, the others being roasting and pressure (autoclave) oxidation. The BIOX® process, however, has shown to have many real advantages, some of which includes:

- Improved gold recovery;
- Significantly lower capital cost;
- Low operating cost;
- Robust technology that is suited to remote locations;
- Low levels of skills required for operation;
- Environmentally friendly;
- Ongoing process development and improvement;
- Backed by more than 24 years of operating experience;
- 13 commercial installations across 4 continents with constant iterative cycling of plant information and improvements by the technology licensees;
- An established refractory technology which has seen the production of over 24 million ounces (Oz) over the last 25 years.

Figure 1.1 : Kirkland Lake Gold Fosterville (211 tpd) BIOX® Plant, Australia

Figure 1.2 : FCF Minerals Runruno (404 tpd) BIOX® Plant, Philippines

The heart of the BIOX® process is a mixed culture of naturally occurring microbes which, under controlled conditions, are able to oxidise gold-bearing sulphide ores or concentrates due to a chemolithotrophic mode of metabolism. This means that they require inorganic compounds for the acquisition of both energy and carbon.

The carbon requirements of the microbes for biosynthesis of cellular biomass are met by CO_2 in the atmosphere or from the dissolution of carbonate minerals in the ore.

The microbial culture in the BIOX® reactors is not controlled but rather allowed to adapt to the concentrate and operating conditions. It is, therefore, important to control the pH, temperature and other operating parameters to within certain prescribed ranges to maintain the right balance of microbial species to optimise the rate of oxidation.

The species, viz. *Acidithiobacillus* ferrooxidans, *Leptospirillum* ferrooxidans, *Leptospirillum* ferriphilum, *Ferroplasma* cupricumulans as well as many archaea species make up the dominant species of the BIOX® microbial consortia. Detailed laboratory and pilot plant studies have indicated that the microbes require a very acidic environment (pH 1.1 to 1.7), a temperature of between 35°C and 45°C and a steady supply of oxygen and carbon dioxide for optimum growth and activity. The unusual operating conditions, which are optimal for the BIOX® microbes, are not favourable for the growth of most other microbes, thus eliminating the need for sterility during the BIOX® process. The BIOX® microbes are non-pathogenic and incapable of causing disease. The microbes employed in the BIOX® process do not, therefore, pose a health risk to humans, animals, or plant life. The microbial oxidation reactions of the important sulphide minerals to achieve gold liberation may be summarised as follows:

Pyrite:	$4FeS_2 + 15O_2 + 2H_2O \rightarrow 2Fe_2(SO_4)_3 + 2H_2SO_4$
Arsenopyrite:	$2FeAsS + 7O_2 + H_2SO_4 + 2H_2O \rightarrow 2H_3AsO_4 + Fe_2(SO_4)_3$
Pyrrhotite:	$4FeS + 9O_2 + 2H_2SO_4 \rightarrow 2Fe_2(SO_4)_3 + 2H_2O$

The oxidation reactions are also highly exothermic. In addition to the direct oxidation of sulphide minerals, several indirect chemical and microbial assisted reactions occur as follows:

Ferric Oxidation of Pyrrhotite and Pyrite:	$FeS + Fe_2(SO_4)_3 \rightarrow 3FeSO_4 + S^o$ $FeS_2 + Fe_2(SO_4)_3 \rightarrow 3FeSO_4 + 2S^o$
Acid Dissolution of Pyrrhotite:	$2FeS + 2H_2SO_4 + O_2 \rightarrow 2FeSO_4 + 2S^o + 2H_2O$
Ferrous Oxidation:	$4FeSO_4 + 2H_2SO_4 + O2 \rightarrow 2Fe_2(SO_4)_3 + 2H_2O$
Elemental Sulphur Oxidation:	$2S^o + 3O_2 + 2H_2O \rightarrow 2H_2SO_4$

Laboratory-based testing provides a valuable indication of the relationship between metal dissolution and sulphide oxidation and the tests are useful in determining preliminary batch reagent consumptions that can be expected as well as providing a benchmark for comparing the relative performance of future variable concentrate samples.

The BIOX® BAT tests on the Velardena concentrate were conducted in Johannesburg at the SGS South Africa Randburg laboratory.

2. OBJECTIVES OF THE BAT PROGRAM

The objectives of the BIOX® BAT test program are summarised as follows:

Standard BIOX® Amenability and Cyanidation Performance

i. To adapt the BIOX® mesophile culture to the Velardena concentrate sample;
ii. To determine the acid / lime requirements of the concentrate during batch BIOX® treatment;
iii. To determine the extents of sulphide oxidations of the concentrate after different BIOX® treatment periods;
iv. To conduct batch cyanidation test work on the BIOX® product samples;
v. To determine the reagent consumptions during batch BIOX® and CIL treatment.

Ancillary BIOX® Test Work

i. To determine the efficiency of arsenic removal and reagent consumption during the batch neutralisation of the BIOX® liquor;
ii. To determine the stability of arsenic precipitates formed during neutralisation of this liquor;
iii. To determine the optimum flocculant addition, settling rate and estimated required thickener area for selected product slurries through static liquid solid separation test work.

3. EXPERIMENTAL PROCEDURE

3.1. Sample Receipt, Preparation and Inoculum Adaptation

During April 2019, approximately 10 kg of Velardena flotation concentrate was received at the SGS South Africa laboratories. The sulphide concentrate sample was to be used in a BIOX® batch amenability test (BAT) work program. The received sample mass is described in the following table.

Table 3.1 : Samples Received for the BIOX® BAT Program

Description I	Description II	Mass (kg)
Conc Fe	CC Vein	4
Conc Fe	C1 Vein	1
Conc Fe	San Mateo	5

The samples were combined, thoroughly blended and dried at 50ºC. The dried bulk concentrate was screened through an 850 micron screen and thereafter split using a rotary splitter into sub-samples and submitted for chemical and particle size analyses.

An Outotec BIOX® microbial stock culture which was adapted to a pyrite / arsenopyrite substrate was used as the inoculum source for the BAT program. The adaptation and inoculum build-up stages were conducted as follows:

- A 9 K nutrient media (contains an iron substrate as ferrous ions) was used and 250 g of the concentrate was slurried to a 7.7% solids concentration and adjusted to a pH of 1.35 with concentrated sulphuric acid and allowed to stabilise overnight;
- The slurry was thereafter inoculated with a ~10% active culture (V_{INOC}/V_{TOTAL});
- The batch test was maintained between 38 °C and 42 °C and was constantly agitated and aerated;
- Daily pH adjustments (through the addition of lime or acid) were made to maintain the pH range between pH 1.2 to 1.4 while ferrous, ferric and slurry potential measurements were taken daily;
- Once the microbial activity started increasing, evidenced by increasing ferric concentrations and decreasing ferrous concentrations, further 250 g concentrate aliquots were added to finally reach a solids concentration of 20%;
- Upon reaching a ferrous concentration of < 0.1 g/L in the adaptation stage, approximately 300 ml of slurry was transferred to a 3-litre beaker containing a 20% (Velardena) concentrate slurry pH stabilised to a pH of 1.40;

- This marked the commencement of the inoculum build-up phase and a 0 K nutrient media (no iron substrate) was used in the stage;
- Identical monitoring methods were utilised as for the adaptation stage.

3.2. Batch Amenability Tests (BATS)

Seven batch bio-oxidation tests were performed on the Velardena concentrate. The objective was to determine the rate and extent of sulphide oxidation and the corresponding metal dissolutions achievable. The tests were conducted in 5-litre vessels on a multi stirrer BAT bench to ensure similar conditions.

The batch tests were carried out under the following conditions:

Liquid/Solid ratio:	4:1;
Nutrient Medium:	0 K (Appendix 7.1);
Slurry Stabilisation:	98% H_2SO_4 to pH 1.40 with constant agitation;
Solids Concentration	20%;
Inoculum:	~10% (V_{INOC}/V_{Total}) addition;
pH:	pH 1.20 to 1.30 (using acid or lime as required);
Temperature:	40°C ± 2°C;
Aeration:	Target dissolved O_2(aq) concentration ≥ 2.0 mg/L;
Agitation:	Pitch blade turbine at approximately 460 rpm;
Monitoring:	Fe^{2+} and Fe^{3+} titrations, pH, Redox and O_2(aq);
Bio-oxidation period:	Various.

On completion of the batch tests, the slurries were filtered and washed with water acidified to a pH of 1.6 using H_2SO_4. The filter cakes were then finally re-washed with potable water, dried at 50°C, crushed and screened and then submitted for post bio-oxidation chemical analyses and cyanide leach (bottle roll) tests. The bio-oxidation liquors (filtrates) were also submitted for chemical analyses. All equations used in the calculations are given in Appendix 7.2. The amenability test log sheets are included in Appendix 7.3.

3.3. Cyanidation Tests

Batch cyanidation leach tests were conducted on all the bio-oxidised products using the bottle roll method. Standard tests were conducted and are described as follows:

- Slurries were prepared at a liquid/solid (L/S) ratio of 4;
- The pH was adjusted to approximately pH 11.0 by the addition of a lime slurry and the final slurry pre-conditioned for 30 minutes to 1 hour;
- This was followed by the addition of 20 g/L activated carbon and a cyanide addition range of 21 to 25 kg NaCN per tonne solids;

- Leaching was allowed to progress for 24 hours. Lime slurry (100 g/L) was used when necessary to maintain the pH in the range 10.5 to 11.0;
- After completion of the leach, the loaded carbon was removed by careful screening and the cyanidation slurries filtered, the filter cake was water washed and dried at 60 °C;
- Filtrates were analysed (by manual titration) for residual cyanide and lime;
- The cyanidation residues, carbon and filtrates were analysed for gold. The gold dissolutions were determined after correction for mass changes during cyanidation.

Cyanidation equations are detailed in Appendix 7.4. The cyanidation test log sheet is included in Appendix 7.5.

3.4. Batch Neutralisation Tests

Two batch neutralisation tests were conducted on the BIOX® liquors obtained from the tests whiech yielded the highest sulphide oxidation extents (BATs 1 and 2). The neutralisation tests were conducted as follows:

i. **Test 1**: A two-stage neutralisation test where analytical (AR) grade limestone is added during Stage 1 to raise the pH to 5. Stage 2 sees the addition of AR grade lime to increase the pH to 7;
ii. **Test 2**: A single stage neutralisation test where only AR grade lime is used to raise the pH gradually to 5, and thereafter gradually to 7.

The batch neutralisation test procedure is described as follows:

- BIOX® liquor was diluted with potable water in the ratio 1 part liquor to 2 parts water to simulate the conditions of the neutralisation circuit's feed (Counter current decantation number 1 (CCD#1) overflow);
- The experiments were conducted in 5 litre beakers with constant aeration and at ambient temperature;
- The AR grade limestone was prepared to a concentration of 200 g/L while the AR grade lime was prepared to a concentration of 100 g/L;
- Reagent additions were made slowly and after each addition, a period was allowed for the reaction to proceed prior to withdrawing samples;
- During each test, solution samples were removed at different pH values, filtered and the solutions were analysed for Fe and As.
- Arsenic stability tests were performed on the final precipitates formed (conclusion of the experiments). The stability(1) tests were conducted according to the standard Toxicity Characteristic Leaching Procedure (TCLP) and the multiple extraction(2) method adopted by the USA Environmental Protection Agency (USA EPA).

3.5. Static Settling Tests

Laboratory scale static settling tests were conducted on samples of fully oxidised bio-oxidation liquor slurries. The tests were conducted to determine:

i. A preferred flocculant;
ii. The settling rate at different flocculant dosages.

The settling rate test work was comprised of two types of tests viz. the flocculant selection tests and the flocculant optimisation tests. The flocculant selection tests were carried out by flocculation of slurry aliquots in a 100 ml glass measuring cylinder and involved the flocculant screening of standard flocculants.

The procedure used is summarised as follows:

- 33 ml of slurry was transferred to a 100 ml measuring cylinder;
- The slurry was topped up to 100 ml using potable water;
- All the solids were suspended by inversion of the cylinder and a known volume of flocculant (concentration of 100 g/t) was rapidly injected;
- Without delay, the cylinder was violently inverted twice and then once gently;
- The cylinder was placed on a horizontal surface and the timer started;
- The mud line was then marked on a masking tape strip (placed on the outside wall of the cylinder) at various intervals;
- The tape strip was then removed and the settling rate calculated from the mud line measurements.

The dosage optimisation settling rate tests were carried out by flocculation of larger slurry aliquots in a 500 ml glass measuring cylinder using different dosages and thereafter applying the same procedure described earlier. Stock solutions were initially made up to 250 g/L and then diluted down to 25 g/L for the actual test work.

4. RESULTS AND DISCUSSION

4.1. Concentrate Analyses

The chemical analyses of the Velardena gold-bearing flotation concentrates are shown in the accompanying table.

Table 4.1 : Velardena Concentrate Analyses

Species / Ratios	Unit	Velardena 2019 Sample	Velardena 2007 Sample
Au	g/t	18.5	23.6
Ag	g/t	124.0	197.8
Hg	g/t	1.14	
Si	%	3.45	
Fe	%	31.80	37.20
S(T)	%	37.50	41.40
S^{2-}	%	33.50	37.00
S^{o}	%	0.80	<0.50
SO_4^{2-}	%	0.48	
Sb	%	0.18	
As	%	4.39	11.00
C(T)	%	1.41	0.10
C(org)	%	0.07	0.10
CO_3^{2-}	%	6.48	<0.05
Solid SG		4.26	
Fe / As Molar Ratio		9.72	4.54
Au / S^{2-}		0.55	0.64
S^{2-} / CO_3^{2-}		5.17	> 740

The table shows that the concentrate contains no deleterious elements which upon dissolution may inhibit the microbial culture. The arsenic level of 4.4% is low and with an iron content of 31.8%, the iron to arsenic molar ratio is determined to be 9.7.

A sub-sample of the bulk concentrate was also submitted for a particle size distribution (PSD). Table 4.2 shows the average PSD for the sample while Figure 4.1 thereafter depicts the data graphically.

Table 4.2 : Particle Size Distribution of the 2019 Velardena Sample

Particle Size [µm]	Cumulative % Passing
150	100.0
106	99.0
90	97.0
75	93.0
53	80.0
45	78.0
38	67.0

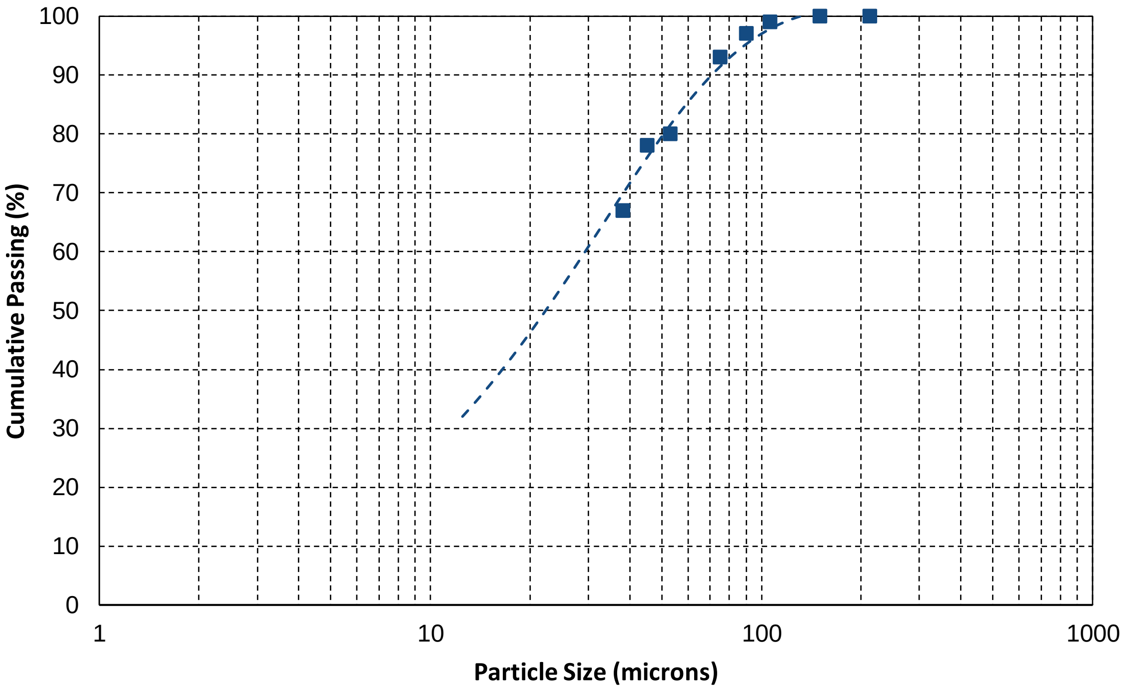


Figure 4.1 : Particle Size Analyses of the 2019 Velardena Sample

Particle size analyses of the Velardena sample showed the sample to have a top size of 100% passing 150 microns with 93% passing 75 microns. The minimum grind for BIOX® is 80% passing 75 microns with < 1% plus 150 microns. The Velardena sample was treated “as received”.

4.2. Batch Amenability Tests (BATs) : Velardena Campaign

The adaptation, inoculum build-up and mesophile batch amenability test log sheets are provided in Appendix 7.3. Figures 4.2, and 4.3 that follow show the progression of the inoculum adaptation and build-up phases.

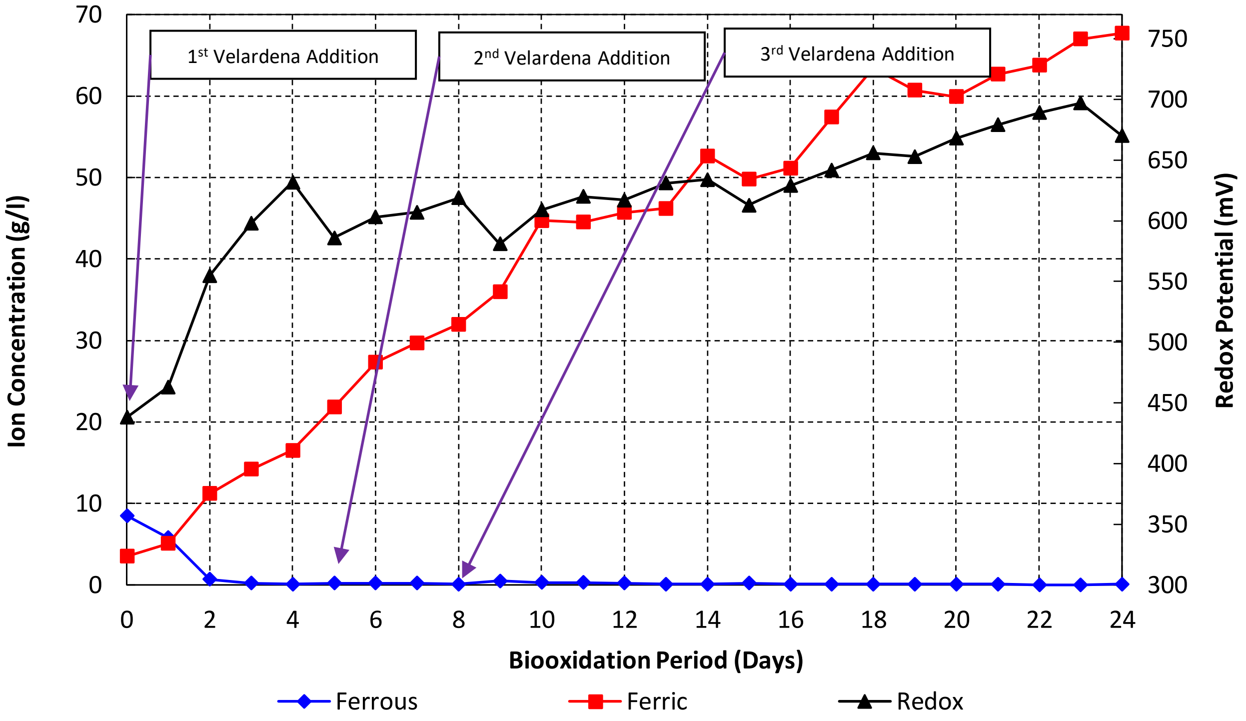


Figure 4.2 : Progression of the Microbial Adaptation for Velardena

The adaptation phase progressed very well with the successive concentrate aliquots post inoculation added on Days 5 and 8 totalling 750 grams of Velardena concentrate (20% solids). The ferrous concentration on Day 10 was measured at ~ 0.1 g/L indicating that the adaptation phase was almost complete. On 02 October an aliquot of this culture was drawn and inoculated in to a fresh batch of Velardena slurry to commence the inoculum build-phase.

Figure 4.3 shows the inoculum build-up phase progression.

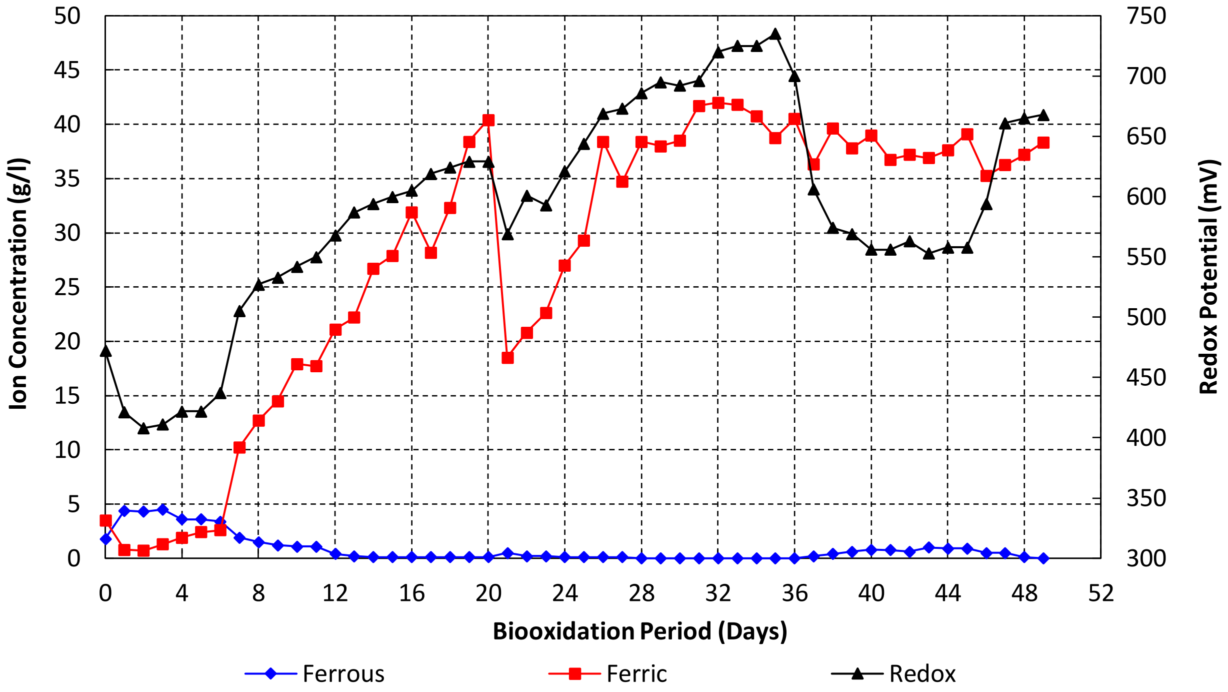


Figure 4.3 : Microbial Inoculum Build-up Progression for Velardena

The Velardena inoculum build-up progressed well. After a 6-day period of dormancy, the microbial activity started to increase evidenced by the ferrous levels dropping to < 2 g/L while the redox potentials increased concomitantly. This trend continued and by Day 10, a ferrous level of 1.1 g/L was recorded with a corresponding ferric concentration of 18 g/L. This corresponded to a slurry potential of 542 mV on the same day. By Day 20 the slurry redox potential reached 629 mV and inoculum was drawn to inoculate the formal seven discrete batch tests. Note the decrease in slurry potential on Day 37 corresponds to a fresh solid substrate addition to the inoculum to maintain the activity of the culture.

The formal Velardena BIOX® BAT program commenced on 02 October 2019 with the inoculation of the 7 discrete batch tests. The tests were inoculated with inoculum drawn from the build-up phase of approximately 300 ml of the Velardena adapted culture into a slurry prepared by using 3 litres of 0K nutrients and 750 grams of concentrate. The figures that follow show the measured parameters of the BATs depicted graphically. Daily parameter measurements of all the BATs are given in Appendix 7.3.

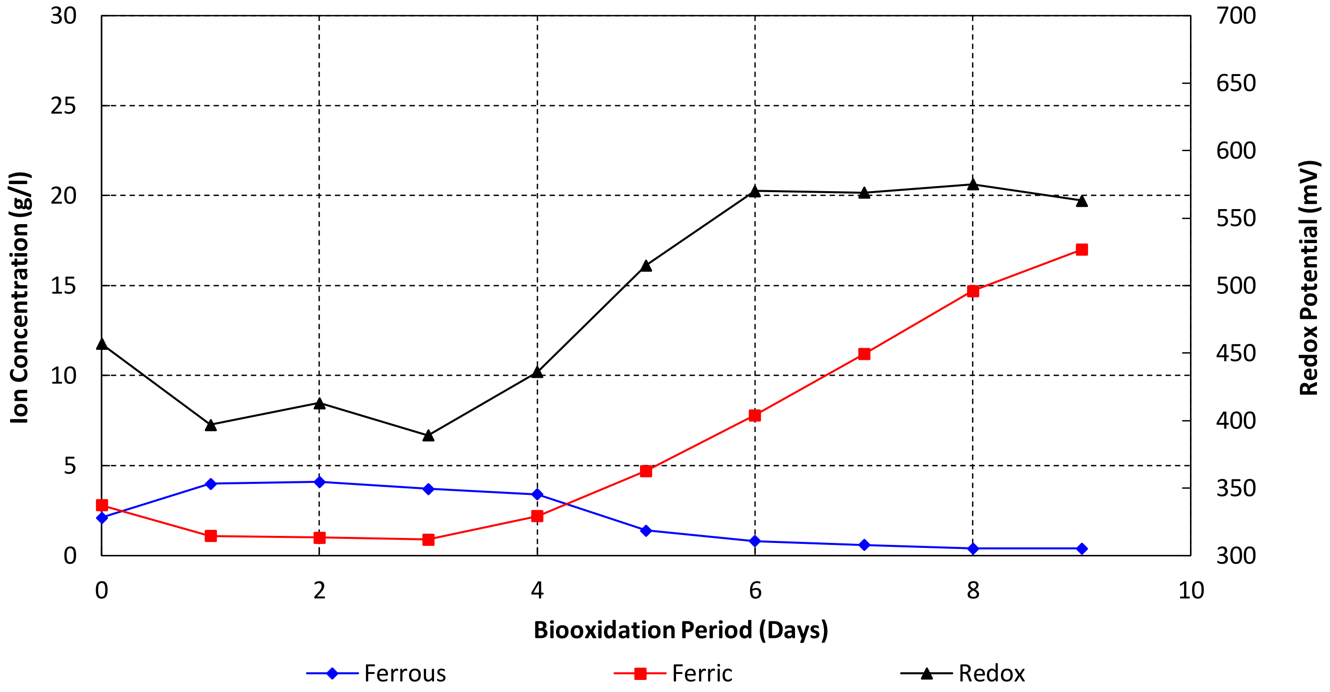


Figure 4.4 : Progression of Velardena BAT 6 (9 Days of Bio-oxidation)

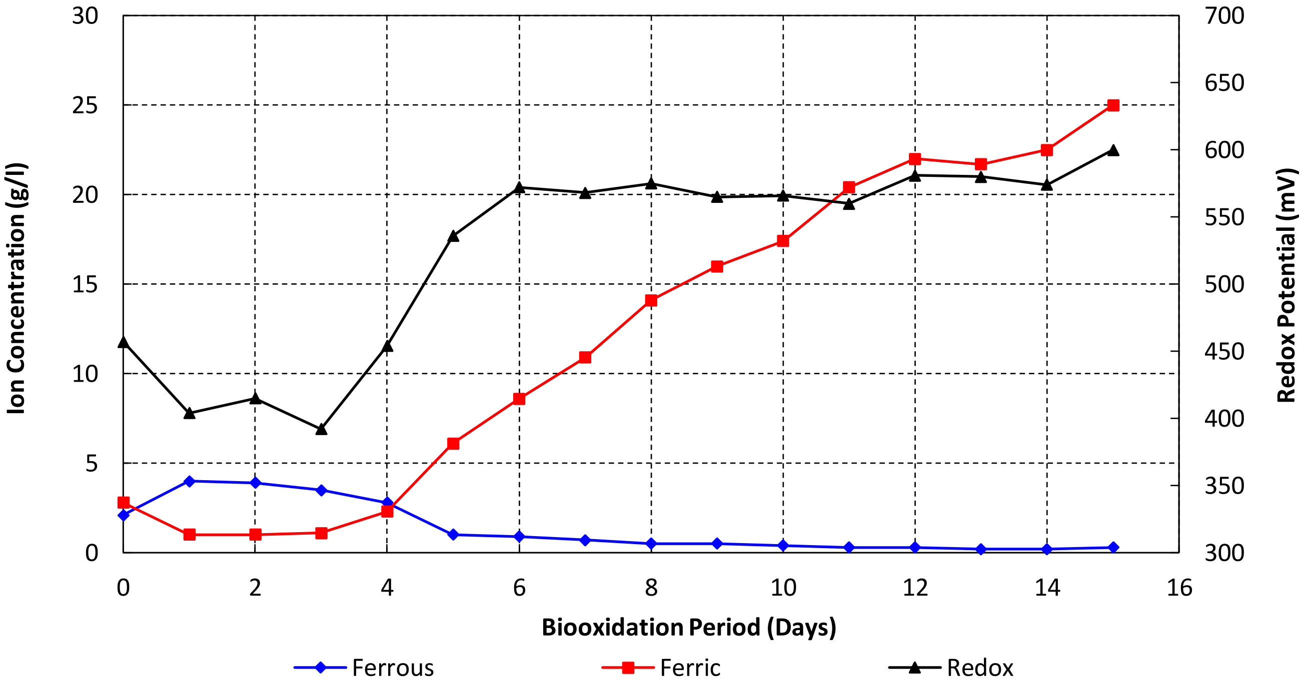


Figure 4.5 : Progression of Velardena BAT 5 (15 Days of Bio-oxidation)

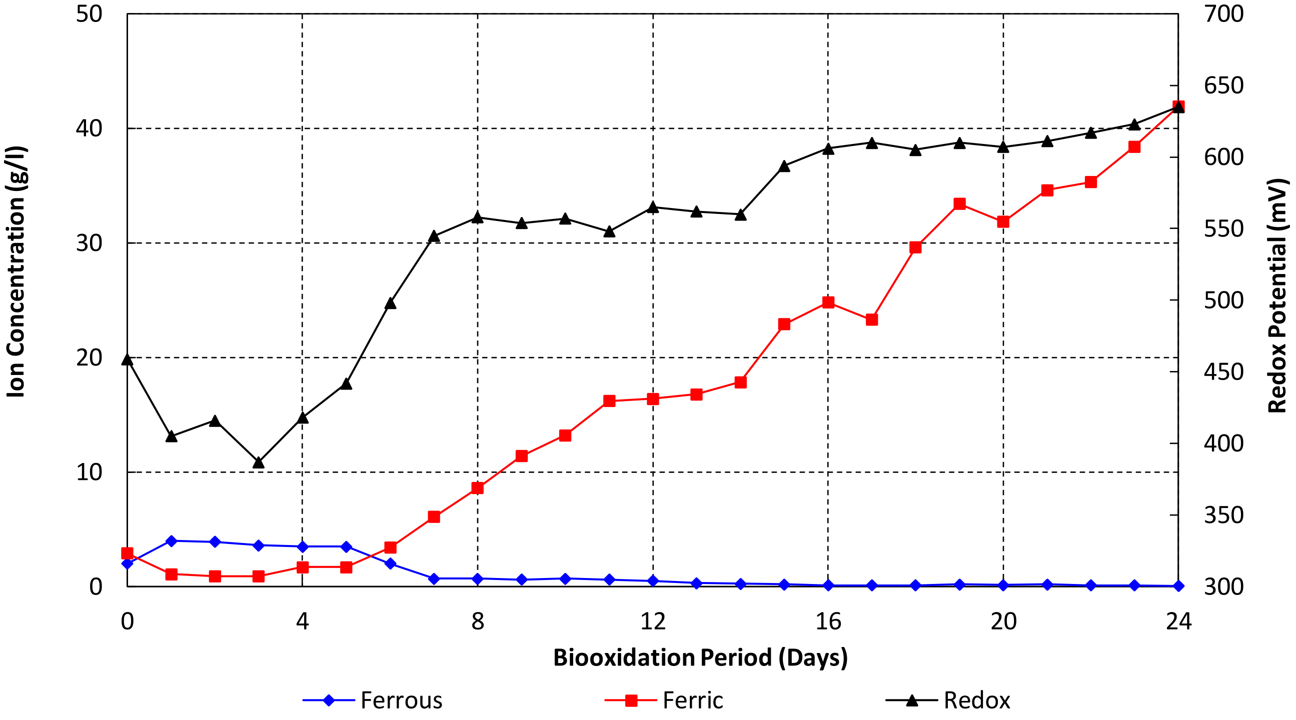


Figure 4.6 : Progression of Velardena BAT 7 (24 Days of Bio-oxidation)

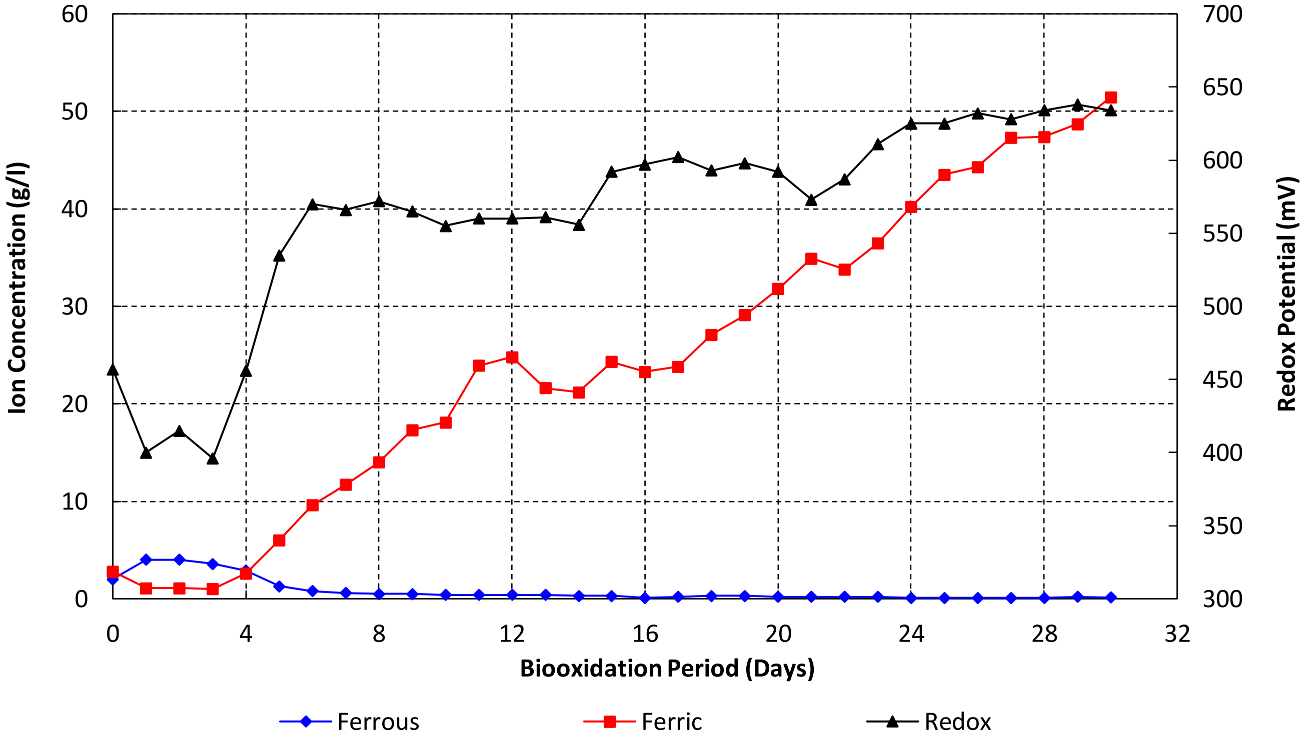


Figure 4.7 : Progression of Velardena BAT 4 (30 Days of Bio-oxidation)

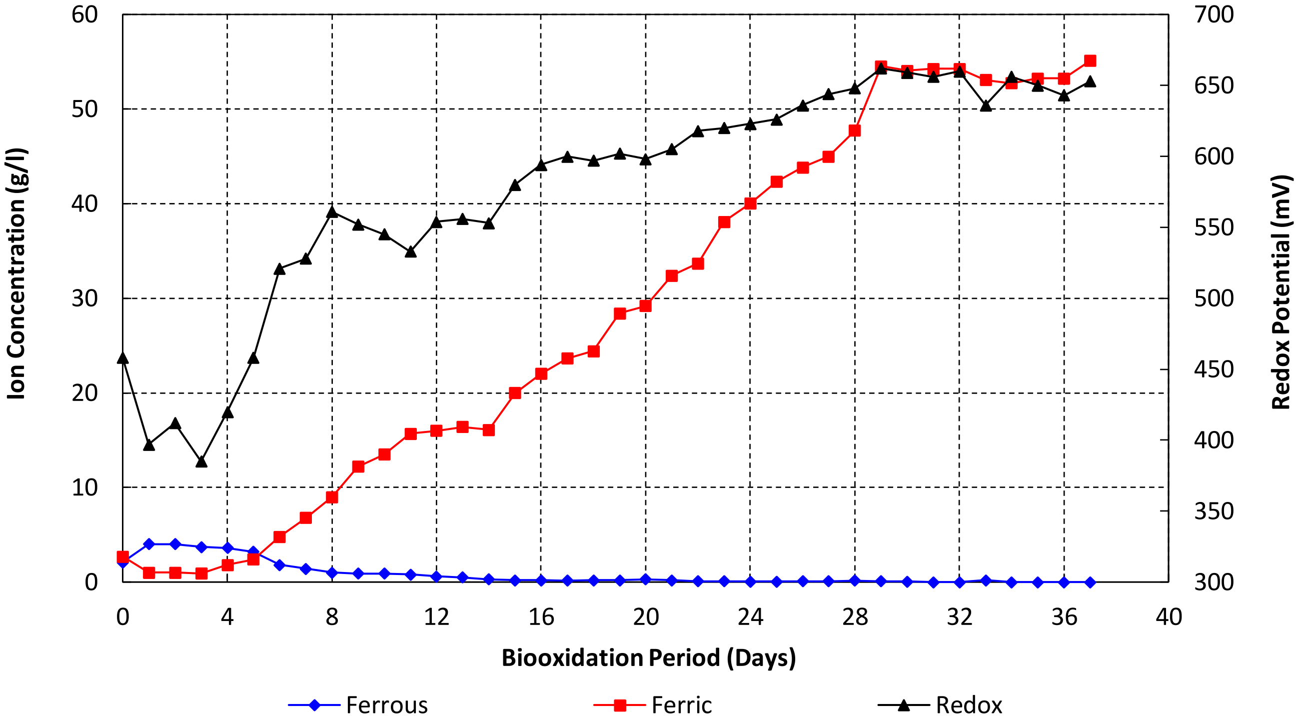


Figure 4.8 : Progression of Velardena BAT 3 (37 Days of Bio-oxidation)

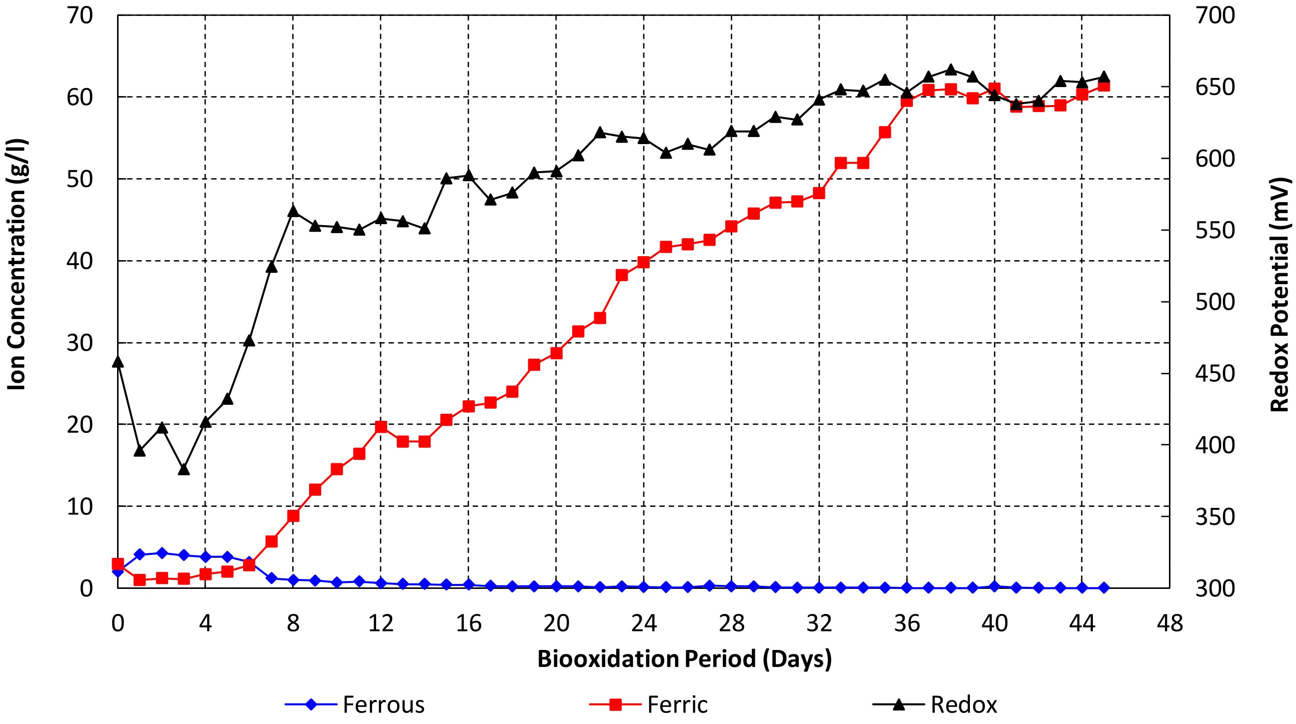


Figure 4.9 : Progression of Velardena BAT 1 (45 Days of Bio-oxidation)

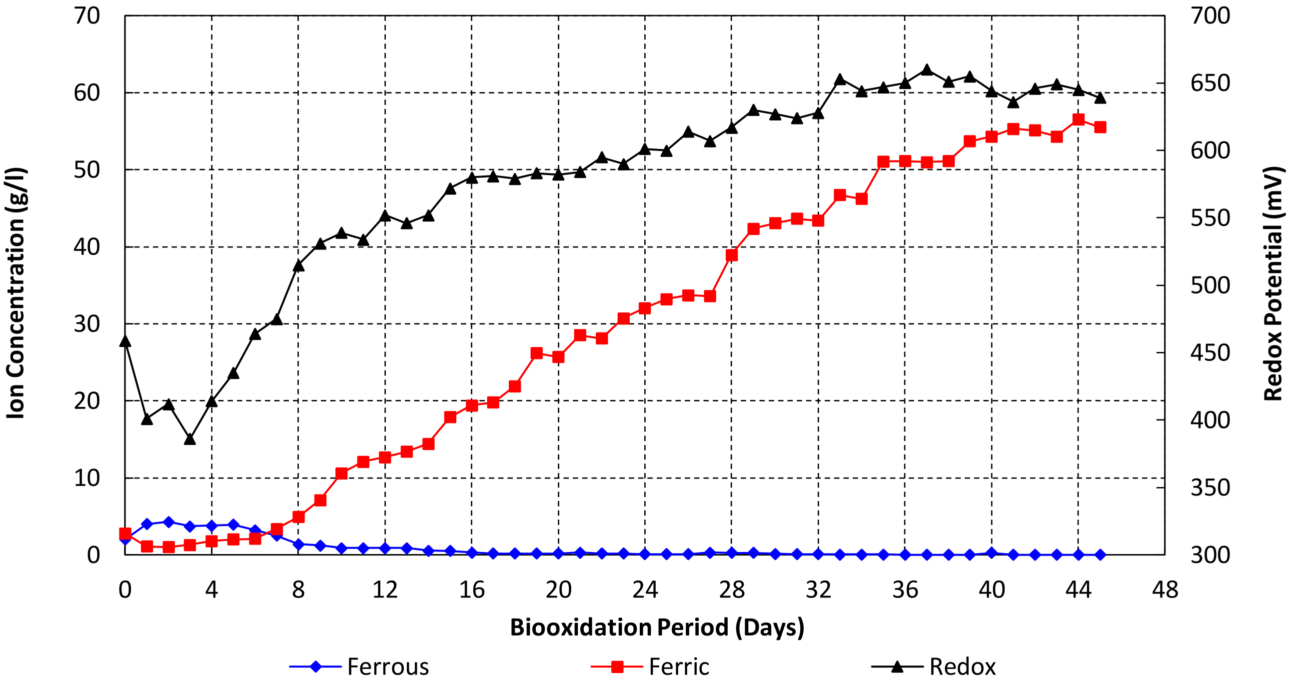


Figure 4.10 : Progression of Velardena BAT 2 (45 Days of Bio-oxidation)

The seven discrete BAT tests on the Velardena sample performed well. A power outage occurred on the 7th of November 2019 and lasted for approximately 8 hours. During this period no agitation and aeration was possible but no impact to microbial activity was seen as by the following day ferrous values were lower and redox potentials had increased. Slight decreases observed in the ferric ion concentrations as the tests progressed were due to the precipitation of iron salts (mainly as jarosite) from the solution when lime was added to control the slurry pH. The mass changes and acid / lime requirements during bio-oxidation follow.

Table 4.3 : Mass Change and Acid / Lime Requirements for Velardena

Biooxidation Treatment Period (days) / (Test)	Mass Change During BIOX® (%)	Initial Acid Addition (kg/t feed)	Total Acid Addition (kg/t feed)	Lime Consumption (kg/t feed)	Net Acid Consumption / Generation[1] (kg/t feed)
9 / BAT Test 6	-12.72	175	190	13	167
15 / BAT Test 5	-5.73	175	190	72	65
24 / BAT Test 7	-4.11	176	190	140	-54
30 / BAT Test 4	3.04	175	189	180	-125
37 / BAT Test 3	4.55	175	190	193	-148
45 / BAT Test 1	-4.47	175	190	182	-129
45 / BAT Test 2	-3.52	176	190	192	-146

Note : (1) A positive value indicates acid consumption and a negative value acid generation.

Mass loss values of between 4% and 13% were recorded in certain BAT tests while mass gain values of 3% and 5% were also measured. The mass gains can be attributed to the lime additions made to regulate the pH. These were subsequently reduced in order to maintain a lower slurry pH as in certain cases, the degree of precipitation can influence metal recoveries in the leach. It follows that as the BAT tests progressed, more pyrite mineral solubilisation would have occurred calling for more lime additions so with a lower pH range selected the metal and salt precipitation could be managed to be not too excessive.

As evidenced in Table 4.3, the bio-oxidation of the concentrate was overall net acid generating. This is as expected and in line with what was achieved in the 2007 Velardena BAT campaign. The high acid consumption in the early stages of the programme can be attributed to the carbonate mineral dissolution during the pH adjustment and stabilisation phase. Arsenopyrite and carbonate minerals are acid consuming and would also have partially offset the acid generation arising from the pyrite oxidation.

The chemical analyses of the bio-oxidation residues and solutions are summarised in the tables that follow.

Table 4.4 : Analyses of the Velardena Bio-oxidation Product Solids(1)

BAT Test	Period (days)	Species (%)								
		S(T)	S^{2-}	S^0	Fe(T)	CO_3^{2-}	As	C(T)	C_{org}	Si
FEED	0	37.50	33.50	0.80	31.80	6.48	4.39	1.41	0.07	3.45
BAT Test 6	9	29.94	25.22	0.44	22.08	0.41	1.36	0.21	0.06	3.35
BAT Test 5	15	30.17	22.63	<0.3	17.06	0.67	0.39	0.32	0.08	3.47
BAT Test 7	24	24.26	13.14	0.85	12.08	1.09	0.25	0.43	0.05	3.38
BAT Test 4	30	20.71	8.53	<0.3	10.61	1.56	0.29	0.50	0.05	3.50
BAT Test 3	37	17.77	3.07	<0.3	6.27	1.67	0.21	0.50	0.05	3.37
BAT Test 1	45	15.76	2.32	2.14	5.22	1.66	0.26	0.53	0.08	3.43
BAT Test 2	45	16.40	2.40	2.28	5.08	1.64	0.20	0.53	0.06	3.41

Note: (1) Corrected for mass change.

Table 4.5 : Analyses of the Velardena Bio-oxidation Product Solutions

BAT Test	Period (days)	Species					
		Fe (T) (g/L)	Fe(II) (g/L)	Fe(III) (g/L)	As (g/L)	S(T) (g/L)	Fe:As Molar Ratio
BAT Test 6	9	17.40	0.40	18.10	8.17	20.30	3.04
BAT Test 5	15	25.30	0.30	26.70	7.99	21.70	4.53
BAT Test 7	24	41.90	0.05	57.35	10.90	48.50	7.06
BAT Test 4	30	51.45	0.15	54.15	8.72	42.20	8.35
BAT Test 3	37	56.92	0.00	54.88	10.33	43.01	7.13
BAT Test 1	45	61.37	0.00	63.70	11.70	48.30	7.30
BAT Test 2	45	55.50	0.00	57.00	10.50	45.50	7.28

Table 4.5 shows that the Fe:As molar ratios of the final BAT BIOX® liquors were all in excess of the recommended value of 3 which is typically required to ensure environmentally stable ferric arsenate will be formed during neutralisation. This was expected owing to the iron and arsenic contents of the concentrate. The analyses of the BAT residue product solids generally indicated a decrease in the iron, sulphide sulphur and arsenic concentrations over time while the inverse trend can be observed in the solution analyses. Figures 4.11 through to 4.13 show these trends.

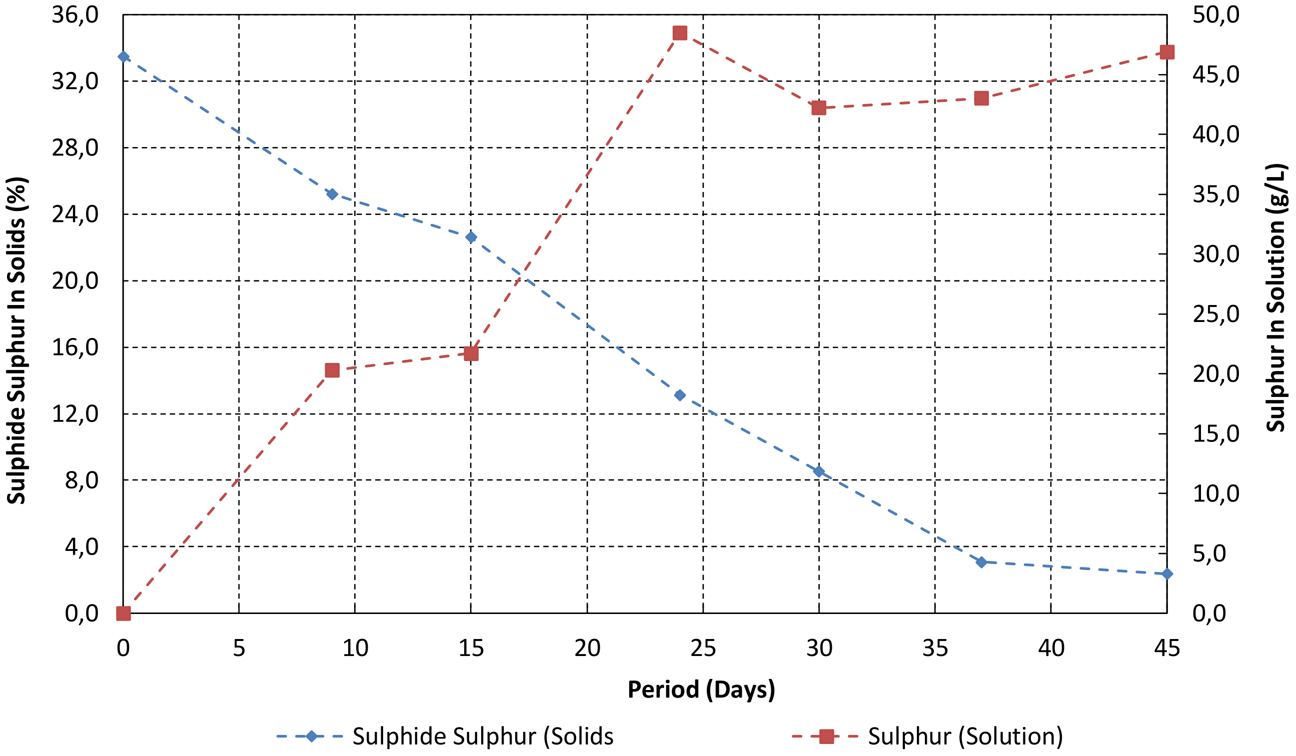


Figure 4.11 : Sulphide Sulphur (Solids) and Total Sulphur (Solution) Profile

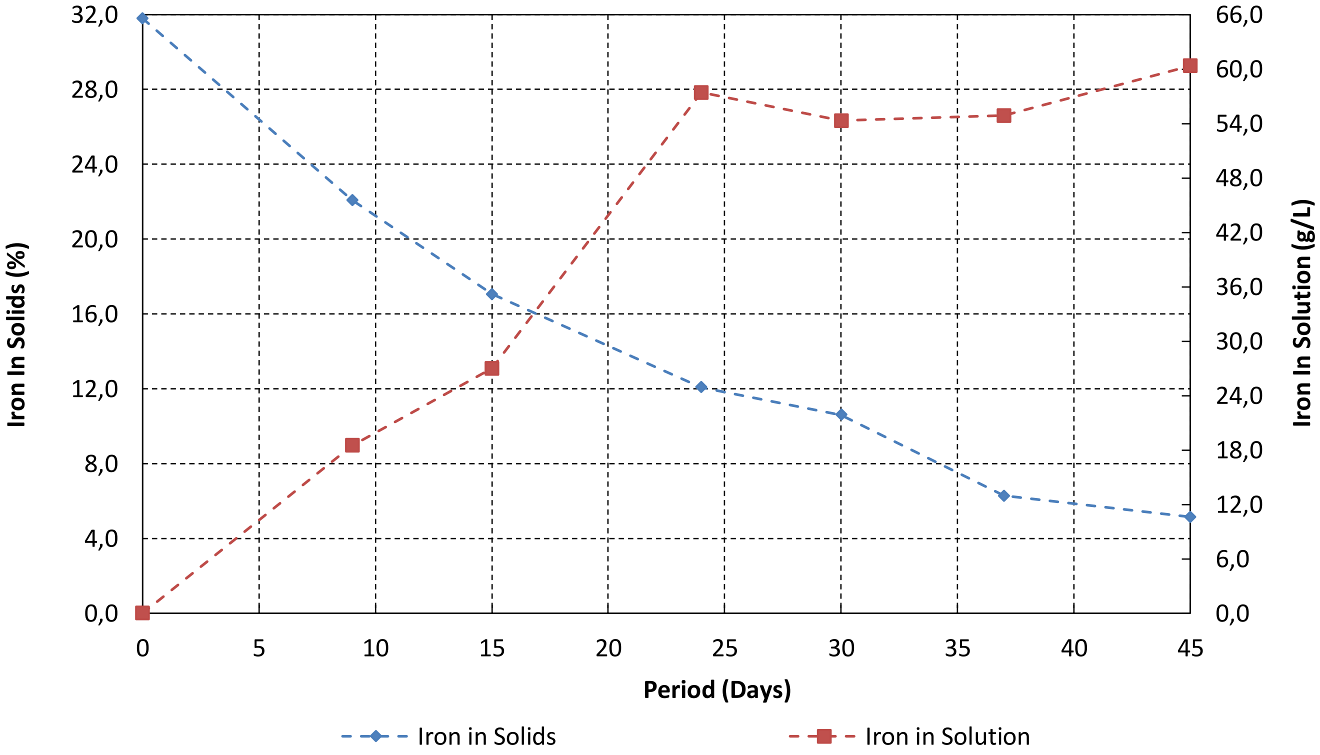


Figure 4.12 : Iron (Solids) and Total Iron (Solution) Profile

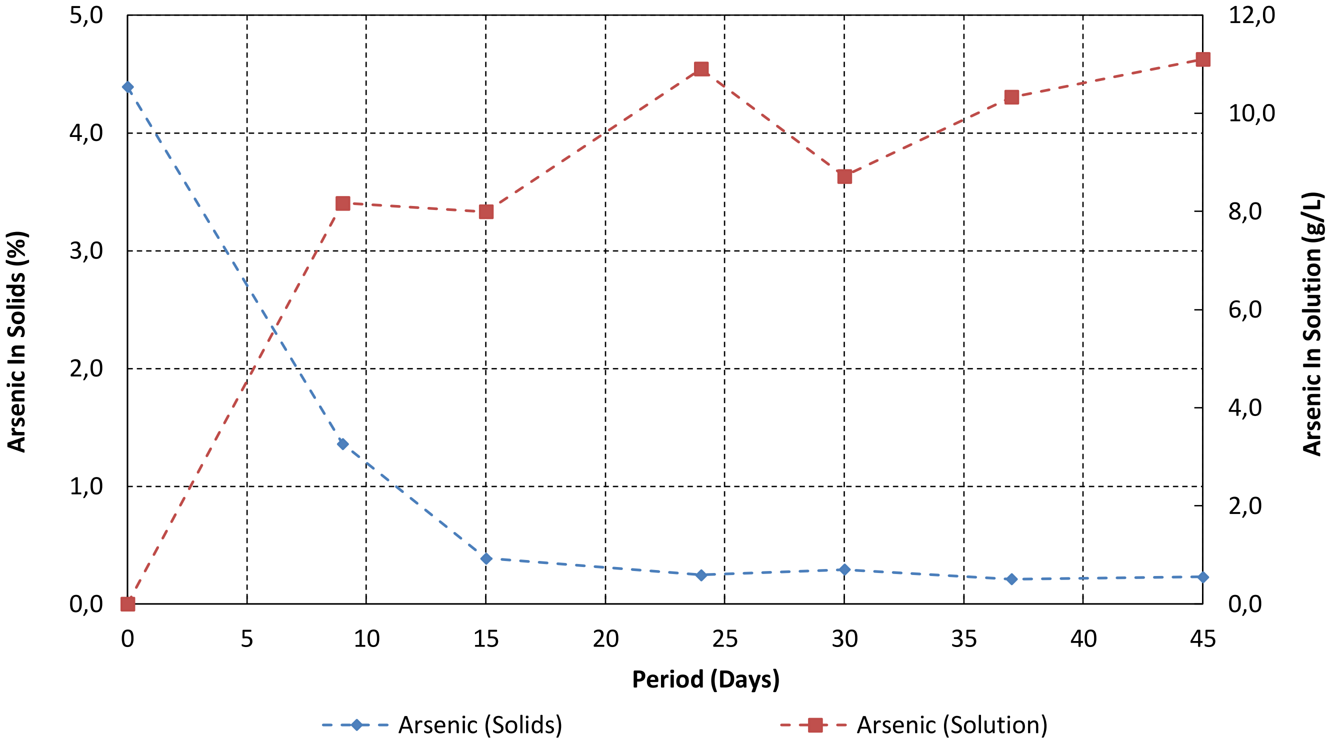


Figure 4.13 : Arsenic (Solids) and Total Arsenic (Solution) Profile

The extent of sulphide sulphur oxidation and metal solubilisations achieved for each discrete Velardena BAT test is shown in the accompanying table. The sulphide oxidation profile is also graphically depicted in the figure thereafter.

Table 4.6 : Velardena Sulphide Oxidations, Iron and Arsenic Solubilisations

BAT Test	Period (days)	Sulphide Oxidation (%)	Arsenic Solubilisation (%)	Iron Solubilisation (%)
BAT Test 6	9	19.9	65.7	26.0
BAT Test 5	15	28.2	90.2	42.9
BAT Test 7	24	58.3	93.8	59.5
BAT Test 4	30	72.9	92.7	64.5
BAT Test 3	37	90.2	94.7	79.0
BAT Test 1	45	92.6	93.5	82.5
BAT Test 2	45	92.4	94.9	83.0

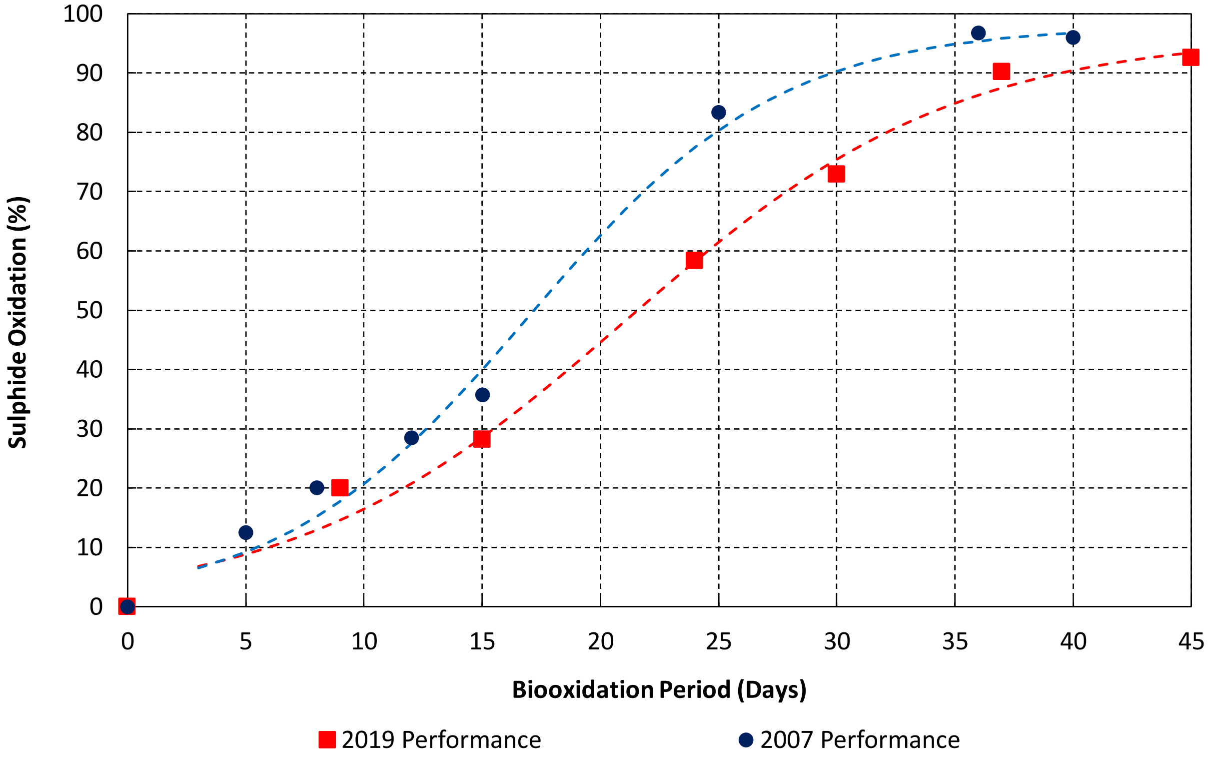


Figure 4.14 : Sulphide Oxidation Profiles of the Velardena Samples I

The sulphide oxidation average of the last two (duplicate) BAT tests was 93%, achieved over a 45-day BIOX® pre-treatment period. The treatment period to achieve the same extent of sulphide mineral oxidation as that achieved under batch conditions for continuous operation is always much quicker.

A commercial BIOX® plant treating the Velardena sample can therefore be expected to achieve similar oxidation levels at overall plant retention times between 5.5 and 7.5 days. The enhanced performance under continuous operation is due to larger and more active microbial populations in the primary and secondary reactors which eliminate the lag phases experienced under batch operation. A minimum retention time of ≥ 3.3 days is prescribed for the primary bio-oxidation stage treating the Velardena concentrate.

The Velardena concentrate is amenable to the BIOX® process.

4.3. Cyanidation Leach Tests on Velardena BIOX® Residues

The results of the Velardena BAT oxidations and subsequent 24-hour cyanidations of the respective BIOX® residues are summarised in Table 4.7. The gold dissolution as a function of sulphide sulphur oxidation is shown in Figure 4.15.

Table 4.7 : Velardena Sulphide Oxidation and Corresponding Gold Dissolutions

BIOX®	Sulphide	Residue Metal Dissolution		Consumption		Au
Period (days)	Oxidation (%)	Gold (%)	Silver (%)	NaCN (kg NaCN / t Concentrate)	Lime (kg CaO / t Concentrate)	Accountability (%)
0	0	5.8	38.62	5.46	0.41	95
9	19.9	62.13	90.52	6.49	2.82	95
15	28.2	87.72	94.44	7.24	7.10	99
24	58.3	86.74	94.98	5.81	4.44	100
30	72.9	88.84	92.03	7.55	4.35	104
37	90.2	90.14	53.22	5.01	5.73	97
45	92.6	89.70	91.25	6.24	4.04	94
45	92.4	91.33	88.51	4.83	4.95	90

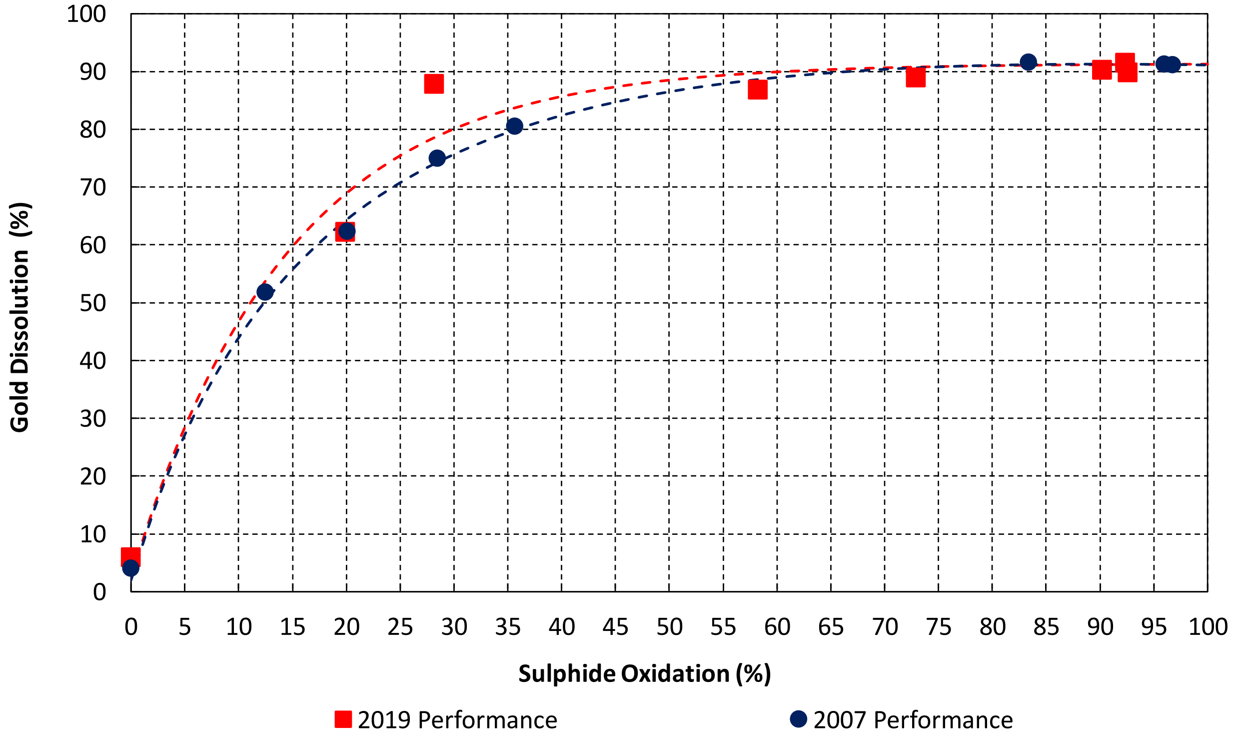


Figure 4.15 : Sulphide Oxidation and Gold Dissolution Relationships

Bio-oxidation of the concentrate increased the gold dissolution from 6% on the un-oxidised concentrate to a maximum of 91% on material that achieved an extent of sulphide oxidation of 93%. The figure shows a marginal dependence of gold dissolution on sulphide oxidation as gold dissolutions already approach 88% at a moderate sulphide oxidation of 28%. Further results of the batch cyanidation tests are shown in the tables that follow.

Table 4.8 : Velardena Cyanidation Results and Sulphide Oxidations

BAT Test	Bio-oxidation Period (days)	Sulphide Oxidation (%)	Leach Feed Au (g/t)		Leach Residue Au (g/t)		Gold Dissolution On (%)[1]	
			Assay	BCH[3]	Assay	Corrected[2]	Residue	BCH
FEED	0	0	18.50	17.65	17.70	17.43	5.80	1.19
BAT Test 6	9	19.9	20.60	19.66	7.86	7.80	62.13	60.32
BAT Test 5	15	28.2	19.80	19.63	2.44	2.43	87.72	87.61
BAT Test 7	24	58.3	15.40	15.43	2.06	2.04	86.74	86.77
BAT Test 4	30	72.9	15.90	16.60	1.83	1.77	88.84	89.31
BAT Test 3	37	90.2	16.30	15.73	1.65	1.61	90.14	89.78
BAT Test 1	45	92.6	17.10	16.14	1.82	1.76	89.70	89.09
BAT Test 2	45	92.4	18.10	16.25	1.59	1.57	91.33	90.34

Notes :
(1) Gold dissolution based on residue and BCH.
(2) Gold residue grade corrected for mass change across leach.
(3) BCH denotes back calculated head.

Table 4.9 : Velardena Silver Cyanidation Results

BAT Test	Biooxidation Period (days)	Silver		
		Feed (g/t)	Tails (g/t)	Dissln.[1] (%)
Concentrate	0	124.00	77.30	38.62
BAT 6	9	112.00	10.70	90.52
BAT 5	15	95.00	5.30	94.44
BAT 7	24	73.00	3.70	94.98
BAT 4	30	85.00	7.00	92.03
BAT 3	37	50.00	24.00	53.22
BAT 1	45	91.00	8.23	91.25
BAT 2	45	62.00	7.22	88.51

Notes : **(1) Silver dissolution based on residue.**

Reagent consumptions and final leach data recorded for the cyanidation of the BIOX® product of the various BAT tests are shown in the following table.

It should be noted here that the NaCN additions were bulk additions made once off and further reductions in the cyanide consumption will be possible through a cyanidation optimisation program.

Table 4.10 : Reagent Consumptions and Final Leach Data for the Velardena Cyanidation Tests

BAT Test	Bio-oxidation Period (days)	Reagent Addition				Reagent Consumption				Residual NaCN (g/L)	End pH	Leach Mass Adjustment Factor
		CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN			
		(kg / t BIOX® Product)		(kg / t BIOX® Feed)		(kg / t BIOX® Product)		(kg / t BIOX® Feed)				
FEED	0	-	-	0.78	20.00	-	-	0.41	5.46	3.41	11.26	0.98
BAT Test 6	9	2.74	20.00	3.08	22.54	2.50	5.76	2.82	6.49	3.41	10.68	0.99
BAT Test 5	15	6.94	20.00	7.33	21.15	6.71	6.85	7.10	7.24	3.34	10.64	1.00
BAT Test 7	24	4.50	20.00	4.68	20.82	4.27	5.58	4.44	5.81	3.48	10.65	0.99
BAT Test 4	30	4.71	25.00	4.57	24.24	4.48	7.78	4.35	7.55	4.21	10.59	0.97
BAT Test 3	37	6.22	25.00	5.94	23.86	6.00	5.25	5.73	5.01	5.00	10.62	0.97
BAT Test 1	45	4.10	25.00	4.28	26.12	3.87	5.97	4.04	6.24	4.64	10.52	0.97
BAT Test 2	45	5.01	25.00	5.19	25.88	4.78	4.67	4.95	4.83	5.00	10.54	0.99

4.4. Neutralisation Tests on the Velardena Product BIOX® Liquor

Two neutralisation tests were conducted on the liquors from the Velardena BAT 1 and BAT 2 tests which yielded the highest sulphide oxidations. The conditions of the batch neutralisation tests using AR grade limestone and lime (Test 1) and AR grade lime only (Test 2) are summarised in Table 4.11. The figures that follow thereafter show the neutralising capacity of the various reagents used and the iron and arsenic profiles at various slurry pH values.

The log sheets for the batch neutralisation tests are provided in Appendix 7.7.

Table 4.11 : Conditions of the Batch Neutralisation Tests

Description	Units	Test 1 Limestone + Lime	Test 2 Lime Only
Analyses of Neutralisation Feed			
Fe(T)	g/L	18.50	18.50
As(T)	g/L	3.14	3.26
Fe / As Molar Ratio		7.90	7.61
Biooxidation Liquor	ml	1000	1000
Water	ml	2000	2000
Total Volume	ml	3000	3000
pH		1.56	1.56
Redox	mV	616	617
Analyses of Limestone			
$CaCO_3$ concentration	g/l	200	-
CO_3	(%)	59.4	-
Ca	(%)	39.6	-
Volume added	ml	588	-
Analyses of Lime			
CaO concentration	g/l	100	100
Ca	(%)	71.4	71.4
Volume added	ml	53	790

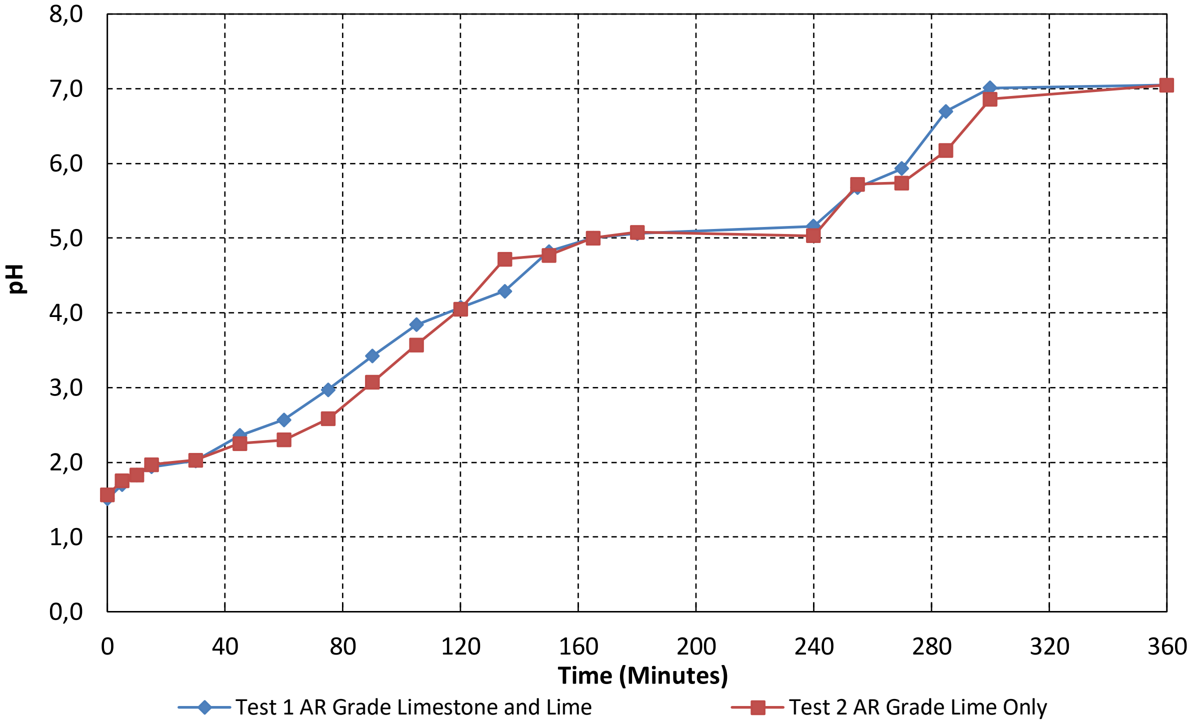


Figure 4.16 : Velardena Batch Neutralisation pH Profiles

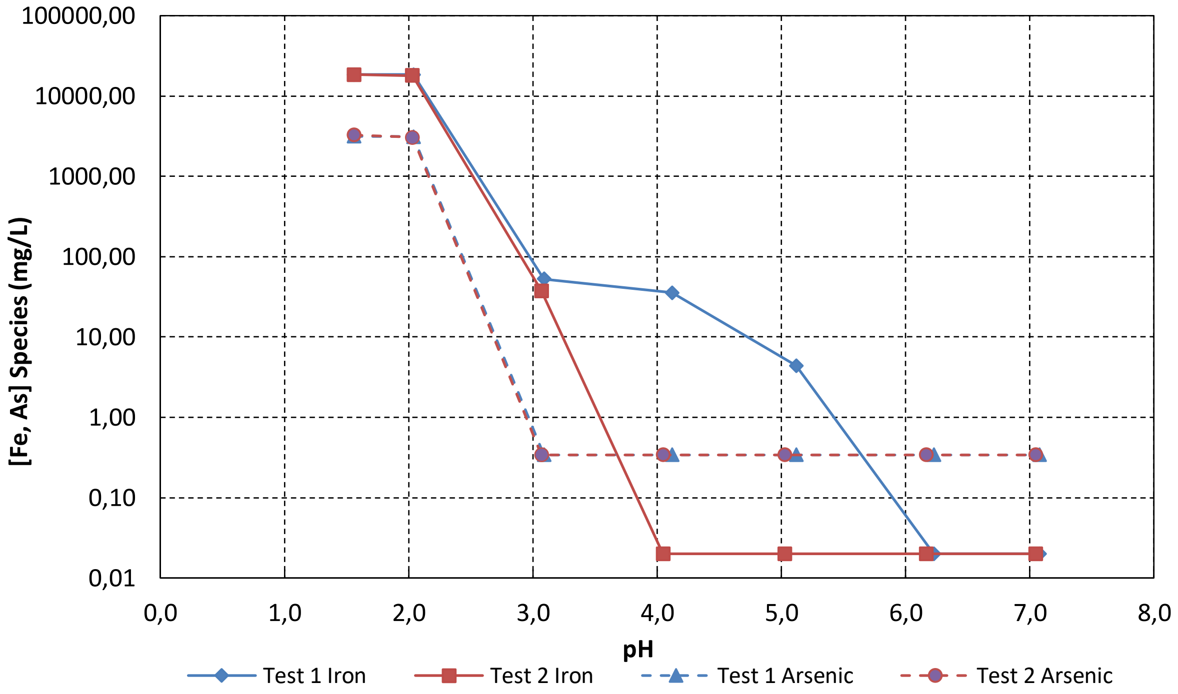


Figure 4.17 : Velardena Batch Iron and Arsenic Profiles

The final solution results for iron and arsenic depicted in Figure 4.17 are also shown in Table 4.12 alongside the concentration of iron and arsenic of the respective neutralised solids. The table also shows the TCLP results conducted on samples of the neutralised solids.

Table 4.12 : Velardena Liquor Neutralisation Product Analyses

Test Number	Neutralisation Products					
	Solids (%)		Solution (mg/L)		TCLP (mg/L)	
	Fe	As	Fe	As	Fe	As
Test 1 (AR Grade Limestone + AR Lime)	16.60	2.50	<0.02	<0.34	< 0.05	0.06
Test 2 (AR Grade Lime Only)	15.50	2.48	<0.02	<0.34	< 0.05	0.07

From Table 4.12 it follows that low levels of iron and arsenic solution values were obtained in the batch neutralisation tests. The results indicate that the iron and arsenic in the BIOX® liquor can be successfully removed from solution by a two-stage treatment with AR grade limestone to pH 5 followed by the addition of AR grade lime to pH 7 (Test 1). A single stage treatment using AR grade lime only was also successful in removing the iron and arsenic to the required levels. Most commercial BIOX® operations however utilise a local limestone and lime source for neutralisation. The Jinfeng and Suzdal BIOX® operations also utilise a portion of their flotation tailings as a neutralising reagent due to the presence of available carbonate minerals to assist in the neutralisation.

With respect to arsenic mobility and its stability as a ferric arsenate precipitate, the neutralised effluents for both tests contained < 0.34 mg/L arsenic, satisfying the limit set by the United States of America's Environmental Protection Agency (USA EPA) which requires neutralised effluent streams to be discharged at < 0.5 mg/L arsenic. The TCLP testing of the neutralisation precipitates produced extracts containing 0.06 and 0.07 mg/L arsenic for both tests. This is well below the 5 mg/L limit set by the United States of America's Environmental Protection Agency (USA EPA). Both precipitates can therefore be considered stable, environmentally compliant and thus acceptable for disposal on a tailings dam.

Further results by way of reagent consumptions for the batch tests using AR grade limestone / lime (Test 1) and AR grade lime only (Test 2) are summarised in Table 4.13.

Table 4.13 : Reagent Consumptions of the Velardena Liquor Tests

Description	Units	Test 1 Limestone + Lime	Test 2 Lime Only
Limestone Consumption	kg/t BIOX® Feed	470	-
Lime Consumption	kg/t BIOX® Feed	21	316
Ca^{2+} Content	kg/t BIOX® Feed	203	223
Final pH		7.08	7.05
Retention Time	hrs	6	6

4.5. Static Settling Tests on Selected Velardena Slurries

The objectives of the static liquid – solids separation tests on the BIOX® product slurry were to determine the optimum flocculant, its dosage and finally, the settling rate and estimated thickener area that could provide a preliminary view of the sizing of the counter current decantation (CCD) section following the BIOX® reactors. The results of the flocculant selection tests on the final bio-oxidation slurry of the BAT 2 slurry are summarised in Table 4.14.

The static settling test log sheets are included in Appendix 7.8.

Table 4.14 : Velardena BIOX® Product Slurry Flocculant Screening Tests

Time	Flocculants Tested (100 g/t)							
(sec)	Magna 10	Magna 155	Magna 333	Magna 919	Magna 345	Magna 455	Magna 336	Magna 405
15	83	14	96	4	7	79	7	73
30	108	29	114	4	19	106	7	108
45	116	47	120	5	22	117	17	118
60	120	59.5	123	6	22	120	21	123
120	126	111	127	9	22	126	44	131
180	128	135.5	129	9	22	128	68	135
240	128	140	129	10	23	130	87	135
300	128	142	129	10	25	130	106	135
Settling time (Seconds)	30	30	30	30	30	30	30	30
Displacement (mm)	108	29	114	4	19	106	7	108
Settling rate (m/h)	12.96	3.48	13.68	0.48	2.28	12.72	0.84	12.96
Overflow Clarity	CLEAR	CLEAR	CLEAR	CLOUDY	CLOUDY	CLEAR	CLOUDY	CLEAR

Magnafiocc 333 produced a superior settling rate compared to the other flocculants tested. The overflow clarity was also quite good for this flocculant. This flocculant was therefore selected for the dosage testing and the results are shown in the figure that follows.

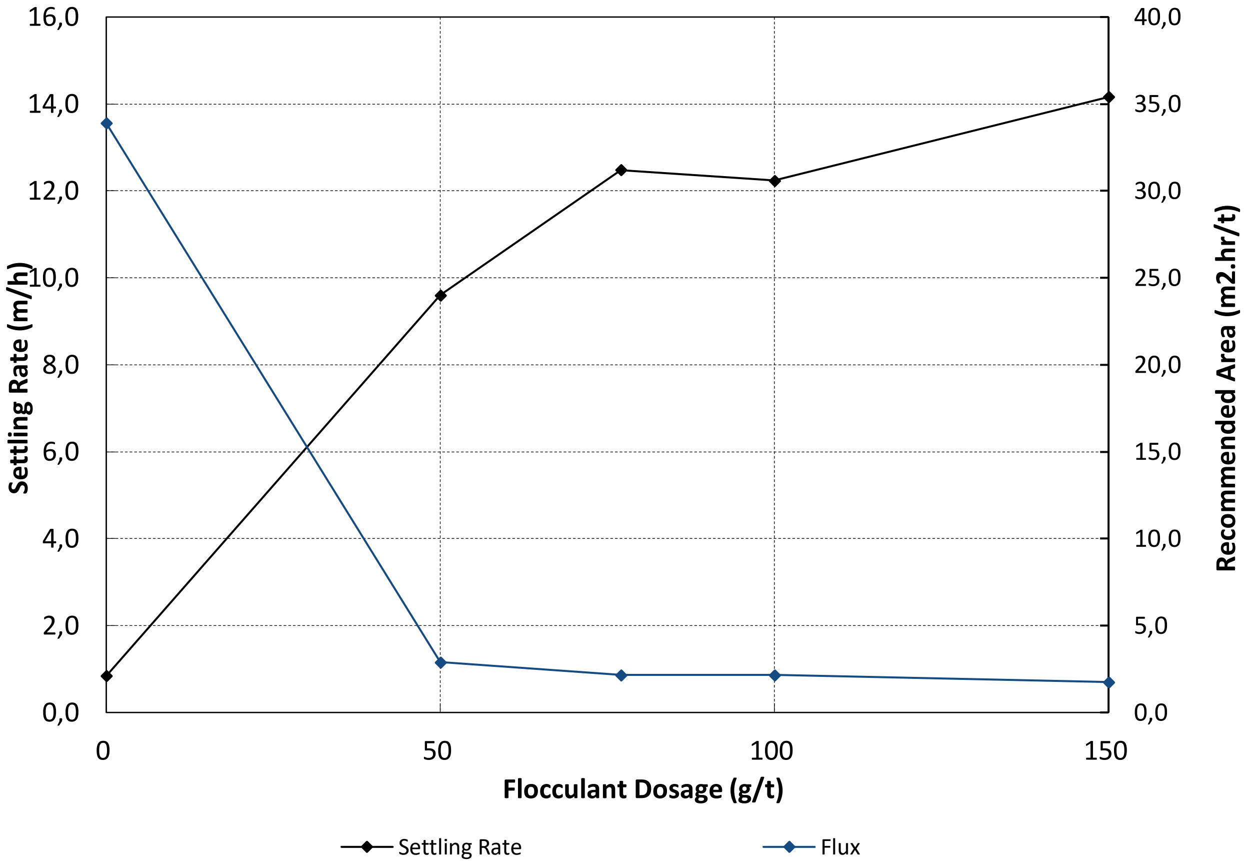


Figure 4.19 : Settling Rates at Various Flocculant (M333) Additions at ~7% Solids

The diluted BIOX® slurry settled quite well across the dosages tested with the highest settling rate of 14 m/hr achieved at a dosage of 150 g/t. For dosages of 50, 77 and 100 g/t the settling rates achieved were 9.6 m/hr, 12.5 m/hr and 12.2 m/hr. A dosage of 50 g/t and a settling rate achieved of 9.6 m/hr is still quite good. This translates to a recommended thickener area flux of ~ 3 m^2.h/t.

5. CONCLUSIONS

The Velardena concentrate is amenable to the BIOX® pre-treatment process. This is apparent by the following:

- The adaptation of the BIOX® culture to the 2019 Velardena sample was very successful;
- Performance data on Velardena concentrate showed that:
    - Biooxidation of the concentrate improved the gold dissolution from 6% on direct cyanidation to a maximum value of 91%. This was achieved after 45 days of biooxidation. The 45-day biooxidation period yielded an average of 93% sulphide oxidation and an average of 91% gold dissolution;
    - Biooxidation of the concentrate showed that under batch operation, the concentrate will be net acid generating during the oxidation process having a net lime consumption of 137 kg CaO / tonne concentrate. This is due to the high pyrite level in the concentrate;
- Cyanidation of the Velardena BIOX® products showed that:
    - The maximum averaged gold dissolution achieved at the averaged sulphide oxidation extent of 93% was 91%. The averaged sodium cyanide consumption for these tests was 5.5 kg NaCN / tonne of concentrate and the averaged lime consumption for the tests was 4.5 kg CaO / tonne of concentrate;
    - Silver dissolutions spanned the range 53% to a maximum of 95% on oxidised residues. A silver dissolution of 39% was achieved on direct concentrate leaching;
- Batch neutralisation results for Velardena BIOX® liquor showed that:
    - The iron and arsenic in the BIOX® liquor can be successfully removed from solution by a two-stage treatment with AR grade limestone to pH 5 followed by the addition of AR grade lime to pH 7. A single stage treatment using AR grade lime only was also successful in removing the iron and arsenic;
    - These neutralised effluents from the two batch tests contained ≤ 0.34 mg/L arsenic satisfying the limit set by the United States of America's Environmental Protection Agency (USA EPA) which requires neutralised effluent streams to be discharged at 0.5 mg/L arsenic;

- The toxicity characteristic leach procedure (TCLP) testing of both the neutralisation precipitates produced extracts well below the 5 mg/L limit set by the United States of America's Environmental Protection Agency (USA EPA).

- The precipitates can be considered stable and thus acceptable for disposal on a tailings facility.

Standard liquid-solid separation test work on BIOX® product slurry showed that a dosage of 50 g/t Magnafloc 333 yielded a suitable settling rate. While higher dosages yielded higher settling rates, the Velardena BIOX settling rate is quite high and the 50 g/t dosage translates to a recommended thickener area flux of ~ 3 m^2.h/t.

Overall, a pleasing set of results have been achieved in the 2019 Velardena BIOX® batch campaign and importantly, the results achieved in this campaign are similar to that achieved in the 2007 test campaign.

The results of the Velardena BAT campaigns therefore demonstrates that BIOX® can be an effective pre-oxidative treatment step to improve the gold dissolutions on the Velardena concentrate.

6. REFERENCES

1. Broadhurst, J L, 'The Nature and Stability of Arsenic Residues from the BIOX® Process', Biomine 1993 Conference Proceedings, Adelaide, March 1993, pp 13.1-10.

2. Broadhurst, J L, 'Recommended Standard Toxicity Characteristic Leaching Procedure for Arsenic-Containing Waste Samples'; GENCOR Process Research, Report No PR 90/49F; August 1990; 29 pp.

7. APPENDICES

7.1. Nutrient Media Make-up

0K NUTRIENT MEDIUM

Chemical	Amount (g/l)
$(NH_4)_2SO_4$	3
K_2HPO_4	0.5
$MgSO_4.7H_2O$	0.5
KCl	0.1
$Ca(NO_3)_2$	0.01

9K NUTRIENT MEDIUM

Chemical	Amount (g/l)
$(NH_4)_2SO_4$	3
K_2HPO_4	0.5
$MgSO_4.7H_2O$	0.5
KCl	0.1
$Ca(NO_3)_2$	0.01
$FeSO_4.7H_2O$	50

7.2. Equations Used for BIOX® Calculations

1. MASS CHANGE DURING BIOX®

$$\text{Mass change (\%)} = \frac{a - (b + c)}{(b + c)} \times 100$$

where a = mass of BIOX® product
b = initial mass of feed added
c = estimated amount of solid added with the inoculum

2. ACID ADDITION TO STABILISE BIOX® SLURRIES

$$\text{Acid addition (kg / ton feed)} = \frac{a \times b}{c}$$

where a = concentration of acid solution used (g/l)
b = total volume of acid added (l)
c = mass feed added to BIOX® (kg)

3. LIME CONSUMPTION DURING BIOX®

$$\text{Lime consumption (kg / ton feed)} = \frac{a \times b}{c}$$

where a = total volume of lime added during BIOX® (l)
b = available CaO concentration in lime (g/l)
c = mass of feed added to BIOX® (kg)

4. NET AMOUNT OF ACID GENERATED/CONSUMED DURING BIOX®

$$\text{Net acid generation / consumption} =$$

$$\text{Acid addition (kg / ton)} - \text{Lime consumption (kg / ton)} \times \frac{\text{molar mass acid}}{\text{molar mass lime}}$$

A negative value indicates a net acid consuming ore/concentrate
A positive value indicates a net acid generating consuming ore/concentrate

5. SULPHIDE REMOVAL

$$Sulphide\ Removal\ (\%) = \frac{[(a_1 \ x\ b_1) + (a_2 \ x\ b_2)] - (a_3 \ x\ b_3)}{[(a_1 \ x\ b_1) + (a_2 \ x\ b_2)]}$$

where a1 = mass feed added to BIOX® batch (g)
b1 = S (pyr) content of pre-BIOX® sample (%)
a2 = mass feed added with inoculum (g)
b2 = estimated S (pyr) content of solids in inoculum (%)
a3 = mass BIOX® product (g)
b3 = S (pyr) content of post-BIOX® sample (%)

7.3. Amenability Test Log Sheets

Table A7.3.1 : Adaptation Phase

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
18-Sep					7.84	1.33	30.0				
18-Sep					1.34						
18-Sep					1.33						
18-Sep					1.43	1.33	2.0				
18-Sep					1.37					41.3	Inoculated 320 ml FV slurry
18-Sep	0	8.5	3.5	438	1.65	1.36	9.0		5.2	41.6	
19-Sep	1	5.8	5.1	463	1.45				5.1	41.0	
19-Sep					1.45						
20-Sep	2	0.7	11.2	555	1.50	1.42	3.0		5.3	41.1	
21-Sep	3	0.2	14.2	598	1.24				4.9	41.0	
22-Sep	4	0.1	16.5	632	1.02				4.8	41.1	Added 250 g of concentrate
23-Sep	5	0.2	21.8	586	1.14	1.23		50	4.0	41.5	
23-Sep	5				1.18	1.24		40			
24-Sep	6	0.2	27.3	603	1.09	1.26		80	4.0	41.2	
25-Sep	7	0.2	29.7	607	1.11	1.27		80	4.0	41.0	
25-Sep	7				1.22	1.24		10			
26-Sep	8	0.1	32.0	619	1.11				3.9	40.6	Added 250 g of concentrate
26-Sep	8				1.37						
27-Sep	9	0.5	36.0	581	1.25				4.2	40.1	
27-Sep	9				1.19	1.28		70			
28-Sep	10	0.3	44.7	609	1.10	1.20		100	4.0	41.4	
29-Sep	11	0.3	44.5	620	1.11	1.23		100	3.7	41.4	
30-Sep	12	0.2	45.7	617	1.15	1.23		60	4.1	41.7	
30-Sep	12				1.19	1.27		60			
01-Oct	13	0.1	46.2	631	1.16	1.21		50	4.0	41.2	
01-Oct	13				1.18	1.28		80			
02-Oct	14	0.1	52.7	634	1.18				4.0	40.7	SCAT. Removed 320 ml slurry, added 320 ml 0K nutriets + 80 g conc
03-Oct	15	0.2	49.8	613	1.1	1.22		100	4.0	41.2	
04-Oct	16	0.1	51.2	629	1.18	1.28		80	4.2	41.2	
05-Oct	17	0.1	57.4	642	1.12				3.4	41.2	
05-Oct	17				1.12	1.25		100			
06-Oct	18	0.1	63.4	656	1.00	1.09		100	2.8	42.0	
06-Oct	18				1.13	1.21		80			
07-Oct	19	0.1	60.7	653	1.14	1.22		100	3.5	41.2	
07-Oct	19				1.18	1.23		60			
08-Oct	20	0.1	59.9	668	1.15	1.23		100	3.9	41.3	Removed 352 ml slurry
08-Oct	20				1.20	1.25		50			

Table A7.3.1 : Adaptation Phase Continued

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(^{0}C)	
09-Oct	21	0.1	62.7	679	1.15	1.25		100	4.0	41.3	
10-Oct	22	0.0	73.8	689	1.16	1.28		100	3.9	41.2	
11-Oct	23	0.0	77.0	697	1.24				4.1	41.0	Removed 200 ml of slurry , added 200 ml 0K +50 g concentrate
12-Oct	24	0.1	67.7	670	1.23				4.0	41.0	
13-Oct	25	0.1	71.8	673	1.15	1.24		100	3.7	40.7	
14-Oct	26	0.1	72.2	664	1.21				4.0	41.3	
14-Oct	26				1.21	1.26		30			
15-Oct	27	0.0	70.5	667	1.26				4.2	41.3	
16-Oct	28	0.0	71.3	661	1.28				4.0	41.3	
17-Oct	29	0.0	65.3	662	1.26				4.1	41.0	
18-Oct	30	0.0	61.3	667	1.25				4.2	41.4	
19-Oct	31	0.0	60.5	674	1.19	1.25		50	3.5	41.6	
20-Oct	32	0.0	65.0	670	1.26				3.1	41.7	
21-Oct	33	0.0	70.5	664	1.26				3.2	41.7	
22-Oct	34										Foarming no measurements taken. SCAT 150 ml, added 150 ml 0K + 15g conc
23-Oct	35	0.1	53.4	632	1.45				3.5	40.8	
24-Oct	36	0.1	57.4	624	1.37				3.3	41.2	
25-Oct	37	0.3	55.2	578	1.36				3.1	41.0	
26-Oct	38	0.3	54.7	586	1.37				3.4	41.3	
27-Oct	39	0.5	52.0	593	1.35				3.7	41.3	
28-Oct	40	0.6	52.2	607	1.36				3.4	41.4	
29-Oct	41	0.8	54.0	594	1.35				3.3	40.9	
30-Oct	42	0.9	56.1	593	1.32				3.3	40.8	
31-Oct	43	0.8	52.0	588	1.37				3.7	40.9	Terminated

Table A7.3.2 : Inoculum Build-up Phase

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
01-Oct					7.76	4.95	42.0				
01-Oct					5.30	1.45	24.0				
01-Oct					1.74	1.35	6.0				
01-Oct					1.40						
02-Oct					1.47	1.34	3.0				
02-Oct					1.49	1.36	3.0				
02-Oct	0	1.8	3.5	472	1.48	1.40	3.0		4.3	40.7	Added 320 ml of Velardena Adapt slurry + 0K salts (3L)
02-Oct	0				1.43						
03-Oct	1	4.4	0.8	421	1.48				4.4	40.6	
04-Oct	2	4.3	0.7	408	1.58	1.47	3.0		4.3	41.0	
05-Oct	3	4.5	1.3	411	1.41				4.5	40.7	
06-Oct	4	3.6	1.9	422	1.38				4.1	40.9	
07-Oct	5	3.6	2.4	422	1.32				4.1	41.0	
08-Oct	6	3.4	2.6	437	1.27				3.8	41.0	
09-Oct	7	1.9	10.2	505	1.29				4.1	40.8	
10-Oct	8	1.5	12.7	527	1.27				4.0	41.0	
11-Oct	9	1.2	14.5	533	1.23				4.2	40.6	
12-Oct	10	1.1	17.9	542	1.15	1.23		100	4.0	40.5	
13-Oct	11	1.1	17.7	550	1.11	1.24		100	3.0	40.0	Increased air
14-Oct	12	0.4	21.1	568	1.17	1.27		60	3.9	41.2	
14-Oct	12				1.21	1.28		30			
15-Oct	13	0.2	22.2	587	1.17	1.26		70	3.7	41.5	
16-Oct	14	0.1	26.7	594	1.12	1.21		100	3.2	41.4	
16-Oct	14				1.18	1.27		100			
17-Oct	15	0.1	27.9	600	1.14	1.28		100	3.4	40.8	
18-Oct	16	0.1	31.9	605	1.15	1.30		100	3.5	41.7	
19-Oct	17	0.1	28.2	619	1.14	1.25		100	2.0	41.6	
20-Oct	18	0.1	32.3	624	1.10	1.20		100	2.2	41.8	
21-Oct	19	0.1	38.4	629	1.11	1.22		100	3.3	41.7	
22-Oct	20	0.1	40.4	629	1.16				4.0	40.7	
23-Oct	21	0.5	18.5	569	1.57	1.41	9.0		3.1	41.2	Removed 2240 ml slurry, added 2 L 0K + 200 g Velardena conc
24-Oct	22	0.2	20.8	601	1.21	1.3		100	3.4	41.4	

Table A7.3.2 : Inoculum Build-up Phase Continued

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
25-Oct	23	0.2	22.6	593	1.22	1.3		50	3.0	41.2	
26-Oct	24	0.1	27.0	621	1.16	1.26		80	3.1	41.5	
27-Oct	25	0.1	29.3	644	1.15	1.27		100	3.3	41.6	
28-Oct	26	0.1	38.4	669	1.14	1.26		150	3.1	41.9	
29-Oct	27	0.1	34.7	673	1.17	1.25		100	2.2	41.1	
30-Oct	28	0.0	38.4	686	1.16				2.3	41.1	
31-Oct	29	0.0	38.0	695	1.21				3.6	41.2	Removed 300 ml ml slurry, added 300 ml 0K + 30 g Velardena conc
01-Nov	30	0.0	38.5	692	1.14	1.24		100	4.2	41.3	
02-Nov	31	0.0	41.7	696	1.21					41.4	
03-Nov	32	0.0	42.0	720	1.10	1.21		50		41.5	
04-Nov	33	0.0	41.8	725	1.16	1.27		100	3.0	41.5	
05-Nov	34	0.0	40.8	725	1.27				5.0	41.3	
06-Nov	35	0.0	38.8	735	1.28				5.0	40.9	
07-Nov	36	0.0	40.5	700	1.33				5.0	40.8	Removed 150 ml ml slurry, added 150 ml 0K + 15 g Velardena conc
08-Nov	37	0.2	36.3	606	1.24	1.31		40	5.2	40.8	
09-Nov	38	0.4	39.6	574	1.32				5.2	40.6	
10-Nov	39	0.6	37.8	569	1.41				5.3	40.2	
11-Nov	40	0.8	39.0	556	1.32				5.4	40.4	
12-Nov	41	0.8	36.8	556	1.28				5.0	40.5	
13-Nov	42	0.6	37.2	563	1.36				6.4	40.5	
14-Nov	43	1.0	36.9	553	1.29					40.3	
15-Nov	44	0.9	37.6	558	1.30				5.7	40.5	
16-Nov	45	0.9	39.1	558	1.35				5.2	40.9	
17-Nov	46	0.5	35.3	594	1.31				4.8	40.9	
18-Nov	47	0.5	36.3	661	1.34				5.3	40.9	
19-Nov	48	0.1	37.2	650	1.36				5.6	40.7	
20-Nov	49	0.0	43.7	645	1.21				5.6	40.9	

Table A7.3.3 : Velardena BAT 1

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(^oC)	
21-Oct					8.03	5.34	33.0				
21-Oct					6.39	3.59	18.0				
21-Oct					5.30	1.34	12.0				
22-Oct					2.25	1.41	6.0				
22-Oct					1.50	1.37	3.0			39.9	
22-Oct	0	2.0	2.9	458	1.53	1.35	6.0		4.0	41.1	Added 320 ml slurry velardena inoc + 0K (3L)
23-Oct	1	4.1	1.0	396	1.44				4.1	39.5	
24-Oct	2	4.3	1.2	412	1.35				4.0	40.0	
25-Oct	3	4.0	1.1	383	1.38				3.7	40.2	
26-Oct	4	3.8	1.7	416	1.38				3.7	40.1	
27-Oct	5	3.8	2.0	432	1.28				3.6	39.7	
28-Oct	6	3.2	2.8	473	1.26				3.9	40.0	
29-Oct	7	1.2	5.7	524	1.29				2.8	38.3	
30-Oct	8	1.0	8.8	563	1.18				1.7	38.7	
31-Oct	9	0.9	12.0	553	1.15				1.5	39.8	
01-Nov	10	0.7	14.5	552	1.05	1.29		100	3.2	39.8	
02-Nov	11	0.8	16.4	550	1.21					40.2	
03-Nov	12	0.6	19.7	558	1.12	1.2		60		40.4	
04-Nov	13	0.5	17.9	556	1.12				3.0	39.8	
05-Nov	14	0.5	17.9	551	1.18				1.5	40.2	
06-Nov	15	0.4	20.5	586	1.13				2.3	40.4	
07-Nov	16	0.4	22.2	588	1.27				1.6	40.0	
08-Nov	17	0.3	22.7	571	1.13	1.21		65	4.0	40.0	
09-Nov	18	0.2	24.0	576	1.22				3.2	40.2	
					1.13	1.26		100			
10-Nov	19	0.2	27.3	590	1.13	1.30		70	2.2	39.8	
11-Nov	20	0.2	28.7	591	1.27				2.8	39.8	
12-Nov	21	0.2	31.3	602	1.17	1.21		50	3.4	38.9	
13-Nov	22	0.1	33.0	618	1.14	1.22		80	3.0	38.7	

Table A7.3.3 : Velardena BAT 1 Continued

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
14-Nov	23	0.2	38.2	615	1.11	1.23		100		38.9	
15-Nov	24	0.2	39.8	614	1.17	1.23		60	3.2	38.6	
16-Nov	25	0.1	41.7	604	1.24				2.7	39.5	
17-Nov	26	0.1	42.0	610	1.11	1.2		90	3.0	39.0	
18-Nov	27	0.3	42.5	606	1.22				3.6	40.4	
19-Nov	28	0.2	48.2	619	1.13	1.23		70		39.6	
20-Nov	29	0.2	52.6	619	1.14	1.19		50	2.5	39.6	
					1.15	1.21		50			
21-Nov	30	0.1	47.1	629	1.21				1.9	40.0	
22-Nov	31	0.1	47.2	627	1.11	1.21		170	2.1	39.7	
23-Nov	32	0.1	48.2	641	1.05	1.27		145	2.7	39.8	
24-Nov	33	0.1	52.0	668	1.19	1.24		50			
25-Nov	34	0.1	52.0	647	1.16					39.1	
26-Nov	35	0.1	55.7	655	1.05					39.5	
27-Nov	36	0.0	59.5	646	1.03	1.07		20	2.8	40.0	
28-Nov	37	0.0	60.8	657	1.03				3.1	40.0	
29-Nov	38	0.0	60.9	662	1.00	1.04		50		41.0	
30-Nov	39	0.0	63.8	657	0.99				2.5	40.9	
01-Dec	40	0.2	61.0	644	0.93				3.9	40.1	
02-Dec	41	0.1	58.8	638	1.09				3.9	40.9	
03-Dec	42	0.0	57.8	640	0.97				4.1	40.3	
04-Dec	43	0.0	56.8	654	0.96				4.3	39.9	
05-Dec	44	0.0	60.3	653	0.97				4.2	39.7	
06-Dec	45	0.0	61.4	637	0.97				4.3	38.6	

Table A7.3.4 : Velardena BAT 2

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
21-Oct					7.99	5.35	33.0				
21-Oct					6.32	3.45	18.0				
21-Oct					5.63	1.38	12.0				
22-Oct					2.00	1.38	9.0				
22-Oct					1.42					39.8	
22-Oct	0	2.0	2.8	459	1.52	1.36	6.0		4.1	40.0	Added 320 ml slurry Velardena inoc + 0K (3L)
23-Oct	1	4.0	1.1	401	1.43				4.0	39.6	
24-Oct	2	4.3	1.0	412	1.35				3.8	39.6	
25-Oct	3	3.7	1.3	386	1.40				3.4	39.8	
26-Oct	4	3.8	1.8	414	1.38				3.5	38.9	
27-Oct	5	3.9	2.0	435	1.38				3.7	38.3	
28-Oct	6	3.2	2.1	464	1.33				3.9	38.5	
29-Oct	7	2.5	3.4	475	1.31				3.4	38.7	
30-Oct	8	1.4	4.9	515	1.29				3.1	37.2	
31-Oct	9	1.2	7.1	531	1.33				2.8	39.7	
01-Nov	10	0.9	10.6	539	1.25				4.2	41.0	
02-Nov	11	0.9	12.1	534	1.17	1.21		30		41.0	
03-Nov	12	0.9	12.7	552	1.13	1.22		40		40.0	
04-Nov	13	0.9	13.4	546	1.17	1.24		50	4.0	40.6	
05-Nov	14	0.6	14.5	552	1.19				4.0	40.9	
06-Nov	15	0.5	17.9	572	1.10	1.2		100	2.8	41.1	
07-Nov	16	0.3	19.4	580	1.20				2.4	40.7	
08-Nov	17	0.2	19.8	581	1.07	1.18		100	3.4	40.8	
09-Nov	18	0.2	21.9	579	1.15	1.22		41	3.2	41.0	
					1.16	1.20		50			
10-Nov	19	0.2	26.2	583	1.11	1.20		78	3.2	40.7	
11-Nov	20	0.2	25.7	582	1.23				3.2	40.9	
12-Nov	21	0.3	28.5	584	1.14	1.24		70	4.7	40.5	
13-Nov	22	0.2	28.1	595	1.18	1.24		70	4.2	40.5	
14-Nov	23	0.2	30.7	590	1.08	1.21		100		40.8	

Table A7.3.4 : Velardena BAT 2 Continued

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
15-Nov	24	0.1	32.1	601	1.23				3.7	39.7	
16-Nov	25	0.1	33.2	600	1.21				4.7	40.8	
17-Nov	26	0.1	33.7	614	1.10	1.20		90	4.1	40.0	
18-Nov	27	0.3	33.6	607	1.20				3.9	41.0	
19-Nov	28	0.3	39.0	617	1.07	1.19		100		41.1	
20-Nov	29	0.3	42.4	630	1.11	1.20		90	2.6	40.1	
					1.16	1.22		40			
21-Nov	30	0.2	43.1	627	1.18				3.4	40.5	
22-Nov	31	0.1	43.7	624	1.09	1.20		80	2.6	40.3	
23-Nov	32	0.1	43.4	628	1.13	1.19		100	2.8	40.4	
24-Nov	33	0.1	46.7	653	1.15	1.18		50		40.5	
25-Nov	34	0.1	46.2	644	1.10	1.18		100	3.0	39.7	
26-Nov	35	0.1	51.1	647	1.06					40.4	
27-Nov	36	0.0	51.1	650	1.01	1.06		20	3.4	40.8	
28-Nov	37	0.0	51.0	660	1.00				3.9	40.9	
29-Nov	38	0.0	51.1	665	0.98	1.01		50		41.3	
30-Nov	39	0.0	53.7	655	1.01				3.4	40.9	
01-Dec	40	0.3	54.3	644	0.97				4.3	40.9	
02-Dec	41	0.0	55.3	636	1.04				4.4	41.0	
03-Dec	42	0.0	55.1	646	0.99				4.6	40.7	
04-Dec	43	0.0	54.3	649	0.96				4.7	40.2	
05-Dec	44	0.0	56.5	645	0.98				4.3	39.9	
06-Dec	45	0.0	55.5	639	0.97				4.4	39.9	

Table A7.3.5 : Velardena BAT 3

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
21-Oct					8.02	5.28	33.0				
21-Oct					6.33	3.40	18.0				
21-Oct					4.99	1.36	12.0				
22-Oct					2.20	1.41	9.0				
22-Oct					1.45					39.8	
22-Oct	0	2.1	2.7	458	1.56	1.39	6.0		3.4	41.0	Added 320 ml slurry velardena inoc + 0K (3L)
23-Oct	1	4.0	1.0	397	1.47				3.3	39.8	
24-Oct	2	4.0	1.0	412	1.39				3.5	40.6	
25-Oct	3	3.7	0.9	385	1.38				2.9	40.5	
26-Oct	4	3.6	1.8	420	1.29				3.1	40.6	
27-Oct	5	3.2	2.4	458	1.29				2.8	40.3	
28-Oct	6	1.8	4.8	521	1.32				3.0	40.7	
29-Oct	7	1.4	6.8	528	1.31				1.3	41.2	
30-Oct	8	1.0	9.0	561	1.18				2.0	40.0	
31-Oct	9	0.9	12.2	552	1.13				2.3	39.9	
01-Nov	10	0.9	13.5	545	1.08	1.33		100	4.2	40.2	
02-Nov	11	0.8	15.7	533	1.27					40.6	
03-Nov	12	0.6	16.0	554	1.14	1.20		30		40.5	
04-Nov	13	0.5	16.4	556	1.18	1.20		40	2.4	40.3	
05-Nov	14	0.3	16.1	553	1.20				4.0	41.0	
06-Nov	15	0.2	20.0	580	1.11	1.22		100	3.1	39.9	
07-Nov	16	0.2	22.0	594	1.22				2.5	39.8	
08-Nov	17	0.2	23.7	600	1.05	1.2		100	3.5	39.5	

Table A7.3.5 : Velardena BAT 3 Continued

Date	Treatment (days)	Ferrous (g/l)	Ferric (g/l)	Redox (mV)	pH Initial	pH Final	Acid add. (ml)	Lime (ml)	DO (ppm)	Temp (°C)	Observation
09-Nov	18	0.2	24.4	597	1.20				2.9	39.5	
					1.08	1.21		130			
10-Nov	19	0.2	28.4	602	1.09	1.2		86	1.8	40.3	
11-Nov	20	0.3	29.2	585	1.24				3.8	40.6	
12-Nov	21	0.2	32.4	595	1.15	1.22		70	4.6	39.7	
13-Nov	22	0.1	33.7	618	1.16	1.22		70	3.5	39.4	
14-Nov	23	0.1	38.1	620	1.14	1.21		73		39.7	
15-Nov	24	0.1	40.0	623	1.11	1.21		70	3.0	39.2	
16-Nov	25	0.1	42.3	626	1.10	1.22		70	3.7	39.4	
17-Nov	26	0.1	43.9	636	1.11	1.21		90	4.0	39.8	
18-Nov	27	0.1	45.0	644	1.21				1.8	40.4	
19-Nov	28	0.2	47.7	648	1.06	1.19		100		41.1	
20-Nov	29	0.1	54.5	662	1.10	1.21		100	2.9	39.0	
					1.14	1.24		60			
21-Nov	30	0.1	54.1	659	1.24				4.1	39.0	
22-Nov	31	0.0	54.3	656	1.12	1.23		70	3.6	39.8	
23-Nov	32	0.0	54.3	660	1.12	1.26		100	2.9	39.9	
24-Nov	33	0.2	53.1	636	1.24					39.3	
25-Nov	34	0.0	52.8	656	1.20				4.0	39.5	
26-Nov	35	0.0	53.3	650	1.18					40.4	
27-Nov	36	0.0	53.3	643	1.12				4.0	40.5	
28-Nov	37	0.0	55.1	653	1.13				3.7	40.5	

Table A7.3.6 : Velardena BAT 4

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
21-Oct					8.06	5.29	33.0				
21-Oct					6.35	3.60	18.0				
21-Oct					4.89	1.35	12.0				
22-Oct					2.08	1.41	9.0				
22-Oct					1.44					39.7	
22-Oct	0	2.0	2.8	457	1.54	1.39	6.0		4.1	40.9	Added 320 ml slurry Velardena inoc + 0K (3L)
23-Oct	1	4.0	1.1	400	1.46				3.2	39.9	
24-Oct	2	4.0	1.1	415	1.36				3.8	40.3	
25-Oct	3	3.6	1.0	396	1.28				2.9	40.5	
26-Oct	4	2.9	2.6	456	1.27				3.1	40.4	
27-Oct	5	1.3	6.0	535	1.28				3.0	40.2	
28-Oct	6	0.8	9.6	570	1.21	1.31		100	3.0	40.6	
29-Oct	7	0.6	11.7	566	1.27				2.3	39.0	
30-Oct	8	0.5	14.0	572	1.09				2.1	40.0	
30-Oct	8				1.07	1.23		100	2.4	39.6	
31-Oct	9	0.5	17.3	565	1.17				2.4	39.6	
01-Nov	10	0.4	18.1	555	1.10	1.25		100	4.3	39.7	
02-Nov	11	0.4	23.9	560	1.17					40.5	
03-Nov	12	0.4	24.8	560	1.04	1.2		100		40.1	
04-Nov	13	0.4	21.6	561	1.21				2.1	40.7	
05-Nov	14	0.3	21.2	556	1.20				3.0	41.0	
06-Nov	15	0.3	24.3	592	1.16	1.28		100	3.0	40.8	
07-Nov	16	0.1	23.3	597	1.29				2.5	40.6	
08-Nov	17	0.2	23.8	602	1.14	1.24		75	3.2	41.6	
09-Nov	18	0.3	27.1	593	1.20				2.6	41.1	
					1.14	1.23		100			
10-Nov	19	0.3	33.8	598	1.11	1.21		100	2.6	40.7	
11-Nov	20	0.2	31.8	592	1.20				3.2	40.6	
12-Nov	21	0.2	34.9	573	1.13	1.21		80	3.4	39.5	
13-Nov	22	0.2	33.8	587	1.21	1.25		70		40.1	
14-Nov	23	0.2	36.5	611	1.16	1.21		46		40.2	
15-Nov	24	0.1	40.2	625	1.14	1.23		50	2.5	39.8	
16-Nov	25	0.1	43.5	625	1.17	1.22		40	3.1	40.3	
17-Nov	26	0.1	44.3	632	1.08	1.2		100	3.3	40.2	
18-Nov	27	0.1	47.3	628	1.18				2.9	40.5	
19-Nov	28	0.1	47.4	634	1.14	1.21		80		40.2	
20-Nov	29	0.2	52.8	638	1.10	1.22		100	2.7	40.0	
					1.18	1.21		20			
21-Nov	30	0.2	51.5	634	1.20					39.9	

Table A7.3.7 : Velardena BAT 5

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
21-Oct					8.08	5.37	33.0				
21-Oct					6.47	3.34	18.0				
21-Oct					4.88	1.38	12.0				
22-Oct					1.90	1.37	9.0				
22-Oct					1.41					39.6	
22-Oct	0	2.1	2.8	457	1.52	1.36	6.0		3.0	41.1	Added 320 ml slurry Velardena inoc + 0K (3L)
23-Oct	1	4.0	1.0	404	1.44				3.2	39.9	
24-Oct	2	3.9	1.0	415	1.34				3.3	40.3	
25-Oct	3	3.5	1.1	392	1.26				3.0	40.2	
26-Oct	4	2.8	2.3	454	1.27				3.9	40.4	
27-Oct	5	1.0	6.1	536	1.29				3.5	40.4	
28-Oct	6	0.9	8.6	572	1.21	1.33		80	3.5	40.7	
29-Oct	7	0.7	10.9	568	1.28				1.9	39.2	
30-Oct	8	0.5	14.1	575	1.08				1.7	40.1	
30-Oct	8				1.06	1.29		140			
31-Oct	9	0.5	16.0	565	1.23				2.1	40.0	
01-Nov	10	0.4	17.4	566	1.13	1.28		100	3.8	40.2	
02-Nov	11	0.3	20.4	560	1.21					40.8	
03-Nov	12	0.3	22.0	581	1.09	1.2		80		40.8	
04-Nov	13	0.2	21.7	580	1.18	1.21		40	2.1	41.0	
05-Nov	14	0.2	22.5	574	1.19	1.21		100	3.9	41.2	
06-Nov	15	0.3	25.0	600	1.11				2.6	40.6	

Table A7.3.8 : Velardena BAT 6

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
21-Oct					8.09	5.30	33.0				
21-Oct					6.51	3.47	18.0				
21-Oct					4.87	1.39	12.0				
22-Oct					1.91	1.36	9.0				
22-Oct					1.41					39.9	
22-Oct	0	2.1	2.8	457	1.53	1.36	6.0		4.0	41.2	Added 320 ml slurry Velardena inoc + 0K (3L)
23-Oct	1	4.0	1.1	397	1.44				3.9	40.0	
24-Oct	2	4.1	1.0	413	1.34				4.0	39.7	
25-Oct	3	3.7	0.9	389	1.27				3.2	40.0	
26-Oct	4	3.4	2.2	436	1.24				3.3	39.8	
27-Oct	5	1.4	4.7	515	1.28				3.2	39.5	
28-Oct	6	0.8	7.8	570	1.28	1.33		100	3.4	40.1	
29-Oct	7	0.6	11.2	569	1.30				1.9	38.7	
30-Oct	8	0.4	14.7	575	1.07				1.7	39.4	
31-Oct	9	0.4	17.0	563	1.05				1.4	39.7	

Table A7.3.8 : Velardena BAT 7

Date	Treatment	Ferrous	Ferric	Redox	pH		Acid add.	Lime	DO	Temp	Observation
	(days)	(g/l)	(g/l)	(mV)	Initial	Final	(ml)	(ml)	(ppm)	(°C)	
21-Oct					8.10	5.28	33.0				
21-Oct					6.33	3.37	18.0				
21-Oct					3.85	1.36	12.0				
22-Oct					1.90	1.36	9.0				
22-Oct					1.36					39.8	
22-Oct	0	2.0	2.9	459	1.47	1.33	6.0		3.9	41.3	Added 320 ml slurry Velardena inoc + 0K (3L)
23-Oct	1	4.0	1.1	405	1.42				4.0	40.5	
24-Oct	2	3.9	0.9	416	1.34				4.0	40.8	
25-Oct	3	3.6	0.9	387	1.35				3.2	41.2	
26-Oct	4	3.5	1.7	418	1.32				3.3	40.8	
27-Oct	5	3.5	1.7	442	1.27				3.1	40.7	
28-Oct	6	2.0	3.4	498	1.29				3.2	40.9	
29-Oct	7	0.7	6.1	545	1.34				2.8	39.3	
30-Oct	8	0.7	8.6	558	1.18				2.5	40.3	
31-Oct	9	0.6	11.4	554	1.14				2.4	39.2	
01-Nov	10	0.7	13.2	557	1.06	1.19		100	4.4	39.9	
02-Nov	11	0.6	16.2	548	1.10	1.2		60		40.8	
03-Nov	12	0.5	20.0	565	1.07	1.21		80		40.7	
04-Nov	13	0.3	16.8	562	1.18	1.22		40	3.9	40.9	
05-Nov	14	0.3	17.9	560	1.19	1.22		100	3.9	41.0	
06-Nov	15	0.2	22.9	594	1.11				3.5	40.8	
07-Nov	16	0.1	24.8	606	1.18				2.8	40.9	
08-Nov	17	0.1	23.3	610	1.03	1.23		153	4.2	40.5	
09-Nov	18	0.1	29.6	605	1.16	1.21		42	3.4	40.9	
					1.10	1.20		80			
10-Nov	19	0.2	33.4	610	1.09	1.2		70	3.8	40.6	
11-Nov	20	0.2	31.9	607	1.20				4.1	40.5	
12-Nov	21	0.2	34.6	611	1.1	1.21		100	2.9	39.7	
13-Nov	22	0.1	35.3	617	1.14	1.23		80	3.6	40.4	
14-Nov	23	0.1	38.4	623	1.11	1.21		79		40.7	
15-Nov	24	0.1	41.9	635	1.12	1.23		70	3.9	40.4	

7.4. Equations Used For Cyanidation Calculations

1. CYANIDE ADDITION

$$\text{Cyanide addition } (kg / t \text{ BIOX product}) = \frac{a}{b} \times 1000$$

where a = mass NaCN added (g)
b = mass BIOX® product used for cyanidation (g)

Cyanide addition values are corrected for mass changes during BIOX® using the following equation:

$$\text{Corrected cyanide addition } (kg / t \text{ feed}) = \text{cyanide addition } (kg / t \text{ BIOX product}) \times F1$$

where F1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) / 100
a = mass change during BIOX® (%)

2. LIME ADDITION

$$\text{Lime addition } (kg / t \text{ BIOX product}) = \frac{a}{b} \times 1000$$

where a = mass lime added (g)
b = mass BIOX® product used for cyanidation (g)

The value are corrected for mass changes during BIOX® using the following equation:

$$\text{Corrected lime addition } (kg / t \text{ feed}) = \text{Lime addition } (kg / t \text{ BIOX product}) \times F1$$

Where F1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) / 100
a = mass change during BIOX® (%)

3. CYANIDE CONSUMPTION

$$\text{Cyanide consumption } (kg / t \text{ of BIOX product}) = \frac{(a - b)}{c} \times 1000$$

Where a = mass NaCN added (g)
b = mass NaCN not consumed (g)
c = mass of BIOX® product used for cyanidation (g)

The residual cyanide concentration is determined by titration of the cyanidation liquor with silver nitrate.

b is calculated as the product of the residual cyanide concentration (g/l) and the end volume of the filtrate.

Cyanide consumption values are corrected for mass changes during BIOX® using the following equation:

$$\text{Corrected cyanide consumption (kg / t feed)} = \text{cyanide consumption (kg / t BIOX product)} \times F_1$$

Where F_1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) / 100
a = mass change during BIOX® (%)

4. LIME CONSUMPTION

$$\text{Lime consumption (kg / t BIOX product)} = \frac{(a - b)}{c} \times 1000$$

Where a = mass lime added (g)
b = mass lime not consumed (g)
c = mass of BIOX® product used for cyanidation (g)

The residual lime (CaO) concentration is determined by titration of the cyanidation liquor with oxalic acid.

b is calculated as the product of the residual lime concentration (g/l) and the end volume of the filtrate.

Lime consumption values are corrected for mass changes during BIOX® using the following equation:

$$\text{Corrected lime consumption (kg / t feed)} = \text{lime consumption (kg / t BIOX product)} \times F_1$$

Where F_1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) / 100
a = mass change during BIOX® (%)

5. GOLD DISSOLUTION AFTER CYANIDATION

$$Gold\ dissolution\ (\%) = (1 - \frac{a}{b})\ x\ 100$$

where a = Au_{tail}(g/t) x $mass_{tail}$(g)
b = Au_{head}(g/t) x $mass_{head}$(g)

6. CYANIDATION RESIDUE GOLD CONTENT AFTER CORRECTION FOR MASS CHANGES DURING BIOX® AND CYANIDATION

The following calculation method is used to correct for mass changes during BIOX® and during cyanidation, such that the corrected gold value is comparable directly to the sample head assay.

$$Corrected\ Au_{tail}\ (g/t) = Assay\ Au_{tail}\ (g/t)\ x\ F_1\ x\ F_2$$

where F_1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) / 100
where a = mass change during BIOX® (%)
F_2 = relative mass ratio cyanidation product to BIOX® product
= (100 + b) / 100
b = mass change during cyanidation (%)

7.5. Cyanidation Log Sheets

Table A7.5.1 : Concentrate & Bio-oxidation Residue Leach Tests with Carbon

Sample ID	Biooxidation (Days)	Leach Time (hrs)	Initial Mass Dry Ore (g)	Mass Water added (g)	Reagent Addition							Residual NaCN				Residual CaO			Reagent Consumption CIL Feed		Final pH	Final Mass (g)	Mass of Bottle	Final Mass of Slurry+Bottle (g)	Final Volume (g)	Mass Adjust	Carbon Assayed (g)	Cyanidation				Head Ag (g/t)	Residue Ag (g/t)	Ag Dissln. (%)	Calculated Head Au (g/t)	Au Dissolution		Au Dissolution	Accountability Au (%)
					Lime	CaO	CaO	Carbon		NaCN		Titre	Aliquot	NaCN	NaCN End	Titre	Conc	CaO End	NaCN	CaO								Head Au	Residue Au	Carbon Au	Solution Au					Based on Residue	Based on Carbon and Solution	Based on Calc Head	
					(ml)	(g)	(kg/t)	(g)	(kg/t)	(g)	(kg/t)	(ml)	(ml)	(g/l)	(g)	ml	(g/l)	(g)	(kg/t	(kg/t								(g/t)	(g/t)	(g/t)	(mg/l)					(%)	(%)	(%)	
Direct CIL	-	24	204.8	828.2	1.6	0.16	0.78	16.37	80	4.0925	20.0	4.70	10.0	3.41	2.97	3.10	0.09	0.08	5.46	0.41	11.26	201.69	1220.52	2311.2	873	0.98	16.38	18.50	17.70	2.70	0.001	124.00	77.30	38.62	17.65	5.80	1.19	1.2	95
BAT 6	9	24	215.58	862.32	5.9	0.59	2.74	17.25	80	4.31	20.0	4.70	10.0	3.41	3.07	0.20	0.06	0.05	5.76	2.50	10.68	213.94	1348.42	2480.46	901	0.99	17.22	20.60	7.86	148	0.008	112.00	10.70	90.52	19.66	62.13	57.55	60.32	95
BAT 5	15	24	227.80	914.82	15.8	1.58	6.94	18.30	80	4.56	20.0	4.60	10.0	3.34	3.00	0.20	0.06	0.05	6.85	6.71	10.64	226.98	1386.70	2529.38	898	1.00	17.69	19.80	2.44	221	0.008	95.00	5.30	94.44	19.63	87.72	86.86	87.61	99
BAT 7	24	24	237.90	952.90	10.7	1.07	4.50	19.06	80	4.76	20.0	4.80	10.0	3.48	3.43	0.20	0.06	0.06	5.58	4.27	10.65	235.80	1350.20	2590.20	985	0.99	18.90	15.40	2.06	168	0.011	73.00	3.70	94.98	15.43	86.74	86.96	86.77	100
BAT 4	30	24	246.19	984.76	11.6	1.16	4.71	19.70	80	6.15	25.0	5.80	10.0	4.21	4.24	0.20	0.06	0.06	7.78	4.48	10.59	238.36	1226.98	2491.42	1008	0.97	18.40	15.90	1.83	198	0.006	85.00	7.00	92.03	16.60	88.84	93.22	89.31	104
BAT 3	37	24	175.15	700.60	10.9	1.09	6.22	14.01	80	4.38	25.0	6.90	10.0	5.00	3.46	0.20	0.06	0.04	5.25	6.00	10.62	170.69	1422.48	2297.52	691	0.97	13.13	16.30	1.65	188	0.008	50.00	24.00	53.22	15.73	90.14	86.66	89.78	97
BAT 1	45	24	236.56	948.80	9.7	0.97	4.10	18.98	80	5.91	25.0	6.40	10.0	4.64	4.50	0.20	0.06	0.05	5.97	3.87	10.52	228.94	1231.79	2449.69	970	0.97	18.97	17.10	1.82	175	0.084	91.00	8.23	91.25	16.14	89.70	84.08	89.09	94
BAT 2	45	24	203.63	815.24	10.2	1.02	5.01	16.30	80	5.09	25.0	6.90	10.0	5.00	4.14	0.20	0.06	0.05	4.67	4.78	10.54	200.96	1409.91	2454.68	828	0.99	16.30	18.10	1.59	179	0.087	62.00	7.22	88.51	16.25	91.33	81.11	90.34	90

7.6. Test Feed and Product Weights & Volume

Table A7.6.1 : Bio-oxidation Product Weights and Volumes and Across Tests

BAT Test Nr.	Mass In (g)	Mass Out (g)	Mass Change (%)	Filtrate Weight (g)	Filtrate Volume (ml)	Gold Balance			Fe Balance			As Balance			S(T) Balance		
						Gold in (mg)	Gold out (mg)	Balance (%)	Fe in (g)	Fe out (g)	Balance (%)	As in (g)	As out (g)	Balance (%)	S(T) in (g)	S out (g)	Balance (%)
BAT 6 - 9 Days	755.9	733.97	-12.72	2141	2000	13.98	15.1	108%	240.4	222.7	93%	33.2	27.8	84%	283.5	292.4	103%
BAT 5 - 15 Days	755.1	792	-5.73	2717	2480	13.97	15.7	112%	240.1	210.3	88%	33.1	23.1	70%	283.1	307.2	109%
BAT 7 - 24 Days	753.4	804	-4.11	2409	2045	13.94	12.4	89%	239.6	218.7	91%	33.1	24.4	74%	282.5	302.6	107%
BAT 4 - 30 Days	758.1	869	3.04	2539	2178	14.03	13.8	98%	241.1	207.7	86%	33.3	21.5	64%	284.3	266.5	94%
BAT 3 -37 Days	756.0	879	4.55	2031	2540	13.99	14.3	102%	240.4	192.2	80%	33.2	28.0	84%	283.5	258.7	91%
BAT 1 - 45 Days	756.5	804	-4.47	2982	2480	14.00	13.7	98%	240.6	201.9	84%	33.2	31.2	94%	283.7	252.4	89%
BAT 2 - 45 Days	754.1	810	-3.52	2823	2530	13.95	14.7	105%	239.8	186.9	78%	33.1	28.3	85%	282.8	252.8	89%

7.7. Neutralisation Log Sheets

Table A7.7.1 : Batch Neutralisation Test : AR Grade Limestone & Lime (Test 1)

Reagent imestone / Lime Test

Sample Velardena

Feed Make-up and Composition

BIOX® Filtrate	1000	ml
Dilution Water	2000	ml
Total Iron	18.50	g/l
As	3.14	g/l
pH	1.56	
Redox	616	mV

Reagent Make-up

Limestone slurry make-up	200	g/l
Lime slurry make-up	100	g/l
Precipitation Temperature		

pH increase

Time (min)	pH	Volume Added	Redox	Temperature (°C)	Comments
Limestone Addition (200 g/L)					
0	1.56	50	616	27.2	50 ml sample
5	1.82	20	609	27.3	
10	1.93	0	604	28.1	
15	1.93	20	604	27.4	
30	2.04	60	602	27.5	50 ml sample
45	2.21	80	592	27.3	
60	2.35	200	583	27.8	
75	2.53	43	539	28.0	
90	3.09	20	460	28.1	50 ml sample
105	3.43	25	404	28.1	
120	4.12	20	315	28.5	50 ml sample
135	4.47	30	262	26.7	
150	4.79	10	281	28.4	
165	4.95	10	284	28.4	
180	5.30	0	242	28.5	
Lime Addition (100 g/L)					
240	5.12	20	330	28.6	50 ml sample
255	5.86	5	328	28.8	
270	5.78	4	540	28.8	
285	6.23	20	456	28.2	50 ml sample
300	6.92	4	409	28.7	
360	7.08	0	484	28.7	50 ml sample

Sampling: Fe and As Assays

Sample No.	pH	Fe (ppm)	As (ppm)
1	1.56	18500	3140
2	2.04	18500	3120
3	3.09	52.7	<0.34
4	4.12	35.4	<0.34
5	5.12	4.41	<0.34
6	6.23	<0.02	<0.34
7	7.08	<0.02	<0.34

$CaCO_3$ Addition (ml)	588
CaO (ml)	53

filtration data	
Wet cake, g	1440.4
Dry cake, g	332.1
Filtrate volume, ml	1980.0
Filtrate mass, g	1969.2

Neutralised Product

Final pH	7.08
Final Total Fe in solution (ppm)	<0.02
Final As in solution(ppm)	<0.34

Neutralised Product Assays

Fe (%)	15.60
As (%)	2.50
Fe in TCLP extract (ppm)	<0.05
As in TCLP extract (ppm)	0.06

Reagent Additions

Limestone (kg/t concentrate)	470
Lime (kg/t concentrate)	21
Dry cake mass (g)	332.1

Table A7.7.2 : Batch Neutralisation Test : AR Grade Lime Only Test 2)

Reagent Lime

Sample Velardena

Feed Make-up and Composition

BIOX® Filtrate	1000	ml
Dilution Water	2000	ml
Total Iron	18.50	g/l
As	3.26	g/l
pH	1.56	
Redox	617	mV

pH increase

Time (min)	pH	Volume Added	Redox	Temperature (°C)	Comments
Lime Addition (100 g/L)					
0	1.56	50	617	27.0	50 ml sample
5	1.75	20	612	27.7	
10	1.83	40	609	28.5	
15	1.97	10	605	28.5	
30	2.03	100	605	28.5	50 ml sample
45	2.25	150	592	29.7	
60	2.30	200	586	31.3	
75	2.58	55	542	33.3	
90	3.07	30	487	33.3	50 ml sample
105	3.57	20	433	32.7	
120	4.05	30	359	32.7	50 ml sample
135	4.72	10	268	32.3	
150	4.77	10	304	32.0	
165	5.00	0	321	31.7	
180	5.08	5	286	31.3	
Lime Addition (100 g/L)					
240	5.03	20	355	30.2	50 ml sample
255	5.72	8	330	30.2	
270	5.74	6	516	30	
285	6.17	19	447	29.5	50 ml sample
300	6.86	7	405	29.5	
360	7.05	0	479	29.3	50 ml sample

Reagent Make-up

Limestone slurry make-up		g/l
Lime slurry make-up	100	g/l
Precipitation Temperature		

Sampling: Fe and As Assays

Sample No.	pH	Fe (ppm)	As (ppm)
1	1.56	18500	3260
2	2.03	17900	3060
3	3.07	37.5	<0.34
4	4.05	<0.02	<0.34
5	5.03	<0.02	<0.34
6	6.17	<0.02	<0.34
7	7.05	<0.02	<0.34

CaO (ml)	790

Neutralised Product

Final pH	7.05
Final Total Fe in solution (ppm)	<0.02
Final As in solution(ppm)	<0.34

filtration data	
Wet cake, g	828.5
Dry cake, g	327.3
Filtrate volume, ml	2720.0
Filtrate mass, g	2720.9

Neutralised Product Assays

Fe (%)	15.50
As (%)	2.48

Fe in TCLP extract (ppm)	<0.05
As in TCLP extract (ppm)	0.07

Reagent Additions

Limestone (kg/t concentrate)	0
Lime (kg/t concentrate)	316

Dry cake mass (g)	327.3

7.8. Static Settling Test Log Sheets

Table A7.8.1 : Flocculant Polymer Selection Test for Bio-oxidation Slurry

	Flocculants Tested							
(Dosage: 100 g/t at 0.05% floc strength) : ~7% Solids	**Magna 10**	**Magna 155**	**Magna 333**	**Magna 919**	**Magna 345**	**Magna 455**	**M336**	**M405**
Time (seconds)								
15	83	14	96	4	7	79	7	73
30	108	29	114	4	19	106	7	108
45	116	47	120	5	22	117	17	118
60	120	59.5	123	6	22	120	21	123
120	126	111	127	9	22	126	44	131
180	128	135.5	129	9	22	128	68	135
240	128	140	129	10	23	130	87	135
300	128	142	129	10	25	130	106	135
Settling time (Seconds)	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**
Displacement (mm)	**108**	**29**	**114**	**4**	**19**	**106**	**7**	**108**
Settling rate (m/h)	**13.0**	**3.5**	**13.7**	**0.5**	**2.3**	**12.7**	**0.8**	**13.0**
Overflow Clarity	CLEAR	CLEAR	CLEAR	CLOUDY	CLOUDY	CLEAR	CLOUDY	CLEAR

Table A7.8.2 : Flocculant Dosage Optimisation for Bio-oxidation Slurry

SETTLING RATE DATA FOR VELARDENA

Time (min)	**Flocculent (Magnafloc 333), 0.05% Strength**				
	0 g/t	**50 g/t**	**100 g/t**	**150 g/t**	**77 g/t**
	Displacement (mm)				
0.25	5	44	54	69	59
0.5	7	80	102	118	104
0.75	21	110	129	137	140
1	26	129	143	147	158
2	26	155	163	165	185
3	26	166	170	171	198
4	26	172	176	175	204
5	26	174	178	178	206
10	41	178	181	180	211
15	68	180	182	182	212
30	195	182	185	184	213
45	196	182	185	185	214
60	196	182	185	185	214
120	199	183	186	186	216
1440	202	184	187	186	217
Clarity	Cloudy	Clear	Clear	Clear	Clear
Final sludge vol (ml)	70	95	110	115	100
Final sludge SG	1.47	1.36	1.32	1.32	1.35
Final sludge % solids	36.35	28.30	25.52	25.26	27.55
Final sludge L/S ratio	1.75	2.53	2.92	2.96	2.63
Solids SG	4.30	4.30	4.30	4.30	4.30
Sample liquor SG	1.19	1.19	1.19	1.19	1.19
Sample Slurry SG (before dilution)	1.34	1.34	1.34	1.35	1.34
Test start SG (after dilution)	0.84	1.12	1.12	1.13	1.12
Slurry aliquot sample (ml)	180.00	180.00	180.00	180.00	180.00
Solids taken (g)	37.38	36.44	37.05	38.28	37.08
Liquid taken (ml)	171.31	171.53	171.38	171.10	171.38
Diluent liquor SG	1.00	1.00	1.00	1.00	1.00
Diluent liquor (ml)	315.69	320.09	320.56	314.70	317.30
Test liquor SG	1.07	1.07	1.07	1.07	1.07
Start L/S ratio	13.91	14.39	14.16	13.55	14.06
Start % solids	6.71	6.50	6.59	6.87	6.64
Settling time (min)	0.50	0.50	0.50	0.50	0.50
Displacement (mm)	7.00	80.00	102.00	118.00	104.00
Settling rate (m/h)	0.84	9.60	12.24	14.16	12.48
Net area required (m^2.h/t)	13.56	1.16	0.86	0.70	0.86
Recommended area (m^2.h/t)	33.89	2.89	2.15	1.75	2.15
Solids loading (t/m^2.h)	0.03	0.35	0.46	0.57	0.47

NOTE: Recommended area includes a safety factor of 2.5

CONFIDENTIAL

REPORT NO BIOMET 07/07

BIOOXIDATION TEST PROGRAM ON VELARDENA CONCENTRATE

20 November, 2007
by

T Stanek : Manager BioMet & Fusion
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Client: GFL Mining Services Ltd

DISTRIBUTION

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SYNOPSIS

Approximately 28.8 kg of the Velardena concentrate was received on the 10 July, 2007 from the client. 750 g of the concentrate was removed to start the culture adaptation. The concentrate was thoroughly mixed and representative samples were removed for chemical and particle size analyses.

The direct cyanidation leach test carried out on the Velardena concentrate yielded a gold dissolution of 3.9%.

The chemical composition of the Velardena concentrate is given in the table below.

Analysis	Unit	Velardena Concentrate
Au	g/t	23.6
Ag	g/t	197.5
As	%	11.0
Fe	%	37.2
S(T)	%	41.4
S(elemental)	%	<0.5
S(sulphide)	%	37.0
C(T)	%	0.1
C(organic)	%	0.1
C(graphitic)	%	0.05
CO_3^{2-}	%	<0.05

A BIOX® mesophile bacterial culture already adapted to a pyrite concentrate was used as inoculum for adaptation to the Velardena concentrate.

The sulphide, arsenic removal and the gold and silver dissolution results for the mesophilic batch biooxidation amenability tests carried out on the Velardena concentrate are summarised in the table below.

Biooxidation Treatment Period (Days)	Sulphide Removal (%)	Arsenic Dissolution (%)	Silver Dissolution (%)	Gold Dissolution (%)
0	-	-	-	3.9
5	12.5	37.1	90.8	51.7
8	20.1	54.1	91.3	62.2
12	28.5	77.9	89.1	74.9
15	35.7	88.0	95.6	80.4
25	83.4	95.2	93.7	91.5
36	96.8	96.5	92.6	91.0
40	96.0	95.5	51.8 (50.1)*	91.1

* (value in brackets is the result obtained on the residue of the repeat CIL test)

The Velardena concentrate is amenable to biooxidation treatment. This was confirmed by the following:

The maximum sulphide oxidation achieved during the batch biooxidation amenability tests of the Velardena concentrate was 96.8% after 36 days biooxidation. The corresponding gold dissolution was 91.0%. A gold dissolution of 91.5% was achieved after 25 days biooxidation, corresponding to 83.4% sulphide oxidation. The results indicate that complete sulphide sulphur oxidation may not be necessary to achieve maximum gold dissolution.

The Velardena concentrate was net acid generating (310.4 kg H_2SO_4/t of feed concentrate).

A mass increase was observed during the biooxidation of the Velardena concentrate.

The NaCN consumptions during cyanidation of the biooxidation products varied between 5.0 and 12.2 kg NaCN/t concentrate feed.

The TCLP testing of the neutralisation precipitates produced extracts containing <3 ppm As, below the 5 ppm limit set by the EPA. The precipitates can be considered stable and thus acceptable for disposal on a tailings dam.

CONTENTS

1. INTRODUCTION

BIOX® is a biohydrometallurgical process for the pre-cyanidation treatment of refractory gold ores. This process offers an alternative to conventional roasting or pressure oxidation techniques.

The nucleus of the BIOX® process is the exploitation of a naturally occurring mixed bacterial population consisting of: *Acidithiobacillus ferrooxidans, At. thiooxidans* (oxidises sulphur compounds only) and *Leptospirillum ferrooxidans* (oxidises iron substrates only). Due to the possession of a chemolithotrophic mode of metabolism, these bacteria are able to oxidise gold-bearing sulphide ores and concentrates under controlled conditions. Inorganic substrates such as sulphides, elemental sulphur and ferrous iron are oxidised by the bacteria to provide chemical energy. This is enzymatically converted, by oxidative phosphorylation, to ATP, a form of metabolic energy utilised by the bacteria for various cellular functions. The carbon requirements of the bacteria for biosynthesis of cellular biomass are met by CO_2 in the atmosphere or from dissolution of carbonate minerals in the ore.

A wide range of metal sulphide minerals can be oxidised by the mixed bacterial population. These include pyrite, arsenopyrite, pyrrhotite, chalcopyrite, chalcocite, covellite, stibnite, pentlandite and galena. The mechanism of sulphide breakdown is usually a combination of direct enzymatic attack and indirect chemical activity of metabolic by-products of the bacteria. For direct enzymatic oxidation, attachment of the bacteria to the sulphide mineral is essential. Bacterial oxidation of pyrite and arsenopyrite are illustrated via the following equations:

$$4FeS_2 + 15O_2 + 2H_2O \longrightarrow 2Fe_2(SO_4)_3 + 2H_2SO_4 \qquad (1)$$

$$2FeAsS + 7O_2 + H_2SO_4 + 2H_2O \longrightarrow 2H_3AsO_4 + Fe_2(SO_4)_3 \qquad (2)$$

The ferric sulphate produced may contribute to further sulphide breakdown by indirect chemical attack:

$$2FeAsS + Fe_2(SO_4)_3 + 6O_2 + 4H_2O \longrightarrow 2H_3AsO_4 + 4FeSO_4 + H_2SO_4 \qquad (3)$$

Partial arsenopyrite oxidation may also occur by acid attack:

$$4FeAsS + 5O_2 + 4H_2SO_4 \longrightarrow 4HAsO_2 + 4FeSO_4 + 4S^\circ + 2H_2O \qquad (4)$$

The ferrous sulphate and elemental sulphur are then bacterially re-oxidised to ferric sulphate and sulphuric acid, respectively:

$$4FeSO_4 + 2H_2SO_4 + O_2 \longrightarrow 2Fe_2(SO_4)_3 + 2H_2O \qquad (5)$$

$$2S^\circ + 3O_2 + 2H_2O \longrightarrow 2H_2SO_4 \qquad (6)$$

The arsenic acid produced from the oxidation of arsenopyrite is efficiently neutralised with limestone and/or lime to form non-polluting ferric arsenate precipitates. These precipitates are stable provided the Fe:As molar ratio in the BIOX® liquor is greater than 3:1.(1,2)

The BIOX® process comprises contacting the refractory sulphide ore/concentrate with a strain of the BIOX® mixed bacterial culture for a suitable treatment period while maintaining an optimum operating environment. The bacteria oxidise the sulphide minerals, thereby liberating the occluded gold for subsequent recovery by cyanidation.

Under controlled continuous plant conditions, the number of bacterial cells and their activity is optimised to attain the highest rate of sulphide oxidation. Detailed laboratory and pilot plant studies have indicated that the bacteria require a very acidic environment (pH 1.4 to 1.8), a temperature of between 30°C and 45°C and a steady supply of oxygen and carbon dioxide for optimum growth and activity. The unusual operating conditions optimum for the BIOX® bacteria are not favourable for the growth of most other microbes thus eliminating the need for sterility during the BIOX® process. Due to the fact that organic substances are toxic to the BIOX® bacteria, they are non-pathogenic and incapable of causing disease. The bacteria employed in the BIOX® process do not, therefore, pose a health risk to humans or other animals.

Batch BIOX® amenability tests evaluate the amenability of a concentrate to the BIOX® process and identify potential process problems prior to pilot testing.

The results of the testwork are summarised in this report.

2. EXPERIMENTAL PROCEDURE

2.1 Sample Preparation

Approximately 28.8 kg of the Velardena concentrate was received on the 10 July, 2007 at **SGS LAKEFIELD** Research Africa (Pty) Ltd. The concentrate was screened through an 850 µm screen, thoroughly mixed and representative samples were taken for chemical and particle size analyses, direct cyanidation and the batch amenability tests.

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2.2 Bacterial Adaptation

A BIOX® mesophile bacterial culture already adapted to a pyrite concentrate was used as inoculum for adaptation to the Velardena concentrate. The adaptation and inoculum build up stages were carried out as follows:

The adaptation of the culture was carried out in 9K nutrient medium (see Appendix I). The batch was set up in a 5 liter beaker, containing 3 liters of nutrient medium. Initially 250 g of the concentrate was added. Sufficient concentrated H_2SO_4 was added and the slurry allowed to acid stabilise overnight, with aeration and agitation to a stable pH of 1.6 - 1.8, before inoculating with the selected culture (10 % v/v).

The adaptation batch was maintained at 39°C - 41°C with constant agitation and aeration. Daily pH adjustments (by addition of acid or lime) to maintain the operating pH between 1.4 and 1.6, Redox measurements, and ferrous and ferric ion titrations were carried out. As bacterial activity increased (when the ferric concentration reached a maximum limit and the ferrous ion concentration decreased to below 1 g/l) another 2 x 250 g of the concentrate was added to the adapted batch. The adaptation stage was completed in 13 days and the resulting culture was used to inoculate the inoculum build-up stage.

The inoculum build-up stage on the concentrate was performed in 0K nutrient media. A 5 liter beaker containing 3 liters of 0K medium and 750 g of the concentrate was set up. The slurry was acid stabilised and then inoculated with 10 % (v/v) inoculum from the adaptation stage. The same conditions and monitoring methods were used as for the adaptation stage. The inoculum build-up batch was ready for inoculation of the batch amenability tests 19 days after inoculation.

2.3 Batch Amenability Tests

The tests were conducted in 5 litre vessels on a Multiple Stirrer Bench (MSB) to ensure similar leaching conditions. Seven batch tests were performed on the Velardena concentrate.

The batch tests were carried out under the following conditions:

L/S ratio:	4:1 (750 g concentrate + 3 liters 0K nutrients)
Acid stabilisation of slurry:	By addition of 98% conc. H_2SO_4 to achieve a stable pH of about 1.6 and allowing to stabilise overnight with agitation and aeration.
Inoculum:	10% (v/v) of active culture, as described in section 2.1.
pH:	Maintained at pH 1.4 - 1.6 by daily additions of acid or lime, as required.
Temperature:	40°C ± 1°C
Aeration:	Dissolved O_2 concentrations of 3.5ppm - 4.5ppm.
Agitation:	Approximately 460rpm : Pitch Blade Turbine.
Monitoring methods:	Fe^{2+} and Fe^{3+} titrations, pH and Redox measurements.

In the 25, 36 and 40 day amenability tests, after 15 and 22 days biooxidation periods, 1 litre of leach solution was removed and replaced with 1 litre of 0K nutrients in order to reduce the total ion concentration in solution.

At the completion of each batch test, the slurries were filtered and washed with an acid solution of 0.3% H_2SO_4 (approximately pH 1.6). The filter cakes were repulped, filtered and re-washed. The final filter cake was dried at 60°C, then crushed, screened and submitted for post-biooxidation chemical analysis and cyanide leach tests. The biooxidation liquors (filtrates) were also submitted for chemical analysis. All equations used in the calculations are given in Appendix II.

2.4 Cyanidation Leach Tests

Cyanidation leach tests were conducted on Velardena batch amenability test products. A direct cyanidation leach test was also carried out on the concentrate. The standard cyanidation procedure adopted for the cyanidations was as follows:

i. Slurries were prepared at a L/S ratio of 4:1.

ii. The pH was adjusted to approximately pH 11.5 by the bulk addition of lime and the slurry preconditioned for 30 minutes to 1 hour.

iii. This was followed by the addition of 12.5 g/l activated carbon and 20 kg/t sodium cyanide to the leach slurry.

iv. Leaching was allowed to progress for 24 hours. The pH was adjusted, when necessary, to maintain the pH at 10.5 - 11.5.

v. After completion of the leach, the loaded carbon was removed by screening, and the cyanidation slurries filtered, washed and dried at 60°C.

vi. Filtrates were analysed for residual cyanide and lime.

vii. The cyanidation residue and the biooxidation product head samples were analysed for gold. The gold dissolutions were calculated after correction for mass changes during cyanidation.

All cyanidation equations are detailed in Appendix III.

2.5 Neutralisation of the Biooxidation Liquor

A single batch neutralisation test was carried out on the 40 day biooxidation liquor using AR-Grade limestone and lime as the neutralising reagents.

The objective of the batch neutralisation test was to confirm that precipitation of arsenic from solution as ferric arsenate will produce a slurry suitable for safe disposal on a tailings dam. The procedure for the neutralisation test was as follows:

i. An aliquot of biooxidation liquor was poured into a beaker and diluted with tap water. The dilution with tap water was done to simulate the effluent produced during CCD (counter-current decantation) operations. Agitation and aeration were introduced and the mixture allowed to stabilise over a 15 minute period.

ii. The iron and arsenic concentrations as well as the pH and Redox potential (mV) of the neutralisation solution was measured. AR-Grade limestone (200 g/l concentration) was added over a period of 3 hours to achieve a pH of 5.

iii. The test was allowed to progress for 1 hour at pH 5. AR-Grade lime (100 g/l concentration) was added to the test to increase to pH of the neutralisation slurry to pH 7. The slurry was allowed to condition at pH 7 for 1 hour before the test was terminated.

iv. Representative portions of the slurry from the test neutralised to pH 2, 2.5, 3, 4, 5 and 6 were filtered. The filtrates were collected and analysed for As and Fe.

v. At the end of the neutralisation tests the slurry was filtered and dried at 60 ^{o}C. The neutralisation product mass was recorded for the test. The neutralised liquor and product were submitted for Fe and As analysis. A representative portion of the neutralised product was submitted for TCLP testing.

The TCLP test is a standard procedure developed by the Environmental Protection Agency (EPA) in the USA to evaluate the stability of arsenic precipitates. The requirement is that the TCLP extract should contain less than 5 ppm soluble arsenic, in order to classify the material as stable with respect to arsenic solubilisation.

2.6 Settling Rate Tests

Laboratory scale settling tests were conducted on the 40 day BIOX® slurry. The objective of the tests was to determine the optimum flocculant addition, settling rate and required thickener area. The settling testwork involved two types of tests, flocculant selection tests and flocculant dosage optimisation tests. The BIOX® slurry was diluted with potable water to contain 11.5% solids.

The flocculant dosage optimisation settling rate tests were carried out by flocculation of slurry aliquots in a 500 ml glass measuring cylinder. The following procedure was used:

i. The prepared settling test slurry was transferred to a 500 ml measuring cylinder.

ii. All the solids were suspended by inversion of the cylinder. The flocculant was rapidly injected and, without delay, the cylinder was violently inverted twice and then once gently.

iii. The cylinder was placed on a horizontal surface and the timer started. At set intervals the mud line was marked on a masking tape strip on the outside wall of the cylinder. The final mud volume and mass were recorded for calculation of the underflow slurry density.

iv. The tape strip was removed and the settling rate calculated from the mud line measurements. The solids content of the settled slurry and the solution and dry solid densities were determined.

3. RESULTS AND DISCUSSION

3.1 Sample Characterisation

The results of the particle size analysis of the Velardena concentrate are given in Table 1.

Table 1 : Particle Size Analysis of the Velardena Concentrate

Size (µm)	% Passing
-300	100.0
-212	99.7
-150	97.3
-106	90.5
-75	76.5
-53	55.9
-45	46.3
-38	37.3

The ideal particle size distribution for BIOX® amenability tests is 80% -75µm (95% passing 150 µm) and a minimum of minus 20µm particles. Although the rate of sulphide oxidation is enhanced by a finer grind, it can cause problems in downstream unit operations, like the CCD and CIP.

The results of the concentrate chemical analyses are given in Table 2.

Table 2 : Pre-BIOX® Chemical Analyses of the Velardena Concentrate

Analysis	Unit	Velardena Concentrate
Au	g/t	23.6
Ag	g/t	197.5
As	%	11.0
Fe	%	37.2
S(T)	%	41.4
S(elemental)	%	<0.5
S(sulphide)	%	37.0
C(T)	%	0.1
C(organic)	%	0.1
C(graphitic)	%	0.05
CO_3^{2-}	%	<0.05

Two important factors in determining the amenability of a sample to BIOX® are the Au:S^{2-} and S^{2-}:CO_3^{2-} ratios. The Au(g/t):S^{2-}(%) ratio of the Velardena concentrate was 0.6:1. As a rule of thumb, a Au(g/t):S^{2-}(%) ratio of >0.7 is required to make biooxidation pre-treatment economically viable. The Velardena concentrate carbonate content was <0.05%. In general a low sulphide to carbonate ratio, indicates that the addition of acid may be required during the initial stages of biooxidation in a commercial plant (primary reactors). The S^{2-}:CO_3^{2-} ratio, the rate of oxidation and the solubility of the carbonates will determine the acid and/or lime requirements during the different stages of the biooxidation operation.

The advantage of having CO_3^{2-} in the concentrate is that the CO_2, released from the carbonate minerals during biooxidation, can be utilised by the bacteria for cell biomass production. In the case where an insufficient amount of CO_3^{2-} is present in the concentrate, an external source of CO_2 may be provided on a continuous plant by the addition of limestone during pH control.

3.2 Batch Amenability Tests

The adaptation and inoculum build-up, the batch tests daily logsheets and the feed and final product weights and volumes are given in Appendices IV, V and VI respectively. The progress of the 40 day test on the Velardena concentrate is illustrated in Figure 1.

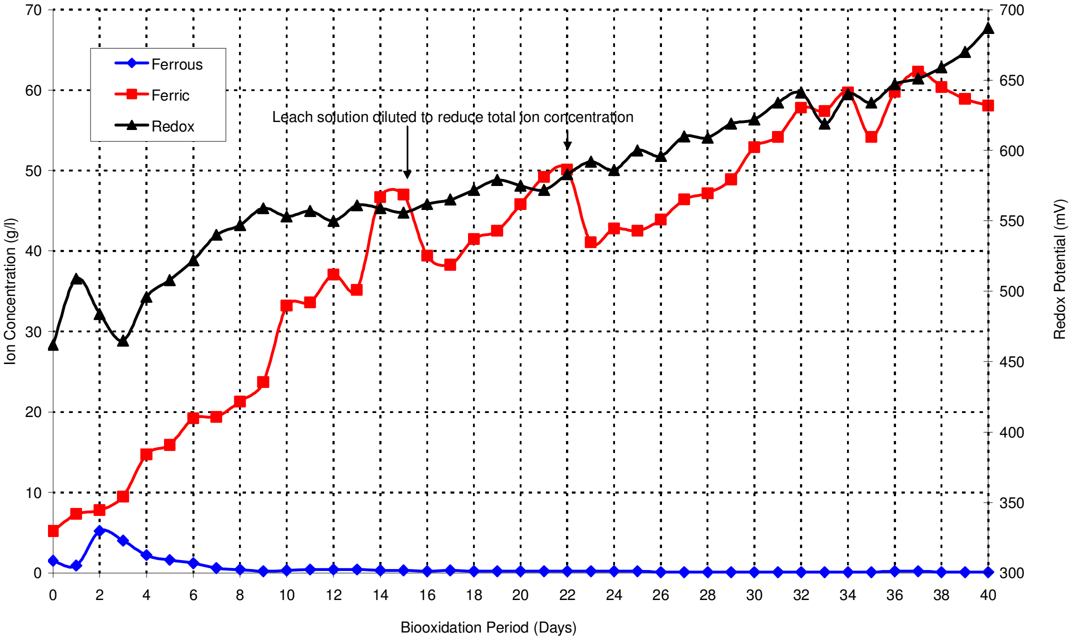

Figure 1 : Progress of the Velardena Concentrate Batch Amenability Test

A decrease in the ferrous ion concentration was observed after inoculation in all the tests. The ferric ion concentration in all the tests increased progressively. Decreases observed in the ferric ion concentrations during the tests could have been due to the precipitation of iron salts (mainly as jarosite) from solution when lime was added to control the slurry pH within the optimum range for bacterial growth.

This, however, does not necessarily indicate that dissolution of the iron sulphides was complete. Ferrous and ferric ion concentrations are used strictly for monitoring purposes during the batch tests. The results give an indication of bacterial activity due to their ability to oxidise ferrous ion to ferric ion. Due to the precipitation of jarosite and ferric arsenate during biooxidation, the iron content of the solution phase cannot be used directly to calculate the solubilisation of the sulphide minerals.

Mass changes and limestone requirements during biooxidation are given in Table 3.

Table 3 : Mass Changes and Lime Requirements During the Biooxidation of the Velardena Concentrate

Biooxidation Treatment Period (Days)	Mass Change During BIOX® (%)	Initial Acid Addition (kg/t feed)	Total Acid Addition (kg/t feed)	Lime Consumption (kg/t feed)	Net Acid Consumption/ Generation* (kg/t feed)
5	-12.4	46.6	46.6	12.4	24.9
8	-5.4	46.6	46.6	55.5	-50.4
12	-10.9	46.6	46.6	69.8	-75.5
15	-8.4	49.1	49.1	101.8	-128.9
25	4.2	49.1	49.1	158.6	-228.2
36	13.6	49.1	49.1	202.3	-304.7
40	5.9	49.1	49.1	205.6	-310.4

*A positive value indicates acid consumption
A negative value indicates acid generation

The Velardena concentrate tests showed an overall mass increase of between 4.2 and 13.6%. The addition of sulphuric acid was required for the pH stabilisation of the concentrate prior to inoculation.

The lime consumption during biooxidation of the 40 day Velardena concentrate test was 205.6 kg/t feed. The Velardena concentrate was net acid generating during biooxidation.

The mass change and lime consumption results for the Velardena concentrate batch amenability tests are presented graphically in Figure 2.

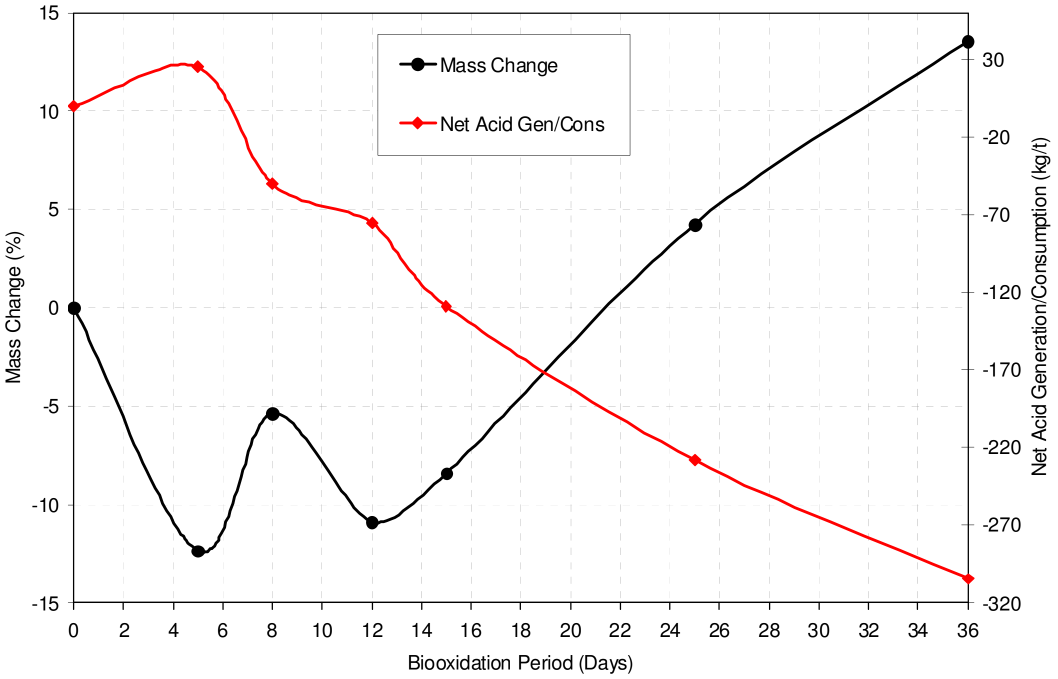

Figure 2 : Mass Changes and Lime Consumptions for the Velardena Concentrate Batch Amenability Tests

The mass change curve for batch amenability tests shows an overall increase in mass. The initial mass loss is attributed to the dissolution of acid soluble minerals during the acid stabilisation stage and sulphide mineral oxidation. Although a lower mass is beneficial in reducing capital cost for the downstream treatment circuits it is not recommended to operate a biooxidation plant at a pH level lower than pH 1.2. Experience has shown that the continuous operation of a biooxidation circuit at pH levels lower than pH 1.2 results in the selection of the iron oxidising bacterium and reduces the overall rate and extent of sulphide oxidation.

Results of analyses of the biooxidation products and solutions are summarised in Table 4.

Table 4 : Analyses of the Products and Solutions of the Velardena Biooxidation Tests

Biooxidation Treatment Period (Days)	Residue Analyses [a]							Solution Analyses (g/l)			
	S(T) (%)	S^{2-} (%)	S^{o} (%)	Fe(T) (%)	As (%)	C(organic) (%)	CO_3^{2-} (%)	S(T)	Fe(II)	Fe(III)	As(T)
0	41.4	37.0	<0.5	37.2	11.0	0.1	<0.05	-	-	-	-
5	35.1	29.9	0.3	29.3	6.6	0.1	0.3	20.2	1.0	23.9	16.8
8	34.6	27.3	0.5	34.6	4.8	0.2	0.4	21.6	0.5	31.0	16.7
12	30.5	24.4	0.6	20.3	2.3	0.2	0.8	28.6	0.4	51.9	28.8
15	29.0	21.9	0.4	17.5	1.3	0.2	0.9	24.9	0.2	50.6	25.3
25	19.6	5.7	0.2	8.1	0.5	0.2	1.3	29.1	0.1	46.4	13.7
36	17.9	1.1	1.7	4.9	0.4	0.2	1.4	44.1	0.1	67.8	15.9
40	17.7	1.4	0.2	22.6	0.5	0.2	1.5	32.8	0.1	56.4	14.7

[a] Corrected for mass change
[b] Assayed values

The analyses of the products from the concentrate biooxidation amenability tests indicate a decrease in iron and arsenic concentrations with time. The pyritic sulphur content also decreased with time. The inverse trend (shown in Figure 3) in the liquor analyses confirms the oxidation of sulphide minerals and the dissolution of sulphur, iron and arsenic.

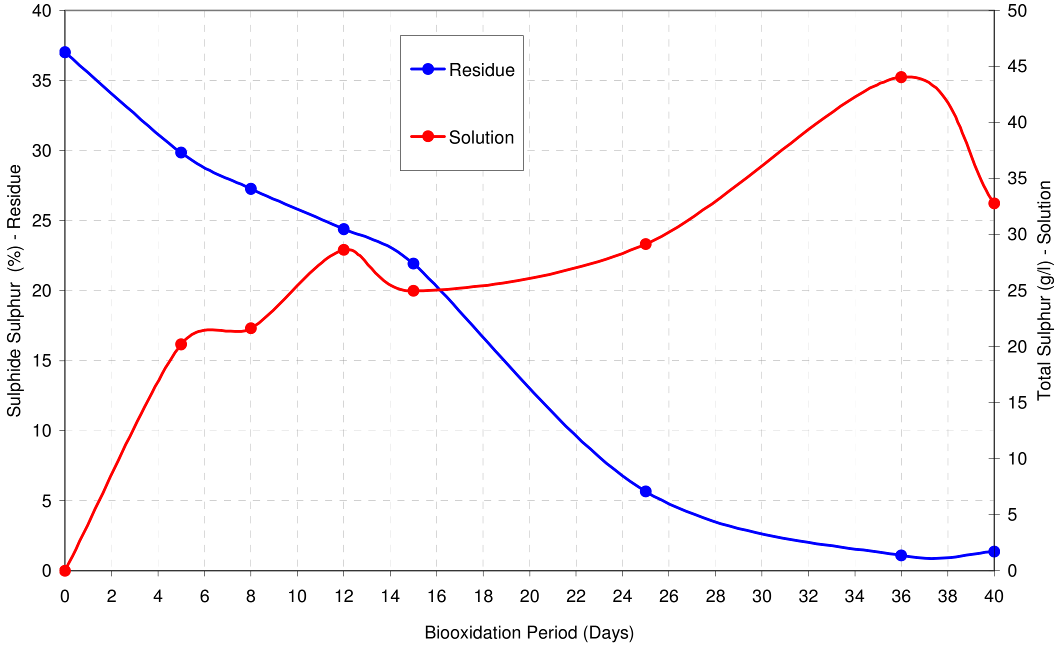

Figure 3 : Sulphide Sulphur in the Velardena Concentrate Product and Total Sulphur in Solution Profiles

The sulphide and arsenic removal and the gold and silver dissolution results are summarised in Table 5.

Table 5 : Sulphide Removal, Arsenic Solubilisation and Corresponding Gold and Silver Dissolution During the Biooxidation of the Velardena Concentrate

Biooxidation Treatment Period (Days)	Sulphide Removal (%)	Arsenic Dissolution (%)	Silver Dissolution (%)	Gold Dissolution (%)
0	-	-	-	3.9
5	12.5	37.1	90.8	51.7
8	20.1	54.1	91.3	62.2
12	28.5	77.9	89.1	74.9
15	35.7	88.0	95.6	80.4
25	83.4	95.2	93.7	91.5
36	96.8	96.5	92.6	91.0
40	96.0	95.5	51.8 (50.1)*	91.1

* (value in brackets is the result obtained on the residue of the repeat CIL test)

The sulphide oxidation and gold dissolution profiles against biooxidation treatment time are presented in Figure 4.

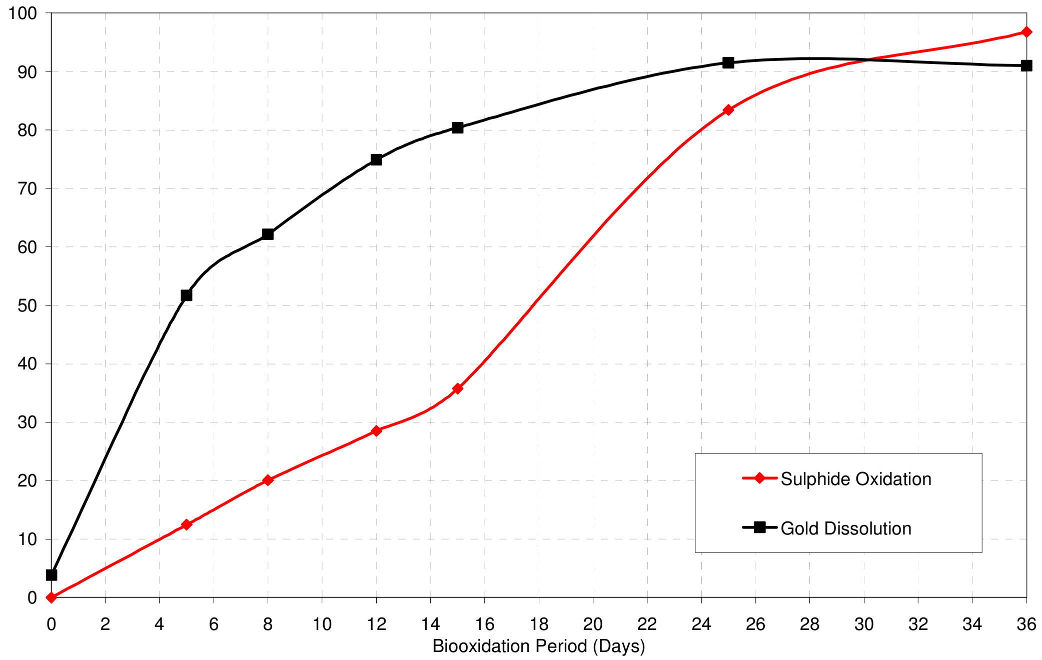

Figure 4 : Sulphide Sulphur Oxidation and Gold Dissolution vs Biooxidation Treatment Time

3.3 Cyanidation Tests

The results of the direct and biooxidation product cyanidation tests are given in Tables 6 and 7. The detailed logsheets for each of the cyanidation leach tests carried out are given in Appendix VII.

Table 6 : Gold Dissolutions vs Treatment Time

Biooxidation Treatment Period (days)	Head Gold Grade (g/t)	Residue Gold Grade (g/t)	Corrected Residue Gold Grade (g/t)	Gold Dissolution (%)	Corresponding Pyritic Sulphur Oxidation (%)
0	23.6	23.1	22.7	3.9	-
5	26.7	13.9	12.9	51.7	12.5
8	22.3	9.2	8.4	62.2	20.1
12	24.3	6.5	6.1	74.9	28.5
15	22.3	4.6	4.4	80.4	35.7
25	20.9	1.9	1.8	91.5	83.4
36	17.3	1.7	1.6	91.0	96.8
40	17.9	1.7	1.6	91.1	96.0

Biooxidation of the Velardena concentrate improved the gold dissolution from 3.9% on direct cyanidation of the unoxidised concentrate to 91.5% after 25 days biooxidation.

Table 7 : Conditions and Reagent Consumptions During the Cyanidation Tests

Biooxidation Treatment Period (days)	Reagent Addition (kg/t BIOX® Product)		Reagent Addition Corrected (kg/t BIOX® Feed)		Reagent Consumption (kg/t BIOX® Product)		Reagent Consumption Corrected (kg/t BIOX® Feed)		Residual Cyanide in Solution	End pH	Mass Change During Cyanidation
	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO	(ppm)		(%)
0 Direct	20.0	2.4	-	-	7.5	1.6	-	-	2930	11.78	-1.57
5	20.0	4.8	17.5	4.2	10.1	4.5	8.9	4.0	2074	10.95	-7.07
8	20.0	6.2	18.9	5.8	11.2	6.1	10.6	5.8	1987	11.00	-8.41
12	20.0	6.2	17.82	5.5	13.6	6.1	12.2	5.4	1378	10.99	-5.45
15	20.0	6.2	18.3	5.7	5.5	5.4	5.0	5.0	3118	11.60	-5.29
25	20.0	12.8	20.9	13.4	11.3	12.6	11.8	13.1	1842	10.62	-4.52
36	20.0	2.9	22.7	3.2	8.7	2.4	9.9	2.8	2683	11.44	-6.37
40	20.0	3.3	21.2	3.5	7.6	2.8	8.1	2.9	2553	11.46	-4.18

The cyanide consumptions during cyanidation of the biooxidation products of the Velardena concentrate varied between 5.0 and 12.2 kg NaCN/ton concentrate feed. The lime consumptions for the same tests varied between 2.8 and 13.1 kg CaO/ton concentrate feed.

The relationship between gold dissolution and sulphide oxidation is shown in Figure 5.

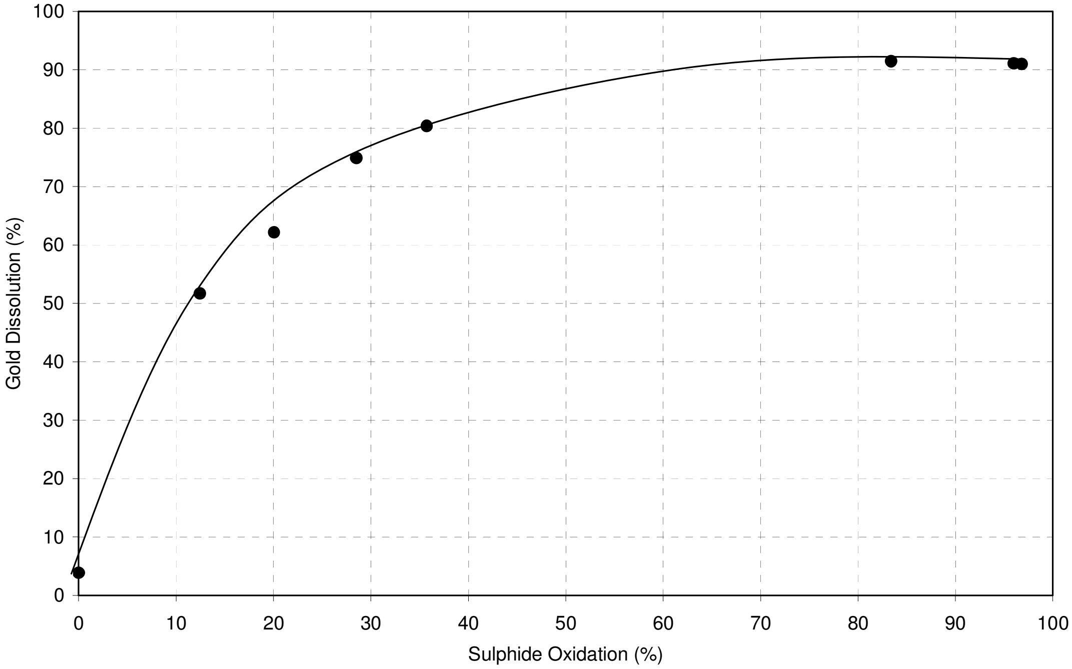

Figure 5 : Gold Dissolution vs Sulphide Sulphur Oxidation

3.4 Neutralisation of the Biooxidation Liquor

The results of the batch neutralisation test conducted on the 40 day test biooxidation liquor sample using AR-Grade limestone and lime are summarised in Table 8. The logsheet for the batch neutralisation test is given in Appendix VIII.

Table 11 : Results of the Batch Neutralisation Tests of the Velardena Biooxidation Liquor

Feature	Units	Results
		AR Grade Limestone / Lime
Analysis of Neutralisation Feed:		
Fe(T)	g/l	18.6
As(T)	g/l	4.5
Fe:As	Molar	5.2
Biooxidation liquor volume	Ml	1000
Water volume	Ml	2000
Total solution volume	ml	3000
Analysis of Limestone Feed:		
$CaCO_3$ concentration	g/l	200
Volume added	ml	400
$CaCO_3$ consumption	kg/t BIOX® feed	320
Analysis of Lime Feed:		
CaO concentration	g/l	100
Volume added	ml	155
CaO consumption	kg/t BIOX® feed	62
Total Ca^{2+} consumption	kg/t BIOX® feed	172.3
Analysis of Neutralisation Precipitate:		
Fe	%	16.8
As	%	4.6
Analysis of Neutralisation Liquor:		
Fe	ppm	0.1
As	ppm	<0.34
Analysis of TCLP extract Neutralisation precipitate:		
Fe	ppm	<0.02
As	ppm	0.59

The results in Table 8 indicate that the arsenic in the biooxidation liquor can be successfully removed from solution by treatment with AR-Grade limestone and lime. The neutralised effluent contained <0.34 ppm As.

The TCLP testing of the neutralisation precipitate produced an extract containing 0.6 ppm As, below the 5 ppm limit set by the EPA. The precipitate can be considered stable and thus acceptable for disposal on a tailings dam.

The results of the neutralisation tests are graphically presented in Figures 6 and 7.

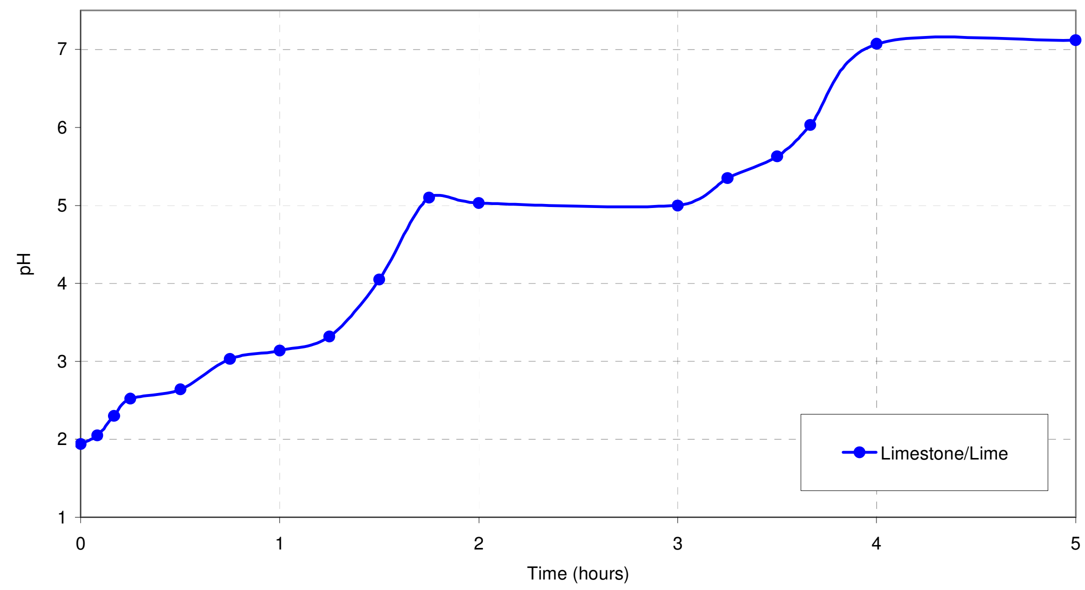

Figure 6 : Velardena Concentrate Biooxidation Liquor Neutralisation – pH Profile

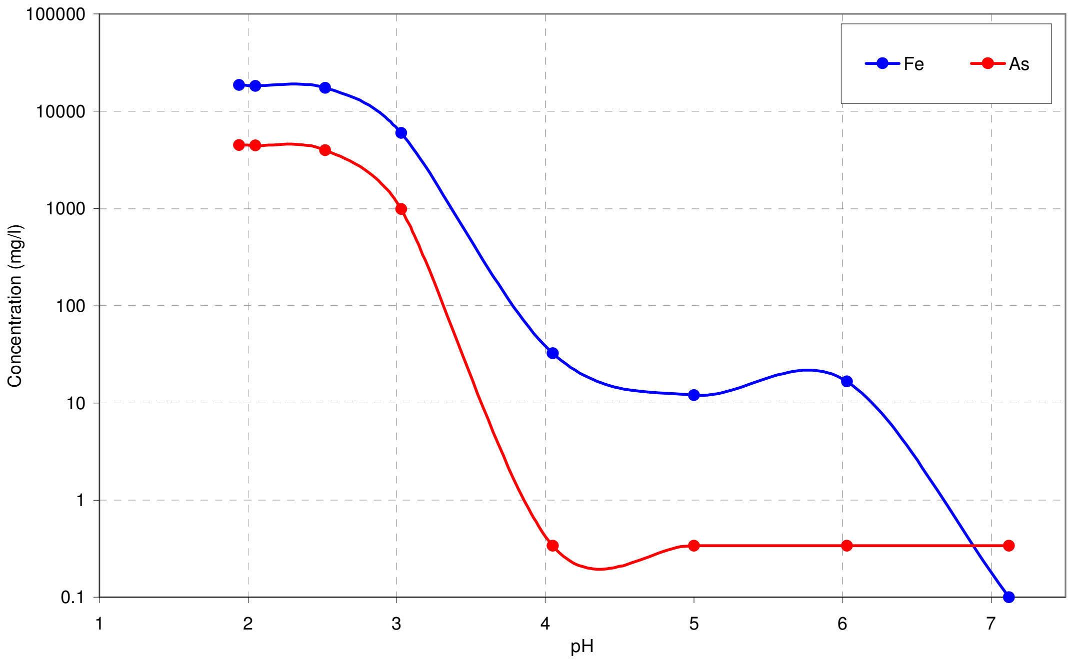

Figure 7 : Velardena Concentrate Biooxidation Liquor Neutralisation – Fe and As Profile

3.5 Settling Test Results

The results of the settling rate tests conducted on the 40 day BIOX® slurry are given in Table 9 and graphically presented in Figure 8. The detailed settling test results for the selection tests and the dosage optimisation tests are given in Appendix IX.

The specific thickener areas were calculated using the method proposed by Coe and Clevenger. The recommended design thickener areas specified in the tables are twice the calculated specific thickener area calculated. This "safety" factor is applied to allow for variation in feedrate and settling rates in a commercial plant.

Table 9 : Settling Test Results

Test Slurry	Flocculant Dosage (g/t solids)	Net Area Required (m^2/h/t)	Recommended Area (m^2/h/t)
BIOX® slurry	0	9.2	18.3
	50	2.5	5.1
	100	1.6	3.1
	150	1.2	2.4
	200	1.0	2.1

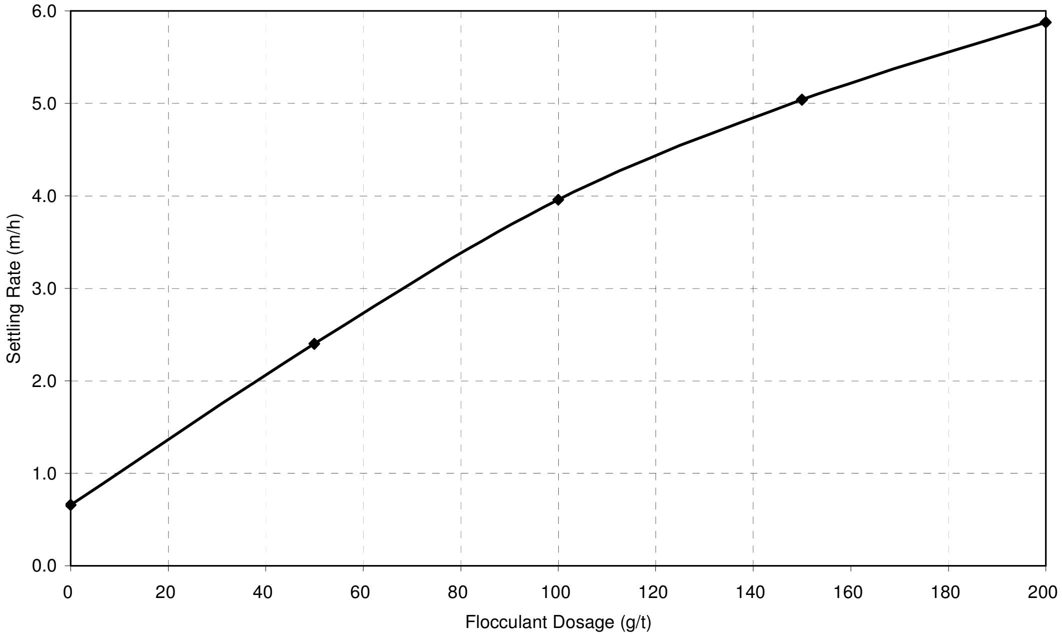

Figure 8 : Velardena BIOX® Product Slurry Settling Test

4. CONCLUSIONS AND RECOMMENDATIONS

The Velardena concentrate is amenable to biooxidation treatment. This was confirmed by the following:

- The direct cyanidation leach test carried out on the Velardena concentrate yielded a gold dissolution of 3.9%.
- The maximum sulphide oxidation achieved during the batch biooxidation amenability tests of the Velardena concentrate was 96.8% after 36 days biooxidation. The corresponding gold dissolution was 91.0%. A gold dissolution of 91.5% was achieved after 25 days biooxidation, corresponding to 83.4% sulphide oxidation. The results indicate that complete sulphide sulphur oxidation may not be necessary to achieve maximum gold dissolution.
- The Velardena concentrate was net acid generating (310.4 kg H_2SO_4/t of concentrate).
- A mass increase was observed during the biooxidation of the Velardena concentrate.
- The NaCN consumptions during cyanidation of the biooxidation products varied between 5.0 and 12.2 kg NaCN/t concentrate feed.
- The TCLP testing of the neutralisation precipitates produced extracts containing 0.6 ppm As, below the 5 ppm limit set by the EPA. The precipitates can be considered stable and thus acceptable for disposal on a tailings dam.

T Stanek	M Mothobi
MANAGER : BIOMET& FUSION	**ACTING LABORATORY SUPERVISOR**

APPENDIX I

NUTRIENT MEDIUM

0K NUTRIENT MEDIUM

Chemical	Amount (g/l)
$(NH_4)_2SO_4$	3
K_2HPO_4	0.5
$MgSO_4.7H_2O$	0.5
KCl	0.1
$Ca(NO_3)_2$	0.01

9K NUTRIENT MEDIUM

Chemical	Amount (g/l)
$(NH_4)_2SO_4$	3
K_2HPO_4	0.5
$MgSO_4.7H_2O$	0.5
KCl	0.1
$Ca(NO_3)_2$	0.01
$FeSO_4.7H_2O$	50

APPENDIX II

EQUATIONS USED FOR BIOX® CALCULATIONS

1. MASS CHANGE DURING BIOX®

$$Mass\ change\ (\%) = \frac{a - (b + c)}{(b + c)} \times 100$$

where a = mass of BIOX® product
b = initial mass of feed added
c = estimated amount of solid added with the inoculum

2. ACID ADDITION TO STABILISE BIOX® SLURRIES

$$Acid\ addition\ (kg / ton\ feed) = \frac{a \times b}{c}$$

where a = concentration of acid solution used (g/l)
b = total volume of acid added (l)
c = mass feed added to BIOX® (kg)

3. LIME CONSUMPTION DURING BIOX®

$$Lime\ consumption\ (kg / ton\ feed) = \frac{a \times b}{c}$$

where a = total volume of lime added during BIOX® (l)
b = available CaO concentration in lime (g/l)
c = mass of feed added to BIOX® (kg)

4. NET AMOUNT OF ACID GENERATED/CONSUMED DURING BIOX®

$$Net\ acid\ generation / consumption =$$

$$Acid\ addition\ (kg / ton) - \left(Lime\ consumption\ (kg / ton) \times \frac{molar\ mass\ acid}{molar\ mass\ lime} \right)$$

A negative value indicates a net acid generating ore/concentrate
A positive value indicates a net acid consuming ore/concentrate

5. **SULPHIDE REMOVAL**

$$Sulphide\ \mathrm{Re}moval\ (\%) = \frac{[(a_1 \ x \ b_1) + (a_2 \ x \ b_2)] - (a_3 \ x \ b_3)}{[(a_1 \ x \ b_1) + (a_2 \ x \ b_2)]}$$

where a_1 = mass feed added to BIOX® batch (g)
b_1 = S (pyr) content of pre-BIOX® sample (%)
a_2 = mass feed added with inoculum (g)
b_2 = estimated S (pyr) content of solids in inoculum (%)
a_3 = mass BIOX® product (g)
b_3 = S (pyr) content of post-BIOX® sample (%)

Arsenic solubilisation values are calculated using the same formula.

APPENDIX III

EQUATIONS USED FOR CYANIDATION CALCULATIONS

1. CYANIDE ADDITION

$$Cyanide\ addition\ (kg\ /\ t\ BIOX\ product)\ =\ \frac{a}{b}\ x\ 1000$$

where a = mass NaCN added (g)
b = mass BIOX® product used for cyanidation (g)

Cyanide addition values are corrected for mass changes during BIOX® using the following equation:

$$Corrected\ cyanide\ addition\ (kg\ /\ t\ feed)\ =\ cyanide\ addition\ (kg\ /\ t\ BIOX\ product)\ x\ F1$$

where F1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) ÷ 100
a = mass change during BIOX® (%)

2. LIME ADDITION

$$Lime\ addition\ (kg\ /\ t\ BIOX\ product)\ =\ \frac{a}{b}\ x\ 1000$$

where a = mass lime added (g)
b = mass BIOX® product used for cyanidation (g)

Lime addition values are corrected for mass changes during BIOX® using the following equation:

$$Corrected\ lime\ addition\ (kg\ /\ t\ feed)\ =\ Lime\ addition\ (kg\ /\ t\ BIOX\ product)\ x\ F1$$

where F1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) ÷ 100
a = mass change during BIOX® (%)

3. CYANIDE CONSUMPTION

$$Cyanide\ consumption\ (kg\ /\ t\ of\ BIOX\ product)\ =\ \frac{(a - b)}{c}\ x\ 1000$$

where a = mass NaCN added (g)
b = mass NaCN not consumed (g)
c = mass of BIOX® product used for cyanidation (g)

The residual cyanide concentration is determined by titration of the cyanidation liquor with silver nitrate.

b is calculated as the product of the residual cyanide concentration (g/l) and the end volume of the filtrate.

Cyanide consumption values are corrected for mass changes during BIOX® using the following equation:

$$\textit{Corrected cyanide consumption (kg / t feed)}$$

$$=$$

$$\textit{cyanide consumption (kg / t BIOX product) x } F_1$$

where F_1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) ÷ 100
a = mass change during BIOX® (%)

4. **LIME CONSUMPTION**

$$\textit{Lime consumption (kg / t BIOX product)} = \frac{(a - b)}{c} \times 1000$$

where a = mass lime added (g)
b = mass lime not consumed (g)
c = mass of BIOX® product used for cyanidation (g)

The residual lime (CaO) concentration is determined by titration of the cyanidation liquor with oxalic acid.

b is calculated as the product of the residual lime concentration (g/l) and the end volume of the filtrate.

Lime consumption values are corrected for mass changes during BIOX® using the following equation:

$$\textit{Corrected lime consumption (kg / t feed)}$$

$$=$$

$$\textit{lime consumption (kg / t BIOX product) x } F_1$$

where F_1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) ÷ 100
a = mass change during BIOX® (%)

5. GOLD DISSOLUTION AFTER CYANIDATION

$$Gold\ dissolution\ (\%) = \left(1 - \frac{a}{b}\right) x\ 100$$

where a = Au_{tail}(g/t) x $mass_{tail}$(g)
b = Au_{head}(g/t) x $mass_{head}$(g)

6. CYANIDATION RESIDUE GOLD CONTENT AFTER CORRECTION FOR MASS CHANGES DURING BIOX® AND CYANIDATION

The following calculation method is used to correct for mass changes during BIOX® and during cyanidation, such that the corrected gold value is comparable directly to the sample head assay.

$$Corrected\ Au_{tail}\ (g/t) = Assay\ Au_{tail}\ (g/t)\ x\ F_1\ x\ F_2$$

where F_1 = relative mass ratio BIOX® product to BIOX® feed
= (100 + a) ÷ 100
where a = mass change during BIOX® (%)
F_2 = relative mass ratio cyanidation product to BIOX® product
= (100 + b) ÷ 100
b = mass change during cyanidation (%)

APPENDIX IV

ADAPTATION AND INOCULUM BUILD UP LOGSHEETS

APPENDIX V

BATCH AMENABILITY TEST LOGSHEETS

APPENDIX VI

AMENABILITY TEST FEED AND FINAL PRODUCT WEIGHTS AND VOLUMES (Incl. Gold Balance)

APPENDIX VII

CYANIDATION TEST LOGSHEETS

APPENDIX VIII

NEUTRALISATION TEST LOGSHEETS

APPENDIX IX

SETTLING TEST LOGSHEETS